Hot Chili Limited

Costa Fuego Project, Chile

Preliminary Feasibility Study NI 43-101 Technical Report

March 2025

Authors:

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Effective Date: March 27 2025
Report Date: May 9 2025

wood.

Disclaimer

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This disclaimer must accompany every copy of this Report, which is an integral document and must be read in its entirety.

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QP Certificates

Certificate of Qualified Person

I, Elizabeth Haren, B.Sc, FAusIMM(CPGeo), MAIG, as co-author of the NI 43-101 Technical Report titled "Costa Fuego Project, NI 43-101 Technical Report Preliminary Feasibility Study" effective date 27th March 2025 (the "Report") do hereby certify that:

1. I am a full-time employee of Haren Consulting with an office at PO Box 1159, Scarborough, Western Australia, 6922, tel. +61 (0) 408 958 259, email elizabeth@haren.com.au.
2. I graduated from the University of Newcastle, Newcastle, Australia with a Bachelor of Science (Geology) Degree, and I have continually practiced my profession since 1996 as a geologist. I have worked in the mining industry for 29 years with experience in underground geology, open pit geology and geostatistics.
3. I am a fellow of The Australasian Institute of Mining and Metallurgy and a Chartered Professional, membership number 208050. I am a member of the Australian Institute of Geoscientists, membership number 6646.
4. I have personally inspected the Costa Fuego project during May 2022.
5. I have read the definition of Qualified Person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfill the requirements to be a Qualified Person for the purposes of National Instrument 43-101 and this technical report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1.
6. I am a "Qualified Person" as defined in National Instrument 43-101 (Standards of Disclosure for Mineral Projects) (the "Instrument").
7. As a Qualified Person, I am independent of the issuer as defined in Section 1.5 of National Instrument 43-101.
8. I am a co-author of this report and responsible for sections 1.2, 1.3, 1.4, 1.5, 1.6, 1.7, 1.8, 1.9, 1.10,1.12, 3.2, 3.3, 3.4, 6, 7, 8, 9, 10, 11, 12.1, 12.4, 14, 24.11, 25.2, 25.3, and 26.3 and accept professional responsibility for those sections of this technical report.
9. I have had prior involvement with the subject property. Since August 2020 I have authored private reports and technical memorandums for Hot Chili Limited.
10. I have read National Instrument 43-101 and confirm that this technical report has been prepared in compliance therewith.
11. I was retained by Hot Chili Limited to prepare the Mineral Resource estimates for the Costa Fuego deposits using CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines and Canadian Securities Administrators National Instrument 43-101 guidelines. The preceding report is based on a review of project files and discussions with Hot Chili personnel.
12. I have not received, nor do I expect to receive, any interest, directly or indirectly, in Hot Chili Limited. I do not beneficially own, directly or indirectly, any securities of Hot Chili Limited or any associate or affiliate of such company.
13. As of the date of this certificate, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated this day 09 May 2025 at Scarborough, Australia.

"Signed and Sealed"

Elizabeth Haren FAusIMM CPGeo, MAIG

Director, Haren Consulting

Certificate of Qualified Person

I, Anton von Wielligh, B.Eng, FAusIMM, as co-author of the NI 43-101 Technical Report titled "Costa Fuego Project, NI 43-101 Technical Report Preliminary Feasibility Study" effective date 27th March 2025 (the "Report") do hereby certify that:

1. I am a consultant engaged by Hot Chili Limited with an office at 33 Chelydra Point, North Coogee, Western Australia, 6163, tel. +61 424 671 380, email anton@abgm.com.au.
2. I graduated from the University of Pretoria, South Africa with a Bachelor of Engineering (Mining) in 2001 and an Engineering honors (Hons) Degree in 2004.
3. I am a Fellow of The Australasian Institute of Mining and Metallurgy, membership number 325251.
4. I have worked as a mining engineer for 22 years since my graduation from university (bachelor's degree in engineering) and completed my honors degree part-time whilst working.
5. I have read the definition of Qualified Person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfill the requirements to be a Qualified Person for the purposes of National Instrument 43-101.
6. I am a co-author of this report and responsible for sections 1.13,1.14,1.15,1.16,1.17, 1.18, 12.3, 15, 16 (except 16.3.4 & 16.4.2), 21.5, 21.15, 23, 24.7, 25.4, 26.4, and 26.5, and accept professional responsibility for those sections of this technical report.
7. I have had prior involvement with the subject property. Since August 2020 I have authored private reports and technical memos for Hot Chili Limited.
8. As a Qualified Person, I am independent of the issuer as defined in Section 1.5 of National Instrument 43-101.
9. I have read National Instrument 43-101 and confirm that this technical report has been prepared in compliance therewith.
10. As of the date of this certificate, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated this day 09 May 2025, Perth, Australia.

"Signed and Sealed"

Anton von Wielligh FAusIMM

Director/Principal Mining Engineer

ABGM PTY LTD

Certificate of Qualified Person

I, David Cuello, BSc, BEng (Mining Engineering), MSc (Geomechanics), as a contributor to the announcement titled Costa Fuego Preliminary Feasibility Study (PFS) March 2025 (the "Announcement"), do hereby certify that:

1. I am a consultant engaged by Hot Chili, with an office at Suite 8, 44 Lakeview Dr, SCORESBY VIC 3179; email: dcuello@gmintec.com.
2. I graduated from Universidad de Atacama with a Bachelor of Science and a Bachelor of Mining Engineering, followed by a Master of Engineering Science (Mining Geomechanics) from Curtin University (Western Australian School of Mines).
3. I am a Fellow Member of the Australasian Institute of Mining and Metallurgy (FAusIMM) (Membership No. 312706).
4. I have over 23 years of experience in mining geotechnical engineering. I have worked extensively in geotechnical engineering for underground and open-pit mining operations. My experience spans both operational roles and multiple feasibility studies. I have participated in geotechnical reviews and due diligence assessments for mining projects and studies worldwide. I have personally inspected the Costa Fuego project.
5. I have read the definition of Qualified Person (QP) as set out in National Instrument 43-101 and certify that, by virtue of my education, professional affiliations, and relevant work experience, I fulfill the requirements to be a Qualified Person for the purposes of National Instrument 43-101 (Standards of Disclosure for Mineral Projects).
6. I am a co-author of this report and responsible for sections 12.6, 16.3.4, 16.4.2, and 26.4. I accept professional responsibility for those sections of this technical report.
7. I have not had prior involvement with the subject property.
8. I am a "Qualified Person" as defined in National Instrument 43-101 (Standards of Disclosure for Mineral Projects) (the "Instrument").
9. As a Qualified Person, I am independent of the issuer as defined in Section 1.5 of National Instrument 43-101.
10. I have not received, nor do I expect to receive, any interest, directly or indirectly, in Hot Chili Limited. I do not beneficially own, directly or indirectly, any securities of Hot Chili Limited or any associate or affiliate of such company.
11. As of the date of this certificate, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated this day 09 May 2025, Melbourne, Australia

"Signed and Sealed"

David Cuello

Principal Geotechnical Engineer - GMT Servicios de ingeniería Ltd.

Certificate of Qualified Person

I, Dean David FAusIMM CP (Met), as co-author of the NI 43-101 Technical Report titled "Costa Fuego Project, NI 43-101 Technical Report Preliminary Feasibility Study" effective date 27th March 2025 (the "Report") do hereby certify that:

1. I am a Senior Consultant with Wood of Level 1, 240 St. Georges Terrace, Perth, Western Australia, which is engaged by Hot Chili Limited. my email is dean.david@woodplc.com
2. I graduated from the South Australian Institute of Technology in 1981 with a B.App.Sc in Metallurgy. I have practiced my profession for 44 years. I have been directly involved in processing flowsheet development and optimisation for studies and projects across multiple commodities for all 44 years. Many of the projects involving relevant aspects such as geometallurgy, comminution and flotation, especially flotation of copper sulphides. I have also been involved in the process and non-process infrastructure aspects of many of those projects.
3. I am a Fellow and Chartered Professional (Metallurgy) with the Australasian Institute of Mining and Metallurgy, membership number 102351.
4. I visited the Costa Fuego Project site from October 23 to 25, 2024. The visit included the site exploration office and core shed, including viewing some requested core. The Productora, Alice and Cortadera mine locations, the commencement point of the Rope Conveyor and, in the same location, the approximate Cortadera primary crusher position. The Productora primary crusher, Productora concentrator, and in the same location, the approximate common end point of the Rope Conveyor and Productora overland conveyor. The Productora heap leach, Productora dump leach and Productora tailings locations. The visit also included the Port of Huasco and the adjacent seawater intake location, the town of Vallenar and several localities between Vallenar and the port. The Panamerican Highway from La Serana to Vallenar was driven as part of the visit.
5. I have read the definition of Qualified Person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfill the requirements to be a Qualified Person for the purposes of National Instrument 43-101.
6. I am a co-author of this report and responsible for sections 1.11.1, 1.11.2, 1.19.1, 1.19.2, 1.24, 2.2, 2.3, 12.2.1, 12.2.2, 13.1, 13.2, 13.3, 13.5, 17.1, 17.2, 17.3, 17.4, 17.5, 17.7, 21.16, 24.9.3, 24.9.4, 24.9.5, 24.9.6, 25.5.1, 25.5.2, 25.5.3, 25.5.5, 26.6, and 27.1 of the Technical Report. I accept professional responsibility for those sections of this technical report.
7. I have been involved with the metallurgical aspects of the Costa Fuego project for five years. In this time I have overseen components of the metallurgical testwork, authored public and private reports, authored assurance reports and authored technical memos for Hot Chili Limited.
8. As a Qualified Person, I am independent of the issuer as defined in Section 1.5 of National Instrument 43-101.
9. I have read National Instrument 43-101 and confirm that this technical report has been prepared in compliance therewith.
10. I have not received, nor do I expect to receive, any interest, directly or indirectly, in Hot Chili Limited. I personally purchased and, for some years I have beneficially owned, a minor parcel of securities of Hot Chili Limited that forms a minor component of a personal superannuation (retirement benefit) fund.
11. As of the date of this certificate, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

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Dated this day 09 May 2025, Perth, Australia

"Signed and Sealed"

Dean David, FAusIMM CP (met)
Senior Consultant
Wood Minerals & Metals Australia

Certificate of Qualified Person

I, Luis Bernal, Civil Mining Engineer, as co-author of the NI 43-101 Technical Report titled "Costa Fuego Project, NI 43-101 Technical Report Preliminary Feasibility Study", effective date 28th March 2025 (the "Report") do hereby certify that:

1. I am a consultant engaged by Hot Chili Limited with an office at Suite 62, 2750 Ricardo Lyon Avenue, Ñuñoa Santiago of Chile; email: luis.bernal@processminerals.cl.
2. I graduated from the University of Chile, Engineering School, Mining Department, in Santiago, Chile, with a Bachelor of Science (Mining & Metallurgical Engineering) in 1978. I have worked in the mining industry for over 40 years in various processes, including iron ore, copper, gold, molybdenum, nickel, and platinum group metals. For over 15 years, I worked in iron and copper mining operations in the two largest companies in Chile. Later, as a process consultant, I developed numerous feasibility studies until their successful completion and participated in the stage of metallurgical tests, design, cost estimates, commissioning and peer review of several of these projects.
3. I am a Registered Qualified Person with the Chilean Mining Commission, membership number 0415.
4. I have personally inspected the Costa Fuego project site and participated in the supervision of metallurgical testing.
5. I have read the definition of Qualified Person in National Instrument 43-101 and certify that, by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfill the requirements to be a Qualified Person for the purposes of National Instrument 43-101.
6. I am a "Qualified Person" as defined in National Instrument 43-101 (Standards of Disclosure for Mineral Projects) (the "Instrument").
7. As a Qualified Person, I am independent of the issuer as defined in Section 1.5 of National Instrument 43-101.
8. I am a co-author of this report and responsible for sections 1.11.3, 1.19.3, 12.2.1.3, 13.4, 17.6, 25.5.4, 26.6.3, 26.6.4, and 27.1. I accept professional responsibility for those sections of this technical report.
9. Also, I participated in the PEA development before acting as a process senior consultant.
10. As a Qualified Person, I am independent of the issuer as defined in Section 1.5 of National Instrument 43-101.
11. I have read National Instrument 43-101 and confirm that this technical report has been prepared in compliance therewith.
12. I have not received, nor do I expect to receive, any interest, directly or indirectly, in Hot Chili Limited. I do not beneficially own, directly or indirectly, any securities of Hot Chili Limited or any associate or affiliate of such company.
13. As of the date of this certificate, to the best of my knowledge, information, and belief, this technical report contains all scientific and technical information that must be disclosed to ensure that the technical report is not misleading.

Dated this day, 09 May 2025, Santiago, Chile

"Signed and Sealed"

Luis Bernal (Mining & metallurgical engineer)

PMC-Process Minerals Consulting

Certificate of Qualified Person

I, David J T Morgan, MAusImm CPEng, as co-author of the NI 43-101 Technical Report titled "Costa Fuego Project, NI 43-101 Technical Report Preliminary Feasibility Study" effective date 27th March 2025 (the "Report") do hereby certify that:

1. I am employed as the Managing Director of Knight Piésold, located at Level 1, 184 Adelaide Terrace, East Perth 6004.
2. I graduated from the University of Manchester, (BSc, Civil Engineering, 1980), and the University of Southampton (MSc, Irrigation Engineering, 1981).
3. I am a registered member in good standing of the Australasian Institute of Mining and Metallurgy (Australasia, 202216) and a Chartered Professional Engineer.
4. I have practised my profession for 43 years. I have been directly involved in the design of the tailings storage facility for the Costa Fuego Copper Project.
5. I have read the definition of Qualified Person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfill the requirements to be a Qualified Person for the purposes of National Instrument 43-101.
6. I am a co-author of this report and responsible for sections 1.20.5, 1.212, 18.4, 24.5, 25.6.3, and 26.9 and accept professional responsibility for those sections of this technical report.
7. I have had previous involvement with the Costa Fuego Project and have visited the Costa Fuego site in October 2015 and October 2024.
8. I am independent of Hot Chili Ltd as independence is described by Section 1.5 of NI 43—101.
9. I have read National Instrument 43-101 and confirm that this technical report has been prepared in compliance therewith.
10. I have not received, nor do I expect to receive, any interest, directly or indirectly, in Hot Chili Limited. I do not beneficially own, directly or indirectly, any securities of Hot Chili Limited or any associate or affiliate of such company.
11. As of the effective date of the Report, to the best of my knowledge, information and belief, the sections of Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

Dated this day 09 May 2025, Perth, Australia.

"Signed and Sealed"

David Morgan, MAusImm, CPEng
Managing Director
Knight Piésold Pty Limited

Certificate of Qualified Person

I, Piers Wendlandt, P.E.), as co-author of the NI 43-101 Technical Report titled "Costa Fuego Project, NI 43-101 Technical Report Preliminary Feasibility Study" effective date 27th March 2025 (the "Report") do hereby certify that:

1. I am employed as Principal Mining Engineer with Wood USA Mining Consulting.
2. I graduated from the Colorado School of Mines, USA with a Bachelor of Mining Engineering in 2005 and a Master of Business Administration, University of Colorado, USA in 2019. I have worked in the mining industry for 20 years with experience in mining engineering, financial modelling, and economic analysis.
3. I am a Registered Professional Engineer in the State of Colorado, number PE.0047235.
4. I have not personally inspected the Costa Fuego project site.
5. I have read the definition of Qualified Person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfill the requirements to be a Qualified Person for the purposes of National Instrument 43-101.
6. I am a "Qualified Person" as defined in National Instrument 43-101 (Standards of Disclosure for Mineral Projects) (the "Instrument").
7. As a Qualified Person, I am independent of the issuer as defined in Section 1.5 of National Instrument 43-101.
8. I am a co-author of this report and responsible for sections 1.22, 1.25, 1.26, 1.27, 1.28, 1.29, 3.5, 3.6, 12.2.4, 12.2.6, 19, 22, 25.8, and 25.11. I accept professional responsibility for those sections of this technical report.
9. I have had prior involvement with the subject property. Since August 2020 I have authored private reports and technical memos for Hot Chili Limited.
10. As a Qualified Person, I am independent of the issuer as defined in Section 1.5 of National Instrument 43-101.
11. I have read National Instrument 43-101 and confirm that this technical report has been prepared in compliance therewith.
12. I have not received, nor do I expect to receive, any interest, directly or indirectly, in Hot Chili Limited. I do not beneficially own, directly or indirectly, any securities of Hot Chili Limited or any associate or affiliate of such company.
13. As of the date of this certificate, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated this day 09 May 2025, Denver, USA

"Signed and Sealed"

Piers Wendlandt, P.E.

Principal Mining Engineer

Wood

Certificate of Qualified Person

I, Jeffrey Peter Stevens, BSc (Chem Eng) Pr. Eng, as co-author of the NI 43-101 Technical Report titled "Costa Fuego Project, NI 43-101 Technical Report Preliminary Feasibility Study" effective date 27th March 2025 (the "Report") do hereby certify that:

1. I am a consultant engaged by Hot Chili Limited with an office at Level 1, 240 St. Georges Terrace, Perth, Western Australia; email: jeffrey.stevens@woodplc.com.
2. I graduated from the University of the Witwatersrand, Johannesburg, South Africa with a Bachelor of Science (Chemical Engineering) in 1989. I have worked in the mining industry for over 30 years in a variety of processes including gold, uranium, base metals, ferrochrome and platinum group metals. Management and successful delivery of feasibility studies through to completion of execution. Experience has included both the implementation and peer review of a significant number of feasibility studies including the compilation, coordination of compilation and review of capital and operating cost estimates.
3. I am a Registered Professional Engineer with the Engineering Council of South Africa, membership number 920272.
4. I have personally inspected the Costa Fuego project.
5. I have read the definition of Qualified Person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfill the requirements to be a Qualified Person for the purposes of National Instrument 43-101.
6. I am a "Qualified Person" as defined in National Instrument 43-101 (Standards of Disclosure for Mineral Projects) (the "Instrument").
7. As a Qualified Person, I am independent of the issuer as defined in Section 1.5 of National Instrument 43-101.
8. I am a co-author of this report and responsible for sections 1.1, 1.20, 1.20.1, 1.20.2, 1.20.3, 1.20.4, 1.23, 1.27, 1.28, 1.29, 2.0, 2.1, 2.2, 2.3, 2.4, 2.5, 2.6, 2.2, 2.3, 2.4, 2.5, 2.6, 3.1, 5, 12.2.3, 12.2.5, 18 (except 18.4), 21 (except 21.5, 21.15, 21.16), 24.1, 24.2, 24.3, 24.4, 24.8, 25.1, 25.6, 25.9,, 25.12, 25.13, 26.1, 26.7, 26.8, 26.10, 26.11, 26.13, 27. I accept professional responsibility for those sections of this technical report.
9. I have had prior involvement with the subject property, being co-author of the report titled "Costa Fuego Copper Project, NI 43-101 Technical Report Preliminary Economic Analysis", Chile, effective date 28th June 2023
10. I have read National Instrument 43-101 and confirm that this technical report has been prepared in compliance therewith.
11. I have not received, nor do I expect to receive, any interest, directly or indirectly, in Hot Chili Limited. I do not beneficially own, directly or indirectly, any securities of Hot Chili Limited or any associate or affiliate of such company.
12. As of the date of this certificate, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated this day 09 May 2025, Perth, Australia

"Signed and Sealed"

Jeffrey Stevens BSc (Chem Eng), Pr. Eng
Senior Study Manager
Wood Minerals & Metals Australia

Certificate of Qualified Person

I, Edmundo J. Laporte, P.E., P.Eng., C.P.Eng., as co-author of the NI 43-101 Technical Report titled "Costa Fuego Project, NI 43-101 Technical Report Preliminary Feasibility Study" effective date 27th March 2025 (the "Report") do hereby certify that:

1. I am the CEO and Principal Consultant with High River Services, LLC, 4440 Walnut Creek Drive, Lexington, KY, 40509 and we are acting as consultants for Hot Chili Limited.
2. This certificate applies to the NI 43-101 Technical Report titled "Costa Fuego Preliminary Feasibility Study (PFS) March 2025" with an Effective Date of 27 March, 2025".
3. I graduated with a Bachelor of Science degree in Civil Engineering from the University of Rafael Urdaneta in Venezuela, in 1987. I am a Registered Member of the Society for Mining, Metallurgy & Exploration Inc. (SME): RM-SME #4150038. I am a Professional Engineer and member in good standing of Engineers Ontario (Member #100508702) as well as in multiple jurisdictions in the United States, and a Chartered Professional Engineer in Australia (Registration #3802017). I have worked as an Engineer for a total of 37 years since my graduation from university.
4. My relevant experience includes exploration, mine planning, mine design, rock mechanics, ground control, environmental analysis and economic modeling for mining projects in the Americas, Europe, Asia and Africa. I have participated in the preparation of numerous NI 43-101 and JORC technical reports, as well as private internal reports for mining projects in North America and abroad.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. I am responsible for the preparation of Sections 1.5.5, 1.21, 4, 12.5, 20, 25.7 and 26.12 of the technical report titled "Costa Fuego Preliminary Feasibility Study March 2025" (the "Technical Report") relating to the Costa Fuego Project.
7. I visited the Costa Fuego Project property between 12 and 17 May, 2024.
8. I participated as Environmental QP during the preparation of the Preliminary Economic Assessment of the Costa Fuego Project.
9. As of the Effective Date, to the best of my knowledge, information and belief, the part of the Technical Report that I am responsible for contains all scientific and technical information that is required to be declared to make the Announcement and eventual Technical Report not misleading.
10. I am independent of the issuer Hot Chili Limited (HCH), its subsidiary Sociedad Minera La Frontera (SMECL) and the property (Costa Fuego Project) applying all of the tests in section 1.5 of National Instrument 43-101.
11. I have read National Instrument 43-101 and confirm that this announcement reflects the eventual technical report has been prepared in compliance therewith.

Dated this day 09 May 2025, Kentucky, USA

"Signed and Sealed"

Edmundo J. Laporte, P.E., P.Eng.
CEO - Principal Consultant

1 Summary

1.1 Introduction

This Report presents the findings of a Prefeasibility Study completed on the Costa Fuego Copper Project ("Costa Fuego", "Costa Fuego Project" or "Project"), located near Vallenar in Chile, South America.

At the request of Hot Chili Limited ("HCH", "Hot Chili Limited" or "Company"), Wood Australia Pty Ltd. ("Wood") has prepared this Report for the Costa Fuego Project, as announced by HCH on 27 March 2025.

The Report was prepared in accordance with the Canadian disclosure requirements of National Instrument 43-101 – *Standards of Disclosure for Mineral Projects* ("NI 43-101") and in accordance with the requirements of Form 43-101F1 – *Technical Report* ("Form 43-101F1").

HCH is a Perth-based copper-gold exploration and development company that undertakes exploration and development of its various copper-gold projects located in Chile's Atacama Region. Employees of the following engineering and geological consultancies contributed to the Report:

Wood Australia Pty Ltd ("Wood")

With 35,000 professionals, across 60 countries, Wood is one of the world's leading consulting and engineering companies operating across Energy and Materials markets.

Haren Consulting

Haren Consulting provide specialist resource geology services for the mining industry including technical mentoring, resource estimation, training, reconciliation, conditional simulation, and QA/QC analysis.

ABGM Pty Ltd. ("ABGM")

ABGM is a niche mining consultancy delivering world class mine technical services to a global client base. ABGM's services are multi-disciplinary and cover precious and base metals, industrial minerals, diamonds, coal and potash for open pit or underground mining methods.

Knight Piésold Pty Ltd. ("Knight Piésold")

Knight Piésold is an employee-owned, global consulting firm providing specialist services to the mining, power, water resources, and infrastructure industries. Knight Piésold has a 1,000-strong team operating from 29 offices across 16 countries.

Geomechanics, Mining and Technology Engineering Services ("GMT")

GMT is a geotechnical/geomechanics engineering consulting firm, which has been providing a range of geotechnical engineering services to the civil and mining industries since 2013. GMT's staff of engineers and senior associated professionals have experience in both open-pit and underground mining.

High River Services ("HRS")

HRS is a social, environmental and engineering consultancy specialising in mining, minerals, and energy, based in the United States. HRS consultants have delivered projects across North and South America, Europe, Asia and Africa.

Process Mineral Consulting ("PMC")

PMC is a Chilean Engineering company that provides mining consulting services worldwide with experience in both large-scale and mid-tier operations. PMC specialises in the development and optimisation of mining operations, processing, water and tailings management.

Doppelmayr Group ("Doppelmayr")

Doppelmayr is a world market leader for rope-propelled mobility. With innovative transport systems, they continue to set standards in the mobility sector. More than 3,000 employees in 50 countries around the world are part of Doppelmayr.

Gestion Ambiental Consultores SA ("GAC")

GAC has more than 30 years of experience as environmental management consultants in the Chilean market. GAC has vast experience in the development of projects associated with the energy, mining, industrial, forestry, agricultural, real estate and auditing sectors, both for the public sector and for private companies.

Piteau and Associates

Piteau and Associates is a global engineering consultancy providing geotechnical, water management, and environmental consulting services to the mining, construction, municipal, first nations, and industrial sectors.

Nova Mineralis

Nova Mineralis develops technological solutions for the mining industry for the recovery of copper and other base metals from primary sulphide ores.

There has been no material change to the Project between the effective date of this Report and the signature date.

Table 1.1 summarises the qualified persons ("Qualified Person" or "QP") responsible for the contents of this Report as that term is defined in NI 43-101, and for preparing this Report in compliance with Form 43-101F1:

Table 1.1 : List of QPs

Qualified Person	Discipline	Company	Section(s)
Elizabeth Haren	Geology	Haren Consulting	1.2, 1.3, 1.4, 1.5, 1.6, 1.7, 1.8, 1.9,1.10,1.12, 3.2, 3.3, 3.4, 6, 7, 8, 9, 10, 11, 12.1, 12.4, 14, 24.11, 25.2, 25.3, 26.3
Anton von Wielligh	Mining	ABGM	1.13,1.14,1.15,1.16,1.17, 1.18, 12.3, 15, 16 (except 16.3.4 & 16.4.2), 21.3.1, 21.5, 21.15, 23, 24.7, 25.4, 26.4, 26.5
David Cuello	Geotechnical	GMT	12.6, 16.3.4, 16.4.2, 26.4
Dean David	Metallurgy – Sulphide	Wood	1.11.1, 1.11.2,1.19.1, 1.19.2, 1.24, 2.2, 2.3, 12.2.1, 12.2.2, 13.1, 13.2, 13.3, 13.5, 17.1, 17.2, 17.3, 17.4, 17.5, 17.7, 21.16, 24.9.3, 24.9.4, 24.9.5, 24.9.6, 25.5.1, 25.5.2, 25.5.3, 25.5.5, 26.6, 27.1
Luis Bernal	Metallurgy – leaching	PMC	1.11.3, 1.19.3, 12.2.1.3, 13.4, 17.6, 25.5.4, 26.6.3, 26.6.4, 27.1
David Morgan	Tailings	Knight Piésold	1.20.5, 1.212, 18.4, 24.5, 25.6.3, 26.9
Piers Wendlandt	Economics	Wood Group USA, Inc.	1.22,1.25,1.26, 1.27, 1.28, 1.29, 3.5, 3.6, 12.2.4, 12.2.6, 19, 22, 25.8, 25.11
Jeffrey Stevens	Infrastructure, Capital Cost Estimates	Wood	1.1, 1.20, 1.20.1, 1.20.2, 1.20.3, 1.20.4, 1.23, 1.27, 1.28, 1.29, 2.0, 2.1, 2.2, 2.3, 2.4, 2.5, 2.6, 2.2, 2.3, 2.4, 2.5, 2.6, 3.1, 5, 12.2.3, 12.2.5, 18 (excluding 18.4), 21 (excluding 21.5, 21.15, 21.16), 24.1, 24.2, 24.3, 24.4, 24.8, 25.1, 25.6, 25.9,, 25.12, 25.13, 26.1, 26.7, 26.8, 26.10, 26.11, 26.13, 27
Edmundo J Laporte	Social and Environmental	HRS	1.5.5, 1.21, 4, 12.5, 20, 25.7, 26.12

1.2 Terms of Reference

This Report has been prepared to describe the proposed methods for exploiting the Cortadera, Productora, San Antonio, and Alice deposits. This Report is based on the mineral resource estimate ("Mineral Resource Estimate" or "MRE") completed in February 2024, and on subsequent technical studies completed during 2024 and 2025, focusing on geotechnical, metallurgy, mining, process plant/infrastructure design, and cost estimation.

Mineral Resource Estimates are reported in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves (May 2014; the 2014 CIM Definition Standards) and were estimated in accordance with the CIM 2019 Best Practices Guidelines, as required by NI 43-101.

Units used in this Report are metric unless otherwise noted. Unless otherwise noted, all dollar figures used are United States of America (US) dollars (US$). The Chilean currency is the Chilean peso (CLP).

Years discussed in the mine and production plan and in the economic analysis are presented for illustrative purposes only, as no decision has been made on mine construction by HCH.

1.3 Project Setting

The Costa Fuego Project is located 17 km south of the regional township of Vallenar, which is the capital of the Huasco Province (population approximately 52,000), approximately 600 km north of Santiago and 160 km north of the coastal city of La Serena, in the low-altitude, coastal range of the Atacama region of Chile.

The Project comprises four Mineral Resources situated within a ~20 km radius: Productora, Alice, Cortadera and San Antonio. The mineral resources are located along the Pan-American Highway with an average elevation of 740 m above sea level and near existing infrastructure of the Huasco valley and the nearby Las Losas port facilities (~60 km distance).

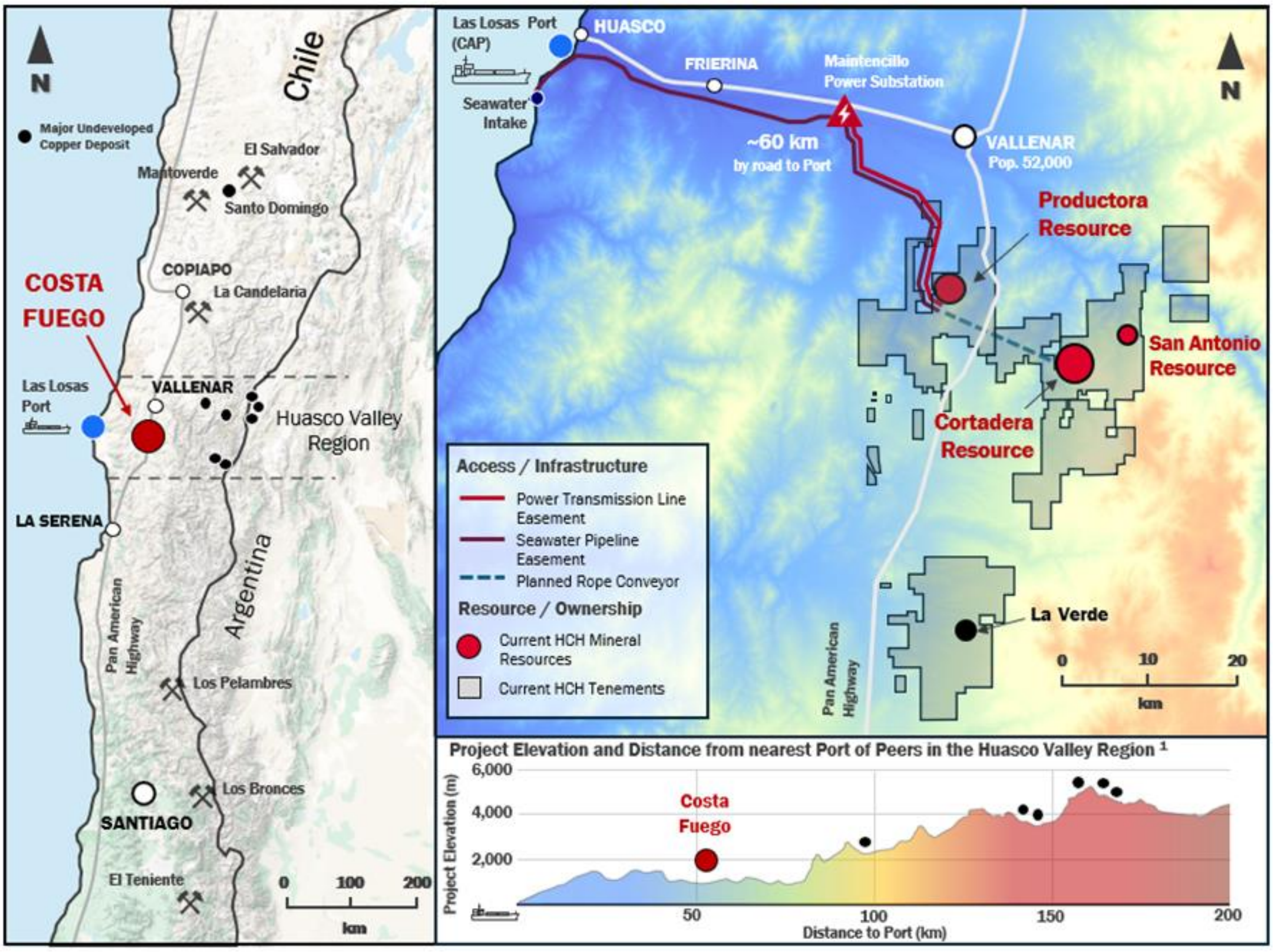
This Report contemplates conventional open pit truck and shovel operations and an underground block cave mine, feeding a conventional copper sulphide mineral process plant and an oxide heap leach and solvent extraction and electro-winning (SX/EW) plant to produce copper and molybdenum concentrates and copper cathode.

The Project will leverage existing surface rights for the proposed central processing facilities and associated infrastructure at Productora, and existing infrastructure access for powerline and sea water pipeline easements.

The Project has a unique location, (Figure 1.1) surrounded by existing infrastructure with the Project centre at Productora, located just 15 minutes by car from Vallenar on the Pan-American Highway.

An airport is located approximately 14 km from Productora, and the Las Losas Port facility and Maitencillo power substation are located 55 km and 20 km away, respectively.

Figure 1.1 : Costa Fuego Copper Project Location in Chile, HCH (2025)



1.3.1 Productora

The Productora deposit is 100% owned by a Chilean incorporated company Sociedad Minera El Aguila SpA (SMEA). SMEA is a joint venture company – 80% owned by Sociedad Minera El Corazón Limitada (a 100% subsidiary of HCH), and 20% owned by CMP Productora (a 100% subsidiary of Compañía Minera del Pacífico S.A (CMP)).

In August 2015, a joint venture (JV) agreement and merger was established between SMEA and one of CMP´s wholly owned vehicles that resulted in the JV company, SMEA. This partnership has enabled security of the majority of surface rights required for developing key infrastructure for the Project, as well as the majority of easements required for water and power transmission lines.

In addition, the JV agreement has consolidated the mining rights required for the development of Productora.

The only economic commitment to keep the Project in good standing are mining patents, which can be summarised as a mining tax paid to the government on a yearly basis (March each year). Total mining patent costs for 2023 were around US$295,000.

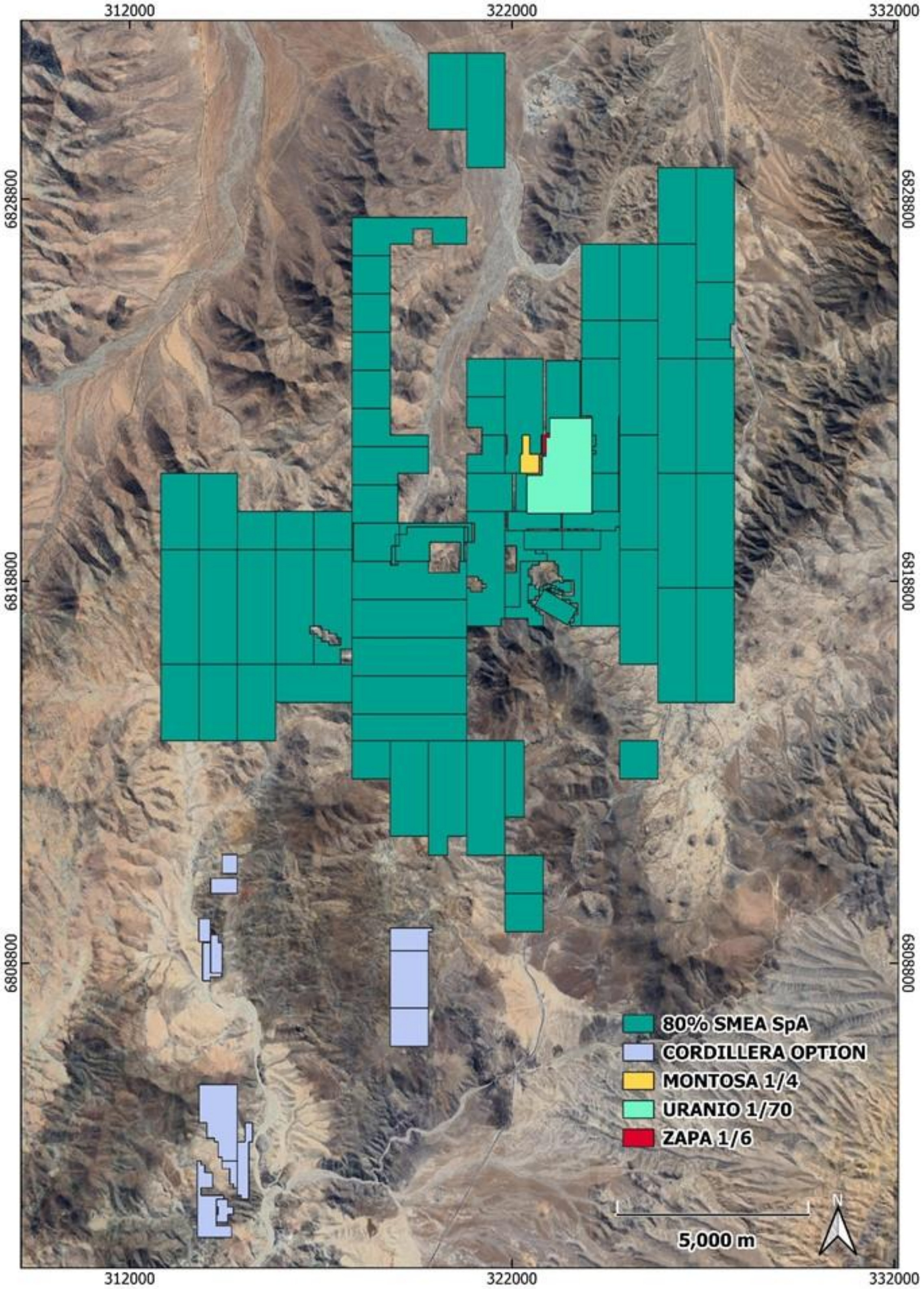
A 30-year lease agreement exists for the mining right Uranio 1-70 between the Chilean Commission of Nuclear Energy (CCHEN) and SMEA, dated 22 August 2012. This agreement incurs an annual lease payment of US$250,000 per year (paid no later than 31 August each year) and expires on 22 August 2042.

All mining rights covering the mining and infrastructure footprint at Productora are exploitation concessions with no risk of expiring if the mining taxes are duly paid annually.

Surface rights are 100% owned by the Company, as are the maritime concession to extract sea water from the coast (in process to be transferred to Huasco Water) and the corridor of easements to construct a pipeline and electrical transmission line to Productora.

A map of HCH mining rights under WGS84 is shown below (Figure 1.2).

Figure 1.2 : Productora Mining Rights Under WGS84, HCH (2025)



Productora has the following annual royalties:

- On the CCHEN mining right "Uranio 1 al 70": Net Smelter Return (NSR) for non-gold = 2%; Gold = 4% and non-metallic = 5%
- On the mining right "Montosa 1 al 4": NSR = 3% all products
- On the mining right "Zapa 1 al 6": Gross Royalty = 1% all products.

1.3.2 Cortadera

HCH, through its 100% subsidiary company Sociedad Minera Frontera SpA (Frontera), controls an area measuring approximately 25,500 ha at Cortadera through various 100% purchase option agreements with private mining title holders and 100% owned tenure.

All mining tenements are in good standing and all mining requirements have been met for the exploration phase. At this stage, there are no legal requirements for any kind of bonds to be issued.

The Cortadera Mineral Resource Estimate is contained within two Mining Rights:

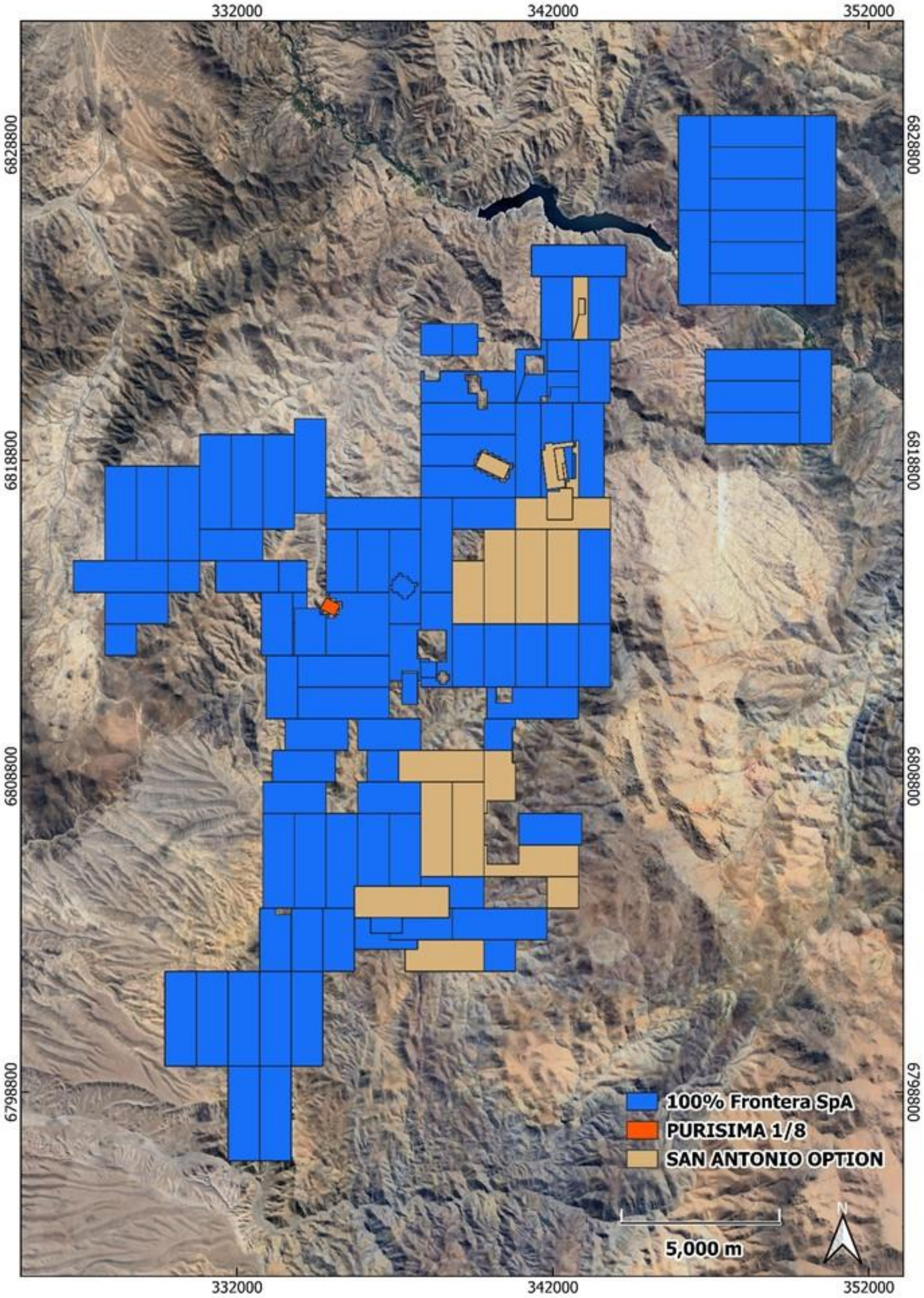
- 'CORTADERA 1/40' (374 ha). Mining tax (or cost per year to keep the mining right) US$2,673. Such mining right 1/40 is owned 100% by Frontera (wholly owned by HCH).
- 'Purísima 1/8 (1/2-5/6)'. (20 ha). Mining tax (or cost per year to keep the mining right) US$142. Such mining right is owned 100% by Frontera (wholly owned by HCH) with a 1.5% NSR attached.

The current exploration activities for Cortadera have been approved under the Environmental Approval number 48, dated 24 March 2021, granted by the Environmental Assessment Service.

Surface land access for exploration activities (and for the future mining operations) has also been reached with the owner, Mr. Pedro Prokurica Morales by agreement acknowledged and approved by the local Court of Vallenar on 30 March 2022.

A map of HCH mining rights under WGS84 is shown in Figure 1.3).

Figure 1.3 : Location of HCH's Cortadera Mining Rights and Surrounding Option Agreements under WGS84, HCH (2025)



1.3.3 San Antonio

HCH, through Frontera, executed an option agreement with a private party to earn a 90% interest in the San Antonio copper-gold deposit. The Option Agreement was renegotiated by HCH in December 2023, with the previous total purchase price of US$11,000,000 decreased to US$4,300,000 for the total El Fuego landholdings (which includes the San Antonio deposit area).

Continuation of existing lease mining agreements to third parties in respect to the San Antonio copper mine (limited to the mining rights San Antonio 1 al 5; Santiago 15 al 19; Santiago 1 al 14/20; San Juan Sur 1 al and San Juan Sur 6 al 23). The lease mining agreements are limited to 50,000 tonnes of material extracted per year and will expire 31st December 2025.

1.3.4 Domeyko

Frontera SpA has entered into a 100% purchase option agreement with payments of US$50,000 payable by 19 April 2025 (already satisfied), US $150,000 payable by 19 April 2026, US$200,000 payable by 19 April 2027, with a final payment of US$3,480,000 payable by 19 April 2028, and a NSR 1% royalty.

Frontera has entered into a 100% purchase option agreement for the mining right Dominoceros 1/20, with an immediate payment of US$320,000, future payments of US$680,000 within 12 months and US$1,000,000 within 24 months. The option may be fully exercised within 36 months for a final payment of US$6,890,000.

The Domeyko cluster of tenements are currently external to this Report as potential exploration targets for the Company, however the Company notes that the development of La Verde within this group as a potential opportunity for the Project.

1.4 Accessibility, Climate, Local Resources, Infrastructure and Physiography

The Project can be accessed by following the main sealed Pan-American Highway connecting Vallenar to Coquimbo in the south.

The Project has a favourable coastal location at a low altitude, surrounded by significant regional infrastructure which will be utilised for developing this greenfield copper project.

The Project benefits from the following infrastructure items:

- Regional township of Vallenar
- Pan-American Highway
- Airport located approximately 3 km south of Vallenar
- Las Losas Port
- Power substations located approximately 20 km northwest at Maitencillo, connected to the Chilean electrical grid.

1.5 Mineral Tenure, Surface Rights, Water Rights, Royalties and Agreements

1.5.1 Mineral Tenure

All mining rights at Productora are exploitation concessions with no risk of expiring if the mining taxes are duly paid annually.

HCH owns the Cortadera deposit through Frontera and controls an area measuring approximately 20 000 ha at the deposit through various 100% purchase option agreements with private mining title holders and 100% owned tenure.

HCH, through Frontera, executed an option agreement with a private party to earn a 100% interest in the San Antonio copper-gold deposit

1.5.2 Surface Rights

Ownership rights to the sub-soil are governed separately from surface ownership. Articles 120 to 125 of the Chilean Mining Code (the "Mining Code") regulate mining easements. The Mining Code grants to the owner of any mining exploitation or exploration concessions full rights to use the surface land, provided that reasonable compensation is paid to the owner of the surface land.

Productora surface rights are 100% owned by SMEA.

1.5.3 Water Rights

Since July 2024, the maritime water extraction license is in the transferring regulatory process to the newly established water company “HW Aguas para El Huasco SpA” (Huasco Water). Huasco Water is a subsidiary of Hot Chili Limited (80% interest) in partnership with CMP (20% interest).

Water easements and costal land accesses held by SMEA will be authorised to be used by Huasco Water for the benefit of Costa Fuego. Separate from this Report, a PFS level Water Supply Business Case Study by Huasco Water has been delivered focusing on establishing a water supply to the Costa Fuego Project and the broader Huasco Valley region.

1.5.4 Royalties and Encumbrances

July 2023, SMEA and Frontera entered into royalty agreements with Osisko Gold Royalties Ltd (Osisko) for the grant of a 1% NSR royalty on HCH’s share of copper and 3% NSR royalty on HCH’s share of gold produced from the Project. The royalties were granted for a USD 15 million in cash consideration pursuant to an investment agreement between the HCH, SMEA, Frontera and Osisko. The royalties are secured by pledges over the concessions comprising the Project.

The “Uranio 1 al 70", the "Montosa 1 al 4" and the "Zapa 1 al 6" mining rights within the Productora project area are subject to the royalty interests as noted in Section 1.3.1 of this Report.

The “Purisima” mining right within the Cortadera project area is subject to the royalty interest as noted in Section 1.3.2 of this Report.

Apart from that the above-mentioned royalty and security interests, the only other third-party encumbrance in Productora is an electric transmission line on the extreme north of the Project. There are no major impediments to developing the Project because this power line sits well outside of the area where most of the Project infrastructure is located.

1.5.5 Environmental Considerations

Environmental baseline studies developed at Productora and Cortadera, commencing in 2012 for Productora and 2019 for Cortadera, cover areas where mining infrastructure is proposed. This includes stockpiles, waste dumps, and the tailings storage facility ("TSF").

Studies comprise archaeological baselines, flora and fauna baselines, and groundwater monitoring and landscape analysis. A surrounding community's study has been carried out to identify the potential impacts on dwellings proximal to the Project.

Additional environmental studies are ongoing at Cortadera, San Antonio, Productora, and Alice.

1.6 Geology and Mineralisation

1.6.1 Productora

The Productora deposit is located within the Chilean Iron Belt, which extends for more than 600 km along a 20 to 30 km wide, north-northeast trending zone at the east side of the Coastal Cordillera. The deposit is hosted in the (lower Cretaceous) Bandurrias Group, a thick volcano-sedimentary sequence comprising intermediate to felsic volcanic rocks and intercalated sedimentary rocks which dips gently (15 to 30°) west to west-northwest. Dioritic dykes intrude the host rocks at Productora, typically along west- to northwest-trending late faults, and probably represent sub-volcanic feeders to an overlying andesitic sequence not represented in the Project area.

At the Productora deposit, major fault zones are associated with extensive tectonic breccia (damage zones) that host copper-gold-molybdenum mineralisation. Late faults offset the host rocks showing a west to north-westerly strike and while generally narrow, are locally up to 20m wide.

The distribution of alteration mineral assemblages at Productora and spatial zonation suggest a gentle northerly plunge for the Productora mineral system, disrupted locally via vertical and strike-slip movements across late faults. These late faults appear to be trans-tensional and nominally normal to the distal Atacama fault system.

Mineralisation at the Productora deposit strikes north-northeast and is structurally controlled, hosted within a hydrothermal tourmaline breccia unit, and generally forms sub-vertical narrow (~2-5 m) zones. Wider high-grade mineralised zones near the upper surface of the tourmaline breccia vary in orientation but tend to dip sub-vertically or ~70° west. There are also some steeply east dipping high grade zones present at Productora (e.g. Habanero lode).

Secondary and relatively lower-grade mineralisation is evident as manto (or manto-like) horizons in the southern, far northern, and far eastern flanks of Productora. Lodes within the manto horizons are typically shallow dipping at 20° to 30° to the east or west and enclosed by lower grade mineralisation.

1.6.2 Alice

The Alice porphyry is located immediately beneath an extensive, pyrophyllite-rich advanced argillic lithocap, with a porphyry stock of quartz diorite to granodiorite, characterised by biotite and hornblende phenocrysts. The lithocap overprints Alice and the regional volcanic stratigraphy and can be seen in multiple silica ridges indicating telescoping in this porphyry system. It is comprised of numerous advanced argillic alteration types, including quartz-alunite, quartz-pyrophyllite, alunite-dominant and pyrophyllite-dominant zones.

The mineralisation at Alice is hosted in a northeast trending dyke-like porphyry with sheeted and stockwork A- and B-type quartz veinlets, within additional locally disseminated background mineralisation. Highest grade mineralisation is associated with alteration overprinting, with replacement of the biotite-altered porphyry by quartz, actinolite, chlorite and magnetite.

1.6.3 Cortadera

Cortadera is a copper-gold-molybdenum porphyry deposit, comprising a series of mineralised centres (Cuerpos 1, 2, and 3) within a northwest striking structural corridor. Mineralisation continues to at least 1.3 km below the surface.

The Cortadera deposit is characterised by early- and intra-mineralisation, porphyritic tonalitic to quartz dioritic intrusions and adjacent volcano-sedimentary wall-rocks that have been recrystallised to hornfels and skarn. Hydrothermal alteration consists of moderate to strong phyllic (+chloritic) alteration, characterised by quartz/silica, sericite, and lesser amounts of chlorite.

Chalcopyrite occurs as disseminations of variable intensity within the porphyritic host rocks, particularly in association with stockwork A- and B-type veins. There is a clear correlation between increased percentage of quartz-bearing stockwork veining and sulphide content with elevated copper-gold grades.

Vein systems at Cortadera are typical of those found within porphyry-style mineralised systems. Early quartz-rich veins observed at Cuerpo 1 and Cuerpo 2 exhibit unidirectional solidification textures (UST) that are commonly associated with high-temperatures during vein emplacement. Veins formed subsequent to UST veins comprise quartz rich A-veins (chalcopyrite-pyrite± magnetite), banded MAB veins (quartz-magnetite-chalcopyrite-pyrite) and B-veins (molybdenite), cut by sericitic/chlorite C-veins (pyrite-chalcopyrite), D-veins (quartz-pyrite-sericite) and late calcite-bearing fractures. Anhydrite is locally present within some of the B and C veins.

1.6.4 San Antonio

The San Antonio deposit has been interpreted as a skarn copper deposit with mineralisation presenting in lodes with strong structural and lithological control. The deposit is characterised by mineralisation along an NNE-SSW trending shear zone through the host rocks, which comprise a shallowly east-dipping sedimentary and volcanic sequence.

Mafic and felsic dyke intrusions are common through the San Antonio deposit, mostly striking NE-SW and dipping steeply to the east. The abundance of structure and dyke is highest in the central section of San Antonio (decreasing to the north and south). Structure at San Antonio is interpreted as being due to the emplacement of an intrusion at depth, rather than crustal scale faulting.

Mineralisation is focussed on the through-going San Antonio shear and associated fault zones (nominally less than 2 m width - striking between N30E and NS) and the cross-cutting mafic dykes. The intersection lineation between these structures is interpreted to plunge approximately 30° to the south and is thought to be a significant control on mineralisation.

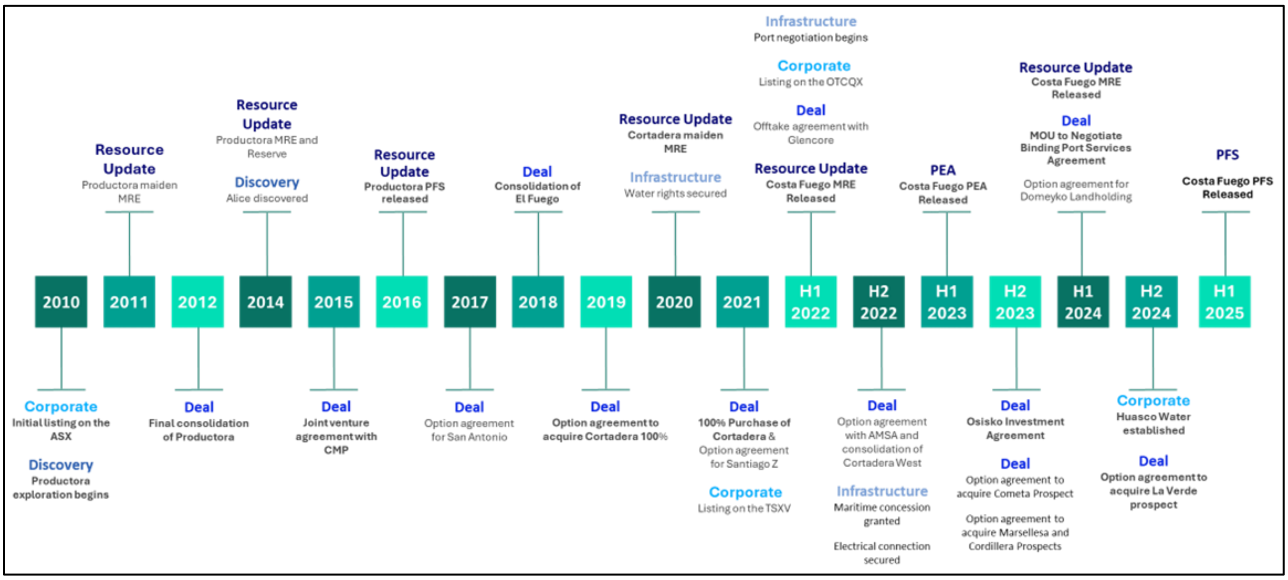
While the mafic dykes can be mineralised (although only displaying weak to moderate alteration), the intensely skarn altered (epidote-chlorite) fault zones are the more significantly mineralised. Mineralisation is observed both as supergene and hypogene principally associated with high levels of epidote-chlorite alteration.

The dominant sulphide species at San Antonio are chalcopyrite and pyrite, which occur as disseminations around the fault zone. High copper grades (up to 2%) occurring along these fault zones is associated with intense epidote > chlorite ± magnetite ± albite ± calcite and minor specular hematite

1.7 History

A summary of the history of the activities by HCH in Chile is shown in Figure 1.4.

Figure 1.4 : Summary of HCH Activities at the Costa Fuego Copper Project Since 2010



1.7.1 Productora

The Productora area has a long mining history for iron, copper and gold extending back to pre-Hispanic times. Copper mining in the past century has occurred regionally and locally at the Productora, Santa Innes, Remolino, and Montserrat mines, as well as at more than 80 smaller pits, workings, or mineralised outcrops in the area containing iron, copper, or gold mineralisation.

Since the 1980s there has been private and publicly listed companies completing exploration at Productora, including Reverse Circulation (RC) and Diamond Drilling (DD), soil sampling and geological mapping and geophysical surveys. Initially exploration was for uranium and in the 1990s the focus turned to Candelaria type iron-oxide-copper-gold (IOCG) deposits. HCH completed acquisition of the main tenement package in 2012.

1.7.2 Cortadera

Near surface oxide mineralisation was identified at Cortadera in the 1990s, with small-scale production completed via trenches and surface excavations. Porphyry style mineralisation was discovered at Cortadera and high-grade copper oxides were mined in the 2000s. Substantial drilling was completed between 2010 and 2013 and was successful in defining three mineralised porphyries (Cuerpo 1, 2 and 3) and resulted in geological modelling, preliminary resource estimation and metallurgical testwork.

Following execution of the Cortadera option agreement in February 2019, HCH undertook a resource drill out focussed on extending and infilling previously defined mineralisation, both near-surface and at depth.

1.7.1 San Antonio

The San Antonio deposit has been privately owned since 1953 and has been mined by several operators over this time via lease from the owners. Documentation regarding discovery and early exploration activities is limited for San Antonio.

1.8 Production History

1.8.1 Productora

Mining for copper at Productora commenced in 2006 and was operational until 2012, before briefly restarting in 2020.

Underground mining was completed via a modern 4.5 m x 4.5 m decline access exploiting 15 m sub-level room and pillar development. The workings extend to ~120 m below surface and produced cupriferous mineralised materials at a head grade of 0.8 to 1.2% Cu over a strike length of 300 m and a width of 50 m.

The mineralised material was stockpiled and then trucked off-site for toll-treatment at Empresa Nacional de Minería (ENAMI) processing facility in Vallenar.

Drilling by the mine operators and by HCH demonstrates that the mineralisation exploited in the underground mine is contiguous with the larger Mineral Resource.

1.8.2 Cortadera

At Cortadera, previous mining took place upon mining leases Purísima 1/8 and Cortadera 1/40, where several small workings (pirquineros) were developed in the oxide zone as trenches, surface excavations and short tunnels.

The largest of these previous workings is the Purísima Mine (Cuerpo 1) where in 1990, previous miners R.G Grego and J.R Alday developed a tunnel approximately 70 m long in the oxide zone.

Further mining at Purísima took place during 2003 to 2004 when a small open pit was developed to extract copper oxides at a grade of approximately 0.9% Cu.

1.8.3 San Antonio

The San Antonio deposit has been privately owned since 1953 and has been mined by several operators over this time via lease from the owners. Limited historic documents provided the following production data:

- 1965-1972: produced 100,000 t at ~2.5% Cu soluble (3% Cu total)
- 1980: 30,000 t of 3.0% oxide and 25,000 t at 2.0% Cu sulphide mineralisation
- 1988-1995: ~399,000 t at 1.6% Cu.

HCH's joint venture partner at San Antonio has indicated that total historic production is approximately 2 Mt of material grading, including approximately 2% Cu and 0.3 g/t Au, however no documentation has been provided that verifies this estimate.

1.9 Drilling and Sampling

Drilling by HCH across Costa Fuego has been completed over the last 14 years, beginning with Productora and Alice in 2010, followed by San Antonio in 2018 and then Cortadera in 2019.

The majority of diamond (DD) and reverse circulation (RC) drilling and assay results used for the Productora and Alice MREs were generated by extensive HCH exploration and resource development drilling programs completed between 2010 and 2015. In 2021 an exploration program of 17 RC drillholes was completed across several regional targets at Productora, followed by a metallurgical campaign of four DD holes across Productora and Alice in 2023.

A summary of all drilling that informs the 2024 Costa Fuego Project Mineral Resource Estimate is detailed in Table 1.2 below.

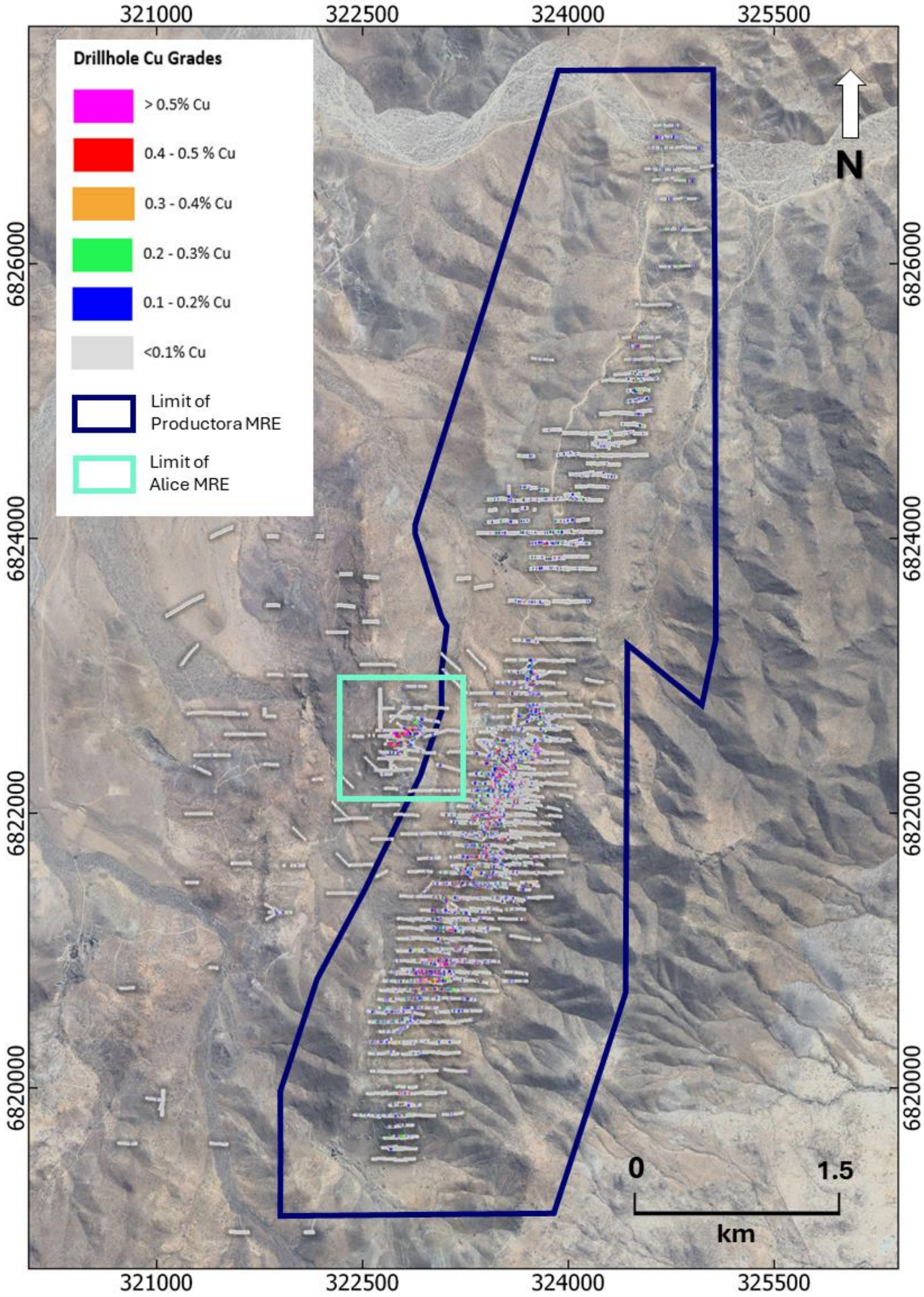
Table 1.2 : Breakdown of the Drilling Completed by HCH Across Costa Fuego Mineral Resources

Project	Year	RC (m)	DD (m)
Productora	2010 - 2023	218,231	29,241
Alice	2010 - 2022	17,156	1,802
Cortadera	2019 - 2024	41,680	44,881
San Antonio	2018 - 2022	6,931	495

1.9.1 Productora

All drilling and assay results used for the Productora and Alice Mineral Resource Estimates were generated by extensive HCH exploration and resource development drilling programs completed between 2010 and 2015, and metallurgical drilling completed in 2022 (Figure 1.5).

Figure 1.5 : Plan View of Productora Showing All Completed Drilling Relative to the Productora and Alice MRE extents, Drill Holes Displayed by Copper Grade, HCH (2025)



RC and DD completed at Productora is logged, with measurements for bulk density and samples for metallurgical testwork taken as required. Drilling at Productora has been completed on a nominal drill collar pattern of 80 m by 40 m to approximately 300 m vertical from surface.

Multi-element ME-MS61 (48 element) analysis was collected on surface soil samples, rock chips and selected down hole samples over several exploration and drilling campaigns.

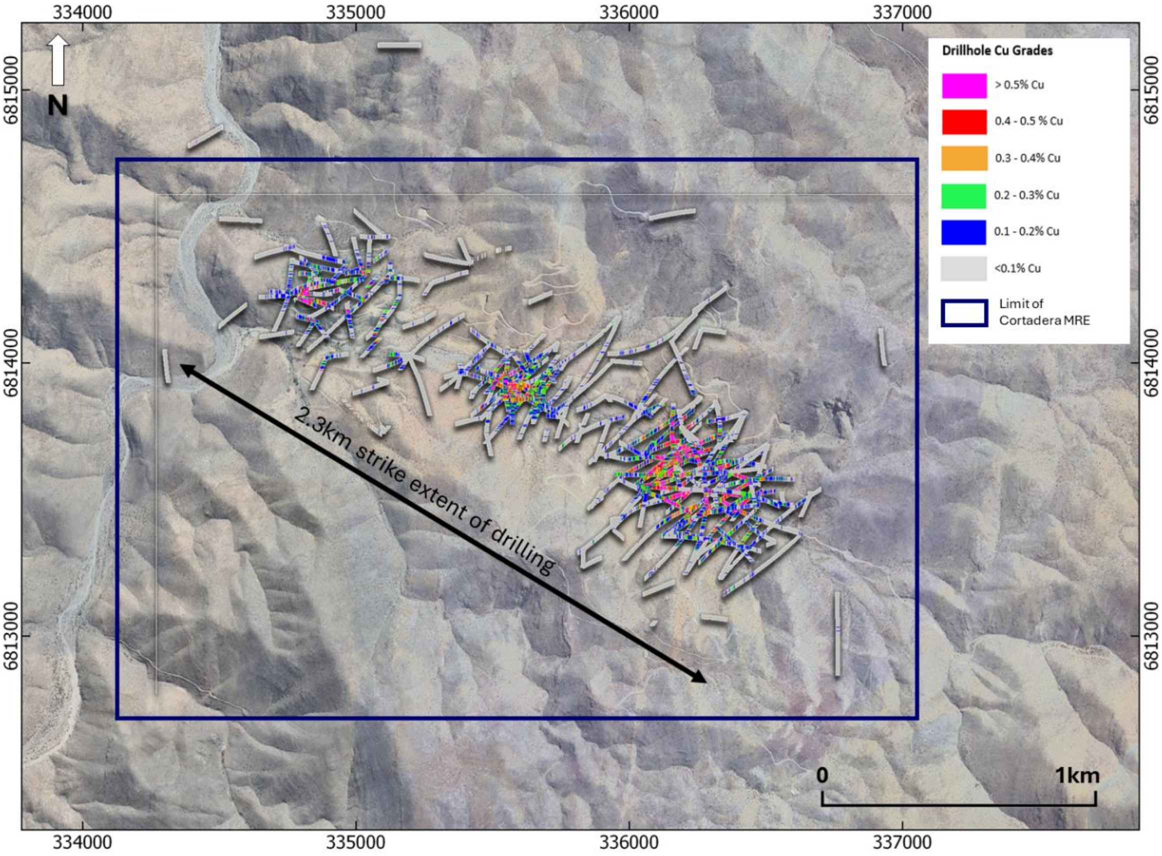
1.9.2 Cortadera

Following execution of the Cortadera option agreement in February 2019, HCH undertook several phases of resource definition drilling focussed on infilling and extending previously defined mineralisation. Drilling improved geological understanding and resulted in significant resource growth, including the discovery of a bulk-tonnage high grade zone at Cuerpo 3. The Mineral Resource Estimate outlined in this Report incorporates resource extension and exploration drilling completed in 2022 and 2023, including six metallurgical/geotechnical drillholes (Figure 1.6).

The spacing and location of much of the drilling at Cortadera is variable and averages approximately 80 m along strike and 150 m across strike. Drill holes dip 60 to 80° toward the northeast or southwest. Additional orientations were used to ensure geological representativeness and to optimise the use of available drill platforms.

Multi-element ME-MS61 (48 element) analysis was collected on surface soil samples, rock chips and selected down hole samples over several exploration and drilling campaigns. This data was used for 3D geochemical modelling completed by Fathom Geophysics in 2021, which utilised the geochemical element zoning models for the Yerington porphyry copper deposit in Nevada (Cohen, 2011; and Halley et al., 2015).

Figure 1.6: Plan View of Cortadera Showing all Drilling Completed up to 2024 relative to MRE extents, Drill Holes are Displayed by Copper Grade, HCH (2025)



1.9.3 San Antonio

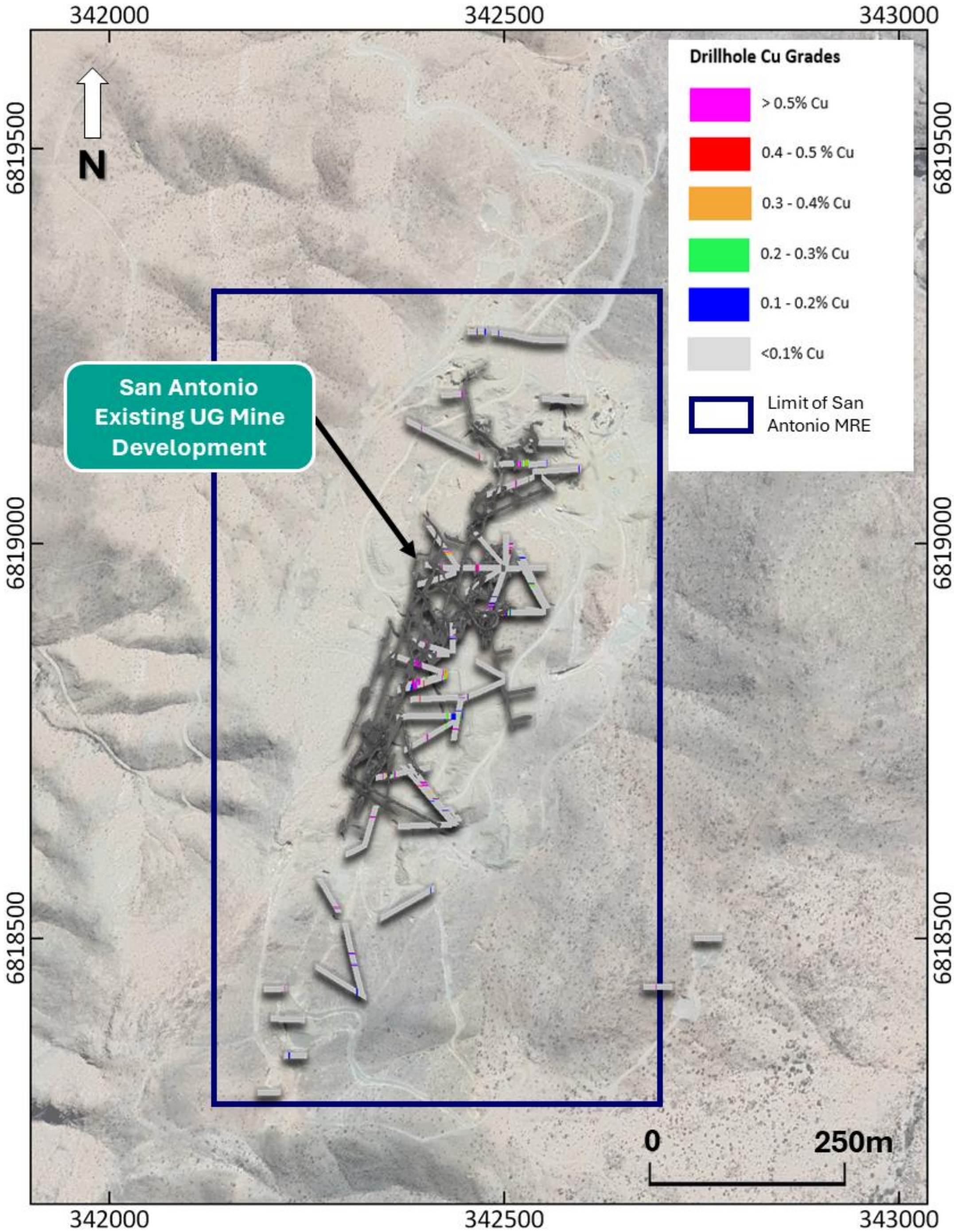
Since executing the San Antonio option agreement in November 2017, HCH has undertaken a range of exploration activities including field mapping, soil and rock sampling, underground mapping, underground chip sampling, an RC drill program, and a survey of accessible mined excavations to verify depletion and void models.

RC drilling completed by HCH at San Antonio is spaced at approximately 40 m x 40 m, with the closest spacing around existing development where mineralisation grades are highest. RC and DD completed at San Antonio is logged, with measurements for bulk density and samples for metallurgical testwork taken as required.

Multi-element ME-MS61 (48 element) analysis was collected on surface soil samples, rock chips and selected down hole samples over several exploration and drilling campaigns.

A plan view showing all drilling completed at San Antonio relative to the MRE extents is shown in Figure 1.7.

Figure 1.7 : San Antonio Drill Program Hole Locations Relative to Existing Underground Mine Development, HCH (2025)



1.10 Data Verification

Drill collars were routinely surveyed following completion of drilling campaigns by a contract surveying company, and downhole surveys were completed by contracting companies using either a mechanical gyroscope instrument (non-north-seeking) or a north-seeking high-speed continuous gyroscopic camera.

Geological logging was recorded in a systematic and consistent manner such that the data was able to be interrogated accurately using geological modelling software programs. Collar, survey, sample register and sample condition, sample recovery data or comments (and commonly magnetic susceptibility) were recorded by a competent field technician.

HCH has implemented rigorous sample preparation and analytical procedures for both RC and DD samples, following consultation with ALS in Chile, to ensure assays were reported with a high degree of confidence and a wide range of appropriate commodities were assessed.

The results from HCH's QA/QC analysis as well as RC vs. DD twinned analysis provides a high level of confidence in the precision and accuracy of the assays used for the resource estimations.

The sample lengths, preparation and assay techniques are considered suitable for the styles of mineralisation and deposit types.

The verification of input data included the use of company QA/QC blanks and reference material, field and laboratory duplicates, umpire laboratory checks and independent sample and assay verification.

An internally managed and audited acQuire geological database was used to capture and manage geological drill data. Strict data flow procedures were followed during drilling. This ensured that all drill data was collected in a systematic, repeatable, and accurate manner allowing consistent and validated data for resource estimation.

The Qualified Person has assessed the drill hole database validation work and QA/QC undertaken by HCH and was satisfied the input data could be relied upon for the MREs.

1.11 Metallurgical Testwork

1.11.1 General

Metallurgical samples were selected as contiguous intervals from diamond drilling completed across Productora (16 drillholes), Alice (2 drillholes), Cortadera OP (15 drillholes), Cortadera UG (15 drillholes) and San Antonio (1 drillhole). The samples provide appropriate coverage of the various mineralisation styles encountered across the Costa Fuego Project.

PFS Metallurgical testwork was conducted at:

- ALS Laboratories in Perth Western Australia (flotation and comminution testwork)
- Nova Mineralis in Santiago, Chile (oxide heap and low-grade sulphide dump leaching)
- Outotec in Perth, Australia (sulphide concentrate thickening and filtration)

- Auralia Metallurgy, Australia (flotation and comminution)
- JKTech in Brisbane, Australia (comminution testwork)

Mineralogical analysis of fresh ores showed that for all four deposits the dominant copper mineral was chalcopyrite. The other sulphide of significance in the deposits was pyrite. There is excellent liberation between chalcopyrite and pyrite after regrinding to 25 µm P_{80}. Molybdenite is also present at low (ppm) levels high enough to recover into a saleable concentrate. Molybdenite is well liberated.

Testwork was conducted on both the sulphide mineralised material and the copper oxide mineralised material, with the aim of providing design criteria for the sulphide and oxide process plants, an indication of crushing and grinding circuit throughputs and metallurgical recoveries for each mining area.

1.11.2 Comminution and Flotation

Three key comminution properties are shown in Table 1.3 for the concentrator feed originating from main deposits plus Alice.

Table 1.3: Summary Comminution Properties for Costa Fuego Project Deposits – Concentrator Feed

	Productora	Alice	Cortadera OP	Cortadera UG
Samples	22	2	20	21
DWI (kWh/m^3)	7.6	7.0	6.6	8.7
BWI (kWh/t)	18.5	17.4	14.2	16.9
Ai	0.28	0.27	0.14	0.17

The metallurgical testwork indicates the sulphide mill feed can be processed by conventional crushing followed by grinding in a SAG and ball mill circuit to a grind P_{80} of 125 µm. Predictive methods were developed to allow the estimation of both DWI and BWI from geological information and these parameters have been incorporated into the mine planning process.

Over 400 flotation tests have been performed on samples since the project commenced and a suitable testing flowsheet has been developed. The ground ore is separated using froth flotation to recover copper, gold and silver into a copper concentrate and recover molybdenum into its own saleable concentrate. A chalcopyrite-specific xanthate ester flotation collector, RTD2086, has proved successful despite being challenged throughout the program and is recommended for use. No pH adjustment is necessary for rougher, cleaner and scavenger flotation of Cu/Mo concentrates with this collector. Regrinding of rougher and scavenger concentrate to 25 µm P_{80} is generally required to achieve, or better, the target concentrate grade of 25% Cu.

The copper concentrates from 8 out of the 33 locked-cycle tests completed for the Costa Fuego Project were comprehensively analysed and they confirm very low arsenic levels and no significant levels of other potential penalty elements. Acceptable chloride levels were obtained in the copper concentrate after fresh water washing. Negligible deleterious elements were reported in what is consistently a high specification clean concentrate.

A combined copper concentrate, remaining after Mo removal by flotation and proportionally representing the three main deposits, was comprehensively analysed giving the result in Table 1.4.

Table 1.4: Final Cu Concentrate for the Costa Fuego Project

Element	Unit	Final Cu Concentrate Costa Fuego Project
Cu	%	25.6
Mo	ppm	586
Au	ppm	3.82
Ag	ppm	23.1
Al_2O_3	%	2.66
As	ppm	18.9
Ba	ppm	96
Bi	ppm	2.6
CaO	%	0.59
Cd	ppm	2.0
Cl	ppm	200
Co	ppm	323
F	ppm	238
Fe	%	28.1
Hg	ppm	0.78
K	ppm	4,568
MgO	ppm	3,599
Mn	ppm	122
Na	ppm	2,611
Ni	ppm	178
P	ppm	134
Pb	ppm	45
S	%	32.6
Sb	ppm	9
Se	ppm	69
SiO_2	%	9,5
Sn	ppm	6
Sr	ppm	33
Te	ppm	3.0
Th	ppm	3.9
Ti	ppm	563
Zn	ppm	301
Zr	ppm	125

[1] Weighted average by copper metal produced by deposit on a LOM basis
[2] Final Cu concentrate stream includes the two streams reporting to the Final Cu concentrate (Molybdenum rougher flotation tail and Molybdenum cleaner flotation tail, as per Figure 17.6).

Average anticipated recoveries for values from sulphide ore (fresh and transition included where appropriate) are shown in Table 1.5.

Table 1.5 : Average Flotation Product Recoveries for Fresh Sulphide Material

Item	Copper	Gold	Molybdenum	Silver
Average (%)	86	54	70	37

In the vast majority of cases the Cu/Mo and Cu concentrates produced in both open circuit and locked cycle testing contained >25% Cu.

1.11.3 Leaching

Metallurgical leaching testwork has been completed on samples from the Costa Fuego project including the Productora and Cortadera deposits, which represent 90% or more of the processing feed considered for leaching technology.

The results confirmed the technical and economic viability for oxide and low-grade sulphide to heap leaching and dump leaching technology, respectively, as well as provided input for PFS level design.

1.11.3.1 Heap Leaching

Preliminary heap leaching characterisation testwork was conducted during 2014-2015 on Productora samples, main deposit discovered at the time. The Cortadera deposit was acquired and included lately as part of the Costa Fuego Project, with leaching testwork completed during 2020.

The initial testwork focused on conventional sulfuric acid heap leaching methods while later in the Project, the focus changed to NOVAMINORE® (NOVAMINORE) hypersaline technology. Thus, the NOVAMINORE technology was tested on Productora and Cortadera samples during 2024 and compared against conventional leaching technology.

According to the results of the testwork completed on Productora and Cortadera samples, copper recovery appeared to be higher when NOVAMINORE® technology was used versus conventional leaching while acid consumption appeared to be slightly higher for conventional technology after initial leaching time.

This study included testing an additional 29 variability samples from the Productora (and Alice) deposit plus 8 samples from the Cortadera deposit, using 1 m height column test. The results from the variability testing were used for copper recovery and acid consumption predictive models update.

Life of Mine copper recovery established from the copper recovery predictive models for heap leaching is presented in Table 1.6.

Table 1.6 : LOM Recovery for Heap Leaching

Recovery	Cu Recovery to Cathode (%)
Life of Mine	65

1.11.3.2 Dump Leaching

Preliminary dump leaching testwork on primary low-grade production material was completed during 2022. The objective of the testwork was to assess the amenability of acid dump leaching via the application of NOVAMINORE® technology to low-grade sulphide material.

The results of the preliminary results were positive and therefore it was considered to progress with the use of the technology for dump leaching to pre-feasibility level.

Additional dump leaching testwork was completed during 2024, including laboratory scale testwork on variability samples and semi-pilot testwork. The variability testing included a total of 8 samples from Productora and 14 samples from Cortadera and were developed using 1 m height columns. A scaling test was also completed using a Productora sample equivalent to a 9 m height column test. The results of the variability samples were used for copper concentrate and acid consumption predictive models update while the semi-pilot test was used for scaling the results.

Preliminary and later dump leaching testwork completed, confirmed Productora material as feed to the low-grade dump leaching while Cortadera material was discarded for low-grade dump leaching at pre-feasibility level design.

Based on the above, the Life of Mine copper recovery for dump leaching is presented in Table 1.7.

Table 1.7 : LOM Recovery for Dump Leaching

Recovery	Cu Recovery to Cathode (%)
Life of Mine	39

1.11.3.3 Laboratory SX testing

Laboratory testing was completed using solutions obtained from the leaching testwork developed and reagents used in SX plants. Two reagent suppliers operating in Chile were selected for this testwork with the objective of validating the solvent extraction process for the project. It is expected that the PLS copper-rich solutions from Heap and Dump leaching will have a minimum chloride content of 120 g/L which was considered to potentially impact the performance of some extractive reagents used in SX plants, therefore solutions around this range and higher were used for the testwork. Similar results were obtained from both suppliers indicating availability of reagents to process the PLS solutions thus validating the design.

1.12 Mineral Resource Estimates

The Costa Fuego Project MRE includes four MREs: Productora, Alice, Cortadera and San Antonio.

The Costa Fuego Project MRE is reported in accordance with the Joint Ore Reserves Committee Code (2012) and the Canadian Institute of Mining, Metallurgy and Petroleum ("CIM") Standards on Mineral Resources and Reserves, Definitions and Guidelines prepared by the CIM Standing Committee on Reserve Definition, as required by NI 43-101.

Productora has had multiple MREs completed since HCH took ownership in 2013, with the 2024 version (2024 MRE) further refining the updated estimation technique used for the March 2022 MRE. The 2024 MRE updates the March 2022 MRE, which informed the 2023 Preliminary Economic Assessment ("PEA").

Alice (which had previously been combined for reporting with the Productora MRE) has had two MREs completed; the maiden resource in 2015 which informed the 2023 PEA, and the current 2024 MRE.

Cortadera has had three MREs completed; the maiden resource in 2020, the March 2022 MRE which informed the 2023 PEA, and the 2024 MRE.

San Antonio has had two MREs completed; the maiden resource in 2022 which informed the 2023 PEA, and the 2024 MRE.

Resource models used Leapfrog® version 2023.1 and Datamine® version 2.0.66, two industry standard commercial geological and mining software packages. The construction of the 3D resource models, and the estimation of mineral resources were performed by HCH personnel following HCH procedures.

The estimation approach for Productora utilised Categorical Kriging, taking advantage of the extensive dataset of the multi-element assays, while the Cortadera estimation employed Ordinary Kriging and Indicator Estimation within mineralisation- and geologically constrained domains. The San Antonio and Alice estimations used Ordinary Kriging.

Extensive validation was completed on the resource estimations, including internal company peer review. The MRE has benefited immensely from the technical guidance of Dr Steve Garwin, one of the leading authorities on porphyry style mineralisation in the circum-pacific region. Mineral Resource practises and the resultant estimates have also undergone detailed external technical review by Scott Dunham (SD2 Consulting, 2024) and Mark Noppe (WH Bryan Mining Geology Research Centre, 2024).

Mineral Resources were classified as either Indicated or Inferred, based on a range of criteria, including but not limited to, geological and grade continuity between drill holes, drill hole spacing, mineralisation type, and data quality.

Independent QP Ms Elizabeth Haren of Haren Consulting was responsible for all data verification, geological and mineralisation interpretation, and three-dimensional surface creation. Work completed by the Company was peer reviewed prior to block model resource estimation and classification by Ms Haren. Ms Haren was responsible for all aspects of geostatistical analysis, variography modelling, and determination of parameters for block model and resource estimation.

Ms. Haren is a Qualified Person within the meaning of NI 43-101. Ms. Haren is a Fellow and Chartered Professional of The Australasian Institute of Mining and Metallurgy ("AusIMM") and a Member of the Australian Institute of Geoscientists ("AIG").

There are currently no known environmental, permitting, legal, title, taxation, socio-economic, marketing, political or other relevant factors which could affect the Costa Fuego Project MRE.

The combined Costa Fuego MRE can be seen in Table 1.8, with Figure 1.8, and Figure 1.9 illustrating the individual block models informing the 2024 MRE.

Table 1.8 : Costa Fuego Project Mineral Resource Summary – Reported by Classification (26 February 2024)

Costa Fuego OP Resource		**Grade**					**Contained Metal**				
Classification	Tonnes	CuEq	Cu	Au	Ag	Mo	Copper Eq	Copper	Gold	Silver	Molybdenum
(+0.20% CuEq*)	(Mt)	(%)	(%)	(g/t)	(g/t)	(ppm)	(tonnes)	(tonnes)	(ounces)	(ounces)	(tonnes)
Indicated	736	0.46	0.37	0.11	0.50	85	3,370,000	2,720,000	2,480,000	11,700,000	62,800
M+I Total	**736**	**0.46**	**0.37**	**0.11**	**0.50**	**85**	**3,370,000**	**2,720,000**	**2,480,000**	**11,700,000**	**62,800**
Inferred	170	0.30	0.25	0.06	0.36	65	520,000	420,000	340,000	1,900,000	11,000
Costa Fuego UG Resource		**Grade**					**Contained Metal**				
Classification	Tonnes	CuEq	Cu	Au	Ag	Mo	Copper Eq	Copper	Gold	Silver	Molybdenum
(+0.27% CuEq*)	(Mt)	(%)	(%)	(g/t)	(g/t)	(ppm)	(tonnes)	(tonnes)	(ounces)	(ounces)	(tonnes)
Indicated	62	0.39	0.31	0.08	0.55	85	250,000	190,000	160,000	1,100,000	5,300
M+I Total	**62**	**0.39**	**0.31**	**0.08**	**0.55**	**85**	**250,000**	**190,000**	**160,000**	**1,100,000**	**5,300**
Inferred	33	0.35	0.29	0.07	0.41	46	120,000	96,000	76,000	430,000	1,500
Costa Fuego Total Resource		**Grade**					**Contained Metal**				
Classification	Tonnes	CuEq	Cu	Au	Ag	Mo	Copper Eq	Copper	Gold	Silver	Molybdenum
	(Mt)	(%)	(%)	(g/t)	(g/t)	(ppm)	(tonnes)	(tonnes)	(ounces)	(ounces)	(tonnes)
Indicated	798	0.45	0.37	0.10	0.50	85	3,620,000	2,910,000	2,640,000	12,800,000	68,100
M+I Total	**798**	**0.45**	**0.37**	**0.10**	**0.50**	**85**	**3,620,000**	**2,910,000**	**2,640,000**	**12,800,000**	**68,100**
Inferred	203	0.31	0.25	0.06	0.36	61	640,000	516,000	416,000	2,330,000	12,500

[1] Mineral Resources are reported on a 100% Basis - combining Mineral Resource Estimates for the Cortadera, Productora, Alice and San Antonio deposits. All figures are rounded, reported to appropriate significant figures and reported in accordance with the Joint Ore Reserves Committee Code (2012) and NI 43-101. MRE practices are undertaken in accordance with CIM Estimation of Mineral Resource and Mineral Reserve Best Practice Guidelines (29 November 2019) and reported in accordance CIM Definition Standards for Mineral Resources and Mineral Reserves (10 May 2014) that are incorporated by reference into NI 43-101.

[2] Mineral Resources are inclusive of the Mineral Reserve.

[3] The Productora deposit is 100% owned by Chilean incorporated company Sociedad Minera El Aguila SpA (SMEA). SMEA is a joint venture (JV) company – 80% owned by Sociedad Minera El Corazón SpA (a 100% subsidiary of Hot Chili), and 20% owned by Compañía Minera del Pacífico S.A (CMP).

[4] The Cortadera deposit is controlled by a Chilean incorporated company Sociedad Minera La Frontera SpA (Frontera). Frontera is a subsidiary company – 100% owned by Sociedad Minera El Corazón SpA, which is a 100% subsidiary of Hot Chili.

[5] The San Antonio deposit is controlled through Frontera (100% owned by Sociedad Minera El Corazón SpA, which is a 100% subsidiary of Hot Chili Liited) and Frontera is party to an option agreement pursuant to which it can earn a 100% interest in the property.

[6] The Mineral Resource Estimates in the tables above form coherent bodies of mineralisation that are considered amenable to a combination of open pit and underground extraction methods based on the following parameters: Base Case Metal Prices: Copper US$3.00/lb, Gold US$1,700/oz, Molybdenum US$14/lb, and Silver US$20/oz.

[7] All MRE were assessed for Reasonable Prospects of Eventual Economic Extraction (RPEEE) using both Open Pit and Block Cave Extraction mining methods at Cortadera and Open Pit mining methods at the Productora, Alice and San Antonio deposits.

[8] Metallurgical recovery averages for each deposit consider Indicated + Inferred material and are weighted to combine sulphide flotation and oxide leaching performance. Process recoveries: Cortadera – Weighted recoveries of 82% Cu, 55% Au, 81% Mo and 36% Ag. CuEq(%) = Cu(%) + 0.55 x Au(g/t) + 0.00046 x Mo(ppm) + 0.0043 x Ag(g/t). San Antonio - Weighted recoveries of 85% Cu, 66% Au, 80% Mo and 63% Ag. CuEq(%) = Cu(%) + 0.64 x Au(g/t) + 0.00044 x Mo(ppm) + 0.0072 x Ag(g/t) Alice - Weighted recoveries of 81% Cu, 47% Au, 52% Mo and 37% Ag. CuEq(%) = Cu(%) + 0.48 x Au(g/t) + 0.00030 x Mo(ppm) + 0.0044 x Ag(g/t). Productora – Weighted recoveries of 84% Cu, 47% Au, 48% Mo and 18% Ag. CuEq(%) = Cu(%) + 0.46 x Au(g/t) + 0.00026 x Mo(ppm) + 0.0021 x Ag(g/t). Costa Fuego – Recoveries of 83% Cu, 53% Au, 71% Mo and 26% Ag. CuEq(%) = Cu(%) + 0.53 x Au(g/t) + 0.00040 x Mo(ppm) + 0.0030 x Ag(g/t)

[9] Copper Equivalent (CuEq) grades are calculated based on the formula: CuEq% = ((Cu% × Cu price 1% per tonne × Cu_recovery) + (Mo ppm × Mo price per g/t × Mo_recovery) + (Au ppm × Au price per g/t × Au_recovery) + (Ag ppm × Ag price per g/t × Ag_recovery)) / (Cu price 1% per tonne × Cu recovery). The base case cut-off grade for Mineral Resources considered amenable to open pit extraction methods at the Cortadera, Productora, Alice and San Antonio deposits is 0.20% CuEq, while the cut-off grade for Mineral Resources considered amenable to underground extraction methods at the Cortadera deposit is 0.27% CuEq. It is the Company's opinion that all the elements included in the CuEq calculation have a reasonable potential to be recovered and sold.

[10] Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. The 2024 PEA includes Inferred Mineral Resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorised as Mineral Reserves. It is reasonably expected that the majority of Inferred mineral resources could be upgraded to Measured or Indicated Mineral Resources with continued exploration.

[11] The effective date of the MRE is 26 February 2024. The 2024 MRE was previously reported in the 2024 PEA. Hot Chili Limited confirms it is not aware of any new information or data that materially affects the information included in the 2024 PEA and all material assumptions and technical parameters stated for the 2024 MRE in the 2024 PEA continue to apply and have not materially changed.

[12] Hot Chili Limited is not aware of political, environmental, or other risks that could materially affect the potential development of the Mineral Resources other than as disclosed in this Report. A detailed list of Costa Fuego Project risks is included in Section 25 of this Report.

Figure 1.8 : Long Section View of the Productora MRE Blocks, with the 2025 PFS Design Pit shape and 2024 MRE RPEEE Pit shape shown for reference. Model coloured by Cu Grade. HCH (2024)

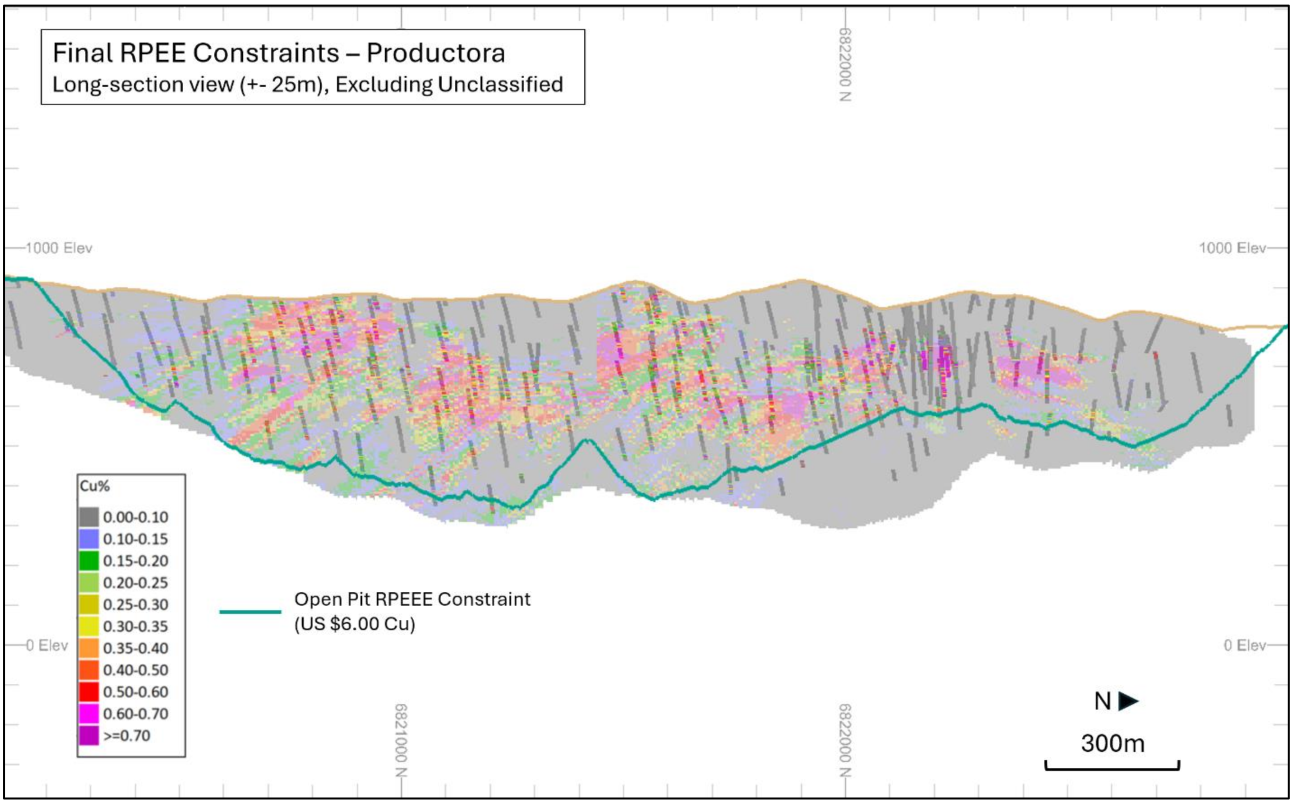
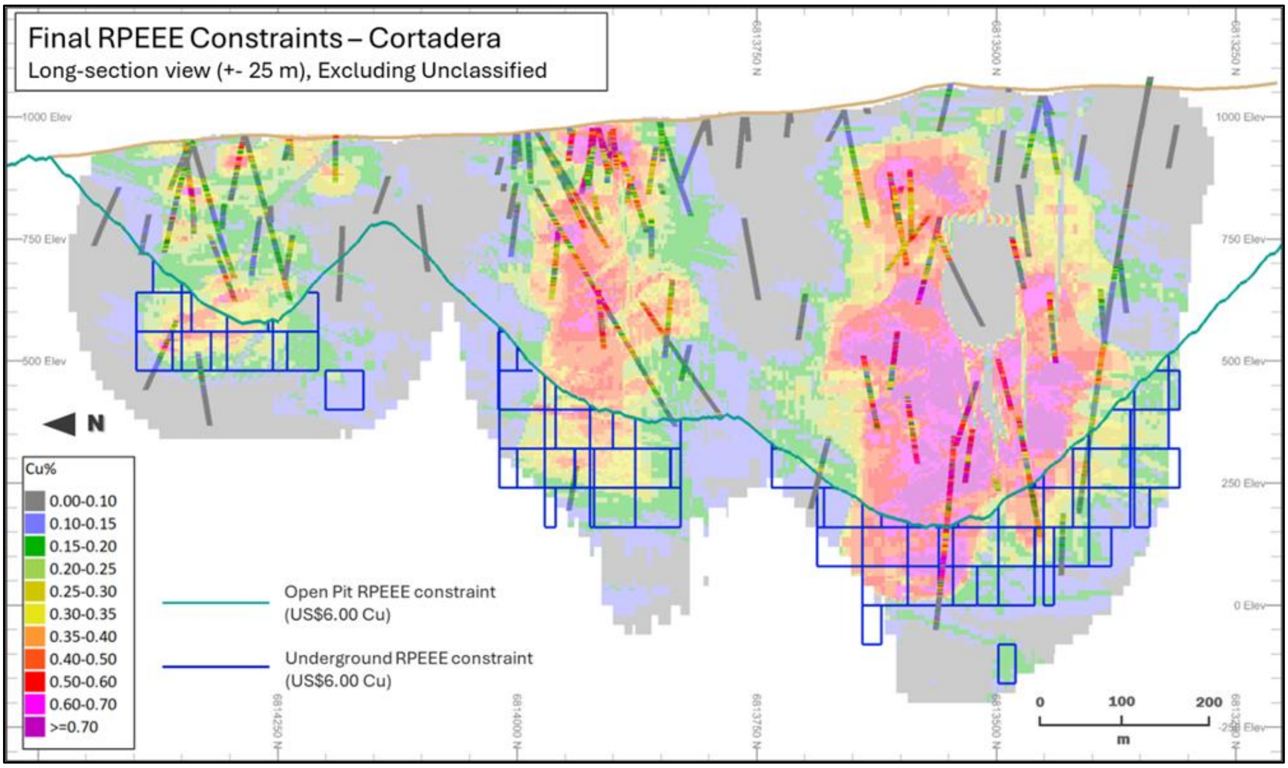


Figure 1.9 : Long Section View of the Cortadera MRE Blocks, with the 2025 PFS Design Pit shape and 2024 MRE RPEEE Pit shape shown for reference. Model coloured by Cu Grade. HCH (2024)



1.13 Mineral Reserve Estimation

1.13.1 Modifying Factors and Material Assumptions for the Mineral Reserve

The Costa Fuego Project at PFS stage envisages conventional open pit, truck and shovel operation from four mineral deposits (Alice, Cortadera, Productora, and San Antonio) and underground block caving from a single mine area (Cortadera, below Cuerpo 3 open pit). Ore would be processed either via heap leach (oxide only), concentrator (transitional and fresh only), or low-grade dump leach (all material classifications).

The Probable Mineral Reserve is based on Indicated Mineral Resources within resource block models regularised to 5 m (x) x 10 m (y) x 5 m (z). Only Indicated blocks have been considered for the Mineral Reserve estimate, with metal grades for Inferred blocks coded to zero before the first stage of model optimisation.

Dilution and ore loss is captured within the Mineral Resource block model due to regularisation to the singular mining unit (“SMU”). This is considered appropriate for the large-scale mineralised systems that comprise the majority of the Costa Fuego Project Mineral Reserve.

Development schedules produced during Mineral Reserve estimation consider study, permitting and construction periods. Environmental, Social and Governance (“ESG”) modifying factors include the assumption that strong relations with local communities and government will be maintained and there will be no material environmental or related issue that impacts the development schedule.

1.13.2 Open Pit Mineral Reserves

Mineral Reserve evaluation was completed using a series of open pit optimisation created using Net Smelter Return ("NSR") cut-offs which were subsequently engineered into mining design stages. This work was completed using Datamine NPVS software. Staged pit designs were developed with geotechnical slope design criteria as assessed by the Geotechnical QP. For scheduling, pit stages were imported into MineMax Scheduler software where the optimal schedule was determined from several iterations.

Following mine design and scheduling, an economic evaluation was completed using NSR cut-offs to determine if a block would be processed. Copper, gold, molybdenum, and silver were considered as economic contributing minerals to be recovered and sold as a concentrate (for the transitional and fresh blocks) and copper cathode to be produced from the oxide leach and low-grade sulphide leach material.

For open pit mining, modifying factors included planned and unplanned mining dilution (incorporated within the SMU consideration), potential ore loss (ore extraction percentage), operating costs, pit slopes, minimum mining width, maximum annual mining rates, vertical rate of bench advance and haulage distances to the plant, stockpile and waste dumps.

Economic modifying factors include the assumption that there will be no unforeseen cost impediments or negative metal price fluctuations that impact the development of the Project. The pit optimisation studies tested various potential economic criteria variations (cost, recovery and metal prices). The final pit shells were generated at a lower copper price assumption to ensure that the open pit economic perimeters will be reasonably robust if changes to economic modifying factors occur.

1.13.3 Underground Mineral Reserves

Block caving has been determined as the optimal exploitation strategy for the Cortadera Mineral Resource that is not optimised into an Open Pit.

Block cave mine shape optimisation developed the cave footprint using the Geovia Footprint Finder software using NSR cut-offs. This process output multiple optimisation footprints and draw heights to ultimately recommend the optimal block cave footprint, undercut elevation and block cave height. Detailed block cave engineering and design work then completed on the optimised block cave in Geovia PCBC software. The block caving mine design followed geotechnical guidelines and accounted for interactions with the Cortadera open pit, reviewed by the Geotechnical QP.

The block cave design is supported by a detailed mine access and mine development design and associated mine schedule completed in Datamine Studio UG software.

Following mine design and scheduling, an economic evaluation was completed using NSR cut-offs to determine if a block would be processed. Copper, gold, molybdenum, and silver were considered as economic contributing minerals to be recovered and sold as a concentrate.

1.13.4 Mineral Reserve Statement

The Costa Fuego Project Mineral Reserve is reported in accordance with the Joint Ore Reserves Committee ("JORC") Code (2012) and the Canadian Institute of Mining, Metallurgy and Petroleum ("CIM") Standards on Mineral Resources and Reserves, Definitions and Guidelines prepared by the CIM Standing Committee on Reserve Definition, as required by NI 43-101. References to "Mineral Reserves" mean "Ore Reserves" as defined in the JORC Code and references to "Proven Mineral Reserves" mean "Proved Ore Reserves" as defined in the JORC Code. There is no material difference between the definitions of Probable Ore Reserves under the 2014 CIM Definition Standards for Mineral Resources and Mineral Reserves and the equivalent definitions in the JORC Code (2012). Terms Mineral Reserve (CIM) and Ore Reserve (JORC) are equivalent, and this study uses Mineral Reserve for consistency.

The competent person for the Mineral Reserve estimate is Mr Anton Von Wielligh, Director of ABGM. Mr Von Wielligh is a Qualified Person within the meaning of NI 43-101 who is a Fellow of The Australasian Institute of Mining and Metallurgy ("FAusIMM"). Table 1.9 summarises the Costa Fuego Project Mineral Reserve estimate by mining methodology (open pit and underground) and by processing methodology (sulphide concentrator, oxide leach, and low-grade sulphide leach).

Table 1.9 Costa Fuego Mineral Reserve Estimate (27 March 2025)

		Grade				Contained Metal			
	Tonnes (Mt)	**Cu (%)**	**Au (g/t)**	**Ag (g/t)**	**Mo (ppm)**	**Cu (kt)**	**Au (koz)**	**Ag (koz)**	**Mo (kt)**
Open Pit									
Concentrator									
Proven	-	-	-	-	-	-	-	-	-
Probable	293	0.36	0.08	0.37	113	1,043	728	3,517	33
Total	**293**	**0.36**	**0.08**	**0.37**	**113**	**1,043**	**728**	**3,517**	**33**
Heap Leach									
Proven	-	-	-	-	-	-	-	-	-
Probable	41	0.35	0.07	0.43	35	142	96	563	1
Total	**41**	**0.35**	**0.07**	**0.43**	**35**	**142**	**96**	**563**	**1**
Dump Leach									
Proven	-	-	-	-	-	-	-	-	-
Probable	22	0.13	0.03	0.23	41	29	20	168	1
Total	**22**	**0.13**	**0.03**	**0.23**	**41**	**29**	**20**	**168**	**1**
Combined									
Proven	-	-	-	-	-	-	-	-	-
Probable	356	0.34	0.07	0.37	98	1,213	844	4,248	35
Total	**356**	**0.34**	**0.07**	**0.37**	**98**	**1,213**	**844**	**4,248**	**35**
Underground									
Concentrator									
Proven	-	-	-	-	-	-	-	-	-
Probable	146	0.44	0.16	0.79	93	645	734	3,704	14
Total	**146**	**0.44**	**0.16**	**0.79**	**93**	**645**	**734**	**3,704**	**14**
Combined (Open Pit and Underground)									
Proven	-	-	-	-	-	-	-	-	-
Probable	502	0.37	0.10	0.49	97	1,858	1,578	7,951	49
Total	**502**	**0.37**	**0.10**	**0.49**	**97**	**1,858**	**1,578**	**7,951**	**49**

[1] Mineral Reserves are reported on a 100% Basis - combining Mineral Reserve Estimates for the Cortadera, Productora, Alice and San Antonio deposits, and have an effective date of 27 March 2025.

[2] An Ore Reserve (declared in accordance with JORC Code 2012) was previously reported at Productora, a component of Costa Fuego, on 2nd March 2016 on the ASX. The Company was not subject to the requirements of NI 43-101 at that time.

[3] Mineral Reserve estimation practices are in accordance with CIM Estimation of Mineral Resource and Mineral Reserve Best Practice Guidelines (29 November 2019) and reported in accordance CIM Definition Standards for Mineral Resources and Mineral Reserves (10 May 2014) that are incorporated by reference into NI 43-101.Mineral Reserve estimates are in accordance with the JORC Code. References to "Mineral Reserves" mean "Ore Reserves" as defined in the JORC Code and references to "Proven Mineral Reserves" mean "Proved Ore Reserves" as defined in the JORC Code.

[4] The Mineral Reserve reported above was not additive to the Mineral Resource. The Mineral Reserve is based on the 26 February 2024 Mineral Resource.

[5] Tonnages and grades are rounded to two significant figures. All figures are rounded, reported to appropriate significant figures and reported in accordance with the Joint Ore Reserves Committee Code (2012) and NI 43-101. As each number is

rounded individually, the table may show apparent inconsistencies between the sum of rounded components and the corresponding rounded total.

[6] Mineral Reserves are reported using long-term metal prices of US$4.30/lb Cu, US$2,280/oz Au, US$27/oz Ag, US$20/lb Mo.

[7] The Mineral Reserve tonnages and grades are estimated and reported as delivered to plant (the point where material is delivered to the processing facility) and is therefore inclusive of ore loss and dilution.

[8] The Productora deposit is 100% owned by Chilean incorporated company Sociedad Minera El Aguila SpA (SMEA). SMEA is a joint venture (JV) company – 80% owned by Sociedad Minera El Corazón SpA (a 100% subsidiary of Hot Chili Limited), and 20% owned by Compañía Minera del Pacífico S.A (CMP).

[9] The Cortadera deposit is controlled by a Chilean incorporated company Sociedad Minera La Frontera SpA (Frontera). Frontera is a subsidiary company – 100% owned by Sociedad Minera El Corazón SpA, which is a 100% subsidiary of Hot Chili Limited.

[10] The San Antonio deposit is controlled through Frontera (100% owned by Sociedad Minera El Corazón SpA, which is a 100% subsidiary of Hot Chili Limited) and Frontera is party to an option agreement pursuant to which it can earn a 100% interest in the property.

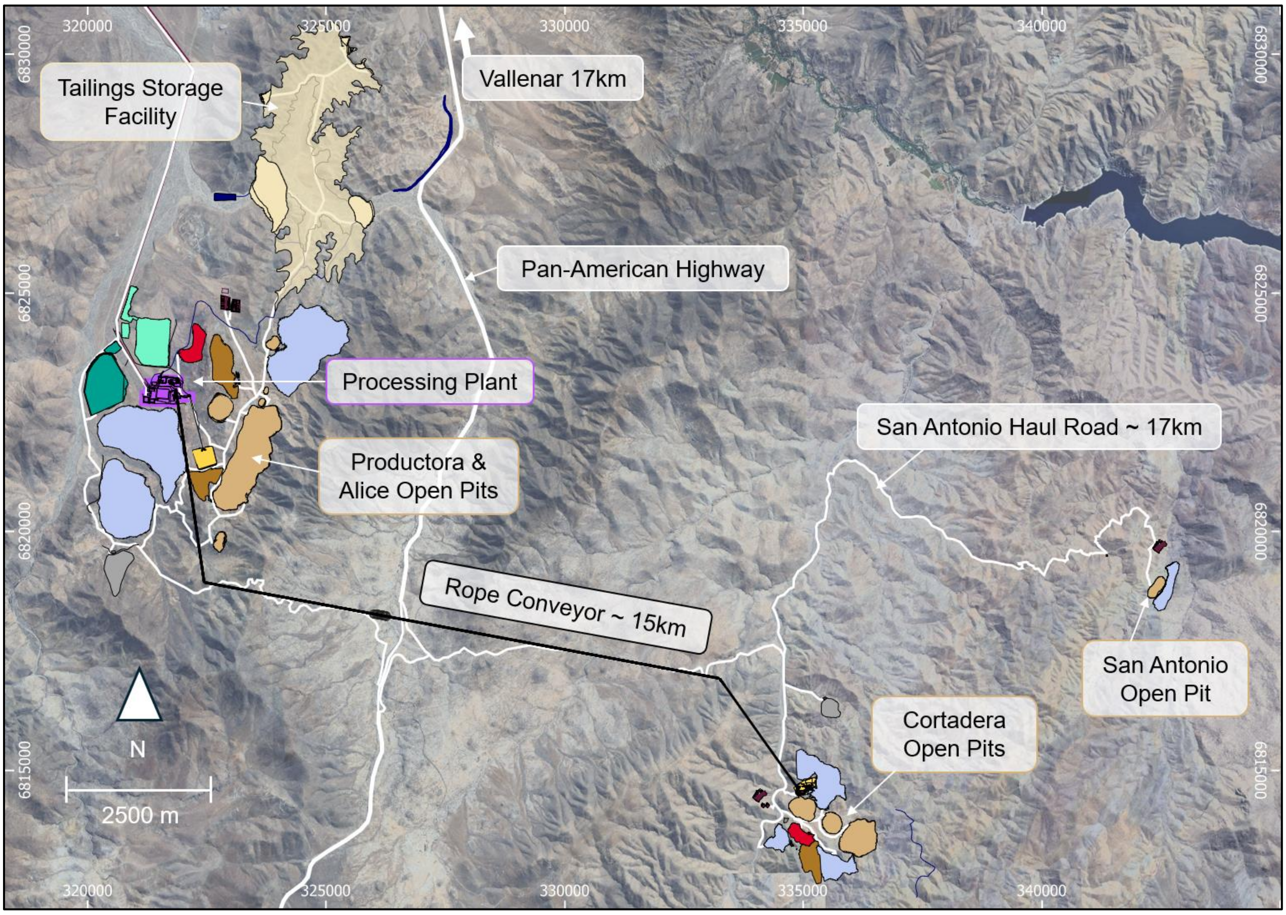
[11] The Mineral Reserve Estimate as of 27 March 2025 for the Costa Fuego Project was prepared by Anton von Wielligh, Fellow with the AUSIMM (FAUSIMM). Mr. von Wielligh fulfils the requirements to be a "Qualified Person" within the meaning of NI 43-101 and is the Competent Person under JORC for the Mineral Reserve.

[12] Hot Chili Limited is not aware of political, environmental, or other risks that could materially affect the potential development of the Mineral Reserves other than those that will be disclosed in a technical report for the PFS. A detailed list of Costa Fuego Project risks is also included in Section 25 of this Report.

1.14 Mining Methods

The Costa Fuego Project includes production from the Cortadera, Productora, Alice and San Antonio deposits. Mining at Cortadera includes three open pits, Cuerpo 1, 2 & 3 and a block cave operation under Cuerpo 3. As shown in Figure 1.10. The mines are envisaged to be operated by a mining contractor, with technical support and direction provided by HCH personnel.

Figure 1.10 : Costa Fuego Mining Areas, HCH (2025)



1.15 Open Pit

1.15.1 Open Pit Geotechnical Analysis

The open pit geotechnical design was conducted by GMT (2024). Existing diamond drillholes were utilised (47 for Productora/Alice, 103 for Cortadera, and 3 for San Antonio) with core reviewed and rock samples selected for laboratory testing. Additional surface geotechnical mapping was also completed, adding to significant existing mapping completed by HCH and previous owners.

The geotechnical analysis for the open pit addresses stability at the bench scale, and designs ensure the containment of inter-bench failures and mitigate rockfall risks. These configurations form the basis for assessing inter-ramp and overall pit wall stability. Pit design parameters are informed by updated geotechnical and structural models, with the application of slope design criteria alongside haul road width considerations determining the overall pit wall angle.

Geotechnical design zones are established based on structural and geotechnical domains, incorporating the dip and strike direction of the pit walls. These design zones summarize the bench-scale parameters, inter-ramp heights, and angles for each geotechnical and structural domain, while accounting for the dip and strike orientation of the pit walls.

GMT applied conservative assumptions to compensate for areas where information is insufficient or limited. These conservative measures, particularly in cases where data gaps exist, help to reduce uncertainty and ensure a more robust and reliable analysis. This judgement is supported by the previous experience dealing with similar situations, the numbers applied can be improved in the next study phase.

GMT defined geotechnical design zones for all pits and recommended inter-ramp angles. These angles were adjusted to account for pit ramps, leading to design inter-ramp angles from 36º to 55º at Productora and 43° to 57° for all other open pits.

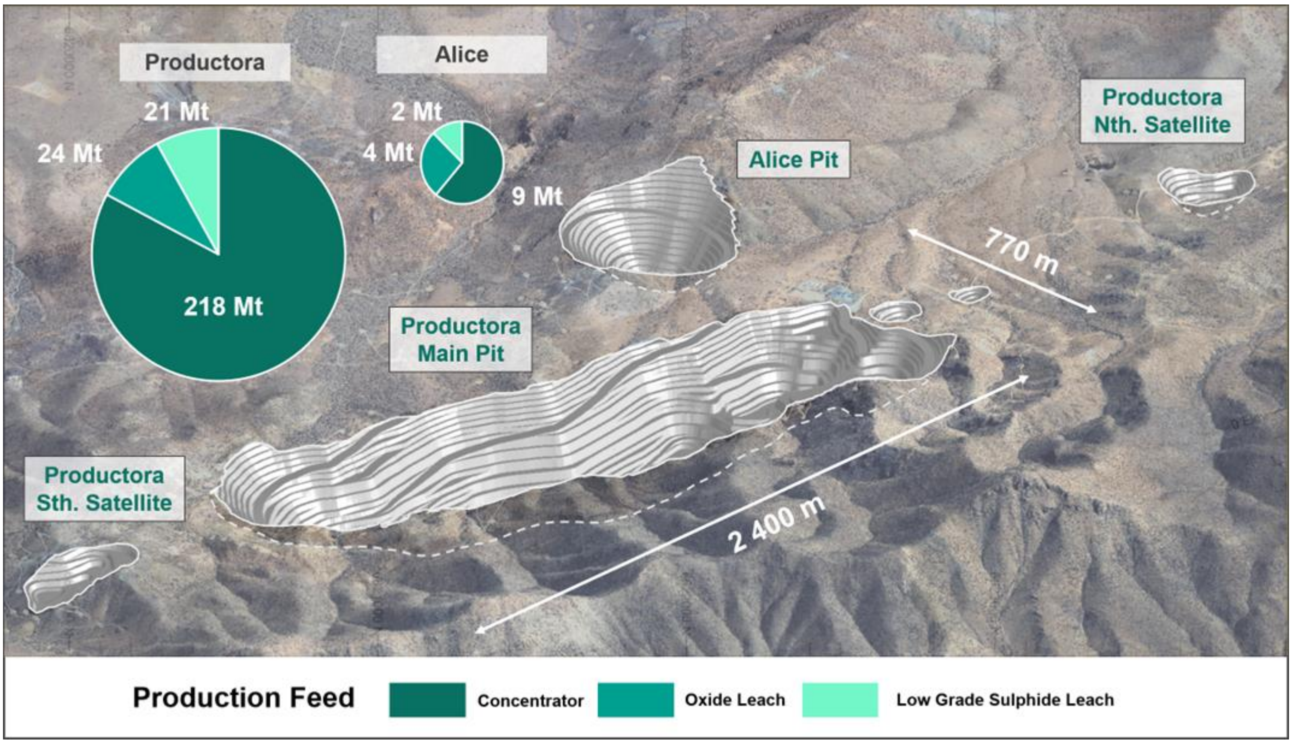
1.15.2 Open Pit Mine Design

The open pit mine designs were prepared for each pit based on outputs from optimisation work. The final open pit shells used to inform the mine design assumed the 100% revenue factor (RF 100%) economic areas as developed within Datamine NPVS.

Open pit mining operating costs were based upon a mining contractor scenario. This resulted in an initial mining cost estimate of approximately US$2.03/t for Productora and Cortadera, and US$2.23/t for Alice and San Antonio, which corresponds to the average mining cost of the referred mining contractor quote plus a preliminary assessment of HCH's mining personnel.

The resultant designs for Productora main pit (and satellite pits) and Alice pit, as shown in Figure 1.11.

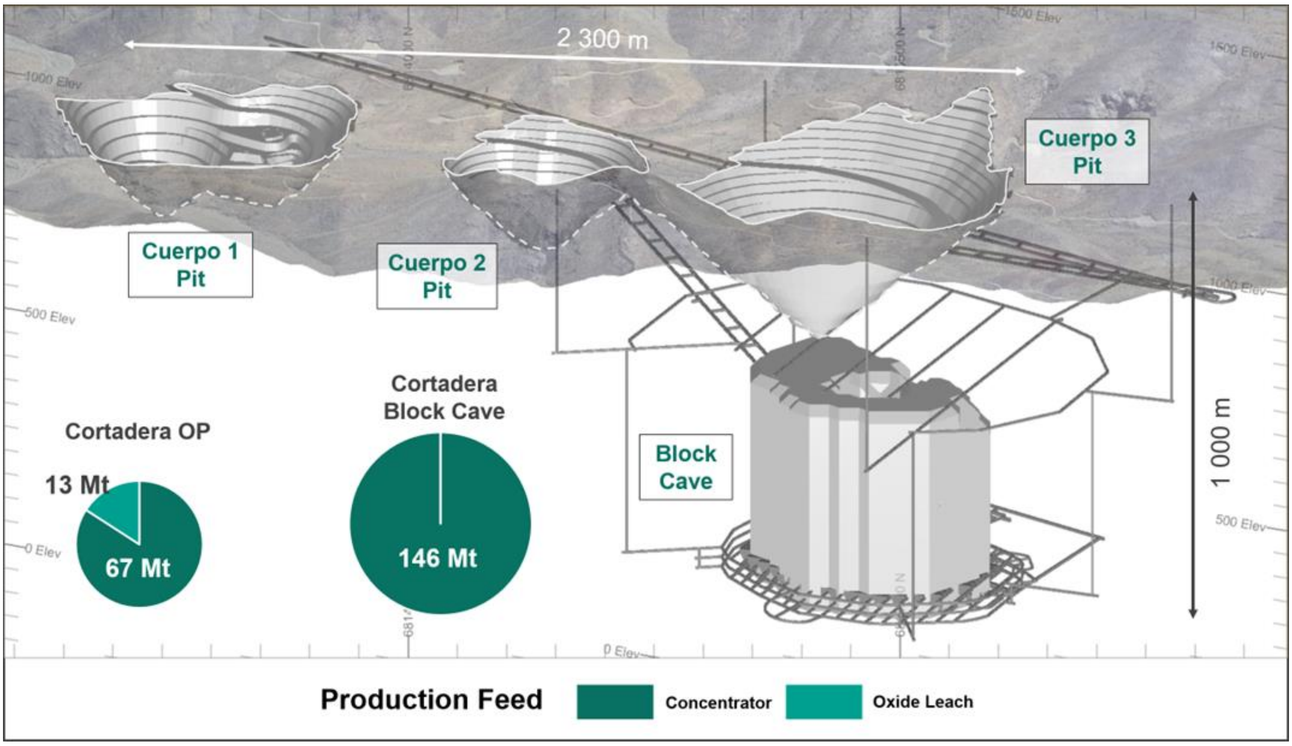
Figure 1.11 : Productora and Alice Open Pit Designs - View Looking North-West, HCH (2025)



Open pit mining at Cortadera deposit (located roughly 15 km east-southeast from Productora) comprises three separate pits, with the mining sequence commencing with the Cuerpo 1 pit, which has the largest volume of higher-grade, near-surface mineralisation. Cuerpo 2 and Cuerpo 3 are mined thereafter, in that order. Mineralisation extends below Cuerpo 3 and is amenable to mass mining methods (Figure 1.12).

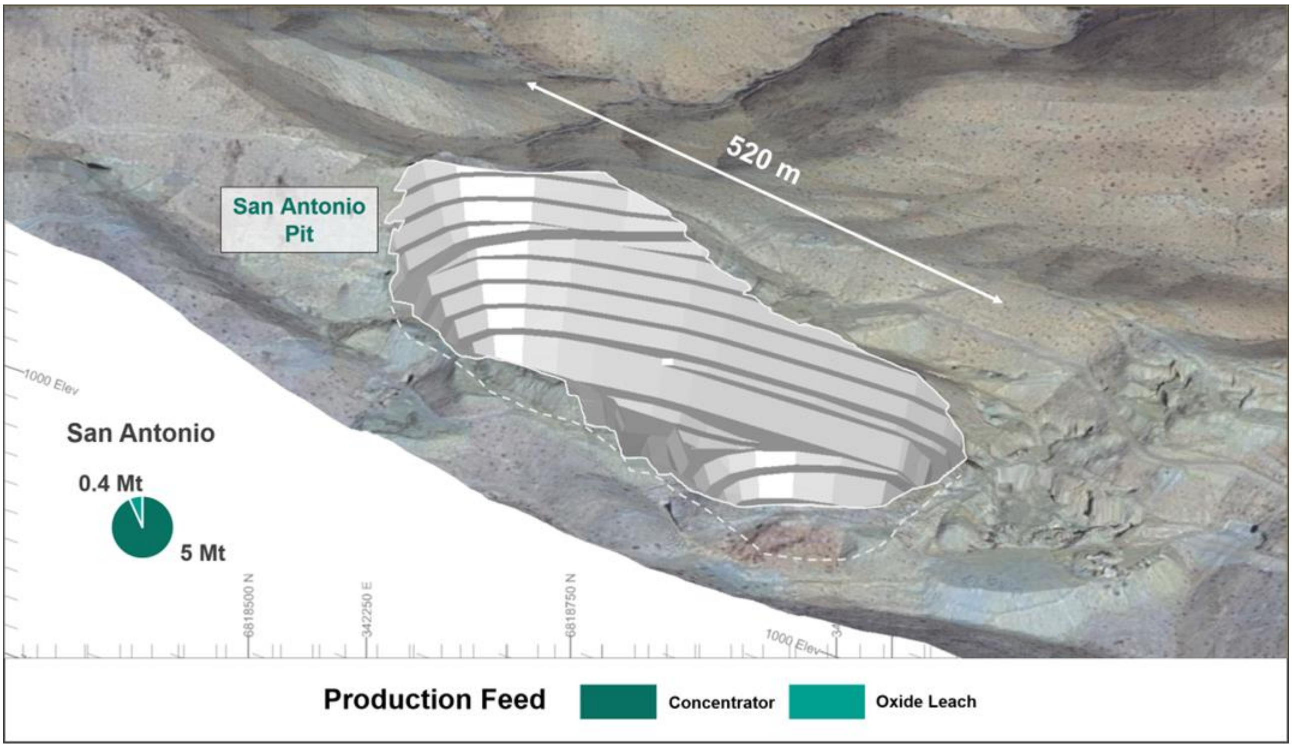
The Project utilised a block cave mining approach to assess the viability of material below the open pits.

Figure 1.12 : Cortadera Open Pits and Block Cave Designs – View Looking North-East, HCH (2025)



The San Antonio pit located roughly 8 km northeast of the Cortadera mining area. The San Antonio open pit design is shown in Figure 1.13.

Figure 1.13 : San Antonio Open Pit Design – View Looking North-East, HCH (2025)



Production feed includes Oxide leach, Low Grade (LG) sulphide leach and Sulphide concentrator material. The PFS developed both pit shells (shown in Section 16) and open pit designs (shown above in Figure 1.11 to Figure 1.13).

Open Pit mining envisions a conventional drill-blast-load-haul method with 15-metre-high benches. For the larger open pits (Cortadera and Productora Main) double benching (30 m) is considered for planning purposes in the upper waste rock benches. These pits will utilise larger diesel-driven bulk loading equipment, before reverting to the smaller mining equipment for the lower ore benches.

The mining study considered 220 t capacity rigid body mining trucks matched with a 60 t capacity bucket size hydraulic excavator or shovel (mainly for the bulk waste and some bulk ore mining benches). As the respective pits gets deeper, there will be a transition from the larger bulk earth moving equipment to smaller trucks and excavators for improved selectivity. The smaller equipment for the deeper mining benches at both Productora and Cortadera is typically 90 t trucks with matching 12 m^3 bucket size hydraulic excavators

Productora pit design includes five nested stages (Stages 1 to 5) to balance stripping requirements while satisfying the processing requirements. Satellite pits at Productora as well as pits at Alice and San Antonio are mined in a single phase.

1.16 Underground

1.16.1 Geotechnical Analysis – Underground

The geological drilling program at Cortadera has been extensive, comprising multiple campaigns with specific objectives, including exploration, resource definition, and geotechnical characterization. The data collected encompasses geotechnical parameters such as Rock Quality Designation (RQD), fracture frequency per metre (FF/m), joint orientation, and vein logging. Recent efforts, particularly the CORMET 2022 campaign, have focused on geotechnical drilling to support block caving studies, emphasizing detailed rock mass characterization.

Drilling at Cortadera intersected Level 220 of the underground mine through 31 drill holes, yielding valuable data on rock mass characteristics. These include RQD, fracture frequency, joint alteration, joint roughness, Geological Strength Index (GSI), Q' values and Rock Mass Rating (RMR). Laboratory tests on intact rock, such as unconfined compressive strength (UCS), Brazilian tensile strength (BTS), triaxial testing, and point load tests (PLT), have been conducted as part of several campaigns.

For geotechnical analysis, the primary geological units were categorized into geotechnical units based on their weathering states: oxide, transitional, and fresh. Each geotechnical unit exhibited distinct geotechnical properties.

Uniaxial Compressive Strength (UCS) values highlight the unique characteristics of each geotechnical unit, reflecting the data source. For oxide rocks, UCS values derived from hardness logging range between 10 – 50 MPa. Transitional rocks, primarily evaluated through hardness logging, show UCS values mostly between 50 – 100 MPa. However, less competent geologies within the transitional unit, such as skarns, early mineralisation porphyries, and post-mineralisation dykes display lower UCS values of 25 – 50 MPa. In contrast, UCS values for fresh units, determined through laboratory testing, typically range from 100 – 137 MPa, except for the post-mineralisation dykes, which exhibit a notably lower UCS of 47 MPa.

A Structural model for Geological structures that are relevant for Geotechnical Analysis have been developed for the Cortadera mine (Carrizo, 2024).

The in-situ stress was determined by the Australian Centre of Geomechanics (ACG) using deformation rate analysis (DRA) on five core samples (Dight, 2022). The stress measurements reveal a consistent trend of increasing stress with depth. The analysis also provides a mean stress orientation for numerical modelling, indicating that the major principal stress aligns SE/NW, while the minor principal stress is oriented sub-vertically.

1.16.1.1 Caveability assessment

The relationship between footprint dimensions and cave column heights, typically used as an initial indicator to assess if a cave can reach the desired height, was evaluated by GMT through benchmarking and empirical methods (GMT, 2024). For the proposed Cortadera underground mine, the column height is estimated to range from approximately 525 m to 820 m, with a footprint width set at 350 m. The H/B ratio varies between 1.5 and 2.5. Analysis indicates that mines with similar ratios have effectively employed rock mass preconditioning (RMPC) to facilitate cave propagation, especially in scenarios involving taller cave columns.

Typically, initial assessments in block caving projects employ Laubscher's nomogram to estimate the minimum dimensions needed to initiate and propagate the cave. Laubscher's (1990) caving chart is widely regarded as the empirical standard method for determining the necessary Hydraulic Radius to ensure caving. This empirical approach adjusts the Intact Rock Mass Rating (IRMR) and uses these modifications to compute the Mining Rock Mass Rating (MRMR) through Equation

For the case of the caveability of geotechnical units of Cuerpo 3: A_M=1 (no weathering), A_O=0.8 (three joint sets define the blocks, three faces inclined from the vertical, joint condition 16-30), A_S=1.0 (medium stress condition), A_T=1 (no blasting), A_W=1 (dry conditions).

1.16.1.2 Cave propagation analysis

A Mine-scale FLAC3D model was developed by GMT (GMT, 2024) to simulate the caving process and associated subsidence.

Cave initiation and propagation were assessed both with and without the incorporation of HF, the results show that the required hydraulic radius (HR) for cave initiation decreases by 47% when HF is applied. Furthermore, the time required for the cave to break through to the open-pit surface is reduced by three-quarters.

For cave propagation, the results indicate that in the last quarter of the first operational year, the cave ratio increased from 0.5 to 3.5 with the incorporation of HF, while maintaining the same HR. By the last quarter of the second operational year, the cave ratio further increased from 3.7 to 5.4 under the influence of HF.

Open-pit interaction is evaluated by analysing the development of the cave shape over time and its interaction with the open-pit bottom surface. The findings suggest that a crown pillar width of at least 300 m is necessary to maintain the integrity of the rock mass beneath the pit floor. The model also highlights the significant influence of geological structures on caving propagation.

Assessing caving-induced subsidence is critical for evaluating its effects on both underground infrastructure and surface mine facilities, as well as understanding potential environmental impacts from landscape changes. Numerical modelling defines subsidence zones to establish appropriate stand-off distances for all underground and surface critical infrastructure.

In the case of post-undercutting, early stages of fragmentation can be managed through drill and blast techniques, such as a high undercut. Fragmentation is estimated using the Block Cave Fragmentation (BCF) software. Primary fragmentation at the Cortadera mine is anticipated to yield uniform block sizes across most geotechnical domains, with P80 sizes ranging from 0.88 to 3.39 meters. Secondary fragmentation analysis indicates that block sizes decrease as the height of draw increases, facilitating manageable sizes for material handling.

Conventional ground support based on analysis by GMT using the empirical Q-System.

The Cuerpo 3 underground block cave was studied in two stages using Geovia PCBC™ caving software. In Stage 1 Geovia PCBC™ caving software's Footprint Finder module (PCBC-FF) was used as a global optimisation tool and determine the cave's optimal extraction level elevation range, approximate geometry and sensitivity to a range of typical caving design and schedule parameters.

The findings and conclusions from this strategic optimisation output stage are then used Stage 2 using Geovia PCBC™ caving software's detailed design and production scheduler (PCBC) to produce the final mine design and schedule. PCBC-FF provides a simplistic schedule and financial evaluation of the results, and these should be treated as relative values to compare between different scenarios rather than absolute values.

During Stage 2 of this study detailed mine design and scheduling and first principles cost modelling and financial evaluation was used to further refine these results and to determine the expected cave value.

The footprint results are not sensitive to the exact final elevation selected, showing less than 3% relative value difference over an 80 m vertical range (200 mRL to 280 mRL highlighted), and about 6% difference over 120 m vertical range (180 mRL to 300 mRL).

The Geotechnical recommended sequence starts in the north and progress south at an azimuth of 158 degrees). This reduces the maximum undercut face length to about 350 m, which has been shown as a manageable face length in various caves and remains constant for most of the undercut duration. It also allows the drawpoints to be orientated near perpendicular with the advancing undercut abutment front, avoiding any drives being orientated parallel with the abutment stress.

1.16.2 Mine Design

The PCBC (now Geovia's Long Term Planner software) results were used to define the extraction level and undercut level elevations, as well as the footprint maximum boundary. From this the perimeter drives and major infrastructure, such as the crushers, were positioned to allow stable geotechnical stand-off distances and to suit materials handling from the cave. Access from surface via declines, cave preconditioning development and primary ventilation circuits were then designed to suit the position of the major infrastructure, allowing for suitable stand-off to remain outside the cave influence zone.

Standard excavation profiles were utilised based on typical cave operations. The decision was made for this study to use the more common 17 t LHD size and drive sizes, especially common for South American operations. This also provides marginally larger and more stable pillars on the extraction level layout due to smaller tunnel requirements for a given drawpoint layout.

The ground conditions and expected stress environment at the planed Cortadera block cave allows for a post-undercutting sequence to be applied. For the planned Cortadera block cave, provision was made in the PFS for both an apex level and undercut level, and the undercut level tunnels are located on top of the drawbells to allows switching from a post-undercut to a pre-undercut sequence if required.

This will be further optimised in future studies and provides flexibility to reduce undercut development by removing the apex level and relocating the undercut tunnels above the major apex should it be required.
Given the geometry of this block cave footprint, both crushers were located on the northern side, with the undercut front moving away from the crushers towards the south. To compensate for crushers being located in the north only, turning bays have been designed in the northern side of each extraction drive between the first drawpoint and the crusher.

The design utilises a twin decline layout from the portal position near the Cuerpo 1 pit, keeping access near long term vertical ventilation rises to facilitate practical development. One decline will accommodate a conveyor belt to transport process feed to surface and serve as a second means of egress, while the other will serve as primary access for mobile plant, personnel, and material.

Cave optimisation results indicated that the block cave extraction level should be located on the 220 m RL level, and the undercut level and apex level located on the 240 m RL and 260 m RL levels respectively. The block cave footprint design makes allowance for a larger footprint by placing long term infrastructure outside the cave abutment zone and ensuring the designed development quantities allow expansion flexibility if the Mineral Resource increases. It also ensures sufficient time is allowed to establish the initial capital infrastructure, with a level of conservatism built-in.

The extraction level design feeds two crushers located on the northern side of the footprint. The current production rate is determined by the sulphide process plant capacity, which were sized for the open pit operations.

The proposed fixed plant construction schedule, with the civil and structural components of each flight of conveyor implemented in parallel with the conveyor decline face advance. This lags the face by three months to separate development and construction activities.

The crusher chambers for the crusher installation and the conveyor feed installation is expected to take six months to develop and support. These large chambers can be developed simultaneously, followed by the raisebore hole that connects the two excavations.

It is recommended that this scope should be well packaged and studied in the Feasibility Study (FS) stage, utilising a EPCM company with the relevant engineering disciplines to complete a first principles cost estimate, and develop a detailed construction schedule.

1.17 Waste Rock Dumps

Waste Rock Dumps were designed in 15 m lifts. Each lift is constructed at an approximate angle of repose of 37°. A 15 m setback between each lift maintains the overall angle at 25° to facilitate reclamation and long-term stability. A constant 35% swell factor, translating into a 2.0 t/m^3 loose density was assumed. All the dumps avoid the quebrada boundaries, except for the Cortadera Cuerpo pit valleys where there are approvals to mine and disturb that area.

Geochemical assessment was completed to assess for potential acid generation and neutralisation. Using these conditions, waste rock was categorised accordingly with the following delivery schedule:

- Stability analysis for waste dump and stockpile designs has considered the maximum capacity option of each structure, under dry substratum.
- The stability of waste dumps and stockpiles was assessed using Limit Equilibrium Analysis, following the acceptability criteria.
- The results indicate that the designs for waste dumps and stockpiles meet acceptability criteria during both the operational and post-closure phases.

The main ROM pad is strategically located near the Productora and Alice open pits, where the primary ROM crusher will feed a transfer (overland) conveyor belt to the Productora plant comminution circuit. A ROM pad with a primary crusher is also designed for Cortadera. The ROM pad at Cortadera is optimally located to connect underground block cave ore via the Cortadera surface primary crusher to the RopeCon loading point.

1.18 Mine Schedule

1.18.1 Open pit Schedule

Scheduling was conducted using the strategic scheduling software Minemax, a Strategic mine scheduling tool used to identify mathematically optimal solutions for scheduling models that define available source material, processing options, and the constraints governing material access and processing.
Only economic material with an Indicated resource classification is considered for processing routes. The schedule results inform final capital and operating cost estimates, using interim cost assumptions that are validated against the financial model.
The schedule model is configured with material pathways dictated by material characteristics such as resource classification and economic viability. The available pathways include:

1. To Concentrator (Primary)
2. To Heap Leach
3. To Concentrator (Secondary – Pre-Crush then fed to Concentrator)
4. To Dump Leach
5. To Stockpile
6. To Waste Dump

1.18.2 Underground Mine Schedule

The underground development scheduling/planning used Datamine Studio UG and EPS software whilst merging schedules with the Geovia Long Term Planner ("PCBC") block cave/cave draw schedules. The mine development and underground infrastructure scheduling, inclusive of the proposed underground materials handling system, contains multiple activities and design aspects which, while complex to plan, are incorporated in sufficient technical design and scheduling detail within Datamine UG and EPS.

The development schedule and the key dates/times when specific underground design locations are reached were scheduled and tested with the mine planning scheduler software, however, these activities were also built in an activity simulator (SimMine) to further test impacts of significant unplanned events and delays on these specific locations becoming available in the underground development timelines. The slower/later date (Studio UG/EPS vs SimMine) of reaching the undercut, underground crusher chambers and production level positions were assumed and adopted in the final development schedule.

Draw bells were scheduled to lead the advancing undercut rings by one to two draw bells. Draw point tunnel development in turn lead the draw bell development by at least one to two draw points, allowing sufficient time for draw point development and the installation of arches and concrete roadways prior to commencing with draw bell mining.

Production ramp-up is a combination of adding new drawpoints to the active production area as the undercutting process continues, and existing drawpoints increasing draw rates as the cave matures. The maximum production rate was constrained in the final schedules to 85% of this maximum ramp-up rate to account for overall system availability and total system throughput capability. This should be confirmed through further simulation work. The Base Case schedule ramps up to the target maximum rate over a period of about four years (16 quarters). Restricting the ramp-up rate to 50% of maximum capacity extends this to approximately seven years.

1.19 Recovery Methods

1.19.1 General

The Project will produce saleable flotation concentrates of copper and molybdenum and will also produce copper cathode from the heap and dump leaching operations

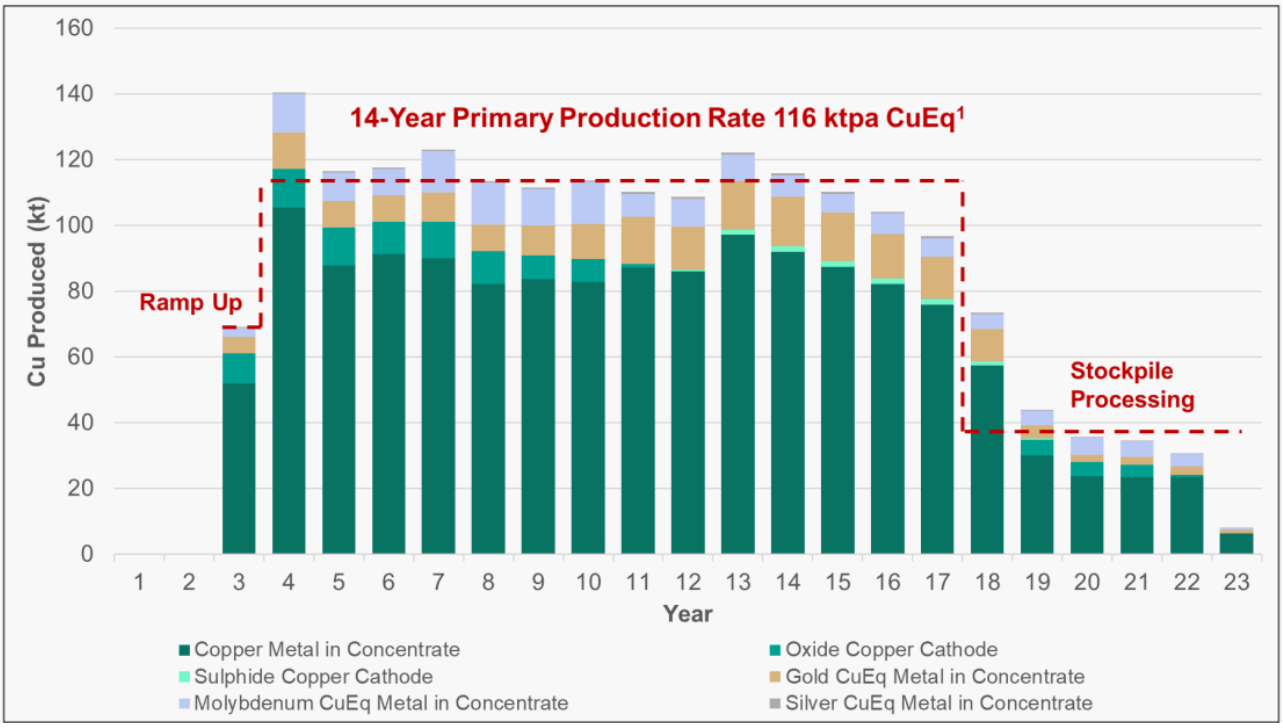
Payable elements for the purposes of the PFS are copper, gold and silver in copper concentrate, copper in cathode and molybdenum in the molybdenum concentrate.

The proposed processing facilities include a concentrator, a heap leaching preparation and stacking and a dump leaching area. Located at Productora, the sulphide concentrator is the centrepiece of the facility and is designed to process nominally 20.7 Mt/a of sulphide process feed and is capable of averaging 21.7 Mtpa across the project life. Concentrator capacity will vary by deposit based on comminution properties and feed crushing approaches.

The deposits will also produce 4 Mt/a of oxide feed to be processed via a crushing-agglomeration-heap leach circuit coupled with a SX/EW plant producing up to 15 kt/a of copper cathode. Low-grade sulphide ores from Productora and Alice will be treated in a dump leach which receives 3.6 Mt/a of feed. The dump leach cathode production is included in the 15 kt/a mentioned above.

The Project has a processing ramp-up time of one year for both the concentrator and oxide heap leach.
Annual metal production across the three processing streams averages 95 kt Cu, 48 koz Au, 158 koz Ag and 4.4 Mlb Mo for the primary production period of the 14 years post commissioning. LOM annual metal production across the 20-year processing life averages 74kt Cu, 37 koz Au, 128 koz Ag and 3.4 Mlb Mo. Figure 1.14 shows a breakdown of the yearly metal production over the life of the Project.

Figure 1.14 : Yearly Copper-Equivalent Production Over Life-Of-Mine[1]

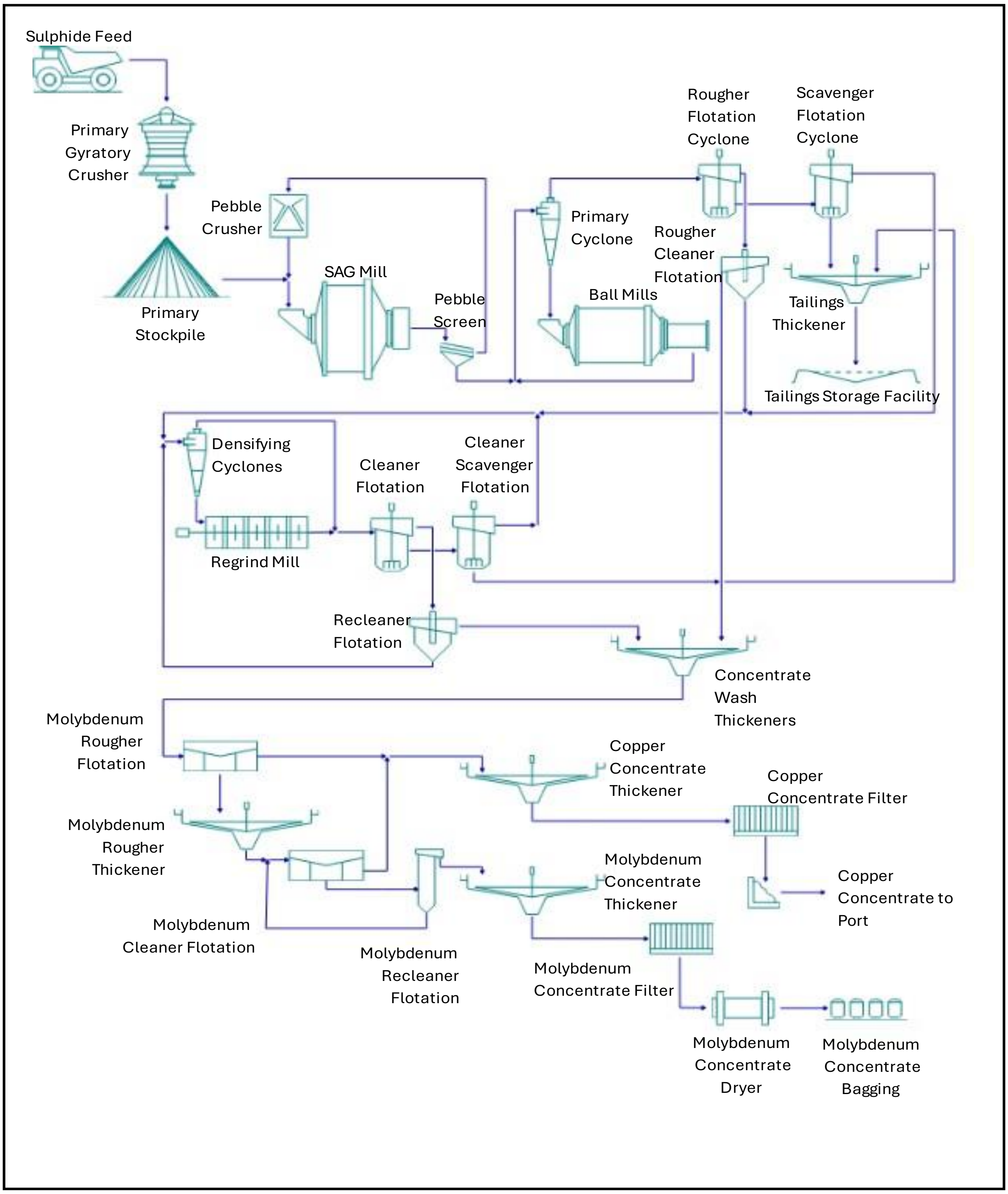


[1] The copper-equivalent (CuEq) annual production rate was based on the combined processing feed (across all sources) and used long-term commodity prices of: Copper US$4.30/lb, Gold US$2,280/oz, Molybdenum US$20/lb, and Silver US$28/oz; and estimated metallurgical recoveries for the production feed to the following processes: Concentrator (86% Cu, 54% Au, 37% Ag, 70% Mo), Oxide Leach (65% Cu only), & Low-grade Sulphide Leach (39% Cu only).

1.19.2 Sulphide Concentrator

Process flowsheet for the sulphide flotation concentrator is shown in Figure 1.15.

Figure 1.15 : Sulphide Process Plant Flowsheet



The major comminution equipment for the Project is listed in Table 1.10.

Table 1.10: Comminution Circuit - Major Equipment Selections

Element	Units	Model	Installed Power (MW)	Set (mm)	Diameter (ft)	Length (ft)	Diameter (m)	Length (m)
Productora Sulphide Primary Crusher	1	Superior 60-89 MK III Gyratory	0.60	175 (open side)				
Cortadera Surface Primary Crusher	1	FLS KB 130-75 DM Pro Jaw Gyratory	0.65	152 (open side)				
Cortadera Underground Primary Crusher	2	Superior 54-75 MKIII-UG	0.6	152 (open side)				
SAG Mill	1	28 MW GMD	28		40	28	12.2	8.53
Pebble Crusher	1	Metso MP 1250	0.94	13 (closed side)				
Ball Mills	2	19 MW Twin Pinion	38		28	40	8.53	12.2

The FLS KB (Jaw Gyratory) unit shown is the double-sided-mouth version (DM) suitable for direct ROM tipping installations.

The comminution circuit throughput has been estimated in mine planning by geometallurgical modelling. Equations were developed to predict two main ore properties (DWi and BWi) on a block basis and the concentrator throughput was then predicted by identifying the comminution bottleneck, SAG or ball mill, when processing that block. The averaged outcomes of the throughput predictions for the main orebodies were determined by the mine planning process and are summarised in Table 1.11.

Table 1.11 : Average Estimated Throughput for SAG and Ball Mills and Grinding Circuit by Deposit

Item	Productora	Alice	Cortadera OP	Cortadera UG	San Antonio
TputBM (t/h)	3,880	3,366	3,744	3,419	2,546
TputSAG (t/h)	2,631	2,810	2,961	2,348	3,141
Tput (t/h)	**2,464**	**2,810**	**2,840**	**2,348**	**2,546**
Mt/a	19.7	22.5	22.7	18.8	20.4

Flotation equipment was selected based on testwork residence times and kinetic response for rougher and scavenger flotation and on the flotation timings used in locked cycle testing for the remainder of the flowsheet.

Regrind milling was selected with reference to the IsaMill online calculator provided by Glencore and with the application of significant conservatism.

Process water quality requirements for flotation have led to three concentrator water systems. The largest process water system is based on seawater and supplies both grinding and Cu/Mo flotation. A smaller system is based around low-saline water and is used to wash most of the chlorides out of the sulphide concentrate. The final system is based on RO water and ensures that the final concentrates have acceptable chloride levels and that Molybdenum cleaning can proceed with high separation efficiency.

Concentrate thickeners and filters were selected based on vendor testwork. Tailings thickening requirements were determined from vendor testwork.

Samples of tailings materials from locked cycle tests have been subjected to targeted testing to allow the design of the TSF.

1.19.3 Leaching

1.19.3.1 Heap Leaching

The heap leaching circuit uses primary, secondary and tertiary crushing, to produce material sized at -15mm to feed the agglomeration process, where sulfuric acid and concentrated brine (as source of NaCl salt) from Electrodialysis process, are added. The agglomerated heap leach feed is then conveyed for constructing the heap pads, where the material is allowed to rest. After the agglomeration and resting stage, intermittent irrigation and resting cycles occur. Cycles of one week of irrigation followed by three weeks of resting continue for the entire leaching timeframe, which spans 240 days. This way, the resulting pregnant leach solution (PLS) reaches 2.0 g/L Cu and 120 g/L Cl, suitable for feeding the SX-EW plant.

Once the piles begin to decrease their Cu concentration, a second cycle of irrigation is initiated with refining solution (Raffinate), which is the spent solution from the SX plant. Since the leaching cycle is established in four weeks cycles, the heaps are organized in four sectors so that there is a continuous flow of irrigation and continuous production of PLS to feed SX-EW. The purpose of the SX plant is to purify the PLS solution coming from the heap leach pads. This solution contains other ions, in addition to Cu^{2+}. Using solvents, which are reagents with the ability to selectively capture Cu and discard other ions, it is then possible to transfer the copper to an electrolyte solution to achieve high Cu concentration of 50 g/L. Washing stages are included throughout the process for salt elimination and temperature increase to 40 C° (by means of a heat exchanger) feeding through to the EW stage, which produces copper cathodes for commercialization, after approximately seven days of deposition cycle.

Process block diagram for heap leaching is shown in Figure 1.16 and heap leaching design basis are presented in Table 1.12.

Figure 1.16 : Process Block Diagram for Heap Leaching

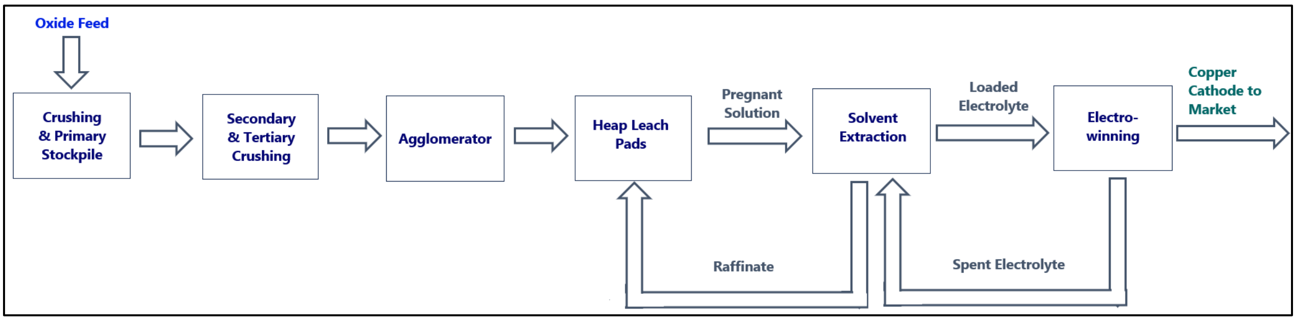


Table 1.12: Heap Leaching Design Basis

Item	Unit	Quantity
Annual Heap Leaching Feed	t/y	4,000,000
Heap Leaching Feed Topsize	mm	15
Design Factor	1/1	1.15
Cu Head Grade (Design)	%	0.40
Cu Recovery (Design)	%	75.0
Heap Leaching Cycle	d	240
Irrigation percentage	%	0.25
Rest time percentage	%	0.75
Acid Consumption	kg/t	20
Salt Consumption (as concentrated brine 200 g/l NaCl)	kg/t	15
Heap Leaching Lift Height	m	6
Heap Leaching Utilisation	%	94
Cu Cathode Production	t/y	12,000
SX-EW PLS Flow	m^3/h	750

1.19.3.2 Dump Leaching

According to the PFS Mine schedule there will be three separate operating periods for heap and dump leaching; this means that these processes will operate in stand-alone mode and therefore, heap leach facilities will be available during the dump leaching operation period. This approach allowed the design to consider the use of pumps designed for the heap leaching to be used for dump leaching, considering the recirculation of raffinate and ILS. Proximity of the installations also favored this option.

Fresh low-grade sulphide ROM ore from Productora and Alice open pits is sent to dump leaching, either direct from the open pits or from a dump leach ROM stockpile. Once the truckload is dumped the ore is irrigated with sulphuric acid and concentrated brine (as source of NaCl salt) obtained from Electrodialysis process initiating the conditioning or curing process. The ore is then left to rest using air injected by blowers for an energetic oxidation process.

The dump leaching cycle is two years and two dump leach facilities should operate simultaneously every year. Considering a 4-week cycle of irrigation and rest, and the 2-year leaching period definition, the annual processing of low-grade ore has been divided into two sectors or modules implemented biannually so that four modules per year are available for processing. This allows for the implementation of irrigation every four weeks.

As a design criterion, the PLS feed flow to the SX-EW plant of 750 m^3/h (heap leaching) is decreased by half. Consequently, the raffinate discharge flow from the SX-EW plant is established as 375 m^3/h. Copper cathodes are produced from the SX-EW plant for commercialization.

Process block diagram for dump leaching is shown in Figure 1.17 and dump leaching design basis are presented in Table 1.13

Figure 1.17 : Process Block Diagram for Dump Leaching

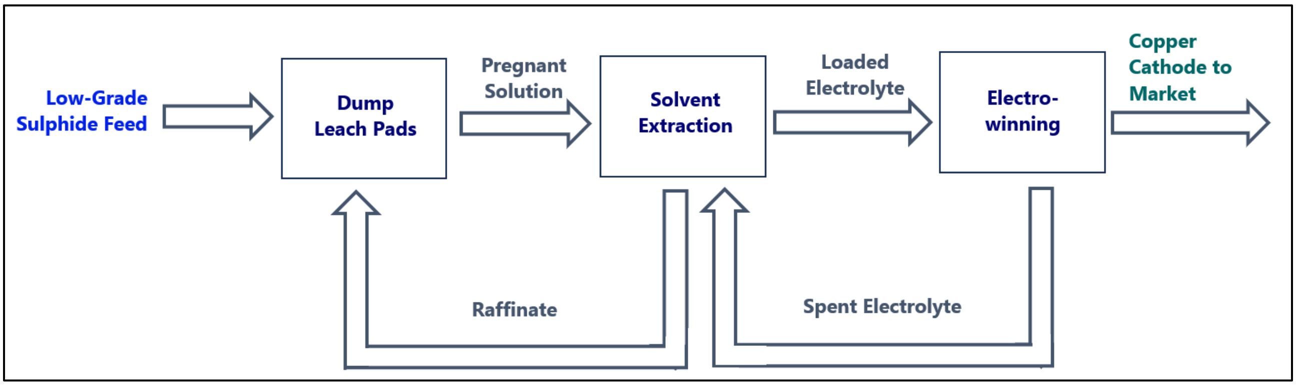


Table 1.13: Dump Leaching Design Basis

Item	Unit	Quantity
Annual Dump Leaching Feed	t/yr	3,600,000
Dump Leaching Particles Size	-	ROM
Design Factor	1/1	1.15
Cu Head Grade (Design)	%	0.16
Cu Recovery (Design)	%	40.0
Dump Leaching Cycle	d	730
Irrigation percentage	%	0.25
Rest time percentage	%	0.75
Dump Leaching Lift Height	m	20
Acid Consumption	kg/t	30 (design) 15-25 (consumption)
Salt Consumption (as concentrated brine 200 g/l NaCl)	kg/t	15
Dump Leaching Utilisation (trucks operation)	%	70
SX-EW PLS Flow	m^3/h	375

1.20 Project Infrastructure

The Project would be able to utilise existing infrastructure and services in the Vallenar/Huasco region. The township of Vallenar (17 km from the mine site) would provide accommodation and services to support the Project.

Other general infrastructure around Vallenar includes the following:

- Aerodrome (3 km south of Vallenar)
- Pan American Highway (5 km east of Productora mine site)
- Access roads from the Pan American Highway (C486 or Algarrobo route) and from Maitencillo (C472) would provide partial access to the mine site
- Main road (C-46) from Vallenar to Huasco
- A 220 kV electrical substation located at Maitencillo, connected to the Chilean electrical grid

1.20.1 Site Development

The proposed plant would be sited at Productora to the west of both the Alice and Productora pits. The ROM pad and primary crusher would be located adjacent to the main haul road.

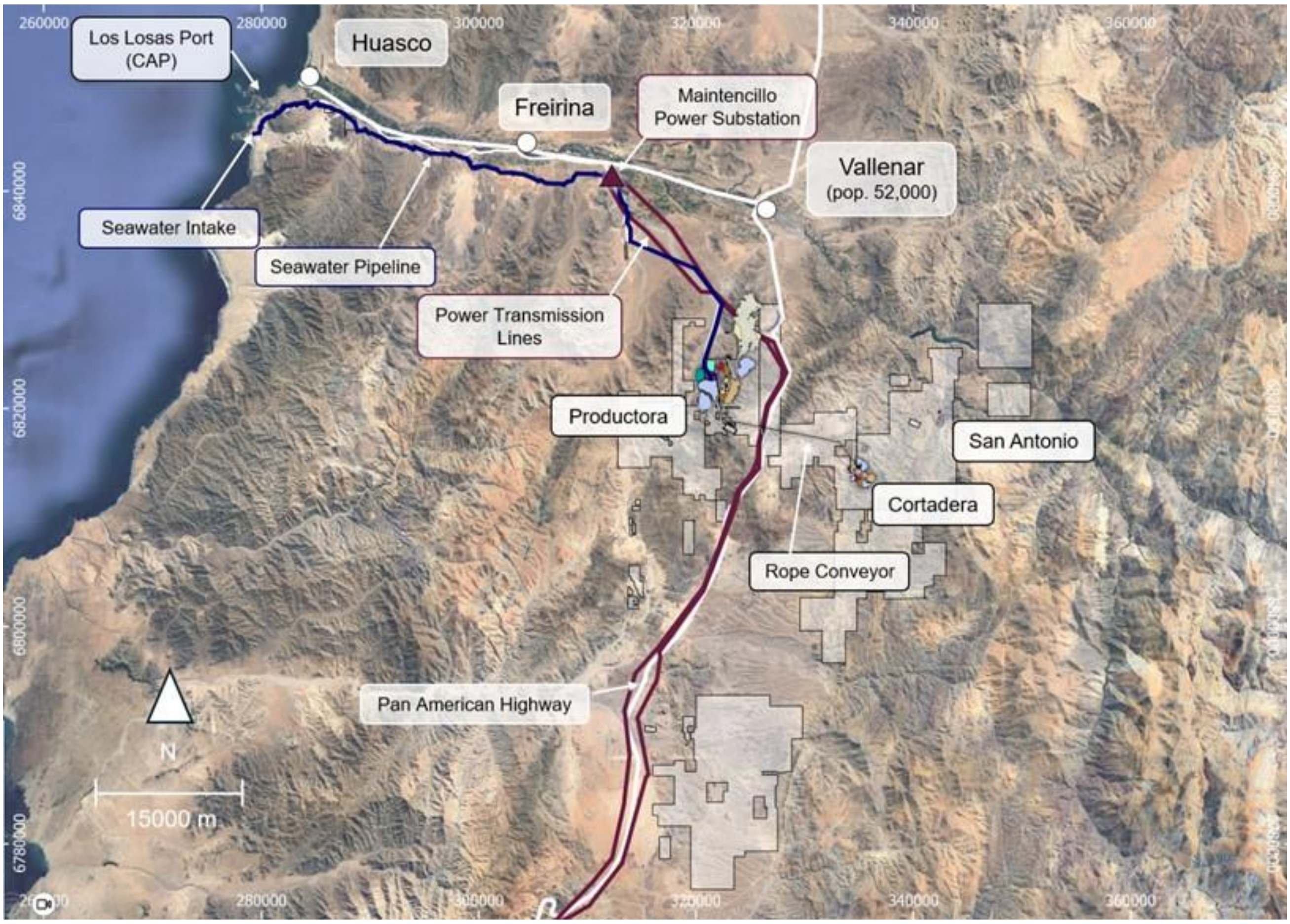
The majority of site buildings would be located in the area adjacent to the sulphide process plant, including the main administration building, main warehouse and change rooms. Smaller support facilities would be located at the copper oxide plant. An area for establishment of mining contractor's facilities has been provided to the north of the Alice pit and the Productora pit.

The key infrastructure required for the Project includes the following:

- Power supply and distribution
- Seawater transfer system
- Rope Conveyor
- Tailings storage facility
- Port facility

Figure 1.18 depicts the existing and planned infrastructure.

Figure 1.18 : Costa Fuego Project Planned and Existing Regional Infrastructure, HCH (2025)



1.20.2 Power Supply and Distribution

Power supply from Maitencillo to Productora will be via a 220 kV overhead powerline. The ± 25 km route for the 220 kV overhead line is nominal and will be detailed in further stages of Project development.

The maximum demand is estimated at 149 MVA for the Project, and the Maitencillo substation represents the nearest location with sufficient capacity to provide power to the Project. The substation has numerous 220 kV power lines which connect to the country's transmission and generation network and has space available for a non-redundant 220 kV point of supply.

A design assumption for the Project is that the Maitencillo substation would not require significant changes to provide the point of supply and that a spare gas insulated switchgear (GIS) circuit breaker will be available for connection at Maitencillo.

At the Productora substation and switchyard, the 220 kV supply is stepped down to 23 kV via an 80 MVA transformer for Cortadera and a 140 MVA transformer for Productora. The Productora substation design includes the main transformers, outdoor 220 kV SF6 switchgear and 23 kV air insulated switchgear. The 23 kV supply is connected to the 23 kV Substation via a 400 mm2 cable; power from the 23 kV substation will be distributed to each of the load centres via switchboard and cabling.

Power would be provided to Cortadera via a power supply line in the infrastructure corridor that houses the RopeCon and other infrastructure.

1.20.3 Water Supply

Seawater supply, as the raw water source for the Project, is transferred from the coast (south of the port of Huasco) to seawater storage ponds located at Productora. The pipeline from coast to the process plant site is approximately 62 km in length.

The seawater transfer system comprises one intake pump station, one seawater transfer pumping station and the above-ground transfer pipeline. The design volumetric capacity of the seawater system is 500 l/s.

- Power supply for the seawater intake pump station and transfer pump station will be via a 33 kV high voltage power line from the main Productora site.
- The seawater transfer system will consist of the following:
- Seawater intake pipeline
- Seawater intake pit and pump station
- Seawater transfer pump stations
- Transfer pump station emergency storage pond and tank
- Seawater transfer pipeline
- Sulphide plant and oxide plant seawater storage ponds.

A reverse osmosis (RO) water plant treats seawater to provide low chloride content water, utilised in both the mining and process plant operations. RO water is further treated through a demineralisation plant, for use within the SX-EW facility of the Oxide process plant. The RO plant will also supply a treatment plant to produce potable quality water for use across the Project.

A separate water treatment facility is included in the design for the treatment of TSF seepage, ARD and mine contact waters.

1.20.4 Rope Conveyor

Sulphide and oxide processing material is planned to be transported 15km, from Cortadera to the Productora site via a rope conveyor.

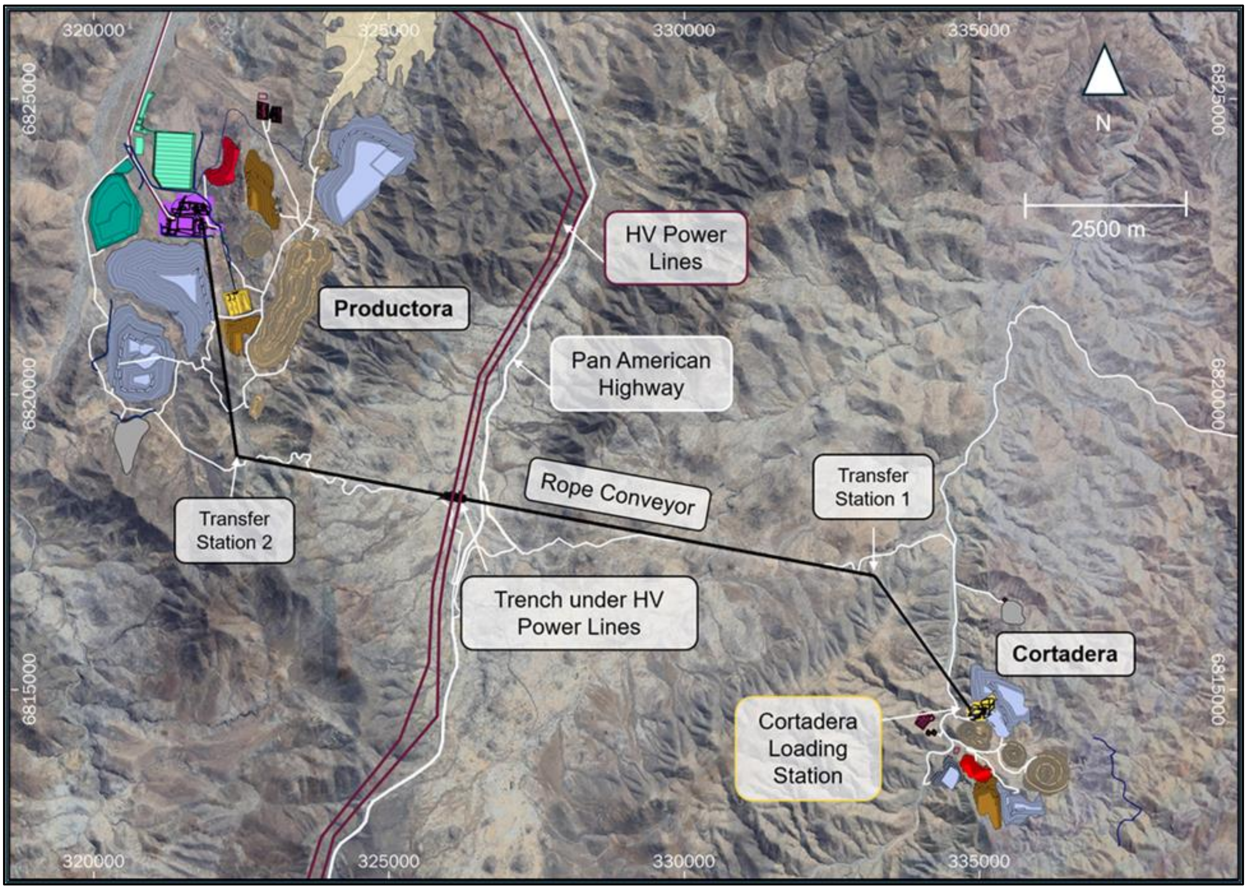
Dopplemayr completed a PFS level Engineering Study to support the design and construction of the rope conveyor (RopeCon). The RopeCon has been designed with a nominal capacity of 25 Mtpa.

The PFS technical deliverables included:

- Technical elaboration of the route, general arrangement of stations and towers, and specifications for the belt, track rope frame and motor rating

- Foundation loads
- Capital and operating cost estimates.

Figure 1.19 : Rope Conveyor Location, HCH (2025)



The material specifications for each section of the rope conveyor design are provided in Table 1.14.

- Section 1 – Cortadera Loading Station to Transfer Station 1 is 2.4 km long and includes an elevation difference of 74 m.
- Section 2 – Transfer Station 1 to Transfer Station 2 is 11 km and includes an elevation difference of 66m. This section is designed as a hybrid RopeCon Low-Line Structure so it can cross highways, secondary roads, and high voltage powerlines.
- Section 3 – Transfer Station 2 to Distribution Conveyor at Productora is 3.8 km long and includes and elevation change of 265 m.
- Section 4 - Distribution Conveyor to separate Sulphides and Oxides Stockpiles at Productora is 12 m long and includes an elevation change of 13m. The Section 4 conveyor is planned to be supported on the track ropes of RopeCon Section 3, allowing discharge onto both ore stockpiles.

Table 1.14: Technical Data Design Criteria.

Criteria	Units	Section 1	Section 2	Section 3	Section 4
Horizontal conveying length	m	2,411	10,995	3,875	119
Difference in elevation	m	74	-66	-265	-13
Position of drive		Both sides	Both sides	Loading	Loading
Belt width	mm	1,050	1,050	1,050	1,200

The RopeCon transports the material on a continuous cross-reinforced flat textile or steel cord belt with corrugated side walls. The corrugated side walls are either bonded or vulcanized to the belt to a height determined by the material to be transported. The conveyor belt performs a haulage function and is driven by a drive and gearbox arrangement and is equipped with two independent mechanical braking systems.

Six fully locked steel wire track ropes form the line structure of the RopeCon system. Track rope frames are mounted at regular intervals to keep the six track ropes in alignment. The track ropes which form the line structure on which the running wheels of the conveyor belt travel are tightly tensioned. The ropes are anchored via a separate concrete block or the track ropes can be anchored in a station building designed for the purpose.

The intended route underwent multiple iterations, taking into consideration environmental sensitivities identified during the winter and spring environmental baseline surveys, interactions with high voltage powerlines and the Pan American highway, topography and anticipated stockpile dimensions.

The design includes the following when considering a RopeCon design to cross the Pan American Highway:

- Ground clearance with Ruta 5 highway of 10 meters.
- Clearance between RopeCon and the auxiliary road running in parallel to Ruta 5 of 7 meters

1.20.5 Tailings

The proposed facility is for valley storage with three multi-zoned embankments and, once mined out, in pit disposal within the Main Productora Pit. The Tailings Storage Facility (TSF) is located approximately 5.4 km northeast of the Project plant site (centroid to centroid) at Productora.

A preliminary seismic hazard assessment confirmed that the Project area is located in an area of high seismic activity. An in-country seismic specialist has been engaged to conduct a site-specific seismic hazard assessment.

The topography of the proposed TSF area is steep and hilly and the perimeter is characterised by steep and undulating ridges. The basin is in a valley that is bounded by ridges to the north and south and falls in the westerly direction towards the main embankment. The main embankment is located at the west of the facility. A second embankment will be constructed on the eastern side of the valley, and four saddle embankments will be constructed to the North (3) and West (1) of the facility.
The TSF is designed to accommodate 386 Mt of tailings within a conventional TSF and 114 Mt within the Productora Open Pit, for a total of 500 Mt.

The TSF will comprise a cross-valley storage facility formed by multi-zoned earth fill embankments, comprising a total footprint area (including the basin area) of approximately 276 ha for the Stage 1 TSF, increasing to

889 ha for the final TSF and 88.5 ha within the Productora Open Pit. A total of 18 raises of the TSF will be required over the life of mine. Tailings will be discharged into the TSF by sub-aerial deposition methods Stage 1 will provide storage for 18 months (31.1 Mt of dry tailings) and have a maximum embankment height of 38.3 m (RL 582.3 m). Subsequent stages will be raised annually to suit storage requirements. Downstream raise construction methods will be utilised for all TSF embankment raises.

Tailings will be delivered to the TSF via the tailing's delivery pipeline contained in a HDPE liner trench, with an access road constructed adjacent to allow for daily inspection by process personnel. The pipeline will be fitted with telemetry and automatic shut offs to prevent continued discharge in the event of pipeline failure.

Knight Piésold previously tested a sample of representative tailings in 2015. During the PFS, two composites were created, one at a P_{80} grind size of 106 µm and the other at 150 µm. Additional physical tailing testwork is currently underway on samples generated for each deposit. These will be incorporated into the next design phase.

A suite of static geochemical testing was conducted on the 2015 tailings samples, which indicated that the tailings were potentially acid forming (PAF), had a moderate number of elemental enrichments and that the supernatant water quality was poor (partially due to seawater used in processing).

Further geochemical characterisation of the various tailings expected from the various deposit is underway. These will be reviewed as part of the next design phase, which will inform the tailings management requirements and whether further geochemical testing is required. Geochemical testing on operational samples will also be continued throughout operations and rehabilitation trials conducted to inform the final closure requirements. At the designed 68% solids, the tailings are expected to exhibit highly thickened behaviour.

A geotechnical investigation of the TSF was conducted, which comprised diamond drilling, test pitting, and a MASW survey. The site typically comprises a compact granular soil layer of variable thickness, underlain by residual soil/extremely weathered rock (granular soil), transitioning to rock. Towards the valley base, Alluvium was encountered to depths of between approximately 15 m and 30 m in some locations. Further geotechnical investigations are planned for the next design phase to expand on these findings.

A groundwater investigation within the TSF footprint was conducted to collect In-situ water quality parameters, installation and testing of groundwater bores. The depth to bedrock ranges from 12 m to 80 m, and the depth to groundwater (where intersected) ranges from 18 m to 66 m. The depth of rock within the eastern embankment footprint is between 12 m and 26 m, and the depth to groundwater is between 28 m to 42 m based on historical drilling.

A basin liner comprising a HDPE liner is proposed beneath all embankments, over alluvium within the basin, and along all supernatant ponds and channels. Options for different configurations of basin linings have been investigated.
The TSF design incorporates an underdrainage system to reduce head pressure near the embankments, reduce seepage, increase tailings densities, and improve the geotechnical stability of the embankments. The underdrainage system comprises a network of finger and collector drains. The underdrainage system drains by gravity to a collection sump located at the lowest point in the TSF basin.

An operational emergency spillway will not be provided during operation as the facility will be designed to attenuate the probable maximum flood (PMF), which is in addition to the capacity of the target operational pond. The facility's excess stormwater capacity is significant, and the containment of the PMF is considered practicable with minimal additional freeboard being applied.

A downstream seepage collection system will be installed downstream of the TSF embankment to capture and return seepage to the plant site from the TSF. The abstraction system will comprise a series of abstraction bores located downstream of the embankments to capture seepage.
Historical groundwater depth measurements indicate that the stabilised water depth in the Productora Pit is around 20 to 80m below the surface (RL 705 m to RL 780 m). Three geological units were and are characterised by fractured rock and fresh bedrock. These units have moderate ($1x10^{-7}$ to $1x10^{-8}$ /s) to low permeabilities ($1x10^{-8}$ m/s) and low groundwater capacity. The structural model indicates that many faults intercept the Productora pit, and these will control the hydrogeological conditions around the pit.

The climate at the site is arid, with an average annual rainfall of about 50 mm, water storage capacity was allowed for 24-hour and 72-hour storm events were estimated based on meteorological data from Santa Juana. As the facility will be designed without an operational spillway, it was designed to contain the probable maximum flood; therefore, a runoff coefficient of 1.0 was adopted.

A deposition model was run for the TSF, assuming deposition off the eastern and western embankments and endpoint discharge at two locations along the southern ridge of the facility and one location along the eastern ridge. The nominated minimum tailings freeboard for tailings was set at 1 m in accordance with Chilean regulations (Decree 248 and 50, Refs. 1 and 2). An additional 0.8 to 0.9 m freeboard was applied, above the supernatant pond generated by the design storm event (Probable Maximum Flood 72 hr), superimposed over the operational pond.

Seepage control measures such as an engineered liner and an above-liner underdrainage system were incorporated based on the testing. Due to the elevated enrichments, a mine waste cover was incorporated to limit access to the tailings and manage dust generation at closure. Furthermore, due to the PAF nature of the tailings and supernatant water quality, an engineered low permeability layer to reduce oxygen diffusion and water ingress was incorporated. Additional measures, such as designing the process flow sheet to remove sulphide-bearing materials, will also be considered as part of the next design phase.

The seepage management for the TSF will focus on containment and recovery of seepage at the main embankment location or downstream of the embankment. The system design includes the following components:

- A cut-off trench
- The TSF basin and upstream embankment faces will be lined with a geosynthetic clay liner (GCL), overlain by a high-density polyethylene (HDPE) liner.
- An underdrainage system comprising finger and collector drains will be installed throughout the Stage 1 TSF basin to reduce the pressure head acting on the liners and assist with drainage of the tailings mass.
- Downstream of the TSF embankment will be a fence line of abstraction bores.

Beyond the drainage control system, several monitoring bore stations (with one shallow and one deep monitoring bore) will be installed to monitor any potential seepage.

To reduce the risk of affecting the Quebrada La Higuera alluvial deposits and the Agua Verde wetland, a combination of measures will likely be required to control infiltration including e.g. liners, cut-off walls, grout injections, while a seepage collection system (SCS), based on a trench and/or pumping wells downstream of the West embankment. Options should be evaluated and designed in post-PFS studies as they will be required in the future EIA submission.

A monitoring programme for the TSF will be developed to monitor for any potential problems which may arise during operations. Abstraction bores around the Productora Pit will be specified to intercept structures to allow groundwater quality to be monitored and pumped, if required.

For closure, it is proposed that the final tailings profile be shaped to direct runoff to the north of the facility, where a closure spillway will be excavated in the northern ridge so that any rainfall runoff will run over the tailings surface to a sediment control area before discharge downstream.

1.20.6 Concentrate Storage and Loadout

An existing port facility near the Huasco township, would be utilised for receival, storage, reclaim and ship loading of copper concentrate. The facility would require upgrading to handle the volume of mine concentrate to be stored and shipped. New facilities required to be constructed for the Project are described include:

- Access roads
- Concentrate storage yards
- Conveyors from concentrate storage yards via wharf to ship loader facility
- Ship loader facility.

1.21 Environmental, Permitting and Social Considerations

1.21.1 Environmental Considerations

The Environmental Impact Assessment (EIA) of the Project will be submitted for approval using the EIA System that is currently being applied in Chile.

Several baseline campaigns have been carried out as part of the EIA since 2012. Around 11,000 ha has been covered and more than eight reports compiled. Seasonal baselines (including flora and fauna surveys) have been completed across Winter, Spring and Summer seasons. Non-seasonal baselines include 19 separate components such as archaeology, landscape, palaeontology, human environment, geomorphology, and natural risks.

In parallel with the EIA work, three Environmental Impact Statements were submitted to the Authority in 2012, 2013 and 2018 to obtain the license needed for exploration at Productora and similarly in 2021 for Cortadera. These drilling campaigns occurred within future mine areas, so the baseline information on flora, fauna, archaeology, noise and vibration, and landscape in these areas contributed to the PFS analysis.

Environmental studies to inform the PFS included hydrogeological studies and waste rock characterisation (ARD) geochemistry.

1.21.2 Tailings Storage Facility

As discussed above, the PFS is based on a conventional tailings facility, valley hosted north of Productora, and an option for in-pit tailings deposition at Productora. The in-pit tailings deposition investigations are not yet complete at the PFS level, with a groundwater and water balance model to be delivered to assess this option. This assessment will be completed prior to the EIA submission.

1.21.3 Water Management

The Costa Fuego Project hydrogeological monitoring network includes standpipe piezometers, open boreholes and surface seepage or shallow wells across the Productora, Cortadera and TSF sites.

A hydrogeological characterisation and conceptual model were developed for the Productora, Alice and Cortadera deposits. These models found that the units showed low permeability and perform as an aquitard system. The phreatic surface is approximately 17 to 85 m below the surface, and groundwater can be pressurised within this system until it can be released by drilling or mining excavations. The low permeability and arid environment result in a hydrogeological recharge assumption that is extremely low, around 1 l/s.

Future phreatic surface modelling (mine dewatering modelling) was conducted on the mine designs, with a minimum, base case, and maximum model. The inflow estimate ranged from 3 to 25 l/s. Standby pumping equipment capable of handling 100 l/s for the 1 in 100-year rainfall event was noted. Pumping specifications were provided in stages with pit progression, and the recommendation noted that the areas high evaporation rate of 1,200 mm/year would accommodate the volumes of water produced. Surface water is addressed with non-contact water diversion channels. Contact water resulting from large scale rain events is managed through either surface drainage back to the open pits for dewatering, or with the use of emergency ponds adjacent to waste rock emplacements.

1.21.4 Closure and Reclamation Planning

The mine closure plan, in compliance with Chilean regulations, will be prepared for the EIA submission.

The PFS considered closure costs for the Project, and findings of the studies in hydrology, biodiversity, acid rock drainage and the TSF design to define a closure approach for the PFS. The Project considers a progressive closure approach, where appropriate, to allow remediation of areas as they are released by the mine schedule, and to allow for period of monitoring as to the effectiveness of remediation before project end of life.

A summary of the mine closure approach, by Project area follows:

- **Open Pit:** Pit lakes are expected to naturally restore upon the cessation of mine dewatering. Earthworks will include safety bunds around the perimeters. In-pit tailings is proposed at the Productora main pit, with studies to determine closure requirements to be completed within the EIA preparation. Waste rock backfill at Cortadera will occur in Cuerpo 1 and 2 pits, rendering them surface stockpiles upon closure.

- **Underground workings:** Entrances to the underground workings (portal, ventilation shafts) will be plugged. Earthworks and fencing will be installed to prevent access to the pit and subsidence zone.

- **TSF**: The PFS design includes capping and the excavation of a closure spillway.

- **Surface Stockpiles:** stockpiles will be reshaped for continuity with the landscape and in consideration of drainage for surface water. ARD testwork to be finalised within the EIA preparation will inform of any additional capping or encapsulation.

- **Buildings and Infrastructure:** Salvage will be engaged in the decommissioning and removal of buildings and infrastructure such as pipelines and the rope conveyor.

- **Water treatment facility:** The duration of operation of the water treatment facility will be determined following a period of operation to quantify water volumes and qualities experienced across the Project life.

- **Revegetation:** The baseline surveys indicate a low vegetation baseline for the Project, on account of the arid environment. Revegetation will be in continuity with the landscape.

1.21.5 Permitting Considerations

Permitting involves securing multiple environmental, water, land, and operational permits required at each stage of the Project's lifecycle, from exploration through to mine closure.

The Environmental Impact Assessment (EIA) is central to the Costa Fuego Project's permitting process, providing a detailed evaluation of the potential environmental impacts the Project may have throughout its lifecycle. As required by Ley 19.300, the EIA systematically addresses issues such as air and water quality, biodiversity, ecosystem health, noise, and the social impacts on surrounding communities. By conducting this rigorous assessment, the Project not only ensures regulatory compliance but also identifies mitigation measures that promote environmental sustainability and community well-being.

The EIA for the Costa Fuego Project is being prepared by a dedicated environmental and social consultancy in collaboration with Hot Chili Limited and with consultation with local stakeholders. It incorporates baseline environmental data collected during the exploration and pre-development stages, ensuring that all potential environmental impacts are thoroughly evaluated.

The EIA also incorporates a Social Impact Assessment (SIA), which evaluates the potential effects of the Costa Fuego Project on local communities, including indigenous groups. The SIA focuses on issues such as land use, cultural heritage, employment opportunities, infrastructure development, and community well-being. Special attention has been given to the needs and concerns of indigenous communities in the region, as required by Chilean law and international standards such as the Free, Prior, and Informed Consent (FPIC) principle outlined in ILO Convention No. 169.

Hot Chili Limited has secured permits for seawater intake, granted through its Maritime Concession permit. Seawater intake, and management of the associated infrastructure, will be addressed within the Huasco Water feasibility studies. Huasco Water is a separate entity to Hot Chili Limited, with Hot Chili Limited holding 80% ownership, and the Costa Fuego Project is anticipated to be a customer of Huasco Water.

Water permits related to the dewatering activities of the proposed mining operations are required separate to the land and mining permits. Hot Chili Limited will apply for these permits in conjunction with the Costa Fuego Project EIA submission.

Chilean law mandates that all mining projects submit a comprehensive mine closure plan as part of the permitting process. The Costa Fuego Project's mine closure plan will be developed in accordance with Law No. 20.551, which outlines the requirements for environmental restoration and financial assurance in the event of mine closure and prepared for its EIA submission.

1.21.6 Social Considerations

Hot Chili Limited has placed a significant emphasis on adopting best practices for community engagement and social responsibility, integrating international standards such as the Free, Prior, and Informed Consent (FPIC) principles from the ILO Convention No. 169, and adhering to the CIM ESG guidelines for social governance. These standards are not only necessary for regulatory compliance but are also fundamental to ensuring that the Project is socially sustainable and that its benefits are shared equitably among all affected communities.

The Project is located on the ancestral lands of several Indigenous groups with the most populous being the Diaguita people. The engagement with these communities has been conducted through a consultation process, where the Project team has shared information on the Project and will continue with consultation regarding input on how the development might affect their lands, resources, and way of life. Formal agreements are required to be reached with all recognised Indigenous Community (IC) groups for the EIA submission, and these are on track for this milestone.

Aligned with the preparation of the companies EIA submission, the company has commenced the recognised community engagement process, the participación ciudadana temprana (PCT), which translates to Early Citizen Participation. This programme includes public presentations and information sessions regarding the Project.

1.22 Markets and Contracts

A copper price of US$4.30/lb has been applied to the calculations for the 2025 PFS, viewed as balanced when compared with market forecasts. A 25-bank assessment, provided by NBF in February 2025 has a long-term copper price range of US$3.45/lb to US$5.00/lb.

Gold is a key by-product for the Costa Fuego Project, with a price of US$2,280/oz being applied for the 2025 PFS, aligned with the long-term gold price forecast.

A discount rate of 8% has been used for net present value ("NPV") calculations.

The mine is expected to produce a clean copper-gold-silver concentrate to be sold to smelters in Asia with offtake terms reflective of that market.

1.23 Capital Cost Estimates

All costs are estimated in United States dollars as at Q4 2024 and are judged to have an accuracy of ±15% to 25% and are at Preliminary Feasibility Study level in accordance with Wood's Estimating Procedures and Class 4 as defined in the AACE document 18R-97.

The estimates are based on a combination of direct quotes, benchmarking, inputs from consultants and QP's, and HCH supplied data.

Construction and expansion capital costs are estimated at $1.27 billion and $1.35 billion, respectively, with LOM sustaining capital costs (including reclamation and closure) estimated at $811 million.

1.24 Operating Cost Estimates

All costs are estimated in United States dollars as at Q4 2024 and are judged to have an accuracy of ±15% to 25% and are at Preliminary Feasibility Study level in accordance with Wood's Estimating Procedures and Class 4 as defined in the AACE document 18R-97.

Processing operating cost estimates have been prepared for the Cortadera surface primary crusher, primary crusher and sulphide concentrator located at Productora, Productora oxide plant (ore preparation, heap leach, SX/EW), sulphide dump leach and port operations (copper concentrate storage and transhipment).

Open pit mining cost estimates were developed using a zero-based model, and all expenditures for the open pit have been classified as operating costs. The open pit plans to employ a contractor for the entire LOM. Underground operating cost estimates include the use of a development contractor for the initial 30 quarters. During this period all expenditures will be capitalised.

The overall life of mine operating cost is US$8,650 M.

1.25 Economic Analysis

Based on the economic analysis, the Project delivers a base-case, post-tax NPV8% of US$1.20 Billion and an IRR of 19% (based on metal price assumptions of US$4.30/lb copper (Cu), US$2,280/oz gold (Au), US$28/oz silver (Ag), and US$20/lb molybdenum (Mo)). On a pre-tax basis, the Project delivers a base-case NPV8% of US$1.71 billion and an IRR of 22%, with a project life of 20 years and a payback period of 4.5 years.

Project after-tax NPV is most sensitive to factors that affect copper revenue - copper price, grade and recovery - and discount rate. NPV is also sensitive to changes in mining cost, processing cost and construction capital.

Tax calculations are based on the tax regime in effect as of the date of this Report.

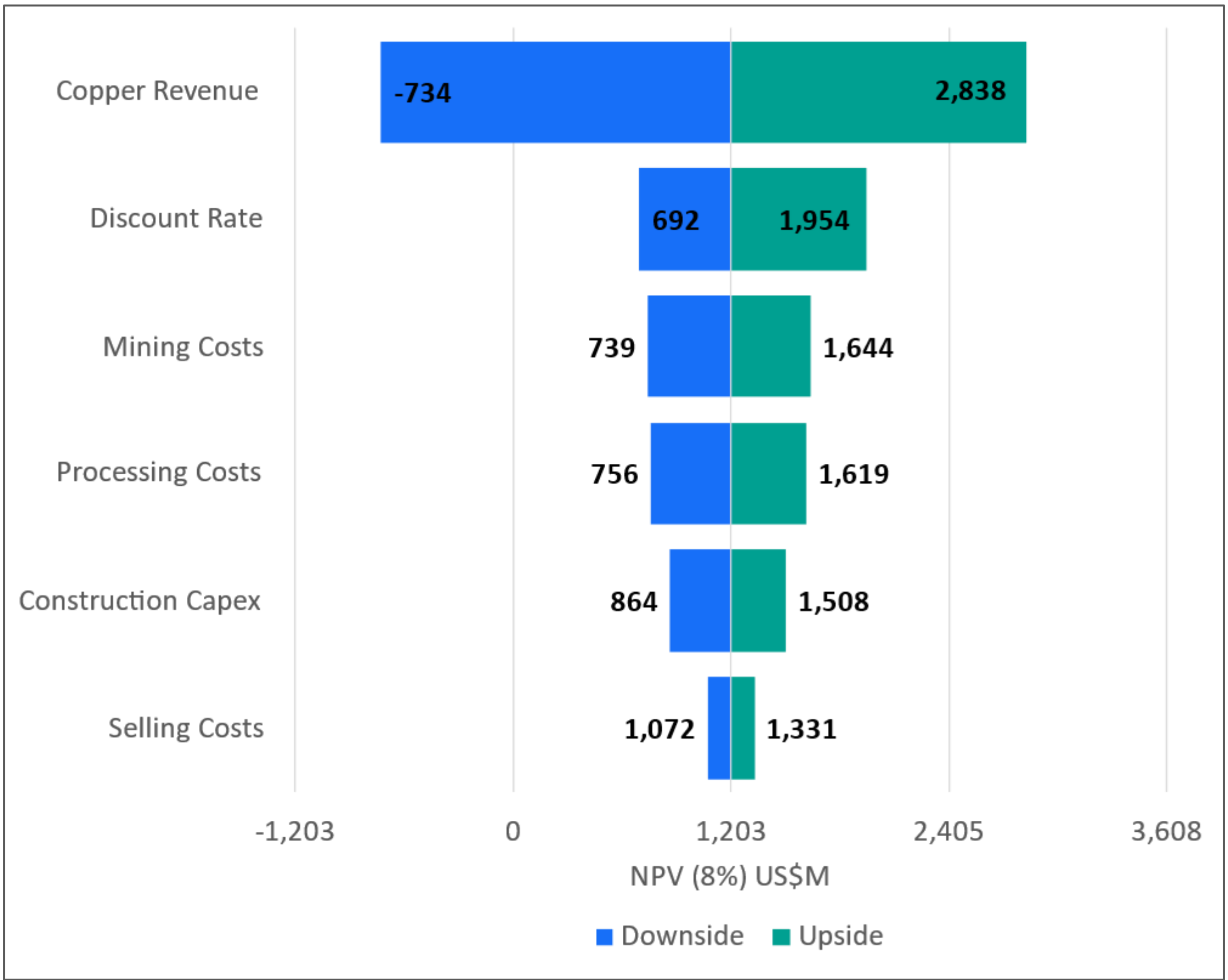
Royalties in the financial model are applied according to the royalty agreements described in Section 4 of this Report.

1.26 Sensitivity Analysis

The Project is most sensitive to copper revenue, followed by discount rate and operating costs, and least sensitive to construction capital expenditure and selling costs.

Figure 1.20 illustrates how the estimated base case $NPV_{8\%}$ of US$1,203M varies using 40% higher and 40% lower assumptions for metal prices, capital and operating costs, noting base case assumptions for the PFS used a copper price of US$4.30/lb.

Figure 1.20 Costa Fuego Project Sensitivity Analysis



1.27 Risk and Opportunity

As part of ongoing risk management HCH monitors general mining risks such as changes in political risks and uncertainties affecting legislation, royalties, labour, and market volatility. These risks are monitored and mitigated by HCH as part of their ongoing Project development.

The most significant risks were evaluated include:

- Capital and operating cost escalation as Project plans and parameters change or are refined, a possible mitigation maybe to consider alternate strategies e.g. procurement/flowsheet or technologies
- Diesel fuel is a significant component of the mine operating costs. Higher fuel prices could impact project returns, a possible mitigation is to consider alternate technologies at the time
- Tailings storage facilities, in particular the PFS design reaching capacity in the event of additional processing volumes, and the Extreme dam break assessment based on proximity to Vallenar. A program for assessment of alternative TSF locations that may also meet the criteria for the PFS and expansion is underway
- Rope conveyor overhead passing of Pan American Highway, with risk of falling objects into a publicly accessible zone and increased risk for development application challenges. An underpass alternative has been investigated within the PFS and additional design works may be reviewed prior to commencement of a FS.
- Environmental social risks which are proposed to be mitigated by ongoing EIA processes and stakeholder engagement

Opportunities identified included:

- Additional revenue streams by inclusion of cobalt in the process feed inventory and/or consideration for the further recovery of acid via pyrite roasting.
- Optimisation of the process flowsheet
- Single pit option for Cortadera vs block cave mining strategy
- Potential development of the La Verde exploration target, for incorporation into the Costa Fuego project

Risks and opportunities will be continuously assessed and reviewed throughout the various phases of the design, construction and operation, in accordance with HCH's Risk Management Framework.

1.28 Interpretation and Conclusion

The results of this PFS at the Costa Fuego Project supports progressing the Project to the Feasibility Study (FS) stage.

- The exploration program continues to demonstrate the potential for future growth of the Mineral Resource, which may further enhance Project economics and/or extend the operating life.
- The sample preparation, security, and procedures followed by HCH are adequate to support the Mineral Resource Estimates contained herein.
- Assay data provided by HCH was represented accurately and is suitable for use in the Mineral Resource estimation.
- There are no known factors related to metallurgical, environmental, permitting, legal, title, taxation, socio-economic, marketing, or political issues which could materially impact the ability to develop the Project.
- The mine design produced within the PFS is robust and adequately informed. Mineral Reserves comprise Indicated Resources.
- The TSF solutions produced within the PFS are adequate for the proposed quantity of tailings within the PFS, with appropriate levels of risk management.

- The Project delivers a robust, positive, economic outcome.
- The metallurgical testwork undertaken is reasonably extensive and suitable for this level of study. The design of the processing circuit is based in the majority on this testwork data and there is minimal use of assumptions.
- The mineralised material is of moderate competency and hardness, and amenable to grinding in a conventional circuit. The mineralogy is fine grained compared to may projects in South America. Testwork indicates a requirement to re-grind to a fine particle size (25 µm P80) to achieve adequate liberation for flotation to produce good copper concentrate grades.
- Overall recoveries from ores to flotation concentrates are estimated at 87% for copper and 71% for molybdenum.
- The Project has been designed to meet current social and environmental management practices.

1.29 Recommendations

The QPs consider that growth and optimisation opportunities outlined in Sections 24 and 25 should be explored to advance the Costa Fuego Project through the next study phases, which will include the submission of an Environmental Impact Assessment (EIA) and Feasibility Study (FS).

The FS would be based on the recommended case presented in the PFS, utilising additional technical work to further improve capital and operating cost estimation accuracy and provide increased confidence in the financial model outcomes.

The QPs have reviewed the proposed program of work and budget and find them to be reasonable and justified considering the observations made in this report. The recommended work program and proposed expenditures are appropriate and well thought out. The proposed budget reasonably reflects the type and scope of the contemplated activities.

The QPs recommend that HCH conduct the planned activities subject to availability of funding and any other matters which may cause the objectives to be altered in the normal course of business activities.

2 Introduction

This Report presents the findings of a Prefeasibility Study ("PFS") completed on the Costa Fuego Copper Project ("Costa Fuego", the "Costa Fuego Project" or "Project"), located near Vallenar in Chile, South America.

At the request of Hot Chili Limited ("HCH," "Hot Chili Limited" or "Company"), Wood Limited ("Wood") has prepared the Report for the Costa Fuego Project, as announced by HCH on 27 March 2025.

This Report was prepared in accordance with the Canadian disclosure requirements of NI 43-101, and in accordance with the requirements of Form 43-101F1.

HCH is a Perth-based copper-gold exploration and development company that undertakes exploration and development of its various copper-gold projects located in Chile's Atacama Region. The responsibilities of the QP’s and/or independent consultants are as follows:

- Haren Consulting is an independent geological consulting firm based in Perth, Australia.
- Geomechanics, Mining and Technology (GMT) is a geotechnical/geomechanics engineering consulting firm based in Chile, which has been providing a range of geotechnical / geomechanics engineering services to the civil/mining industry since 2013.
- ABGM is an independent mining consulting firm based in Perth, Australia
- Knight Piésold Consulting is a global consulting firm, with expertise in tailings management, based in Perth, Australia
- Wood is one of the world’s leading consulting and engineering companies operating across Energy and Materials markets. QP’s were based in Perth, Australia and Denver, USA
- Process Mineral Consulting (PMC) is a Chilean Engineering Company that provides mining consulting services worldwide with experience in both large-scale and mid-tier operations. PMC specialises in the development and optimisation of mining operations, processing, water and tailings management.. PMC were engaged from Santiago Chile.
- High River Services is an independent environmental consulting company, based in Kentucky, USA
- Knight Piésold is an employee-owned, global consulting firm providing specialist services to the mining, power, water resources, and infrastructure industries. Knight Piésold has a 1,000-strong team operating from 28 offices across 16 countries. Knight Piesold delivered the tailings storage facility design and tailings geochemical analysis within the PFS and were engaged from Perth Australia.
- Doppelmayr is a world market leader for rope-propelled mobility. With innovative transport systems, they continue to set standards in the mobility sector. More than 3,000 employees in 50 countries around the world are part of the Doppelmayr Group. Doppelmayer provided the Rope Conveyor design within the Infrastructure remit, as an independent consultant.
- Gestion Ambiental Consultores SA (“GAC”) has more than 30 years of experience as environmental management consultants in the Chilean market. GAC has vast experience in the development of projects associated with the energy, mining, industrial, forestry, agricultural, real estate and auditing sectors, both for the public sector and for private companies. GAC were engaged from Santiago Chile and provided environmental and social consulting as independent consultants.

- Nova Mineralis develops technological solutions for the mining industry for the recovery of copper and other base metals from primary sulphide ores. Nova Mineralis were engaged from Santiago Chile as independent consultants.
- Piteau and associates is a global engineering consultancy providing geotechnical, water management, and environmental consulting services to the mining, construction, municipal, first nations, and industrial sectors. Piteau and associates provided the hydrogeological studies, and waste rock geochemical analysis as independent consultants.

There has been no material change to the Project between the effective date of this Technical Report and the signature date.

2.1 Terms of Reference

This Report has been prepared to describe the proposed methods for exploiting the Cortadera, Productora, San Antonio and Alice deposits. This Report is based on mineral resource estimates ("Mineral Resource Estimate" or "MRE") completed in February 2024, and on subsequent technical studies completed during 2024 and 2025, focusing on geotechnical, metallurgy, mining, process plant/infrastructure design, and cost estimation

Mineral Resource Estimates are reported in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves (May 2014; the 2014 CIM Definition Standards) and were estimated in accordance with the CIM 2019 Best Practices Guidelines, as required by NI 43-101.

Units used in this Report are metric unless otherwise noted. Unless otherwise noted, all dollar figures used are United States of America (US) dollars (US$). The Chilean currency is the Chilean peso (CLP).

Years discussed in the mine and production plan and in the economic analysis are presented for illustrative purposes only, as no decision has been made on mine construction by HCH.

2.2 Authors of the Report / Qualified Persons

Table 2.1 summarises the Qualified Persons responsible for the contents of this Report as that term is defined in NI 43-101, and for preparing this Report in compliance with Form 43-101F1:

Table 2.1 List of QPs

Qualified Person	Discipline	Company	Section(s)
Elizabeth Haren	Geology	Haren Consulting	1.2, 1.3, 1.4, 1.5, 1.6, 1.7, 1.8, 1.9,1.10,1.12, 3.2, 3.3, 3.4, 6, 7, 8, 9, 10, 11, 12.1, 12.4, 14, 24.11, 25.2, 25.3, 26.3
Anton von Wielligh	Mining	ABGM	1.13,1.14,1.15,1.16,1.17, 1.18, 12.3, 15, 16 (except 16.3.4 & 16.4.2), 21.3.1, 21.5, 21.15, 23, 24.7, 25.4, 26.4, 26.5
David Cuello	Geotechnical	GMT	12.6, 16.3.4, 16.4.2, 26.4
Dean David	Metallurgy – Sulphide	Wood	1.11.1, 1.11.2,1.19.1, 1.19.2, 1.24, 2.2, 2.3, 12.2.1, 12.2.2, 13.1, 13.2, 13.3, 13.5, 17.1, 17.2, 17.3, 17.4, 17.5, 17.7, 21.16, 24.9.3, 24.9.4, 24.9.5, 24.9.6, 25.5.1, 25.5.2, 25.5.3, 25.5.5, 26.6, 27.1
Luis Bernal	Metallurgy – leaching	PMC	1.11.3, 1.19.3, 12.2.1.3, 13.4, 17.6, 25.5.4, 26.6.3, 26.6.4, 27.1
David Morgan	Tailings	Knight Piésold	1.20.5, 1.212, 18.4, 24.5, 25.6.3, 26.9
Piers Wendlandt	Economics	Wood Group USA, Inc.	1.22,1.25,1.26, 1.27, 1.28, 1.29, 3.5, 3.6, 12.2.4, 12.2.6, 19, 22, 25.8, 25.11
Jeffrey Stevens	Infrastructure, Capital Cost Estimates	Wood	1.1, 1.20, 1.20.1, 1.20.2, 1.20.3, 1.20.4, 1.23, 1.27, 1.28, 1.29, 2.0, 2.1, 2.2, 2.3, 2.4, 2.5, 2.6, 2.2, 2.3, 2.4, 2.5, 2.6, 3.1, 5, 12.2.3, 12.2.5, 18 (excluding 18.4), 21 (excluding 21.5, 21.15, 21.16), 24.1, 24.2, 24.3, 24.4, 24.8, 25.1, 25.6, 25.9,, 25.12, 25.13, 26.1, 26.7, 26.8, 26.10, 26.11, 26.13, 27
Edmundo J Laporte	Social and Environmental	HRS	1.5.5, 1.21, 4, 12.5, 20, 25.7, 26.12

2.3 Site Visits and Scope of Personal Inspection

HCH has had an ongoing presence on site since 2010, undertaking extensive exploration and resource definition drilling programs over the last ten years with offices, drill core yards and associated infrastructure at both Productora and Cortadera.

Table 2.2 summarises the Qualified Persons responsible for the contents of this Report as that term is defined in NI 43-101, and for preparing this Report in compliance with Form 43-101F1:

Table 2.2 Site Visits by QPs

Qualified Person	Discipline	Date of Site Visit
Elizabeth Haren	Geology	May 2022
Anton van Wielligh	Mining	October 2024
Dean David	Metallurgy – Sulphide	October 2024
Loui Bernal	Metallurgy – leaching	July 2024
David Morgan	Tailings	July 2024
Jeffrey Stevens	Infrastructure	October 2024
Edmundo J Laporte	Social and Environmental	May 2024

2.4 Effective Dates

This Report has an effective date of 27 March 2025.

The 2024 Technical Report, reproduced in this Report was issued with an effective date of 26 February 2024. This Report updated the 2024 Technical Report and 2024 PEA.

2.5 Information Sources and References

HCH has engaged specialist consultants and information from these reports, prepared by independent consultants, has been utilised in the compilation of this Report.

This Report relies on historic and recent data presented in historical reports, including:

- Information provided by qualified geologists employed by HCH regarding the geology, drilling, sampling and other exploration procedures and processes adopted by HCH
- Information provided by HCH pertaining to the history of the Project
- Information provided by HCH in relation to the property location, property title and ownership and mineral tenure.

In so far as other persons that have had input into the preparation of this Report, the authors have conducted appropriate due diligence and consider such reliance to be reasonable.

Section 27 provides a list of references relied upon in preparation of this Report.

2.6 Previous Technical Reports

This Report supersedes and replaces the technical reports as listed below and they should no longer be relied upon.

- Von Wielligh, A, Haren, E, Resource Report for the Costa Fuego Copper Project Located in Atacama Chile, Technical Report NI 43-101 prepared by Haren Consulting and ABGM, with an effective date of 29 October 2021.
- Von Wielligh, A, Haren, E, Resource Report for the Costa Fuego Copper Project Located in Atacama Chile, Technical Report NI 43-101 prepared by Haren Consulting and ABGM, with an effective date of 31 March 2022.
- Haren, E., von Wielligh, A., Laporte, E. J., Morgan, D., David, D., Stevens, J., Wendlandt, P., Preliminary Economic Assessment for the Costa Fuego Copper Project, Technical Report NI 43-101 prepared by Haren Consulting, ABDM, Wood, Knight Piesold, and High River Services, with an effective date of 28 June 2023.
- Haren, E., von Wielligh, A., Laporte, E. J., Morgan, D., David, D., Stevens, J., Wendlandt, P., Preliminary Economic Assessment for the Costa Fuego Copper Project, Technical Report NI 43-101 prepared by Haren Consulting, ABDM, Wood, Knight Piesold, and High River Services, with an effective date of 26 February 2024

3 Reliance on Other Experts

3.1 Introduction

This Report is based, in part, on the review, analysis, interpretation and conclusions derived from information which has been provided by HCH staff and its consultants and have opined upon it.

The QPs used their experience to review such information and determine if the information was suitable for inclusion in this Report. The QPs take responsibility for this information.

This Report includes technical information which required subsequent calculations to derive subtotals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, the QPs do not consider them to be material.

3.2 Mineral Tenure

The QPs have not reviewed the mineral tenure, nor independently verified the legal status, ownership of the Property, underlying property agreements or permits. The QPs have fully relied upon, and disclaim responsibility for, information derived from experts retained by HCH for this information through the following documents:

- Expert: Correa Squella, Chile, 18 July 2023
- Report, opinion or statement relied upon: letter prepared by Correa Squella – Productora Project Title Opinion, Legal opinion in connection to the current status of the Productora deposit mining rights
- Grasty, Quintana, Majlis – Legal opinion in connection to the current status of the Cortadera deposit mining rights.

This information is used in Section 4 of the Report. It is also used in support of the Mineral Resource statement in Section 14 and the economic analysis result in Section 22 and Section 24.9 of this Report.

3.3 Legal Title

The QPs have relied upon the following various independent consulting companies that contributed to the development of this Report:

- Expert: Correa Squella, Chile, 18 July 2023
- Report, opinion or statement relied upon: letter prepared by Correa Squella – Productora Project Title Opinion, Legal opinion in connection to the current status of the Productora deposit mining rights
- Expert: Clyde & Co, Chile, 14 June 2023
- Report, opinion or statement relied upon: letter prepared by Clyde & Co – Legal opinion regarding the validity of Sociedad Minera La Frontera SpA; incorporation and its power of attorney
- Expert: Grasty, Quintana, Majlis, Chile, 14 July 2021
- Report, opinion or statement relied upon: Legal opinion regarding Hot Chilis Titles Opinion

- Grasty, Quintana, Majlis – Legal opinion in connection to the current status of the Cortadera deposit mining rights.

This information is used in Section 4 of this Report.

3.4 Permitting

The QPs have fully relied upon and disclaim responsibility for information supplied by HCH staff and experts retained by HCH for information related to permitting for the seawater intake as follows:

- Expert: Ministerio de Defensa Nacional, Chile, 29 September 2020
- Report, opinion or statement relied upon: statement prepared by Ministerio de Defensa Nacional – Otorga Concesion Maritima Mayor, sobre un sector de terreno de playa, playa y fondo de mar en la comuna de Huasco, a Sociedad Minera el Aguila SPA (translation: Grants Concession Maritima Mayor, on a sector of beach, beach and seabed land in the commune of Huasco, to Sociedad Minera el Aguila SPA)

This information is used in Section 4 of this Report.

3.5 Taxation

The QPs have fully relied upon and disclaim responsibility for information supplied by HCH staff and experts retained by HCH for information related to taxation as applied to the financial model as follows:

- Expert: Aninat Abogados, Chile, 11 August 2022
- Report, opinion or statement relied upon: letter prepared by Aninat Abogados – Audit on Specific Mining Tax (IEM) and RIOM (Taxable Income from Mining Operations) calculations.

This information is used in the financial model in Section 22 and Section 25.11 of the Report.

3.6 Markets

The QPs have fully relied upon, and disclaim responsibility for, information supplied by experts retained by HCH for copper marketing and pricing through the following document:

- Wood 2022, Long Term Market Outlook, October 2022, report prepared by Wood.

This information is used in Sections 19 and 22 of this Report. It is also used in Section 25.8 and Section 25.11 of this Report.

4 Property Description and Location

4.1 Introduction

The Project, which includes the Productora, Alice, Cortadera, and San Antonio deposits, is centred 17 km south of Vallenar in the Atacama Region of Chile (Región de Atacama). The Project is located approximately 155 km south of Copiapó, the capital of Atacama Region, and 140 km north of the city of La Serena.

Productora lies 5 km off the main sealed Pan-American Highway which connects Vallenar to La Serena in the south and can be accessed via the Quebrada Arenas gravel road 2.2 km towards the west. From that point, the Project can be accessed by continuing along the Quebrada Verde track for 5 km. Alice sits approximately 500 m to the west of Productora, with access via established trails.

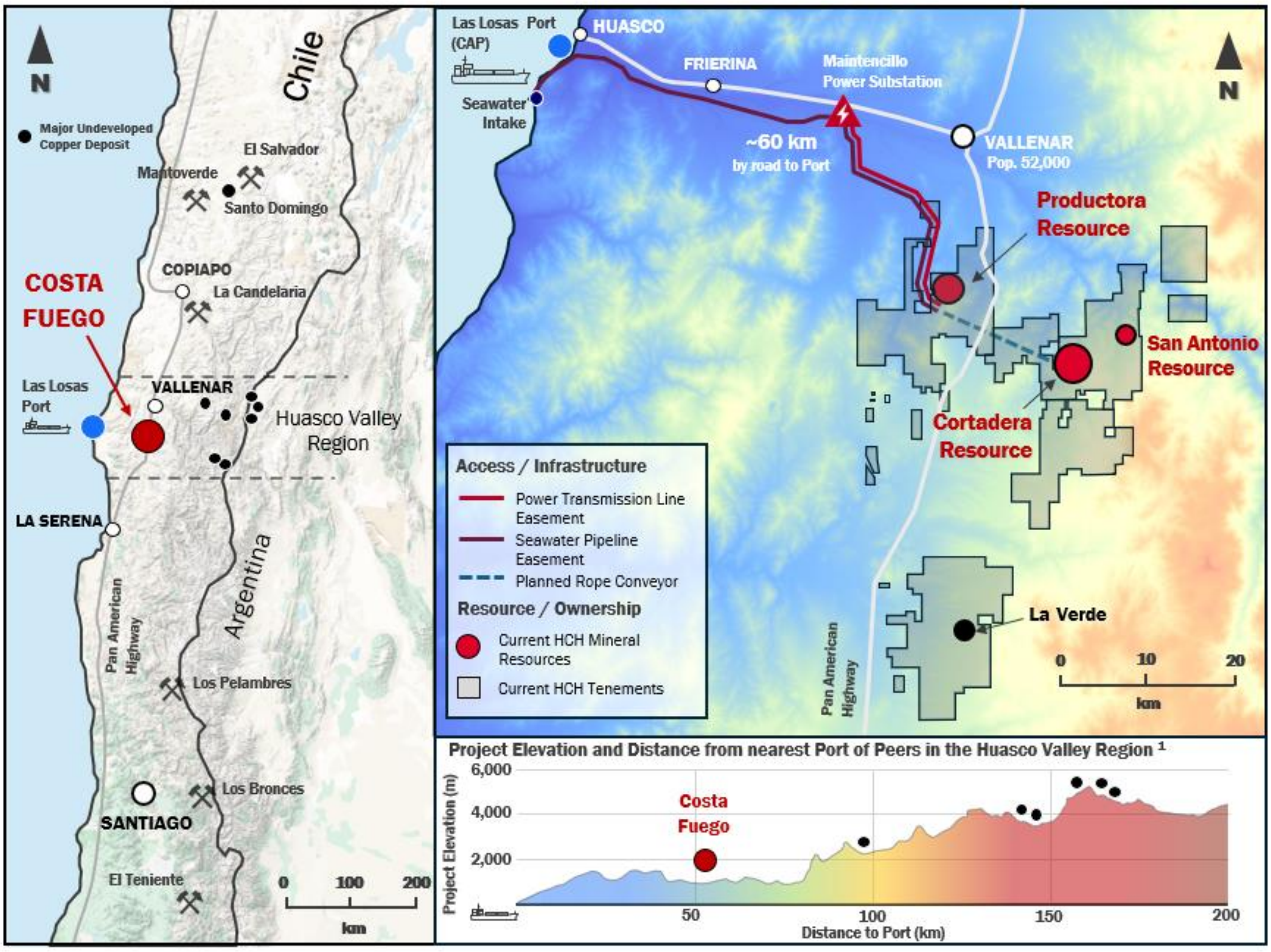
Cortadera and San Antonio lie approximately 15 km to the east of the Pan-Highway and are accessed through gravel roads and trails.

The Project lies within the low altitude coastal range belt and has an average altitude of 740 m above sea level. The Costa Fuego Project has good access to infrastructure and facilities in the regional mining town of Vallenar.

Figure 4.1 outlines the location of the Project relative to Vallenar and the surrounding infrastructure.

Established in July 2024, Huasco Water (HW Aguas para El Huasco SpA) – 80% owned by HCH, 20% owned by Chilean iron ore company Compania Minera del Pacifico (CMP), now is in the transferring regulatory process to hold the valid maritime water extraction license from Sociedad Minera EL Aguila SpA (SMEA). Water easements, costal land accesses are secured by authorization given by SMEA and the second maritime application previously held by Sociedad Minera EL Aguila SpA (SMEA) will be also transfer to Huasco Water once legal requirements have been fulfilled. Huasco Water provides water supply security as potential foundation off-taker to the Costo Fuego Project.

Figure 4.1: Costa Fuego Project Location (HCH, 2025)



4.2 Property and Title in Chile

Information in this subsection is based on data in the public domain and Chilean law (Chilean Civil Code, Mining Code, Chilean Tax Law, Fraser Institute, 2017) and has not been independently verified by additional QPs.

Mining concessions (or mining rights, as they are also known in Chile) are granted by a judicial award issued by a court of justice in the context of a non-litigious proceeding. Mining rights are protected by the Chilean Constitution as well as many different legal bodies, of which the Mining Code is the most important legislation. The territorial extension of a mining concession takes on the shape of a solid, the surface of which is a horizontal parallelogram of right angles, and the depth of which is indefinite within the vertical planes that establish its boundaries.

In general, the Political Constitution of the Republic and the provisions of Chilean law make no distinction among Chileans and non-Chileans regarding the enjoyment of basic rights, the acquisition of property, and the development of economic activities.

According to Article 2 of the Constitutional Law on Mining Concessions (Law 18,097), a mining concession is: "an in-rem property right, different and independent from ownership of the surface land, even if it belongs to one and the same owner; enforceable against the State and any other person; transferable and transmissible; subject to mortgage and other in rem rights and, in general, to any act or contract".

Under Chilean law, there are two types of mining concessions:

- Exploration concessions which are for the purpose of exploring and expire after a period of four years according to the amendments incorporated into the Mining Code by Law No. 21,420 and the new Law No. 21,649, which came into effect on January 1, 2024. All the exploration concessions that expired in 2024 have been extended to 2026.
- Exploitation concessions which are for the exploitation of minerals and have no expiry date, some annual rent payments are required.

4.3 Project Ownership

The Productora deposit (including Alice) is 100% owned by a Chilean incorporated company named Sociedad Minera El Aguila SpA (SMEA). SMEA is a joint venture company – 80% owned by Sociedad Minera El Corazón SpA (a 100% subsidiary of Hot Chili Limited (HCH) – ASX: HCH), and 20% owned by Compañía Minera del Pacífico S.A (CMP), a major Chilean iron ore producer.

This partnership has enabled securement of the majority of the surface rights required for developing key infrastructure for the Project, as well as the majority of easements required for water and power transmission lines.

The Cortadera deposit is controlled by a Chilean incorporated company named Sociedad Minera Frontera SpA (Frontera). Frontera is a subsidiary company – 100% owned by Sociedad Minera El Corazón SpA, which is a 100% subsidiary of Hot Chili Limited (HCH) – ASX: HCH.

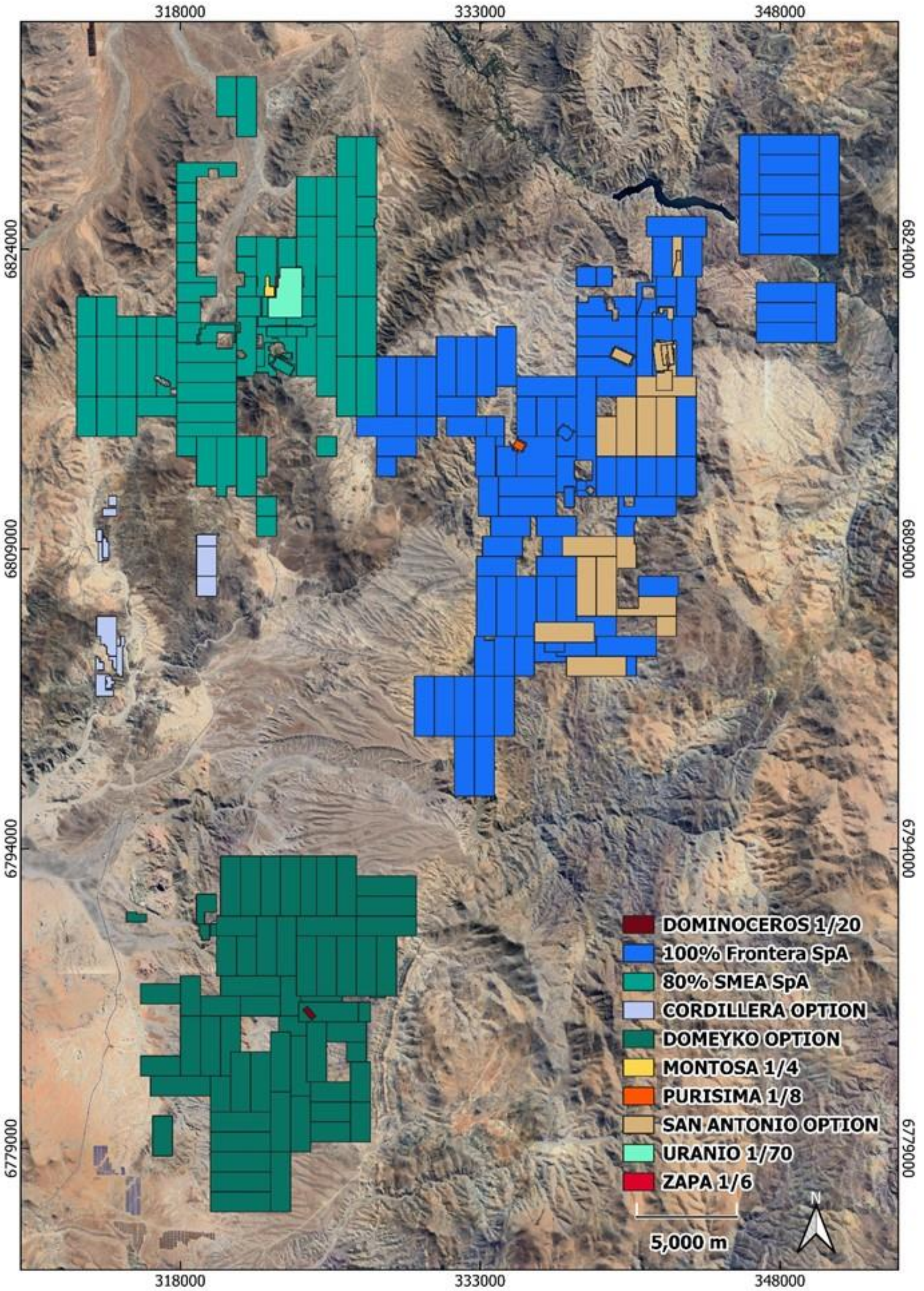
Frontera owns the Cortadera deposit and controls an area approximately 12.5 km north-south by 7 km east-west through various option earn in agreements with private landholders, and through 100% ownership of certain leases.

The San Antonio deposit is controlled through Frontera (100% owned by Sociedad Minera El Corazón SpA, which is a 100% subsidiary of Hot Chili Limited) and has an option agreement with a private party to earn a 100% interest.

4.4 Mineral Tenure

The location of all tenements is shown in Figure 4.2

Figure 4.2 Mineral Tenements Summary, HCH (2025)



4.4.1 Productora

The Productora Project is approximately 18 km long by 14 km wide, with a total landholding of approximately 14,500 ha.

SMEA controls Productora primarily through direct ownership, except for one exploitation concession (Uranio 1/70), in which a 30-year lease agreement has been executed with The Chilean Nuclear Energy Commission (CCHEN), which commenced in 2012.

Two small underground copper mines were in operation within the central mining lease of Productora from late 2006 until mid-2013 and again briefly in 2020. SMEA executed a purchase option agreement with owners of the Productora 1/16 concession where the underground mines were located. In February 2013, following exercise of the purchase option, mining activities ceased, and all equipment was removed from the Project area.

The current exploration activities for the deposits have been approved under the Environmental Approval number 2, dated 11 January 2018, granted by the Environmental Assessment Service.

Table 4.1 details the SMEA tenement holding at Productora, tenement ownership, and mining concession type.

Table 4.1 : SMEA SpA Mining Tenement Holding for Productora

Licence ID	Holder	% Interest	Licence Type	Area (ha)
ALGA 7A 1/32	SMEA SpA	80%	Exploitation concession	89
ALGA VI 4	SMEA SpA	100%	Exploitation concession	2
ALGA VI 5/24	SMEA SpA	80%	Exploitation concession	66
ARENA 1 1/6	SMEA SpA	80%	Exploration concession	40
ARENA 2 1/17	SMEA SpA	80%	Exploration concession	113
AURO HUASCO I 1/8	SMEA SpA	80%	Exploration concession	35
CABRITO CABRITO 1/9	SMEA SpA	80%	Exploitation concession	50
CACHIYUYITO 1 1/20	SMEA SpA	80%	Exploitation concession	100
CACHIYUYITO 2 1/60	SMEA SpA	80%	Exploitation concession	300
CACHIYUYITO 3 1/60	SMEA SpA	80%	Exploration concession	300
CARMEN I 1/50	SMEA SpA	80%	Exploitation concession	222
CARMEN II 1/60	SMEA SpA	80%	Exploitation concession	274
CF 12	Frontera SpA	100%	Exploration concession	100
CF 13	Frontera SpA	100%	Exploration concession	200
CF 14	Frontera SpA	100%	Exploration concession	300
CHICA	SMEA SpA	80%	Exploitation concession	1
CHOAPA 1/10	SMEA SpA	80%	Exploitation concession	50
CUENCA A 1/51	SMEA SpA	80%	Exploitation concession	255
CUENCA B 1/28	SMEA SpA	80%	Exploitation concession	139
CUENCA C 1/51	SMEA SpA	80%	Exploitation concession	255
CUENCA D	SMEA SpA	80%	Exploitation concession	3
CUENCA E	SMEA SpA	80%	Exploitation concession	1

Table 4.1 : SMEA SpA Mining Tenement Holding for Productora				
Licence ID	**Holder**	**% Interest**	**Licence Type**	**Area (ha)**
ELEANOR RIGBY 1/10	Frontera SpA	100%	Exploitation concession	100
ELQUI 1/14	SMEA SpA	80%	Exploitation concession	61
ESPERANZA 1/5	SMEA SpA	80%	Exploitation concession	11
FRAN 1 1/60	SMEA SpA	80%	Exploitation concession	220
FRAN 12 1/40	SMEA SpA	80%	Exploitation concession	200
FRAN 13 1/40	SMEA SpA	80%	Exploitation concession	200
FRAN 14 1/40	SMEA SpA	80%	Exploitation concession	200
FRAN 15 1/60	SMEA SpA	80%	Exploitation concession	300
FRAN 18 1/60	SMEA SpA	80%	Exploitation concession	273
FRAN 2 1/20	SMEA SpA	80%	Exploitation concession	100
FRAN 21 1/46	SMEA SpA	80%	Exploitation concession	226
FRAN 3 1/20	SMEA SpA	80%	Exploitation concession	100
FRAN 4 1/20	SMEA SpA	80%	Exploitation concession	100
FRAN 5 1/20	SMEA SpA	80%	Exploitation concession	100
FRAN 6 1/26	SMEA SpA	80%	Exploitation concession	130
FRAN 7 1/37	SMEA SpA	80%	Exploitation concession	176
FRAN 8 1/30	SMEA SpA	80%	Exploitation concession	120
JULI 10 1/60	SMEA SpA	80%	Exploration concession	300
JULI 11 1/60	SMEA SpA	80%	Exploration concession	300
JULI 12 1/42	SMEA SpA	80%	Exploration concession	210
JULI 13 1/20	SMEA SpA	80%	Exploration concession	100
JULI 14 1/50	SMEA SpA	80%	Exploration concession	250
JULI 15 1/55	SMEA SpA	80%	Exploration concession	275
JULI 16 1/60	SMEA SpA	80%	Exploration concession	300
JULI 17 1/20	SMEA SpA	80%	Exploration concession	100
JULI 19	SMEA SpA	80%	Exploration concession	300
JULI 20	SMEA SpA	80%	Exploration concession	300
JULI 21 1/60	SMEA SpA	80%	Exploration concession	300
JULI 22	SMEA SpA	80%	Exploration concession	300
JULI 23 1/60	SMEA SpA	80%	Exploration concession	300
JULI 24 1/60	SMEA SpA	80%	Exploration concession	300
JULI 25	SMEA SpA	80%	Exploration concession	300
JULI 27 1/30	SMEA SpA	80%	Exploration concession	146
JULI 27 B 1/10	SMEA SpA	80%	Exploration concession	48
JULI 28 1/60	SMEA SpA	80%	Exploration concession	300
JULI 9 1/60	SMEA SpA	80%	Exploration concession	300
JULIETA 1/4	SMEA SpA	80%	Exploration concession	4
JULIETA 10 1/60	SMEA SpA	80%	Exploration concession	300
JULIETA 11	SMEA SpA	80%	Exploration concession	300
JULIETA 12	SMEA SpA	80%	Exploration concession	300

Table 4.1 : SMEA SpA Mining Tenement Holding for Productora				
Licence ID	**Holder**	**% Interest**	**Licence Type**	**Area (ha)**
JULIETA 13 1/60	SMEA SpA	80%	Exploration concession	298
JULIETA 14 1/60	SMEA SpA	80%	Exploration concession	269
JULIETA 15 1/40	SMEA SpA	80%	Exploration concession	200
JULIETA 16	SMEA SpA	80%	Exploration concession	200
JULIETA 17	SMEA SpA	80%	Exploration concession	200
JULIETA 18 1/40	SMEA SpA	80%	Exploration concession	200
JULIETA 5	SMEA SpA	80%	Exploration concession	200
JULIETA 6	SMEA SpA	80%	Exploration concession	200
JULIETA 7	SMEA SpA	80%	Exploration concession	100
JULIETA 8	SMEA SpA	80%	Exploration concession	100
JULIETA 9	SMEA SpA	80%	Exploration concession	100
LA PRODUCTORA 1/16	SMEA SpA	80%	Exploitation concession	75
LEONA 2A 1/4	SMEA SpA	80%	Exploitation concession	10
LIMARÍ 1/15	SMEA SpA	80%	Exploration concession	66
LOA 1/6	SMEA SpA	80%	Exploration concession	30
MAIPO 1/10	SMEA SpA	80%	Exploitation concession	50
MONTOSA 1/4*	SMEA SpA	80%	Exploitation concession	35
ORO INDIO 1A 1/20	SMEA SpA	80%	Exploitation concession	82
PEGGY SUE 1/10	Frontera SpA	100%	Exploitation Concession	100
TOLTÉN 1/14	SMEA SpA	80%	Exploration concession	70
URANIO 1/70	CCHEN	100%	Exploration concession	350
SUERTE 1/7	SMEA SpA	100%	Exploitation Concession	21
SUERTE II 1/15	SMEA SpA	100%	Exploitation Concession	15
ZAPA 1 1/10	SMEA SpA	80%	Exploitation concession	100
ZAPA 1/6**	SMEA SpA	80%	Exploration concession	6
ZAPA 3 1/23	SMEA SpA	80%	Exploitation concession	92
ZAPA 5A 1/16	SMEA SpA	80%	Exploitation concession	80
ZAPA 7 1/24	SMEA SpA	80%	Exploitation concession	120

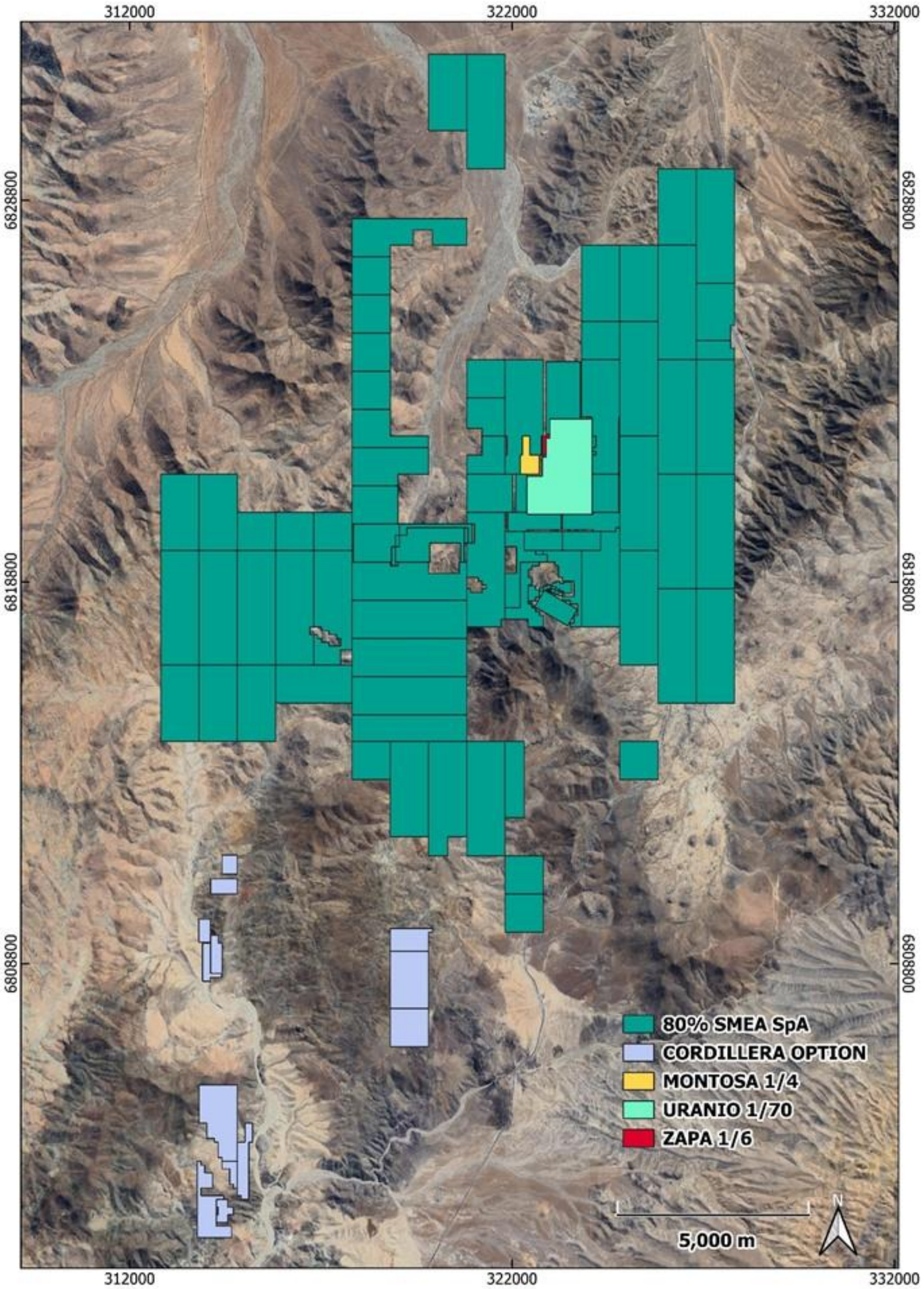
* Subject to a 3% Net Smelter Return (NSR).
**Subject to a 1% Gross Royalty.

All mining rights covering the mining and infrastructure footprint at Productora are exploitation concessions with no risk of expiring if the mining taxes are duly paid annually.

Surface rights are 100% owned by SMEA (except an area where SMEA has an option to acquire the surface rights), as is the electrical transmission line to the Costa Fuego Project. All maritime concessions and pipeline easements have been secured with Huasco Water, as discussed in Section 4.6 of this Report.

A map of HCH´s mining rights at Productora under WGS84 is shown in Figure 4.3.

Figure 4.3 : Productora Mining Rights Under WGS84 (HCH, 2025)



SMEA is the owner of the majority of the surface rights required for the infrastructure of Productora (an area of approximately 4,111 ha), including the surface rights for the mining area, sulphide processing plant, tailings storage facility (TSF) area, access roads and associated supporting facilities.

4.4.2 Cortadera

HCH Limited owns the Cortadera deposit through Frontera and controls an area measuring approximately 22,500 ha at the deposit through various 100% purchase option agreements with private mining title holders and 100% owned tenure.

Figure 4.4 show Frontera's consolidated position over the mining rights at the Cortadera.

As a summary:

- Frontera has full ownership over the mining rights coloured in dark blue.
- Frontera has an option agreement over the mining rights coloured in brown (El Fuego (San Antonio) option agreement) for 100% of the mining rights expiring in September 2026.

All mining tenements are in good standing and all mining requirements have been met for the exploration phase. At this stage, there are no legal requirements for any kind of bonds to be issued.

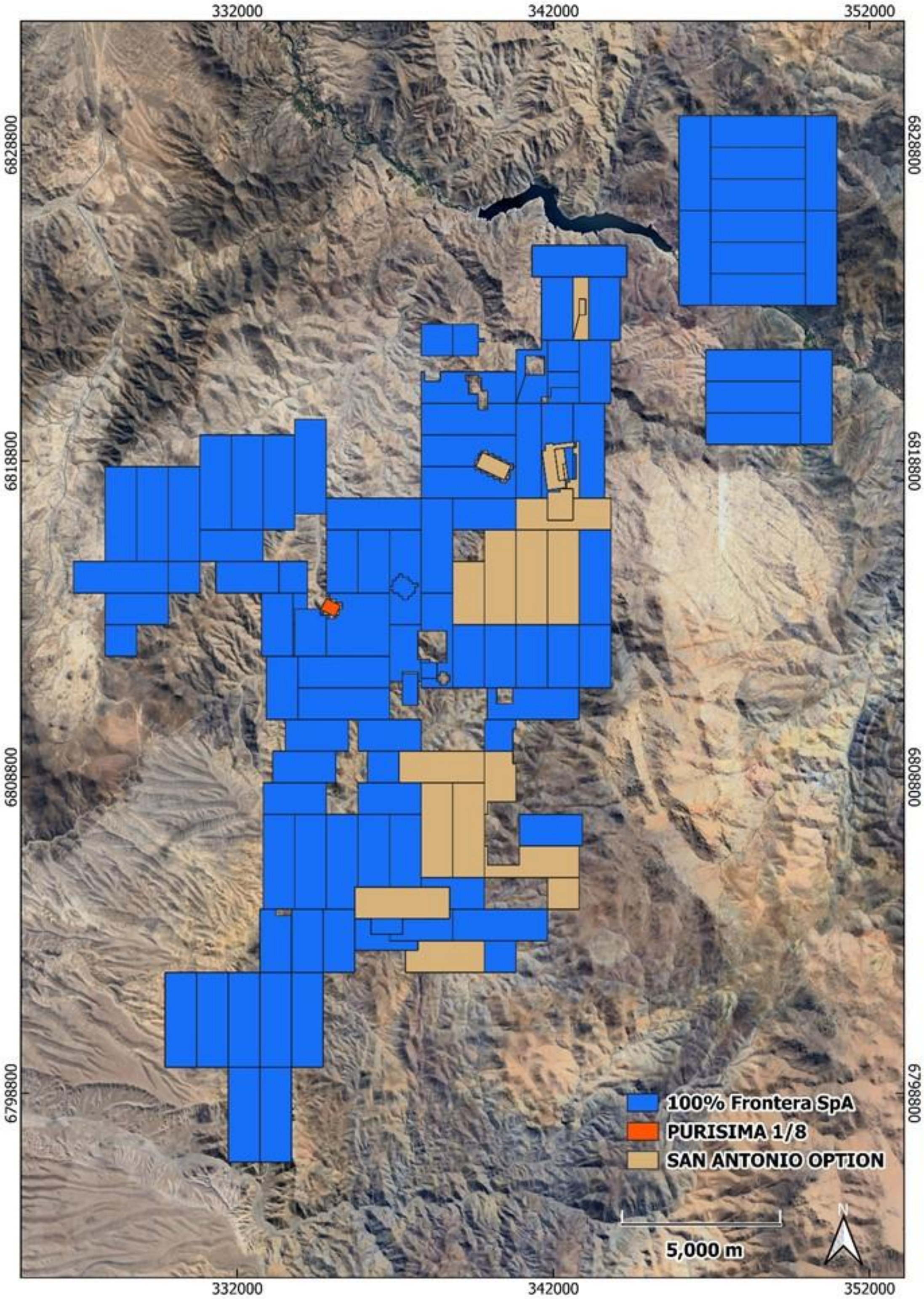
Surface land access for exploration activities has also been granted by the owner, Mr. Pedro Prokurika Morales by agreement acknowledged and approved by the local Court of Vallenar on 30 March 2022.

The area covered by the surface rights are sufficient for any potential open pit and underground mining operation together with the potential area for waste disposal and potential dump leach pads.

All power, water and personnel will be sourced from Productora, and any mineralised material will be transported to the Productora site for processing and subsequent tailings disposal.

Cortadera has no current social and/or community requirements.

Figure 4.4 : Location of HCH's Cortadera Mining Rights with Surrounding Option Agreements under WGS84 (HCH, 2024)



From the 1990s, beginning with initial small-scale mining, several companies explored and mined copper oxides in the Cortadera area. Some small open pit mining was completed in the 2000s, but significant exploration using modern exploration techniques did not occur until Minera Fuego took ownership in 2009.

Since securing the deposit in 2019, HCH has continued advancing Cortadera with modern exploration techniques, such as diamond drilling (DD), reverse circulation drilling (RC), surface geochemical sampling and litho-structural mapping.

Figure 4.4 shows in dark blue colour, Frontera's 100% ownership of the mining rights in the area.

Table 4.2 details the Frontera tenement holding at Cortadera, tenement ownership, and mining concession type.

Table 4.2 : Frontera SpA Mining Tenement Holding for Cortadera

Licence ID	Holder	% Interest	Licence Type	Area (ha)
ALCENIA 1/10	SMEA SpA	100%	Exploitation Concession	50
AMALIA 942 A 1/6	Frontera SpA	100%	Exploitation Concession	53
ATACAMITA 1/82	Frontera SpA	100%	Exploitation Concession	82
CORROTEO 1 1/260	Frontera SpA	100%	Exploitation Concession	260
CORROTEO 5 1/261	Frontera SpA	100%	Exploitation Concession	261
CORTADERA 1 1/200	Frontera SpA	100%	Exploitation Concession	200
CORTADERA 1/40	Frontera SpA	100%	Exploitation Concession	374
CORTADERA 2 1/200	Frontera SpA	100%	Exploitation Concession	200
CORTADERA 41	Frontera SpA	100%	Exploitation Concession	1
CORTADERA 42	Frontera SpA	100%	Exploitation Concession	1
LAS CANAS 1/15	Frontera SpA	100%	Exploitation Concession	146
LAS CANAS 16	Frontera SpA	100%	Exploitation Concession	1
LAS CANAS ESTE 2003 1/30	Frontera SpA	100%	Exploitation Concession	300
PAULINA 10 B 1/16	Frontera SpA	100%	Exploitation Concession	136
PAULINA 11 B 1/30	Frontera SpA	100%	Exploitation Concession	249
PAULINA 12 B 1/30	Frontera SpA	100%	Exploitation Concession	294
PAULINA 13 B 1/30	Frontera SpA	100%	Exploitation Concession	264
PAULINA 14 B 1/30	Frontera SpA	100%	Exploitation Concession	265
PAULINA 15 B 1/30	Frontera SpA	100%	Exploitation Concession	200
PAULINA 22 A 1/30	Frontera SpA	100%	Exploitation Concession	300
PAULINA 24 1/24	Frontera SpA	100%	Exploitation Concession	183
PAULINA 25 A 1/19	Frontera SpA	100%	Exploitation Concession	156
PAULINA 26 A 1/30	Frontera SpA	100%	Exploitation Concession	294
PAULINA 27A 1/30	Frontera SpA	100%	Exploitation Concession	300
PURISIMA*	Frontera SpA	100%	Exploitation Concession	20
MAGDALENITA 1/20	Frontera SpA	100%	Exploitation Concession	100
CF 1	Frontera SpA	100%	Exploration Concession	300
CF 2	Frontera SpA	100%	Exploration Concession	300

Table 4.2 : Frontera SpA Mining Tenement Holding for Cortadera				
Licence ID	**Holder**	**% Interest**	**Licence Type**	**Area (ha)**
CF 3	Frontera SpA	100%	Exploration Concession	300
CF 4	Frontera SpA	100%	Exploration Concession	300
CF 5	Frontera SpA	100%	Exploration Concession	200
CF 6	Frontera SpA	100%	Exploration Concession	200
CF 7	Frontera SpA	100%	Exploration Concession	100
CF 8	Frontera SpA	100%	Exploration Concession	200
CF 9	Frontera SpA	100%	Exploration Concession	100
CF 10	Frontera SpA	100%	Exploration Concession	200
CF 11	Frontera SpA	100%	Exploration Concession	200
CHAPULIN COLORADO 1/3	Frontera SpA	100%	Exploitation Concession	3
CHILIS 1	Frontera SpA	100%	Exploration Concession	200
CHILIS 10 1/38	Frontera SpA	100%	Exploitation Concession	190
CHILIS 11	Frontera SpA	100%	Exploration Concession	200
CHILIS 12 1/60	Frontera SpA	100%	Exploitation Concession	300
CHILIS 13	Frontera SpA	100%	Exploration Concession	300
CHILIS 14	Frontera SpA	100%	Exploration Concession	300
CHILIS 15	Frontera SpA	100%	Exploration Concession	300
CHILIS 16	Frontera SpA	100%	Exploration Concession	300
CHILIS 17	Frontera SpA	100%	Exploration Concession	300
CHILIS 18	Frontera SpA	100%	Exploration Concession	300
CHILIS 3	Frontera SpA	100%	Exploration Concession	100
CHILIS 4	Frontera SpA	100%	Exploration Concession	200
CHILIS 5	Frontera SpA	100%	Exploration Concession	200
CHILIS 6	Frontera SpA	100%	Exploration Concession	200
CHILIS 7	Frontera SpA	100%	Exploration Concession	200
CHILIS 8	Frontera SpA	100%	Exploration Concession	200
CHILIS 9	Frontera SpA	100%	Exploration Concession	300
CORTADERA 1	Frontera SpA	100%	Exploration Concession	200
CORTADERA 2	Frontera SpA	100%	Exploration Concession	200
CORTADERA 3	Frontera SpA	100%	Exploration Concession	200
CORTADERA 4	Frontera SpA	100%	Exploration Concession	200
CORTADERA 5	Frontera SpA	100%	Exploration Concession	200
CORTADERA 6 1/60	Frontera SpA	100%	Exploration Concession	265
CORTADERA 7 1/20	Frontera SpA	100%	Exploration Concession	93
CRISTINA 1/40	SMEA SpA	100%	Exploitation Concession	40
DIABLITO 1/5	SMEA SpA	100%	Exploitation Concession	25
DONA FELIPA 1/10	Frontera SpA	100%	Exploitation Concession	50
DORO 1	Frontera SpA	100%	Exploration Concession	200
DORO 2	Frontera SpA	100%	Exploration Concession	200
DORO 3	Frontera SpA	100%	Exploration Concession	300

Table 4.2 : Frontera SpA Mining Tenement Holding for Cortadera				
Licence ID	**Holder**	**% Interest**	**Licence Type**	**Area (ha)**
FALLA MAIPO 2 1/10	Frontera SpA	100%	Exploitation Concession	99
FALLA MAIPO 3 1/8	Frontera SpA	100%	Exploitation Concession	72
FALLA MAIPO 4 1/26	Frontera SpA	100%	Exploitation Concession	26
MINORI 1	SMEA SpA	100%	Exploration Concession	300
MINORI 2	SMEA SpA	100%	Exploration Concession	300
MINORI 3	SMEA SpA	100%	Exploration Concession	300
MINORI 4	SMEA SpA	100%	Exploration Concession	300
PORFIADA II	Frontera SpA	100%	Exploration Concession	300
PORFIADA III	Frontera SpA	100%	Exploration Concession	300
PORFIADA IV	Frontera SpA	100%	Exploration Concession	300
PORFIADA V	Frontera SpA	100%	Exploration Concession	200
PORFIADA VI	Frontera SpA	100%	Exploration Concession	100
PORFIADA X	Frontera SpA	100%	Exploration Concession	200
PORFIADA B	Frontera SpA	100%	Exploration Concession	200
PORFIADA D	Frontera SpA	100%	Exploration Concession	300
PORFIADA G	Frontera SpA	100%	Exploration Concession	200
PORFIADA I	Frontera SpA	100%	Exploration Concession	300
SAN ANTONIO 1	Frontera SpA	100%	Exploration Concession	200
SAN ANTONIO 2	Frontera SpA	100%	Exploration Concession	200
SAN ANTONIO 3	Frontera SpA	100%	Exploration Concession	300
SAN ANTONIO 4	Frontera SpA	100%	Exploration Concession	300
SAN ANTONIO 5	Frontera SpA	100%	Exploration Concession	300
SOLAR 1	Frontera SpA	100%	Exploration Concession	300
SOLAR 10	Frontera SpA	100%	Exploration Concession	300
SOLAR 2	Frontera SpA	100%	Exploration Concession	300
SOLAR 3	Frontera SpA	100%	Exploration Concession	300
SOLAR 4	Frontera SpA	100%	Exploration Concession	300
SOLAR 5	Frontera SpA	100%	Exploration Concession	300
SOLAR 6	Frontera SpA	100%	Exploration Concession	300
SOLAR 7	Frontera SpA	100%	Exploration Concession	300
SOLAR 8	Frontera SpA	100%	Exploration Concession	300
SOLAR 9	Frontera SpA	100%	Exploration Concession	300
SOLEDAD 1	Frontera SpA	100%	Exploration Concession	300
SOLEDAD 2	Frontera SpA	100%	Exploration Concession	300
SOLEDAD 3	Frontera SpA	100%	Exploration Concession	300
SOLEDAD 4	Frontera SpA	100%	Exploration Concession	300

*Subject to a 1.5% NSR

4.4.3 San Antonio

Figure 4.4 shows, in brown colour, the San Antonio mining rights which include three now terminated options for Valentina, San Antonio and Santiago Z and cover a total of approximately 3,700 ha. The previously proposed JV option agreement with a private party was to earn a 90% interest in the San Antonio copper-gold deposit over a six-year period. The three option agreements were renegotiated by HCH in December 2023, with the previous total purchase price of US$11,000,000 decreased to US$4,300,000 for the total San Antonio landholdings, including the Valentina, San Antonio and Santiago Z landholdings.

The proposed JV involves an option agreement over 27 exploitation leases (~4727 ha), whereby full ownership of 100% of the mining rights of the deposit will be transferred upon satisfaction of a payment of US$1,000,000 by September 2024, US$1,000,000 by September 2025 and then a final payment of US$2,000,000 a year after.

If the new option agreement is exercised, additional payments of up to US$4,000,000 in total are conditional on the following matters:

Additional payment of US$2,000,000, if the copper price average US$ 5.00/lb or above for a period of 12 consecutive months, within a period that expires 1 January 2030.

- Additional payment US$2,000,000, if an independently estimated Mineral Resource Estimate reported in accordance with CIM guidelines, as required by NI43-101, by Hot Chili Limited or its subsidiaries containing 200 million tonnes or greater within the El Fuego landholdings, within a period that expires January 1st 2030.
- An additional payment is to be made by March 2027, if compliance of the condition that justifies payment is verified until 30 September 2026. From October 2026, payment is to be paid within 70 days after the relevant condition is satisfied.

Continuation of existing lease mining agreements to third parties in respect to the San Antonio copper mine limited to the mining right San Antonio 1 al 5; The lease mining agreements are limited to 50,000 tonnes of material extracted per year and will expire 31 December 2025.

Table 4.3 details the San Antonio agreement mining rights.

Table 4.3 : Summary of San Antonio Agreement Mining Rights

Licence ID	Holder	% Interest	Licence Type	Area (ha)
MERCEDES 1/3	Del Campo Family	100%	Exploitation Concession	50
PORFIADA A 1/33	Del Campo Family	100%	Exploitation Concession	160
PORFIADA C 1/60	Del Campo Family	100%	Exploitation Concession	300
PORFIADA E 1/20	Del Campo Family	100%	Exploitation Concession	100
PORFIADA F 1/50	Del Campo Family	100%	Exploitation Concession	240
PORFIADA IX 1/60	Del Campo Family	100%	Exploitation Concession	300
PORFIADA VII 1/60	Del Campo Family	100%	Exploitation Concession	270
PORFIADA VIII 1/60	Del Campo Family	100%	Exploitation Concession	300
PRIMA 1	Del Campo Family	100%	Exploitation Concession	1
PRIMA 2	Del Campo Family	100%	Exploitation Concession	2

Table 4.3 : Summary of San Antonio Agreement Mining Rights				
Licence ID	**Holder**	**% Interest**	**Licence Type**	**Area (ha)**
ROMERO 1/31	Del Campo Family	100%	Exploitation Concession	31
SAN ANTONIO 1/5	Del Campo Family	100%	Exploitation Concession	25
SAN JUAN SUR 1/5	Del Campo Family	100%	Exploitation Concession	10
SAN JUAN SUR 6/23	Del Campo Family	100%	Exploitation Concession	90
SANTIAGO 1/4 Y 20	Del Campo Family	100%	Exploitation Concession	75
SANTIAGO 15/19	Del Campo Family	100%	Exploitation Concession	25
SANTIAGO 21/36	Del Campo Family	100%	Exploitation Concession	76
SANTIAGO 37/43	Del Campo Family	100%	Exploitation Concession	26
SANTIAGO A, 1/26	Del Campo Family	100%	Exploitation Concession	244
SANTIAGO B, 1/20	Del Campo Family	100%	Exploitation Concession	200
SANTIAGO C, 1/30	Del Campo Family	100%	Exploitation Concession	300
SANTIAGO D, 1/30	Del Campo Family	100%	Exploitation Concession	300
SANTIAGO E, 1/30	Del Campo Family	100%	Exploitation Concession	300
SANTIAGO Z 1/30	Del Campo Family	100%	Exploitation Concession	300

4.4.4 AMSA Option Agreement

The company abandoned the AMSA option agreement during Q1 of 2025 and the mining rights have been returned to the owners.

4.4.5 Cometa Option Agreement

The company abandoned the Cometa option agreement during Q1 of 2025 and the mining rights have been returned to the owners.

4.4.6 Marsellesa and Cordillera Option Agreements

Figure 4.3 shows, in light blue colour, the mining rights of an option agreements with private parties to acquire 100% of the historical copper mine area Cordillera, located approximately 10 km southwest of Productora and covering an area of 776 ha.

The option agreement for Marsellesa was abandoned in Q1 2025.

The option agreement for Cordillera with Mr Arnaldo Del Campo (ADC), the holder of a 100% interest in the concession comprising Cordillera, may be exercised within 48 months of the date of grant of the option for a final non-refundable cash payment of US$3,700,000. ADC will also be granted a 1.5% NSR royalty over any material extracted from underground operations, and a 1.5% NSR royalty over any material extracted from open pit operations, on exercise of the Cordillera option. Frontera will have a right of first refusal to buy-back the NSR royalties.

Table 4.4 details the Cordillera agreement mining rights.

Table 4.4 : Summary of Cordillera Agreement Mining Rights				
Licence ID	**Option Holder**	**% Interest**	**Licence Type**	**Area (ha)**
ALBORADA III 1/35	Del Campo Family	100%	Exploitation concession	162
ALBORADA IV 1/20	Del Campo Family	100%	Exploitation concession	54
ALBORADA VII 1/25	Del Campo Family	100%	Exploitation concession	95
CAT IX 1/30	Del Campo Family	100%	Exploitation concession	150
CATITA IX 1/20	Del Campo Family	100%	Exploitation concession	100
CATITA XII 1/13	Del Campo Family	100%	Exploitation concession	61
CORDILLERA 1/5	Del Campo Family	100%	Exploitation concession	20
HERREROS 1/14	Del Campo Family	100%	Exploitation concession	28
MINA HERREROS III 1/6	Del Campo Family	100%	Exploitation concession	18
MINA HERREROS IV 1/10	Del Campo Family	100%	Exploitation concession	23
PORSIACA 1/20	Del Campo Family	100%	Exploitation concession	20
QUEBRADA 1/10	Del Campo Family	100%	Exploitation concession	28
VETA 1/28	Del Campo Family	100%	Exploitation concession	17

4.4.7 Domeyko Option Agreements

Figure 4.5 shows, in green, the mining rights of an option agreement with private parties to acquire 100% of the Domeyko project, located approximately 35 km south of Productora and covering an area of approximately 16,000 ha.

Frontera SpA has entered into a 100% purchase option agreement with payments of US$50,000 payable by 19 April 2025 (already satisfied), US $150,000 payable by 19 April 2026, US$200,000 payable by 19 April 2027, with a final payment of US$3,480,000 payable by 19 April 2028, and a NSR 1% royalty.

Also shown, in dark red, is the mining right Dominoceros 1/20, where Frontera SpA has entered into a 100% purchase option agreement with immediate payment of US$320,000, future payments of US$680,000 within 12 months and US$1,000,000 within 24 months. The option may be exercised within 36 months for a final payment of US$6,890,000.

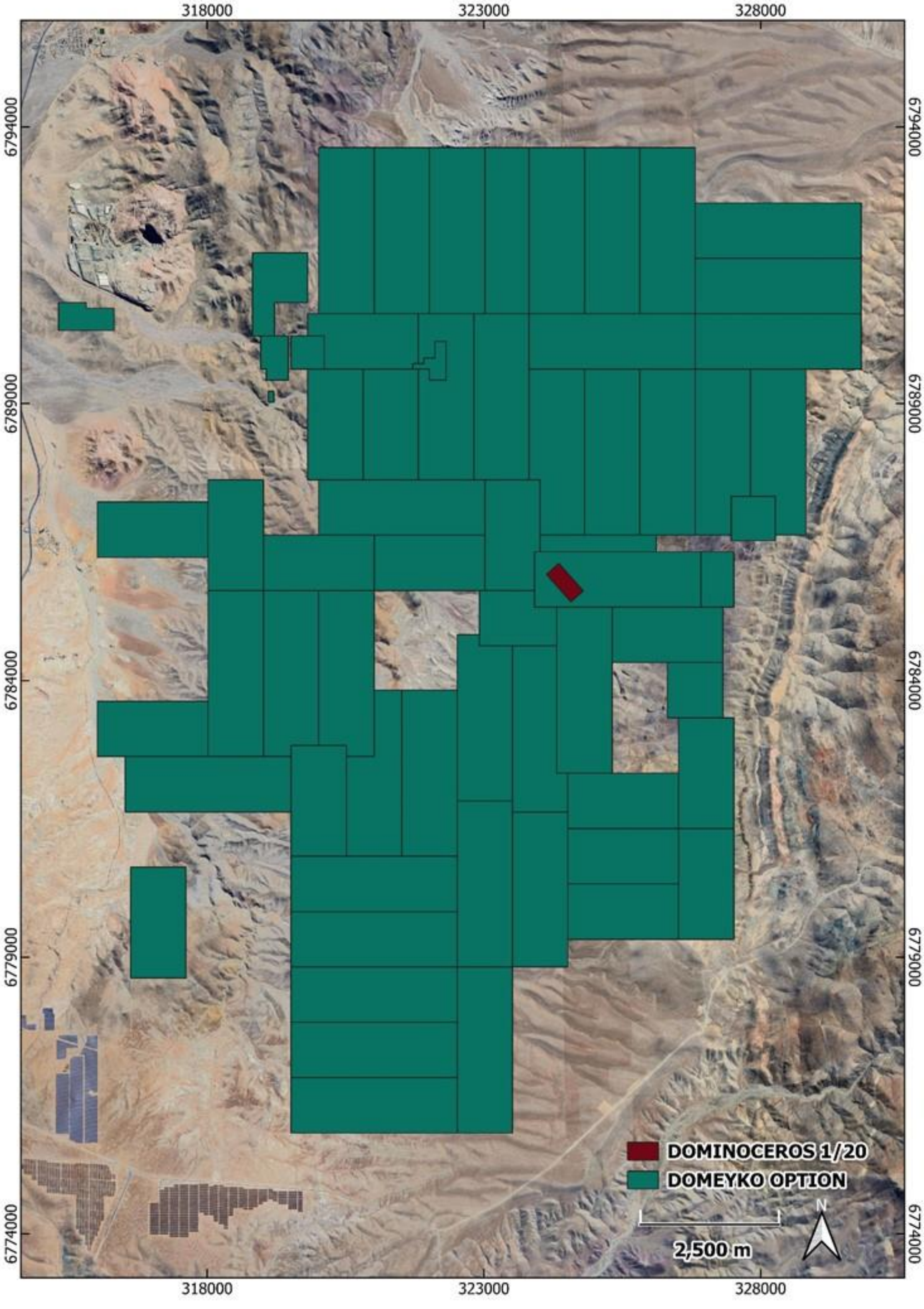
Table 4.5 details the Domeyko agreement mining rights.

Table 4.5 Summary of Domeyko Cluster Agreement Mining Rights				
Licence ID	**Option Holder**	**% Interest**	**Licence Type**	**Area (ha)**
ANTONIO 1 1/56	SLM Ines 1 De Quebrada San Antonio	100%	Exploitation Concession	280
ANTONIO 1/40	SLM Ines 1 De Quebrada San Antonio	100%	Exploitation Concession	200
ANTONIO 10 1/21	SLM Ines 1 De Quebrada San Antonio	100%	Exploitation Concession	63
ANTONIO 19 1/30	SLM Ines 1 De Quebrada San Antonio	100%	Exploitation Concession	128
ANTONIO 21 1/20	SLM Ines 1 De Quebrada San Antonio	100%	Exploitation Concession	60
ANTONIO 5 1/40	SLM Ines 1 De Quebrada San Antonio	100%	Exploitation Concession	200
ANTONIO 9 1/40	SLM Ines 1 De Quebrada San Antonio	100%	Exploitation Concession	193
ANTONIO 36 1/15	SLM Ines 1 De Quebrada San Antonio	100%	Exploitation Concession	74

Table 4.5 Summary of Domeyko Cluster Agreement Mining Rights				
Licence ID	**Option Holder**	**% Interest**	**Licence Type**	**Area (ha)**
CAZURRO 1	CIA MRA Algarrobo Limitada	100%	Exploration Concession	200
CAZURRO 2	CIA MRA Algarrobo Limitada	100%	Exploration Concession	200
CAZURRO 3	CIA MRA Algarrobo Limitada	100%	Exploration Concession	300
CAZURRO 3 1/60	CIA MRA Algarrobo Limitada	100%	Exploitation Concession	300
CAZURRO 4	CIA MRA Algarrobo Limitada	100%	Exploration Concession	300
CAZURRO 4 1/60	CIA MRA Algarrobo Limitada	100%	Exploitation Concession	300
CAZURRO 5	CIA MRA Algarrobo Limitada	100%	Exploration Concession	100
CAZURRO 6	CIA MRA Algarrobo Limitada	100%	Exploration Concession	200
CAZURRO 7	CIA MRA Algarrobo Limitada	100%	Exploration Concession	200
CAZURRO 7 1/40	CIA MRA Algarrobo Limitada	100%	Exploitation Concession	200
CAZURRO 8	CIA MRA Algarrobo Limitada	100%	Exploration Concession	200
CERRO MOLY 1	CIA MRA Algarrobo Limitada	100%	Exploration Concession	300
CERRO MOLY 2	CIA MRA Algarrobo Limitada	100%	Exploration Concession	300
CERRO MOLY 3	CIA MRA Algarrobo Limitada	100%	Exploration Concession	300
CERRO MOLY 4	CIA MRA Algarrobo Limitada	100%	Exploration Concession	300
EMILIO 1 1/8	CIA MRA Algarrobo Limitada	100%	Exploitation Concession	38
EMILIO 3 1/9	CIA MRA Algarrobo Limitada	100%	Exploitation Concession	45
INES 1/40	SLM Ines 1 De Quebrada San Antonio	100%	Exploitation Concession	200
LORENA 1/2	CIA MRA Algarrobo Limitada	100%	Exploitation Concession	2
MERCEDITA 1/7	CIA MRA Algarrobo Limitada	100%	Exploitation Concession	22
PRIMO 1 1/6	CIA MRA Algarrobo Limitada	100%	Exploitation Concession	36
SANTIAGUITO 5 1/24	CIA MRA Algarrobo Limitada	100%	Exploitation Concession	114
KRETA 1/4	Del Campo Family	100%	Exploitation Concession	16
MARI 1/12	Del Campo Family	100%	Exploitation Concession	64
CF 12	Frontera SpA	100%	Exploration Concession	100
CF 13	Frontera SpA	100%	Exploration Concession	200
CF 14	Frontera SpA	100%	Exploration Concession	300
CF SUR 1	Frontera SpA	100%	Exploration Concession	300
CF SUR 10	Frontera SpA	100%	Exploration Concession	200
CF SUR 11	Frontera SpA	100%	Exploration Concession	300
CF SUR 12	Frontera SpA	100%	Exploration Concession	300
CF SUR 13	Frontera SpA	100%	Exploration Concession	300
CF SUR 14	Frontera SpA	100%	Exploration Concession	300
CF SUR 15	Frontera SpA	100%	Exploration Concession	200
CF SUR 16	Frontera SpA	100%	Exploration Concession	300
CF SUR 17	Frontera SpA	100%	Exploration Concession	300
CF SUR 18	Frontera SpA	100%	Exploration Concession	300
CF SUR 19	Frontera SpA	100%	Exploration Concession	300
CF SUR 2	Frontera SpA	100%	Exploration Concession	300
CF SUR 20	Frontera SpA	100%	Exploration Concession	300

Table 4.5 Summary of Domeyko Cluster Agreement Mining Rights				
Licence ID	**Option Holder**	**% Interest**	**Licence Type**	**Area (ha)**
CF SUR 21	Frontera SpA	100%	Exploration Concession	300
CF SUR 22	Frontera SpA	100%	Exploration Concession	300
CF SUR 23	Frontera SpA	100%	Exploration Concession	200
CF SUR 24	Frontera SpA	100%	Exploration Concession	200
CF SUR 25	Frontera SpA	100%	Exploration Concession	300
CF SUR 26	Frontera SpA	100%	Exploration Concession	300
CF SUR 27	Frontera SpA	100%	Exploration Concession	300
CF SUR 28	Frontera SpA	100%	Exploration Concession	200
CF SUR 29	Frontera SpA	100%	Exploration Concession	300
CF SUR 3	Frontera SpA	100%	Exploration Concession	300
CF SUR 30	Frontera SpA	100%	Exploration Concession	200
CF SUR 31	Frontera SpA	100%	Exploration Concession	300
CF SUR 32	Frontera SpA	100%	Exploration Concession	300
CF SUR 33	Frontera SpA	100%	Exploration Concession	300
CF SUR 34	Frontera SpA	100%	Exploration Concession	300
CF SUR 35	Frontera SpA	100%	Exploration Concession	300
CF SUR 4	Frontera SpA	100%	Exploration Concession	300
CF SUR 5	Frontera SpA	100%	Exploration Concession	200
CF SUR 6	Frontera SpA	100%	Exploration Concession	300
CF SUR 7	Frontera SpA	100%	Exploration Concession	300
CF SUR 8	Frontera SpA	100%	Exploration Concession	300
CF SUR 9	Frontera SpA	100%	Exploration Concession	200
DOMINOCEROS 1/20	SLM Los Dominoceros una de la Sierra Los Chiqueros	100%	Exploitation Concession	100
MARI 1	Frontera SpA	100%	Exploration Concession	300
MARI 6	Frontera SpA	100%	Exploration Concession	300
MARI 8	Frontera SpA	100%	Exploration Concession	300

Figure 4.5 Location of HCH's Domeyko Mining Rights with Surrounding Option Agreements under WGS84 (HCH 2025)



4.5 Surface Rights

Ownership rights to the sub-soil are governed separately from surface ownership.

Articles 120 to 125 of the Mining Code regulate mining easements. The Mining Code grants to the owner of any mining exploitation or exploration concessions full rights to use the surface land, provided that reasonable compensation is paid to the owner of the surface land.

4.6 Water Rights

A mining concession grants its holder the right to use the water resources found while developing exploration and/or exploitation works, whichever the case may be, but only for the purposes of such exploration and/or exploitation works.

Water can be bought from certified suppliers (as HCH does for the exploration phase works).

As per the community engagement strategy, the exploitation of the Project contemplates operating with sea water. Water permits related to the dewatering activities of the proposed mining operations are being applied for and are not related to the grant of new water rights.

Since July 2024, the maritime water extraction licence is in the transferring regulatory process to, the newly established water company "HW Aguas para El Huasco SpA" (Huasco Water). Huasco Water is a subsidiary of Hot Chili Limited (80% interest) in partnership with CMP (20% interest). Water easements and costal land accesses held by SMEA will be authorized to be used by Huasco Water for the benefit of Costa Fuego. Separate to this Report, a pre-feasibility level Water Supply Business Case Study by Huasco Water was delivered focussing on establishing a water supply to the Costa Fuego Project and the broader Huasco Valley region.

4.7 Royalties and Encumbrances

On July 2023, a 1% NSR agreement over copper (and 3% over gold products) was granted to the Royalty and Streaming company Osisko Resources for a USD 15 million in cash consideration. Also, CCHEN mining lease named Uranio 1-16; the mining right Purisima, and the mining right Zapa have a royalty over those properties as is explained in the mining rights section.

Apart from that, the only third-party encumbrance in Productora is an electric transmission line on the extreme north of the Project.

4.8 Property Agreements

4.8.1 Productora

Correa Squella Legal, Chile, was requested to render a legal opinion in connection to the status of the HCH mining rights constituted or acquired by the Company in the Productora deposit, including its superficial rights, such as easements, and its maritime concession:

- SMEA was duly incorporated and transformed into a company by shares, in both cases under the laws of Chile. Therefore, it legally exists and is in good standing.

- HCH's investment in Chile is fully protected under national foreign investment protection legislation.
- To this point, SMEA holds all licenses, certificates, and permits from public entities necessary for conducting its business at the Project.

Regarding the legal status of the mining concessions:

- The mining exploitation concessions constituted by SMEA in the Project have been duly constituted and are currently in force.
- The mining claims currently being processed by SMEA have followed, to date, all the legal steps outlined in the Mining Code. Therefore, there has not been any error in their constitutive proceedings.
- Both exploitation concessions and mining claims are duly registered on behalf of SMEA. Therefore, such mining rights are the exclusive property of SMEA.
- Mining patents and fees of these mining rights have been fully and timely paid.
- As of July 2021, the previously referred rights are not subject to liens, prohibitions, embargoes, encumbrances, or lawsuits of any kind.
- The CCHEN Agreement constitutes a valid and binding agreement, and all obligations stipulated therein are enforceable against the parties bound thereby.

Therefore, the Uranium Property:

- Has been duly constituted and is currently in force
- Is duly registered under the name of CCHEN. Consequently, it has the sole and exclusive ownership over the Uranium Property
- From 1990 onward, all mining patents have been paid
- Is not subject to liens, prohibitions, embargoes or lawsuits of any kind, except for the prohibition registered in favour of SMEA under the Uranio Lease Agreement
- Does not grant to its holder the right to explore or exploit other minerals than uranium.

The maritime concession has been lawfully granted:

- To this date, SMEA has followed all the legal steps outlined in the maritime concessions regulation to enter in possession
- All superficial rights are valid, binding, and enforceable against third parties.

4.8.2 Cortadera

Clyde and Co. were requested by HCH to issue a legal opinion based on Chilean law in regard to mining titles that the Company currently holds in Chile by means of its subsidiary, Frontera, a company duly incorporated under Chilean law.

La Frontera is the holder of rights in five mining projects called "Cortadera", "San Antonio", "Santiago Z", "Valentina" and "Purísima".

- Frontera was duly incorporated in accordance with the laws of Chile, it legally exists and is in good standing.
- The option agreements constitute valid and binding agreements and all obligations stipulated therein are enforceable against the parties respectively bound thereby.
- All payments accrued with regards to the option agreements have been timely and fully paid.

Regarding to the mining rights:

- All mining rights included in option agreements were established in accordance with the requirements of the Mining Code. There were no third parties' concessions overlapping the mining rights.
- The mining rights granted in option agreements in favour of La Frontera and the mining rights constituted directly by La Frontera have an approximately surface area of 16 000 ha.
- Based on the review, there are no agreements regarding the mining concessions that could affect Frontera's tenure or Frontera's right to acquire the mining rights.
- The Cortadera deposit and the Purísima deposit have provisional surface rights for the exploration stage approved by the Environmental Approval number 48 dated 24 March 2021 granted by Environmental Assessment Service. However, in order to execute the exploitation project, it will be necessary to obtain a surface ownership, surface easement or other similar title that allows the operator of the Project to occupy the superficial terrain so as to cover and protect all manner of installations such as offices, mineral deposits, clearings, dumps, tailings, mineral extraction plants, communication systems, channels, dams, pipes, housing, transportation routes, aqueducts, and electrical lines, among others. Such easements can be requested to the local Court or agreed with the owner of the surface rights. The document that would confer these benefits would need to be registered in the Registry of Mortgages and Liens of the Mining Registry of Vallenar in order to enforce it against third parties.

4.9 Permitting Considerations

The Permitting status is discussed in Section 20.

4.10 Environmental Considerations

Productora and Cortadera have robust environmental baseline studies, completed over a 13-year period, commencing in 2012 (in the case of Productora).

Environmental baseline studies developed at Productora and Cortadera cover areas where mining infrastructure is proposed. This includes stockpiles, waste dumps, and TSF.

Studies comprise archaeological baselines, flora and fauna baselines, and groundwater monitoring and landscape analysis. A surrounding community's study has been carried out to identify the potential impacts on dwellings proximal to the Project.

Full environmental studies are currently underway for Cortadera, San Antonio and Productora/Alice.

Further discussion of the Environmental and closure plan status is discussed in Section 20.

4.11 Social Licence Considerations

The Permitting status is discussed in Section 20 of this Report.

4.12 Significant Factors and Risks

As part of ongoing assessment, HCH monitors significant factors and risks that may affect access, title, or the right or ability to perform work on the property.

A description of the main risks for this Project is presented in Section 25.12 of this Report.

4.13 Comments on Section 4

The Costa Fuego Project has in place the necessary regulatory licenses and authorisations required for its current status of Exploration according to company officials.

5 Accessibility, Climate, Local Resources, Infrastructure, and Physiography

5.1 Accessibility

The Project has a favourable location, surrounded by infrastructure which can be utilised for developing a greenfield copper project, including:

- Regional township of Vallenar
- Pan-American Highway
- Access to the Las Losas Port at Huasco
- Power substations located approximately 20 km northwest at Maitencillo, connected to the Chilean electrical grid.

The Project is 17 km south of Vallenar, a city and commune in Atacama Region, Chile. It is the capital of the Huasco Province, located in the valley of the Huasco River. Vallenar has a population of approximately 52 000 people and its main activities are farming and mining. The Vallenar airport is located approximately 3km to the south of the city centre.

The Project can be accessed from the main sealed Pan-American Highway connecting Vallenar to Coquimbo in the south.

5.2 Climate

The Project is located in two Koeppen climate classifications: BWh (dry arid desert hot) and BWk (dry arid desert cold). The biggest difference between both climates is the maritime influence in the former, that generates frequent episodes of high humidity throughout the year, creating abundant cloudiness that penetrates from the coastal sector through the Huasco river valley.

For Huasco, Freirina and Vallenar, the annual mean temperatures correspond to 14.9ºC, 15.7ºC, and 15.6ºC respectively, observing an annual mean amplitude of 4ºC, 3.4ºC and 4.5ºC respectively. The highest amounts of accumulated annual average rainfall reach 50.4 mm/year, the wettest months being between May and August. Associated with warm El Niño-Southern Oscillation (ENSO) events, strong rainfall events are concentrated over a few hours, and can bring climate hazards related to river floods, mudflows and the like.

Operations at the Costa Fuego Project would be able to continue year-round.

5.3 Local Resources and Infrastructure

The Project already has temporary facilities available to service approximately 50 people on site. These facilities may be used for initiating the Project development activities, with the site infrastructure items already in place being:

- Access road
- Temporary offices

- Power diesel generator
- Toilets and sanitation facility (water treatment plant)
- Small, fenced storage area
- Diamond drill core storage facility
- Reverse circulation (RC) drill pulp storage facility
- Drill core logging area.

5.3.1 Seawater Extraction

As discussed in Section 4.6, all water extraction rights previously held by HCH's Chilean subsidiary company Sociedad Minera El Aguila (SMEA) were transferred to Huasco Water in July 2024.

Huasco Water now holds the only active granted maritime water concession and most of the necessary permits to provide non-continental water supply to the Huasco Valley, following over a decade of permitting advance for Hot Chili Limited's coastal Costa Fuego Project. Huasco Water provides Hot Chili Limited water supply security as potential foundation off-takers and secures a sufficient water supply to support a large-scale conventional copper-gold operation.

5.3.2 Port Services

In March 2024, HCH executed a Memorandum of understanding (MOU) with Puerto Las Losas SA (PLL) for the right to negotiate a binding Port Services Agreement for the Costa Fuego Project.

HCH will fund 20% of an estimated two-year, US$4.6 million Feasibility Study for a bulk tonnage copper concentrate facility to be developed at Las Losas Port, 50 km west of the Costa Fuego Project.

In consultation with HCH, PLL shall select and commission a top-tier independent engineering company to commence and undertake the port Feasibility Study.

Following completion of the port Feasibility Study, HCH shall have a right of first refusal (ROFR) to ship copper concentrates through Puerto Las Losas facilities for three years, provided that a shipping solution is agreed at existing or potential infrastructure of PLL.

The FS will include bulk loading alternatives for copper concentrates from existing facilities, potentially with or without modifying the existing infrastructure for the operating port.

5.3.3 Electric Connection

In August 2022, Chile's Central Authority Electrical Regulator approved proceedings for HCH's subsidiary SMEA SpA to get connection to room five of Sub Electrical Power Station Maitencillo, located approximately 17 km from the Costa Fuego Project.

The application grants Hot Chili Limited the right to use the last operational room available in Sub Electrical Power Station Maitencillo and secures access to renewable energy from Chile's National Energy Grid.

Several non-binding electrical quotes have been received from market providers. HCH will be able to run the Project on a 100% renewable mix of power (certified by I-Recs), utilising nearby solar generators, wind turbines and hydroelectric sources.

5.4 Physiography

The Project is located in a transition zone between the desert plains and pre-altiplanic mountain ranges and coastal plains. The geomorphological units directly involved in the Project are characterised by a rocky and mountainous desert landscape with elevations ranging from sea level to altitudes above 1,500 m above sea level, which causes a medium to gentle relief.

The landscape is dominated by a system of mountain ranges with little vegetation consisting of bushes and cacti, typical of the coastal desert. Overall, the Project is entirely within a desert area under the influence of tropical bio-climate. Despite having strong water restrictions, due to its wide geographical spread and range of thermal conditions, it benefits from a large variety of vegetation floors. Because of the latitude and the proximity of the ocean, it receives favourable conditions for vegetation growth, such as some xerophytic plants communities. However, different vegetation communities (shrubs, cacti, amongst others) have been strongly impacted by human activity including charcoal production, heavy grazing of goats and mining developments. No permanent or seasonal courses of water are observed in the Project area, however sporadic water courses superficially flow after heavy rains.

The coastal zone is represented by coastal plains and a coastal mountain range with hills up to 1,500 m high. The Huasco river, at its mouth in the town of Huasco, has geoforms associated with river erosion, such as fluvial terraces and meanders. The interior zone is represented by moderately sloping alluvial desert plains.

5.5 Seismicity

A screening assessment of the seismic hazard at the site confirmed that the Project area is located in an area of high seismic risk. In accordance with the Chilean Guidelines the site falls within the Seismic Zone 3 of the Chilean national Design Code (NCL 2369) which indicates a peak ground acceleration of 0.4 g. However, recent seismicity in Chile has resulted in earthquakes ranging from 6.3 to 8.8 on the seismic magnitude scale. Therefore, for the purpose of this Report, the Maximum Design Earthquake (MDE) is likely to be in the order of 0.65 g.

5.6 Comments on Section 5

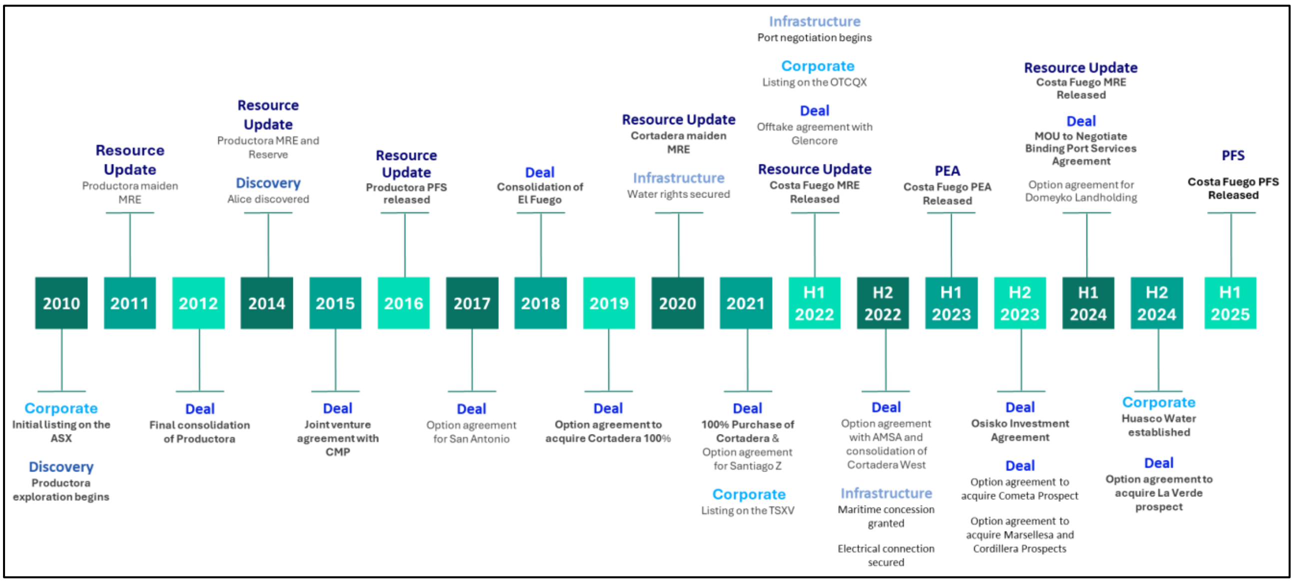
In the QP's opinion:

- There is sufficient suitable land available within the exploitation concessions for the planned tailings disposal, mine waste disposal and mining-related infrastructure such as the open pit, process plant, workshops and offices.
- Mining, processing, desalination and port activities can be conducted year-round with ready access.

6 History

A summary of the history of the activities by HCH in Chile is shown in Figure 6.1.

Figure 6.1 : Summary of HCH activities at the Costa Fuego Project since 2010 (HCH, 2025)



6.1 Productora Historical Exploration and Development Work

The Productora area has a long mining history for iron, copper and gold extending back to pre-Hispanic times. Copper mining in the past century has occurred regionally and locally at the Productora and Santa Innes mines (operated by Playa Brava and ENAMI), Remolina, and Montserrat mines.

Ownership, exploration, and development activities at Productora are summarised in Table 6.1. HCH completed acquisition of the main tenement package in 2012.

Table 6.1 : Productora Ownership and Activity History

Year	Activity	Result
1995 - 1999	General Minerals Corporation (GMC) acquired the Project and explored for Candelaria-type iron-oxide-copper-gold deposits, drilling eight RC drill holes and completing mapping, soil sampling and geophysics surveys.	Nothing publicly released.
1999 - 2005	GMC (in joint venture with Teck Corp) completed eleven RC drill holes targeting secondary copper enrichment zones in the southern portion of Productora.	A 2 km long copper-gold-molybdenum-uranium trend was defined along a north-northeast trending structure with northwest cross-cutting faults.

Table 6.1 : Productora Ownership and Activity History		
Year	**Activity**	**Result**
2008 - 2015	HCH through its wholly owned Chilean subsidiary Sociedad Minera El Aguila options and consolidates the area.	Discovery and resource drill out of the Productora copper-gold deposit. Multiple Mineral Resource Estimates completed, and development streams commenced for Pre-Feasibility Study.
2015 - 2016	Joint venture (JV) agreement between HCH (80%) and CMP (20%) owners of Sociedad Minera El Águila SpA. HCH completed extensive drilling campaigns (RC and DD), soil sampling and mapping, geophysical surveys. Limited small scale underground mining at Productora also took place at Santa Innes and Habanero.	Discovery and resource drill out of Alice copper-gold porphyry deposit. Mineral Resource Estimate completed, Post-Doctoral Study on Productora Calculated Mineralogy, and Productora Project PFS completed.
2016 - present	HCH completed drilling for exploration and metallurgical purposes and completed a significant pulp resampling campaign for silver and soluble copper.	Updated Mineral Resource Estimate released and the advancement of development streams for a PFS.

6.2 Cortadera Historical Exploration and Development Work

Ownership, exploration and development activities at Cortadera are summarised in Table 6.2. HCH completed acquisition of Cortadera in 2019.

Table 6.2 : Cortadera Ownership and Activity History		
Year	**Activity**	**Result**
1990s	Mining was completed by R.G. Grego and J.R. Alday at Purísima (Cuerpo 1).	Production included a 70 m long tunnel in the oxide zone, trenches and surface excavations.
<1993	ENAMI completed four percussion drill holes.	Defined near-surface oxide mineralisation.

Table 6.2 : Cortadera Ownership and Activity History		
Year	**Activity**	**Result**
1993 - 1994	Minera Mt Isa, Chile (MMIC) explored under a purchase option agreement with two Chilean owners (Minera Carola and Raul Flores). Work included mapping, trenching, outcrop sampling, geophysical surveys, magnetometry and drilling of 10 diamond drill holes.	Porphyry-style copper-gold-molybdenum mineralisation and propylitic alteration detected along a 2 km long by 1 km wide northwest-southeast trending mineralised corridor. Late mineral and post mineral dykes were also identified striking north-south and north-northeast to south-southwest.
1994 - 2009	Ownership by Sociedad Contractual Minera Carola (SCM Carola), mining of a small open pit completed at Purísima (Cuerpo 1) between 2003 - 2004.	Copper oxides at a grade ~ 0.9% Cu were extracted.
2009 - 2013	Mineral Fuego Limitada (Minera Fuego) under an option agreement with Sociedad Contractual Minera Carola (SCM Carola) completed 39 diamond drill holes during 2011, as well as mapping, soil sampling, rock chip sampling, geophysical surveys including magnetometry, conductivity and Induced Polarisation (IP).	Geological modelling, preliminary Mineral Resource estimation (not released) and metallurgical testwork for preliminary recovery data.
2019 - Current	Hot Chili Limited – Cortadera and Purísima through Option agreements. HCH completed extensive drilling campaigns (RC and DD), soil sampling and mapping, geophysical surveys, metallurgical and hydrogeological drilling, as well as extensive geotechnical sampling.	Discovery of depth extensions at Cortadera. Resource drill out completed resulting in three Mineral Resource Estimates being released and the advancement of development streams for a PFS.

6.3 San Antonio Historical Exploration and Development Work

There has been very limited exploration activity in areas beyond the San Antonio mine.

Historic drilling was undertaken in two periods; initially Chilean government company ENAMI completed four drill holes in 1993, and then a drilling program by company Minera Tauro (between 1998 and 2002) completed four further holes.

Hot Chili Limited completed drilling campaigns in 2019 and 2022 to define the San Antonio Resource extent, verify and infill data from previous operators.

6.4 La Verde Historical Exploration and Development Work

La Verde sits within the Domeyko landholding, which has a long history of exploration and mining completed by various companies. Documentation of exploration programs conducted by previous operators is limited. Previous exploration across the Domeyko project includes:

- 1990 – 1995, Cominco Resources – Seven RC holes of unknown length completed, soil sampling. No data available.
- 2008, BHP and Teck Cominco – Geological mapping and soil sampling. No data available.
- 2009, Rio Tinto – site visit and project appraisal. Report supplied to HCH.
- 2010 – 2014, Hudbay Minerals Inc - geological mapping, 116 rock chip samples taken (no data available), 3.4 km2 of ground magnetic surveys, 67.2-line km of Titan IP/MT surveys, eight Reverse Circulation (RC) holes drilled for a total of 2,299 m (drilled in 2010), and twelve Diamond Core (DD) holes drilled for a total of 5,774 m (drilled between 2012 and 2014). Final images and reports supplied to HCH.
- 2012, International Copper Corporation – geological mapping, trenching, rock chip sampling, final report available without raw data.

HCH acquired La Verde and commenced an RC drilling campaign in November 2024.

6.5 Production

6.5.1 Productora Production History

HCH, through its Chilean subsidiary company SMEA, entered into a lease mining and processing agreement with Chilean government agency ENAMI, with underground mining recommencing at Santa Ines mine within Productora in July 2020. As part of the agreement, mineralised material was processed at ENAMI's Vallenar processing facility, located 15 km north of Productora.

The agreement provided a low-risk pathway to bring forward first production from Productora whilst also providing certainty of mineralised material supply and employment at ENAMI's nearby processing facility in the township of Vallenar.

The Productora joint venture company SMEA (80% HCH) was paid US$2 per tonne for mineralised material purchased by ENAMI and a 10% royalty on the sale value of extracted minerals subject to ENAMI toll treatment conditions.

Under the agreement, ENAMI had a two-year concession for lease mining and processing approximately 180,000 t/a of mineralised material (through ENAMI's Vallenar plant) over a two-year period with an option to extend the agreement by a further year.

Lease mining has subsequently ceased from the two small underground mines at Productora (Santa Innes and Productora), with depletion of HCH's Mineral Resource not considered material.

6.5.2 San Antonio Historical Production

The San Antonio deposit has been privately owned since 1953 and has been mined by several operators over this time via lease from the owners. Limited historic documents provided the following production data:

- 1965-1972: produced 100,000 t at ~2.5% Cu soluble (3% Cu total)
- 1980: 30,000 t of 3.0% oxide and 25,000 t at 2.0% Cu sulphide mineralisation
- 1988-1995: ~399,000 t at 1.6% Cu.

HCH's joint venture partner has indicated that total historic production is approximately 2 Mt of material grading approximately 2% Cu and 0.3 g/t Au, however no documentation has been provided that verifies this estimate.

6.5.3 La Verde Historical Production

La Verde comprises significant historical open pit workings, where shallow porphyry copper oxide mineralisation was previously exploited by private interests across a strike extent of approximately 800m, widths of up to 200m and depths of up to 15m.

No historic documents are available regarding production.

6.6 Previous Mineral Resource Estimates

Previous Resource and Reserve estimates have been undertaken by HCH and reported either under the Joint Ore Reserves Committee of the Australasian Institute of Mining and Metallurgy, Australian Institute of Geosciences and Minerals Council of Australia (JORC Code 2012 Edition) code or to the standards of NI 43-101.

These previous estimates are provided for historical context only and the Issuer is not treating these estimates as current and they should not be relied upon.

6.6.1 Productora Resource, HCH, September 2011 (reported under JORC 2004)

Following the completion of initial project assessment, HCH commenced an extensive Resource definition drilling programme in August 2010 which was completed in early July 2011. This programme recorded several significant intercepts in extensional areas along strike from existing underground development. A total of 141 RC holes for 28,308 m and 22 DD holes for 5,012 m was drilled by HCH and were used to define the maiden Resource (Table 6.3).

Table 6.3 : Productora JORC 2004 Mineral Resource, September 2011

Reported at > 0.3% Cu

Classification	Tonnes (Mt)	Grade			Contained Metal		
		Cu (%)	Au (g/t)	Mo (ppm)	Cu (tonnes)	Gold (ounces)	Molybdenum (tonnes)
Indicated	31	0.6	0.1	159	185,000	110,000	4,900
Total	**31**	**0.6**	**0.1**	**159**	**185,000**	**110,000**	**4,900**
Inferred	54	0.6	0.1	138	298,000	180,000	7 500

Figures in the above table are rounded, reported to appropriate significant figures, and reported in accordance with the JORC Code - Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves. Metal rounded to nearest thousand, or if less, to the nearest hundred.

The information above was based on information compiled by Alf Gillman, who is a fellow of the Australasian Institute of Mining and Metallurgy. Alf Gillman was a director of Odessa Resources Pty Ltd, and had sufficient experience in mineral resource estimation, which was relevant to the style of mineralisation and type of deposit under consideration. He was qualified as a Competent Person as defined in the 2004 edition of the "Australasian Code for Reporting of Mineral Resources and Ore Reserves".

6.6.2 Productora Resource, HCH, February 2013 (reported under JORC 2004)

Following on from the estimation of the maiden Resource, HCH commenced an extensive exploration and Resource definition drilling programme at Productora in October 2011 to test for mineralisation along strike. This programme was completed in December 2012. Several significant intercepts were recorded in areas along strike (both south and north) from the existing Central Resource area. A total of 398 RC holes for 97,756 m and 27 RCDD for 11,538 m were drilled by HCH and used to define the further extents to mineralisation.

Updates were exclusively outside the extents of the September 2011 Resource. The final public reporting of the February 2013 Resource was from the combined figures of the 2011 Resource and the newly defined 2013 Resource (Table 6.4).

Table 6.4 : Productora JORC 2004 Mineral Resource, February 2013							
Reported at > 0.3% Cu							
Classification	**Tonnes (Mt)**	**Grade**			**Contained Metal**		
		Cu (%)	Au (g/t)	Mo (ppm)	Cu (tonnes)	Gold (ounces)	Molybdenum (tonnes)
Indicated	71	0.6	0.1	139	420,000	260,000	10,000
Total	**71**	**0.6**	**0.1**	**139**	**420,000**	**260,000**	**10,000**
Inferred	95	0.5	0.1	126	500,000	310,000	12,000

Figures in the above table are rounded, reported to appropriate significant figures, and reported in accordance with the JORC Code - Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves. Metal rounded to nearest thousand, or if less, to the nearest hundred.

The information above was based on information compiled by Alf Gillman, who is a fellow of the Australasian Institute of Mining and Metallurgy. Alf Gillman was a director of Odessa Resources Pty Ltd, and had sufficient experience in mineral resource estimation, which was relevant to the style of mineralisation and type of deposit under consideration. He was qualified as a Competent Person as defined in the 2004 edition of the "Australasian Code for Reporting of Mineral Resources and Ore Reserves".

6.6.3 Productora Resource, HCH, March 2014 (Reported under JORC 2012)

Following the February 2013 Mineral Resource, HCH commenced an extensive Resource definition drilling programme to infill and test near-resource mineralisation. A total of 351 RC holes for 85,645 m and 41 RCDD for 9,926 m were drilled and used to define the further extents to mineralisation. This provided a nominal 40m x 80m drillhole coverage across the majority of the Productora Resource (Table 6.5).

Table 6.5 : Productora JORC 2012 Mineral Resource, March 2014							
Reported at > 0.25% Cu							
Classification	**Tonnes (Mt)**	**Grade**			**Contained Metal**		
		Cu (%)	Au (g/t)	Mo (ppm)	Cu (tonnes)	Gold (ounces)	Molybdenum (tonnes)
Indicated	159	0.5	0.1	152	800,000	541,000	24,000
Total	**159**	**0.5**	**0.1**	**152**	**800,000**	**541,000**	**24,000**
Inferred	56	0.4	0.1	97.2	229,000	229,000	6,000

Figures in the above table are rounded, reported to appropriate significant figures, and reported in accordance with the JORC Code - Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves. Metal rounded to nearest thousand, or if less, to the nearest hundred.
The information above was based on information compiled by Mr. J Lachlan Macdonald and Mr. N Ingvar Kirchner. Mr. Macdonald was a full-time employee of Hot Chili Ltd. Mr. Macdonald was a Member of the Australasian Institute of Mining and Metallurgy. Mr. Kirchner was employed by Coffey Mining Pty Ltd (Coffey). Mr. Kirchner was a Fellow of the Australasian Institute of Mining and Metallurgy and a Member of the Australian Institute of Geoscientists. Both Mr Macdonald and Mr. Kirchner had sufficient experience that was relevant to the style of mineralisation and type of deposit under consideration and to the activity being undertaken to qualify as a Competent Person as defined in the 2012 Edition of the 'Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves' (the JORC Code 2012).

6.6.4 Costa Fuego Project Mineral Resource, October 2020 (reported under JORC 2012)

The first combined the Costa Fuego Project estimate was released in October 2020, combining the Productora, Alice, and Cortadera Resource models for the first time.

Following execution of the Cortadera option agreement in February 2019, HCH undertook a resource drill out focussed on extending and infilling previously defined mineralisation. HCH drilling was successful in improving geological understanding, growing the deposit size, and discovering a bulk-tonnage high grade zone at Cortadera. Drilling completed by HCH between February 2019 and July 2020 comprised 32 RC holes and 11 RC-DD holes for a total of 10,126 m of RC and 7,064 m of DD (Table 6.6).

Table 6.6 : Costa Fuego JORC 2012 Mineral Resource, October 2020												
Reported at > 0.25% CuEq												
Deposit	**Classification**	**Tonnes (Mt)**	**Grade**					**Contained Metal**				
			CuEq (%)	Cu (%)	Au (g/t)	Ag (g/t)	Mo (ppm)	CuEq (tonnes)	Cu (tonnes)	Gold (ounces)	Silver (ounces)	Molyb. (tonnes)
Cortadera	Indicated	183	0.49	0.40	0.15	0.70	43	905,000	728,000	889,000	4,227,000	7,900
	Total	**183**	**0.49**	**0.40**	**0.15**	**0.70**	**43**	**905,000**	**728,000**	**889,000**	**4,227,000**	**7,900**
	Inferred	267	0.44	0.35	0.12	0.70	73	1,181,000	935,000	1,022,000	5,633,000	19,400
Productora	Indicated	208	0.54	0.46	0.10	-	140	1,122,000	960,000	643,000	-	29,200
	Total	**208**	**0.54**	**0.46**	**0.10**	**-**	**140**	**1,122,000**	**960,000**	**643,000**	-	**29,200**
	Inferred	67	0.44	0.38	0.08	-	109	295,000	255,000	167,000	-	7,200
Combined Costa Fuego	Indicated	391	0.52	0.43	0.12	-	95	2,027,000	1,688,000	1,533,000	-	37,000
	Total	**391**	**0.52**	**0.43**	**0.12**	**-**	**95**	**2,027,000**	**1,688,000**	**1,533,000**	**-**	**37,000**
	Inferred	334	0.44	0.36	0.11	-	80	1,476,000	1,191,000	1,189,000	-	26,700

Reported at or above 0.25% CuEq*. Figures in the above table are rounded, reported to appropriate significant figures, and reported in accordance with the JORC Code - Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves. Metal rounded to nearest thousand, or if less, to the nearest hundred.
Copper Equivalent (CuEq) reported for the resource were calculated using the following formula:
CuEq% = ((Cu% × Cu price 1% per tonne × Cu_recovery)+(Mo ppm × Mo price per g/t × Mo_recovery)+(Au ppm × Au price per g/t × Au_recovery)+

(Ag ppm × Ag price per g/t × Ag_recovery)) / (Cu price 1 % per tonne × Cu_recovery).
The Metal Prices applied in the calculation were: Cu=3.00 USD/lb, Au=1,550 USD/oz, Mo=12 USD/lb, and Ag=18 USD/oz.
For Cortadera (Inferred + Indicated), the average Metallurgical Recoveries were: Cu=83%, Au=56%, Mo=82%, and Ag=37%.
For Productora (Inferred + Indicated), the average Metallurgical Recoveries were: Cu=83%, Au=43% and Mo=42%.
For Costa Fuego (Inferred + Indicated), the average Metallurgical Recoveries were: Cu=83%, Au=51%, Mo=67% and Ag=23%.
Note: Silver (Ag) is only present within the Cortadera Mineral Resource estimate
The information above that relates to the Productora Project Mineral Resources, was based on information compiled by Mr. N Ingvar Kirchner. Mr. Kirchner was employed by AMC Consultants (AMC). AMC had been engaged on a fee for service basis to provide independent technical advice and final audit for the Productora Project Mineral Resource estimates. Mr. Kirchner was a Fellow of the Australasian Institute of Mining and Metallurgy (AusIMM) and a Member of the Australian Institute of Geoscientists (AIG). Mr. Kirchner had sufficient experience that is relevant to the style of mineralisation and type of deposit under consideration and to the activity being undertaken to qualify as a Competent Person as defined in the 2012 Edition of the 'Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves' (the JORC Code 2012).
The information above that relates to Mineral Resources for the Cortadera Project is based on information compiled by Elizabeth Haren, a Competent Person who was a Member and Chartered Professional of the Australasian Institute of Mining and Metallurgy and a Member of the Australian Institute of Geoscientists. Elizabeth Haren was employed as an associate Principal Geologist of Wood, who was engaged by Hot Chili Limited. Elizabeth Haren had sufficient experience that is relevant to the style of mineralisation and type of deposit under consideration and to the activity being undertaken to qualify as a Competent Person as defined in the 2012 Edition of the "Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves".

6.6.5 Costa Fuego Project Resource, HCH, March 2022 (Reported under NI 43-101)

The March 2022 Resource followed 52,000 m of additional resource drilling at Cortadera. Productora was re-estimated following review of the 2014 estimate, completion of underground mine development and exploration drilling in 2021. A maiden San Antonio Resource was also added to Costa Fuego (Table 6.7).

Table 6.7 : Costa Fuego Mineral Resource, March 2022 (reported under NI 43-101)

Open Pit Resource Reported at > 0.21% CuEq*, Underground Resource Reported at > 0.30% CuEq*

Deposit	Classification	Tonnes (Mt)	Grade					Contained Metal				
			CuEq (%)	Cu (%)	Au (g/t)	Ag (g/t)	Mo (ppm)	CuEq (tonnes)	Cu (tonnes)	Gold (ounces)	Silver (ounces)	Molyb. (tonnes)
Cortadera OP	Indicated	323	0.44	0.34	0.12	0.66	53	1,411,000	1,102,000	1,284,000	6,808,000	17,100
	Total	**323**	**0.44**	**0.34**	**0.12**	**0.66**	**53**	**1,411,000**	**1,102,000**	**1,284,000**	**6,808,000**	**17,100**
	Inferred	53	0.32	0.25	0.08	0.46	62	168,000	132,000	135,000	778,000	3,300
Cortadera UG	Indicated	148	0.51	0.39	0.12	0.78	102	750,000	578,000	559,000	3,702,000	15,000
	Total	**148**	**0.51**	**0.39**	**0.12**	**0.78**	**102**	**750,000**	**578,000**	**559,000**	3,702,000	**15,000**
	Inferred	56	0.38	0.30	0.08	0.54	61	211,000	170,000	139,000	971,000	3,400
Productora	Indicated	253	0.49	0.41	0.08	-	139	1,247,000	1,043,000	646,000	-	35,100
	Total	**253**	**0.49**	**0.41**	**0.08**	**-**	**139**	**1,247,000**	**1,043,000**	**646,000**	**-**	**35,100**
	Inferred	90	0.34	0.29	0.03	-	75	305,000	259,000	91,000	-	6,800
San Antonio	Indicated	-	-	-	-	-	-	-	-	-	-	-
	Total	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
	Inferred	4.2	1.2	1.1	0.01	2.1	1.5	48,100	47,400	2,000	287,400	6
Combined Costa Fuego	Indicated	725	0.47	0.38	0.11	0.45	93	3,408,000	2,755,000	2,564,000	10,489,000	67,400
	Total	**725**	**0.47**	**0.38**	**0.11**	**0.45**	**93**	**3,408,000**	**2,755,000**	**2,564,000**	**10,489,000**	**67,400**
	Inferred	202	0.36	0.30	0.06	0.31	66	731,000	605,000	359,000	2,032,000	13,400

Reported on a 100% Basis - combining Mineral Resource estimates for the Cortadera, Productora and San Antonio deposits. Figures are rounded, reported to appropriate significant figures, and reported in accordance with CIM and NI 43-101. Metal rounded to nearest thousand, or if less, to the nearest hundred. Total Resource reported at +0.21% CuEq for open pit and +0.30% CuEq for underground.
Copper Equivalent (CuEq) reported for the resource were calculated using the following formula: CuEq% = ((Cu% × Cu price 1% per tonne × Cu_recovery)+(Mo ppm × Mo price per g/t × Mo_recovery)+(Au ppm × Au price per g/t × Au_recovery)+ (Ag ppm × Ag price per g/t × Ag_recovery)) / (Cu price 1% per tonne x Cu_recovery). The Metal Prices applied in the calculation were: Cu=3.00 USD/lb, Au=1,700 USD/oz, Mo=14 USD/lb, and Ag=20 USD/oz. For Cortadera and San Antonio (Inferred + Indicated), the average Metallurgical Recoveries were: Cu=83%, Au=56%, Mo=82%, and Ag=37%. For Productora (Inferred + Indicated), the average Metallurgical Recoveries were: Cu=83%, Au=43% and Mo=42%. For Costa Fuego (Inferred + Indicated), the average Metallurgical Recoveries were: Cu=83%, Au=51%, Mo=67% and Ag=23%.

The information above that relates to Mineral Resources for Cortadera, Productora and San Antonio which constitute the combined Costa Fuego Project is based on information compiled by Ms Elizabeth Haren, a Competent Person who was a Member and Chartered Professional of The Australasian Institute of Mining and Metallurgy and a Member of the Australian Institute of Geoscientists. Ms Haren was a full-time employee of Haren Consulting Pty Ltd and an independent consultant to Hot Chili. Ms Haren had sufficient experience, which is relevant to the style of mineralisation and types of deposits under consideration and to the activities undertaken, to qualify as a Competent Person as defined in the 2012 Edition of the 'Australasian Code of Reporting of Exploration Results, Mineral Resources and Ore Reserves'

6.7 Previous Ore Reserves

Following completion of a Prefeasibility Study on the standalone Productora Project in March 2016, an ore reserve for the Productora and Alice deposits was declared according to JORC 2012 as shown in Table 6.8.

At this time the Company was not listed on the TSXV nor reported according to NI-43101.

This previous estimate is provided for historical context only and the Issuer is not treating this estimate as current and it should not be relied upon.

Table 6.8 : Productora Ore Reserve, March 2016 (reported under JORC 2012)

Ore Type	Reserve Category	Mt	Grade			Contained Metal			Payable Metal		
			Cu (%)	Au (g/t)	Mo (ppm)	Cu (kt)	Au (koz)	Mo (kt)	Cu (kt)	Au (koz)	Mo (kt)
Oxide	Probable	24.1	0.43	0.08	49	103.0	59.6	1.2	55.6	-	-
Transitional		20.5	0.45	0.08	92	91.3	54.7	1.9	61.5	24.4	0.8
Sulphide		122.4	0.43	0.09	163	522.5	356.4	20.0	445.8	167.5	10.4
Total	**Probable**	**166.9**	**0.43**	**0.09**	**138**	**716.8**	**470.7**	**23.1**	**562.9**	**191.9**	**11.2**

Cu price - US$3.00/lb; Au price US$1,200/oz; Mo price US$14.00/lb

Weighted average metallurgical recoveries for sulphide and transitional are 86.1% for Cu; 51.9% for Au; 52.2% for Mo.

Heap leach average recoveries are 54.0% for Cu and nil for Au and Mo.

Payability factors for metal contained in concentrate are 96% for Cu; 90% for Au; and 98% for Mo. Payability factor for Cu contained in Cu cathode is 100%.

The information above that relates to Productora Ore Reserves is based on information compiled by Mr. Carlos Guzmán who was a Fellow of the Australasian Institute of Mining and Metallurgy (FAusIMM), a Registered Member of the Chilean Mining Commission (RM- a 'Recognised Professional Organisation' within the meaning of the JORC Code 2012) and a full time employee of NCL Ingeniería y Construcción SpA. NCL was engaged on a fee for service basis to provide independent technical advice and final audit for the Productora Ore Reserve estimate. Mr. Guzmán had sufficient experience which is relevant to the style of mineralisation and type of deposit under consideration, and to the activity which he is undertaking to qualify as a Competent Person as defined in the 2012 Edition of the 'Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves'.

7 Geological Setting and Mineralisation

7.1 Regional Geology

The Costa Fuego Project region lies at the boundary between the Coastal Cordillera and the Atacama fault system. During the Cretaceous, a thick sequence of andesite and minor sediments (Bandurrias Group) developed in an extensional regime within volcanic island-arc settings. A variety of porphyritic intrusions have been emplaced in this sequence, some of which are probably contemporaneous with the host volcanic rocks. These porphyritic intrusions appear to be responsible for most of the alteration and mineralisation observed in the area.

The Productora project area encompasses a small part of the Chilean Iron Belt. The Iron Belt extends for more than 600 km along a 20 to 30 km wide, north-northeast trending zone at the east side of the Coastal Cordillera. Several large copper deposits within this belt are currently in production including Candelaria, Manto Verde, and Punta del Cobre.

Beeson (2012) and Escolme et al., (2020) provide the most recent summary of the regional setting of the Productora deposit, which lies within the Chilean Coastal Cordillera, a broadly longitudinal calc-alkaline magmatic arc marking the start of eastward-migrating magmatic activity during the Andean tectonic cycle (Coira et al. 1982). This early-Andean magmatic arc comprises Jurassic-Cretaceous volcanic and minor sedimentary rocks of predominantly intermediate-felsic composition intruded by intermediate granitoids of the Coastal Batholith (Cretaceous age – from 130±1 Ma to 87±3 Ma – Reyes 1991). An ensialic back-arc basin, within which thousands of metres of shallow marine calcareous and siliciclastic sediments were deposited, is located east of the magmatic arc. Fox (2000) describes the main components of this magmatic arc in the Productora region as:

- The Bandurrias volcanic arc
- The Chañarcillo back-arc basin
- The Cretaceous coastal batholith
- The syn-arc, strike-slip Atacama Fault Zone.

The Cortadera and San Antonio prospects are located within a late Mesozoic volcano-sedimentary host-rock sequence that ranges in age from uppermost Jurassic to the middle of the early-Cretaceous. The regional stratigraphy comprises the following litho-stratigraphic units of the Chañarcillo Group (Szakács & Pop, 2001), presented in order from youngest to oldest (after Dietrich, 2012); Cainozoic gravels, colluvial and alluvial deposits overlie the now dissected Mesozoic sequence described below:

- Cerrillos Formation: well stratified basal sequence of volcano-sedimentary sequence comprising andesitic lava, volcaniclastics and tuff; overlying siliciclastic and calcareous rocks
- Pabellón Formation: lower sequence of calcareous mudstone, wackestone, marl and tuff and an upper sequence of calcarenite and fossiliferous limestone, capped by volcaniclastic rocks
- Totoralillo Formation: alternating and well stratified sequence of calcarenites, volcarenites and breccia with bioclastic lenses; similar to the Nantoco Formation in the project tenements. The uppermost portion of this sequence comprises interbedded andesite and bioclastic rocks

- Nantoco Formation: well stratified chalky sequences of fine to medium grained of limestone to marl, calcarenites and calcilutites
- Punta del Cobre Formation: variable sequence of lava, tuffs and epiclastic rocks of dacitic to andesitic composition, interspersed by siliciclastic rocks and locally limestone banks.

Much of the exposed stratigraphic succession has been variably affected by deformation. The most significant structural elements include the north-south to north-northeast-trending Agua de los Burros fault system and the sub-parallel, (presumably) linked Las Cañas fault system, collectively forming a deformation corridor up to 10 km wide. These generally steeply east-dipping and deep-tapping fault zones show both reverse and strike-slip offsets and are likely to have been reutilised during multiple episodes of deformation.

A series of northwest-trending and subordinate northeast to north-northeast-trending faults cross-cut and locally offset the larger longitudinal fault zones, forming important corridors for mineralisation, particularly at or near fault intersections. Individual fault zones comprise domains of strong brecciation that may be >10 m wide. Some of the cross-cutting faults might also be inherited structures that may have acted as transfer faults during basin formation. These fault systems are also likely to have had a significant influence on the distribution of subsequent intrusive rocks, hydrothermal alteration, and mineralisation. Dietrich (2012) proposes that the Agua de los Burros fault significantly influences the distribution of upper Cretaceous plutonic complexes that intrude the Mesozoic volcano-sedimentary sequences. These plutonic complexes form lobate, kilometre-scale bodies that vary in composition between granite, granodiorite, tonalite, and monzodiorite.

The Cortadera region shows regionally extensive sodic-calcic alteration related to the thermal aureoles of large intrusive bodies and major fault corridors. The sodic-calcic alteration observed regionally typically comprises albite- actinolite- epidote- chlorite ± magnetite/haematite ± garnet (Dietrich, 2012).

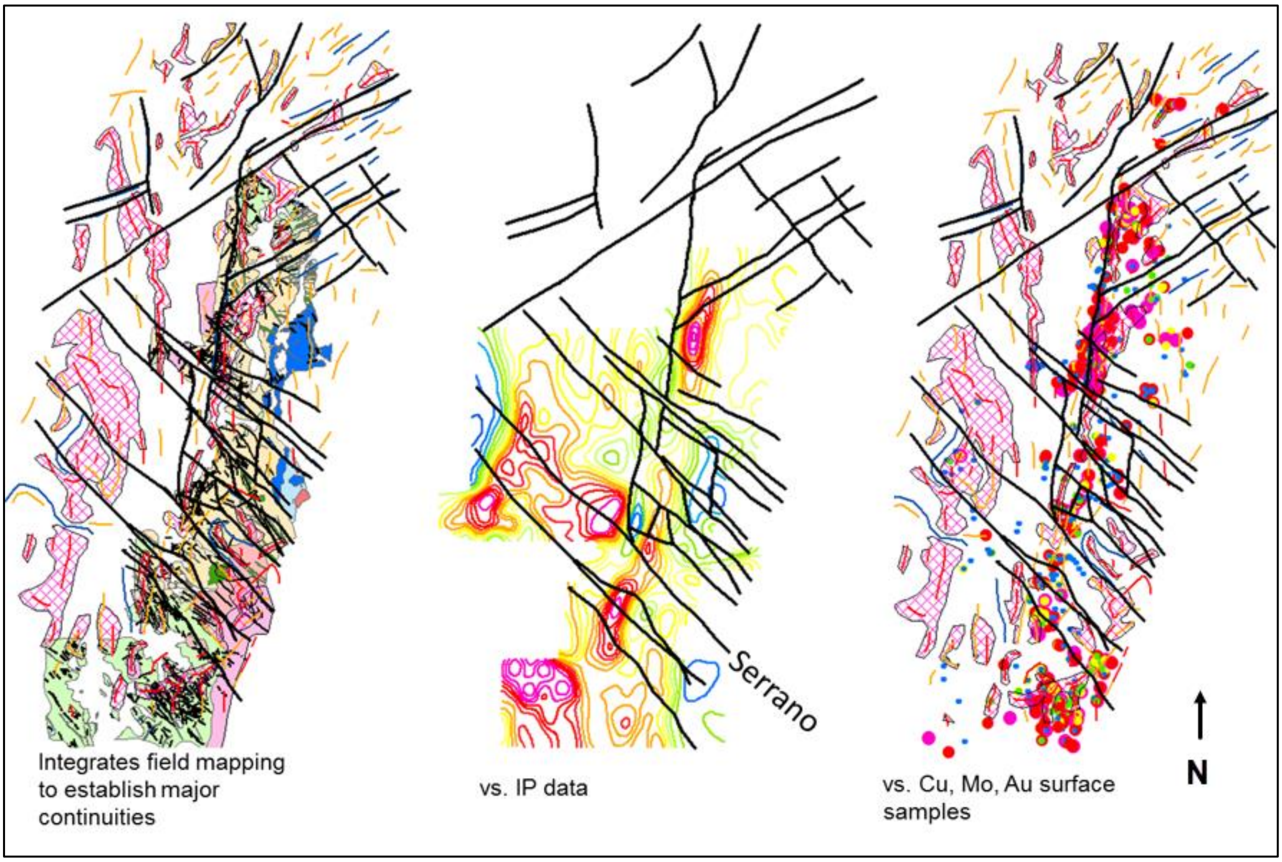
7.2 Property Geology

7.2.1 Productora-Alice Local Geology

The host volcanic and sedimentary sequence dips gently (15 to 30°) west to west-northwest and is transected by several major north- to northeast-trending faults zones, including the Productora fault zone, which coincides with the main mineralised trend. These faults are likely sympathetic to the nominally parallel but distal Atacama fault system. In the Productora deposit, these major fault zones are commonly associated with extensive tectonic breccia (damage zones) that host copper-gold-molybdenum mineralisation. Later faults crosscut and offset the volcano-sedimentary sequence together with the Productora (and sub-parallel) major faults. Late faults generally show a west to north-westerly strike and while generally narrow, are locally up to 20 m wide.

The volcano-sedimentary sequence at Productora is extensively altered, particularly along major faults and associated damage zones, and a distinctive hydrothermal alteration zonation is evident. The distribution of alteration mineral assemblages and spatial zonation suggest a gentle northerly plunge for the Productora mineral system, disrupted locally by vertical and strike-slip movements across late faults. These late faults appear to be trans-tensional and oriented at a high angle to the distal Atacama fault system (Figure 7.1).

Figure 7.1 : Broad 2D Structural Framework – Productora Deposit



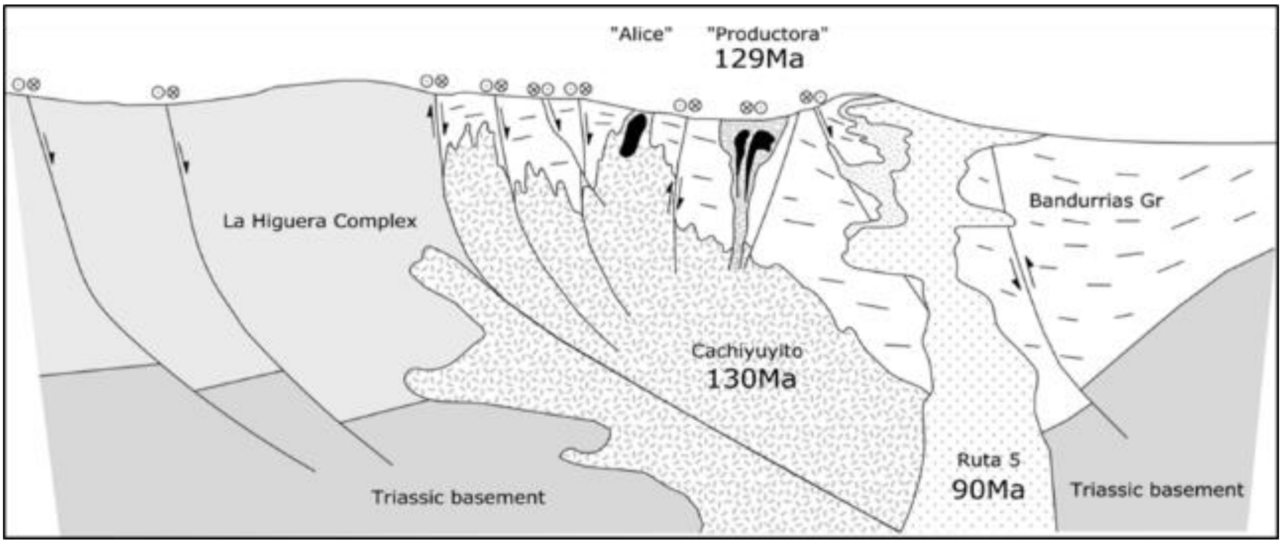
Note : Left; Compared to Ground Mapping, Centre; Compared to Previous IP Data, Right; Compared to Rock Chip Sampling. (HCH, 2016)

The following sections summarise the work described by Fox (2000), Beeson et al. (2012), Escolme et al. (2020) and HCH geologists.

The Productora copper-gold-molybdenum deposit is hosted by the Neocomian (lower Cretaceous) Bandurrias Group, a thick volcano-sedimentary sequence comprising intermediate to felsic volcanic rocks and intercalated sedimentary rocks. The Bandurrias Group consists of variably plagioclase-porphyritic and amygdaloidal andesitic rocks overlain by a felsic volcanic sequence composed of weakly porphyritic rhyolitic to rhyodacitic lavas, tuffs and volcanic breccias. Intercalated sedimentary rocks are primarily well-bedded volcanic sandstones and volumetrically minor. Surface exposures in the Project area are highly silicified reflecting significant surficial leaching. Dioritic dykes intrude the volcano-sedimentary sequence at Productora, typically along west- to northwest-trending late faults, and probably represent sub-volcanic feeders to an overlying andesitic sequence not represented in the resource area. Regionally, the Bandurrias Group is preserved as a series of linear to radiating belts and remnant blocks intruded by middle-upper Cretaceous granitoids varying from granodioritic to dioritic in composition. A large dioritic-granodioritic intrusion lies just to the east of Productora (Figure 7.2).

The volcano-sedimentary sequence at Productora is extensively altered, particularly along major faults and associated damage zones, and a distinctive hydrothermal alteration zonation is evident. Common alteration assemblages include K-feldspar-tourmaline-magnetite-silica-(hematite), typically associated with higher-grade poly-metallic mineralisation along the trace of the Productora fault zone, with flanking distal zones of relatively lower-temperature alteration comprising chlorite-magnetite-epidote-albite-silica-carbonate-(hematite).

Figure 7.2 : Stylised Regional Type Section Across the Productora Project Area. Image Looking North (Escolme, 2016)



Preferred sites for mineralisation are interpreted to be associated with fault jogs, fault intersections, fault bifurcations, damage zones adjacent to faults, and permeable volcanic units located adjacent to any of these fault-related features. It is suspected that copper-gold-molybdenum mineralisation may migrate along permeable stratigraphic units as well as fault zones, giving the impression that mineralisation is developed both along the fault zone and adjacent gently dipping stratigraphic units.

The Alice deposit is thought to be spatially and temporally linked to the Cachiyuyito/Florida system. Alice is porphyry-hosted with the causal intrusion dated to the late Cretaceous.

Mineralisation at Alice is constrained to the west by the Alice Fault. This fault dips steeply towards the west and strikes north to north-northeast. The Alice porphyry is located immediately beneath an extensive, pyrophyllite-rich advanced argillic lithocap, with a porphyry stock of quartz diorite to granodiorite composition.

Alice contains predominantly copper, with silver and molybdenum also present at lesser concentrations. Unlike the Cortadera porphyry system, little gold is present. Mineralisation within the Alice porphyry comprises sheeted and stock work quartz veinlets, within additional locally disseminated background mineralisation. Post-mineralisation albitisation can decrease mineralisation grades locally.

7.2.2 Cortadera Local Geology

A west-northwest-trending fault corridor hosts three porphyry-style mineralised centres at Cortadera. An associated colour anomaly and domain of hydrothermal alteration also extends along this trend for at least 2 300 m. It is possible the structural corridor may also extend further along strike to the west-northwest and east-southeast, influencing the location of additional porphyry-style intrusive centres and related alteration cells with associated mineralisation.

At Cortadera, the porphyry-style vein systems are associated with a multi-phase tonalitic intrusive complex showing hydrothermal alteration zonation typical of such mineralised systems. Tobey (2013b) provides a comprehensive discussion regarding the mineralised porphyry-style vein systems at Cortadera Propylitic (chlorite + epidote) alteration has been mapped at the kilometre-scale surrounding more discrete potassic (biotite and secondary feldspar) alteration zones. The extensive west-northwest-trending colour anomaly, mappable in Quebrada Cortadera and well exposed along the base of Breccia Hill (Cuerpo 3), is most likely associated with a late-stage phyllic (quartz-sericite-pyrite) alteration.

The vein systems at Cortadera appear typical of those found within porphyry-style mineralised systems elsewhere. Tobey describes some of the early quartz-rich veins observed at Purísima (Cuerpo 1) and Stockwork Hill (Cuerpo 2) as showing unidirectional solidification textures (UST). Chalcopyrite also occurs as disseminations of variable intensity within the porphyritic host rocks, particularly in association with stockwork A- and throughgoing B-veins.

The early discontinuous A-veins (quartz-chalcopyrite-pyrite±magnetite) represent the beginning of the main mineralisation event at Cortadera. The main mineralisation stages are associated with through going B-veins (quartz-chalcopyrite-pyrite±molybdenite) and the later chalcopyrite-pyrite-bearing C-veins. A- and B-type quartz veins are most abundant in the centre of the porphyry system, where the copper metal tonnage is largest. Sulphide mineral-bearing C-veins are associated with mineralisation but are volumetrically less significant within the deposit than the A- and B-type quartz veins. D-veins (quartz-pyrite-sericite) with feldspar-destructive selvedges and late calcite-bearing fractures formed subsequent to C veins. Anhydrite is locally present within some of the B- and C- veins and disseminated within the wall-rock.

Multiple phases of tonalite intrusion are present at Cortadera, distinguished primarily by weighted volume of A- and B- type quartz veining present. At least two mineralised phases exist (early- and intra-mineral) as well as late-stage, post-mineral dykes which cross-cut the mineralisation. These dykes contain <1% A- and B-veins and host extremely low copper-gold-molybdenum grades unless xenoliths of the mineralised porphyry are present within.

Late-stage andesite dykes also post-date the mineralisation and are easily identified by their darker colour compared to the tonalitic intrusions, and their lack of A- and B-veins. These also host extremely low copper-gold-molybdenum grades.

7.2.3 San Antonio Local Geology

The San Antonio deposit has been interpreted as a skarn copper deposit with mineralisation presenting in lodes with strong structural and lithological control. The deposit is characterised by mineralisation along an NNE-SSW trending shear zone through the host rocks, which comprise a shallowly east-dipping sedimentary and volcanic sequence.

On the eastern margin of the deposit is an andesitic volcano-sedimentary sequence that consists of massive to porphyritic, vesicular/amygdaloidal lava flows and tuffs. This is underlain to the west by an extensive sedimentary sequence. These predominantly siliciclastic rocks comprise fine- to medium-grained arenitic sandstone and plagioclase-rich volcanic-derived wacke interbedded with siltstone and shale that may be weakly graphitic. The siliciclastic rocks are locally calcareous and include fine grained limestone sequences south of the San Antonio mine. These rocks typically show planar and graded bedding at millimetre- to centimetre-scale and local crossbedding. Sedimentary structures suggest that the sequence is broadly upright and shows eastwards younging.

Mafic and felsic dyke intrusions are common through the San Antonio deposit, mostly striking NE-SW and dipping steeply to the east. The abundance of structure and dyke is highest in the central section of San Antonio (decreasing to the north and south). Structure at San Antonio is interpreted as being due to the emplacement of an intrusion at depth, rather than crustal scale faulting.

Alteration is strongest proximal to the mineralisation, with chlorite-epidote being the most common assemblage.

Mineralisation is focussed on the through-going San Antonio shear and associated fault zones (nominally less than 2 m width - striking between N30E and NS) and the cross-cutting mafic dykes. The intersection lineation between these structures is interpreted to plunge approximately 30° to the south and is thought to be a significant control on mineralisation.

While the mafic dykes can be mineralised (although only displaying weak to moderate alteration), the intensely skarn altered (epidote-chlorite) fault zones are the more significantly mineralised. Mineralisation is observed both as supergene and hypogene principally associated with high levels of epidote-chlorite alteration.

Two main mineralisation events have been identified from petrography:

1. Skarnification of the host-rocks following diorite intrusions - mineralisation comprising pyrite with trace chalcopyrite
2. Intergrowth of chalcopyrite and specular hematite, and fracture-filling quartz-calcite veinlets

7.3 Property Mineralisation

7.3.1 Productora Mineralisation

The Productora copper-gold-molybdenum deposit is an enigmatic breccia complex that presents characteristics consistent with both the porphyry and Iron Oxide Copper-Gold (IOCG) models (Escolme, 2016).

Mineralisation in the Productora deposit comprises two contrasting styles. The predominant style is characterised by narrow, north to northeast trending tourmaline-cemented breccia bodies. Sub-vertical feeder stocks, 2 to 5 m width at depth, increase with elevation, to wider high-grade mineralisation zones. These wider brecciated zones vary in orientation, with central lodes tending to be sub-vertical with an upper flex in wider mineralised zones to dip approximately 70° towards the west and flanking shallower eastern and western lodes dip moderately west and east respectively. There are also some locally steeply east dipping lodes e.g. Habanero.

The host breccia has been modelled from drill hole data over a strike length of 7,900 m. The breccia does not outcrop within the lease area although it has been observed extensively in drill core and in the underground workings.

In structurally favourable dilation zones, these discrete breccia zones hydraulically propagate outward and can commonly coalesce to become larger zones of hydrothermal damage. These larger damage zones are most probably defined by a combination of structural and intra-lithological controls. Drilling at deeper levels at Productora has demonstrated thinning breccia lodes, with some ductile features, that continue to a greater depth.

The copper, gold, and molybdenum mineralisation are also strongly coincident with the potassic alteration. Determining the detailed primary host lithology, within and proximal to mineralisation, is problematic due to the extent of brecciation and hydrothermal alteration.

Secondary and relatively lower-grade mineralised material controls are evident as manto or manto-like horizons in the southern, far northern, and far eastern flanks of Productora. Manto mineralisation appears to be locally focused along the upper part of the volcanic breccia and intercalated, weakly-foliated volcanic and sedimentary rocks. Lodes within the manto horizons are typically gently-dipping at 20° to 30° to the east or west and enclosed by lower grade mineralisation. Also, relative to the Productora breccia mineralisation, manto mineralisation typically exhibits elevated levels of iron (in hematite or magnetite) and calcium (in calcite).

The Productora deposit mineralisation is currently considered to have formed (relatively) distal and higher than at Alice. Although porphyry-type mineralisation has not been recognised to date at the Productora deposit, it is postulated that the tourmaline-cemented breccia and copper-gold-molybdenum signature strongly favours a porphyry model rather than an IOCG model (Sillitoe 2015 – internal company report).

7.3.2 Productora Supergene Mineralisation

The depth of supergene profile at Productora appears directly related to local porosity. The porosity itself is a function of lithology and structure and protection provided by topographic relief. At the Productora deposit, the impact of supergene weathering and alteration is deeper and shows a generally downward zonation in terms of decreasing oxide context. This is a product of the brecciated and fractured nature of the mineralisation, as well as the lack of any significant topographic protection for much of the geological timeframe. In the areas of supergene alteration closest to surface, there appears to be some minor mobility of (low grade) copper, but higher grades remain locally intact. In terms of mineral alteration, the supergene overprint of the pervasive project scale hydrothermal alteration is such that magnetite, sulphides, and hematite are commonly altered to goethite, copper oxides, carbonates and silicates, and chalcocite, covellite and digenite.

7.3.3 Alice Mineralisation

The Alice copper-molybdenum porphyry mineralisation likely formed deeper than the Productora mineralisation, in terms of genetic emplacement, and has a single porphyry body near a remnant lithocap.

The lithocap is physically disconnected with the Alice porphyry. The lithocap overprints the regional volcanic stratigraphy and can been seen in multiple silica ridges. It is comprised of numerous advanced argillic alteration types, including quartz-alunite, quartz-pyrophyllite, alunite-dominant and pyrophyllite-dominant zones.

Within the zones of mineralisation, there appears to be a distinct domain difference between chalcopyrite-dominant and pyrite-dominant areas. Chalcopyrite-dominant zones (i.e. low pyrite : chalcopyrite ratio) correlate with intense A- and B-veins and higher copper grades. Copper mineralisation appears both within veining and disseminated within the groundmass proximal to veining.

Late albite (±epidote ±sericite) alteration appears to have removed chalcopyrite (copper, sulphur) and biotite. Albitic alteration also appears to locally reduce the amount of pyrite in the quartz vein network. This can also be seen in the sodium and sulphur chemistry in the Alice drilling; both correlate with areas of significantly lower copper grade.

Molybdenum mineralisation appears discretely associated with vein networks and appears to be less impacted by the late albite alteration compared to the chalcopyrite (most impacted) and pyrite (moderately impacted). This may be primarily driven by the majority of molybdenite being contained within discrete quartz veins which may offer some protection to late fluids.

Late supergene weathering has impacted some of the Alice mineralisation. Overall, the higher-grade domain appears to be effectively in situ, with perhaps some minor down-fault upgrading via fluid movement. The low-grade mineralisation appears slightly more impacted, with minor lateral spread within the oxide domain.

7.3.4 Cortadera Mineralisation

An interpreted WNW-trending fault corridor hosts the three porphyry-style mineralised centres at Cortadera (Cuerpo 1, 2, and 3). Mineralisation continues to at least 1 km below the surface and 2.3 km along strike.

The Cortadera deposit is characterised by early- and intra-mineralisation, porphyritic tonalitic to quartz dioritic intrusions and adjacent volcano-sedimentary wall-rocks that have locally been recrystallised to hornfels and skarn. Mineralisation tenor and distribution is consistent with that seen in similar porphyry copper-gold-molybdenum deposits; a strong correlation with A- and B- quartz veining and associated chalcopyrite.

The presence of a calcium-rich alteration front is considered to exert a significant geological control on mineralisation and appears to correlate well with zones of higher A- and B-type quartz vein abundances and copper grades that extend outward from the mineralised porphyry intrusions. This geometrical relationship is consistent with the addition of potassium and sodium to the porphyry core (along with copper, gold, molybdenum, silver and other metals), where calcium has been depleted. The calcium has been remobilised and driven outwards along permeable pathways that developed in zones of higher fracture- and vein-abundance and within adjacent competent hornfels and permissive stratigraphic units.

7.3.5 Cortadera Supergene Mineralisation

The presence of near surface limonites with varying copper content occur at Cortadera, as well as minor chalcocite, tenorite, and clays with copper oxides. These limonites are wholly contained within the oxide and transition and are considered a supergene enrichment zone, particularly at Cuerpos 1 and 2.

7.3.6 San Antonio Mineralisation

At San Antonio, mineralisation presents as a main structurally controlled lode, and a series of splay lodes. The main lode is the most significant along strike and in width and has been the focus of both open pit and underground mining at San Antonio. Splay lodes are generally lower grade, with continuity ranging from tens to hundreds of metres.

The dominant sulphide species at San Antonio are chalcopyrite and pyrite, which occur as disseminations around the fault zone. High copper grades (up to 2%) occurring along these fault zones is associated with intense epidote > chlorite ± magnetite ± albite ± calcite and minor specular hematite. Away from the primary structure, alteration observed in the surrounding volcano-sedimentary host rocks occurs as weak chlorite > epidote ± quartz ± calcite ± sericite, associated with a pyrite-dominant halo (up to 25%) with magnetite (up to 10%).

This broader alteration zone likely relates to the first skarnification event discussed in Section 7.2.3 and contains only relatively low copper grades (up to 0.3%).

Chrysocolla and malachite are observed as oxide copper mineralisation near surface, however a continuous oxide defined blanket is not observed across the deposit.

8 Deposit Types

8.1 Costa Fuego Project Deposits

The Costa Fuego Project being advanced by HCH encompasses several different deposit types and mineralisation styles. Cortadera and La Verde mineralisation are associated with a classic copper-gold-molybdenum porphyry deposits, while San Antonio mineralisation is interpreted to be a lode-style copper skarn deposit often found proximal to porphyry deposits. Productora displays characteristics of both IOCG and Manto-type copper mineralisation, but subsequent structural deformation and alteration had made classification complicated. The Productora proximal Alice mineralisation is a copper-molybdenum porphyry.

A summary of the general characteristics of these mineralisation styles is found below.

8.1.1 Iron Oxide Copper-Gold and Manto Deposits

Geology and mineralisation at the Productora deposit were first documented in an MSc study by Fox (2000) titled ‘Fe-oxide (Cu-U-Au-REE) mineralisation and alteration at the Productora prospect’. This provided the most comprehensive documentation of the geology of the Productora deposit prior to ownership by HCH. Fox (2000) concluded that Productora was similar to the Candelaria deposit — a magnetite-dominant IOCG with significant sulphide mineralisation associated with potassic alteration.

A variety of IOCG mineralisation styles have been documented including veins and stockworks (hosted by intrusive rocks, particularly equigranular gabbrodiorite and diorite), hydrothermal breccias, calcic skarns, replacement horizon (mantos) and ‘composite’ styles, which generally include veins and a combination of other styles (Sillitoe 2003). Composite deposits hosted within volcano-sedimentary sequences, such as Candelaria, tend to be the largest and have the most complicated alteration assemblages and over printing relationships (Sillitoe 2003).

Alteration in IOCG deposits is typically complex and varied. A generalised upward and outward zonation is typically observed in deposits that have sodic, calcic and/or potassic alteration, deep magnetite-actinolite-apatite transitions to shallow specular hematite-chlorite-sericite (Escolme 2016). In general, tourmaline is common, and quartz is sparse, although not totally absent (Sillitoe 2003).

Productora also displays characteristics of Manto-type copper mineralisation; they are generally hosted in Mesozoic basaltic and andesitic, volcanic and volcano-sedimentary rocks in peripheral locations to coeval dioritic to granodioritic plutons (Maksaev and Zentilli 2002; Sillitoe and Perrelló 2005). Mineralisation occurs in stratabound disseminated bodies and steeply dipping hydrothermal breccias surrounding barren diorite intrusions with associated veins (Sillitoe and Perrelló 2005). The highest grades are typically found in zones of high permeability, such as permeable faults, hydrothermal breccia, dyke contacts, vesicular flow tops and flow breccia. The dominant hypogene sulphide phases are bornite, chalcopyrite, chalcocite, and pyrite plus occasional covellite and digenite, with additional minor sphalerite and galena identified in the Early Cretaceous deposits (Maksaev and Zentilli 2002).

Hypogene gangue minerals include quartz, hematite, pyrite, chlorite, albite, calcite, and local magnetite as well as zeolites, epidote, and bitumen in the Early Cretaceous deposits (Maksaev and Zentilli 2002). Hypogene mineral zonation has been observed at a number of deposits, including Mantos Blancos, Santo Domingo, Lo Aquirre and El Soldado, with high grade cores centred on redox fronts in the host stratigraphy (Sillitoe 1992;

Sillitoe and Perrelló 2005). The deposits are characterised by a core of copper-rich minerals (chalcocite – bornite ± digenite), surrounded by successive zones of chalcopyrite ± bornite, chalcopyrite - pyrite and pyrite (Maksaev and Zentilli 2002; Sillitoe and Perrelló 2005).

Most of the manto-type deposits are weathered and have an upper oxidised zone. However, supergene enrichment has only occurred at some of the larger deposits. This is probably due to low pyrite abundance, which limited acid production, and abundant calcite gangue which neutralised acid fluids (Maksaev et al. 2007). Copper oxide minerals include atacamite, minor chrysocolla, malachite, copper sulphate and rare cuprite and native copper (Maksaev et al. 2007).

8.1.2 Skarn Deposit

San Antonio presents as a skarn-like deposit, with mineralisation controlled by narrow (1-2 m width) north-east trending fault zones. Skarn deposits are developed due to replacement, alteration and contact metasomatism of typically carbonate host rocks by ore-bearing hydrothermal solutions adjacent to an intrusive body.

Geologically, the first mineralisation event which occurred at San Antonio was related to the intrusion of a quartzodiorite, which generated stratabound bodies related to the metasomatic replacement of the calcareous and volcano-sedimentary units, with silica-epidote-garnet, silica-epidote, and garnet-silica.

The second event related to the presence of quartz-dioritic and dacitic porphyries, which cut through the skarnified areas and outcrop in the vicinity of the San Antonio mine. This second mineralisation event is spatially related to an epidote-chlorite-sericite alteration, in addition to a few quartz and calcite veinlets. The degree of alteration varies from moderate to intense, based on proximity to the mineralised zones.

The third event, and the most important from an economic point of view, is associated with the filling of veinlets and fractures by chalcopyrite-specular hematite-magnetite-quartz-calcite within fault zones. Petrological studies indicate that this mineralisation event is associated with the replacement of chalcopyrite by specular hematite and the replacement of pyrite by chalcopyrite. These replacements are spatially related with quartz-dioritic porphyry, but the genetic relationship is not clear.

8.1.3 Porphyry Deposits

Cortadera is a copper-gold-molybdenum porphyry hosted mineral deposit type, comprising a series of mineralised centres (Cuerpo 1, 2 and 3) within a NW-trending structural corridor.

The Alice copper-molybdenum porphyry deposit is situated 400 m to the west of Productora.

Porphyry systems are typically characterised by:

- Deposits occurring in clusters
- Small diameter (0.5 to 2 km) causative intrusions of intermediate to felsic composition
- Shallow depth of emplacement (1 to 4 km)
- Porphyritic texture, being millimetre scale phenocrysts in a sub-millimetre scale groundmass
- Multiple phases of intrusion, including pre-, early-, syn-, late- and post mineralisation

- Extensive hydrothermal alteration associated with each mineralising intrusion
- Fracture and vein-controlled alteration and mineralisation
- Metal zoning, comprising a central zonation of iron-copper-gold, with proximal molybdenum and distal gold-silver-lead-zinc
- Sulphide and oxide minerals which vary from early magnetite-(bornite) through transitional chalcopyrite-pyrite-(hematite) to late pyrite, pyrite-bornite or pyrite-enargite-(covellite)
- Temporal fluid evolution, with early higher temperatures with high salinity to later lower temperatures of lower salinity (Garwin 2019).

An example of the geometry of a porphyry system in arc settings is shown in Figure 8.1 and the hydrothermal alteration of the same system in Figure 8.2.

Figure 8.1 : Example Cross Section of Porphyry Systems in an Arc Setting, Showing Geological Variations (Sillitoe, 2010)

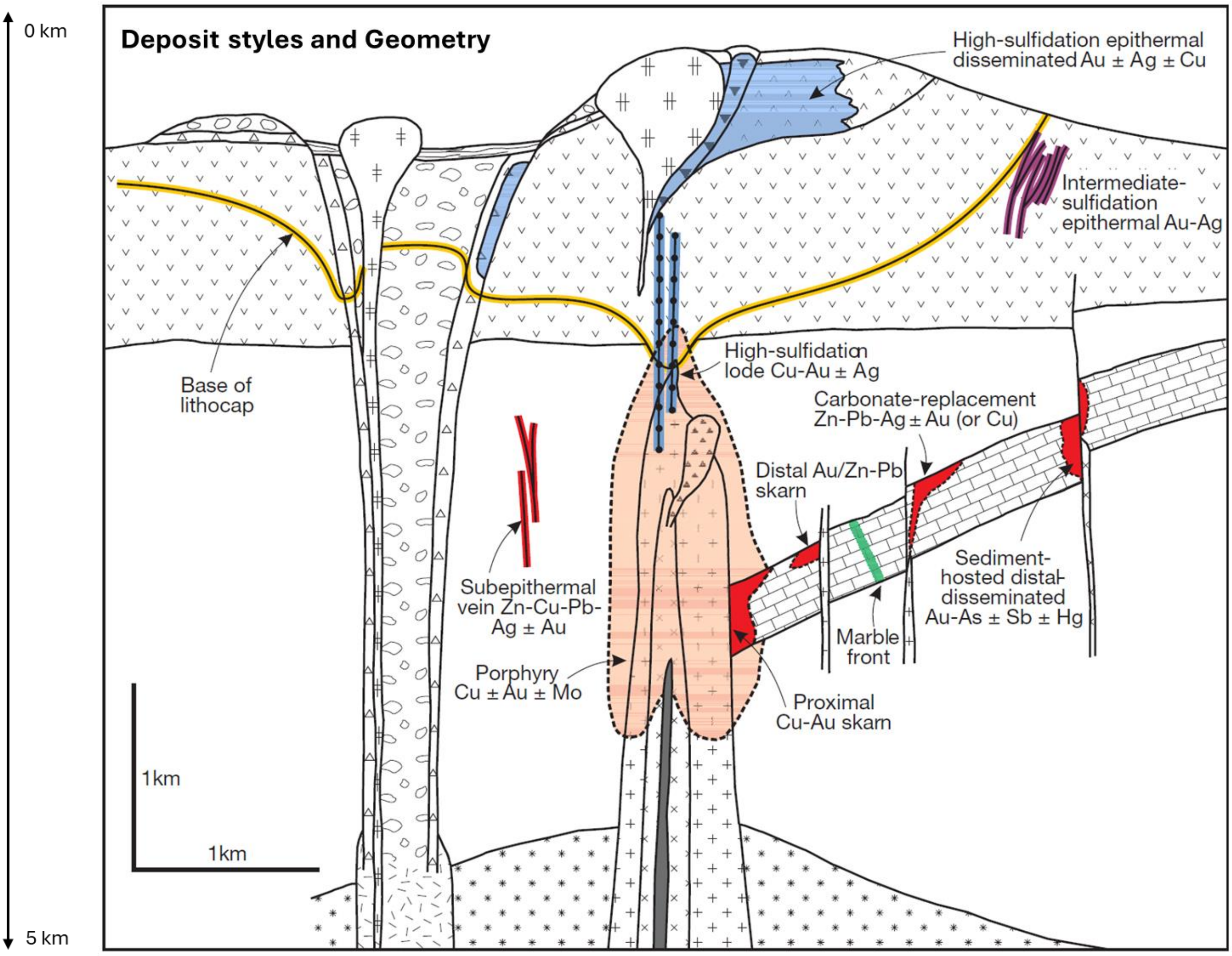
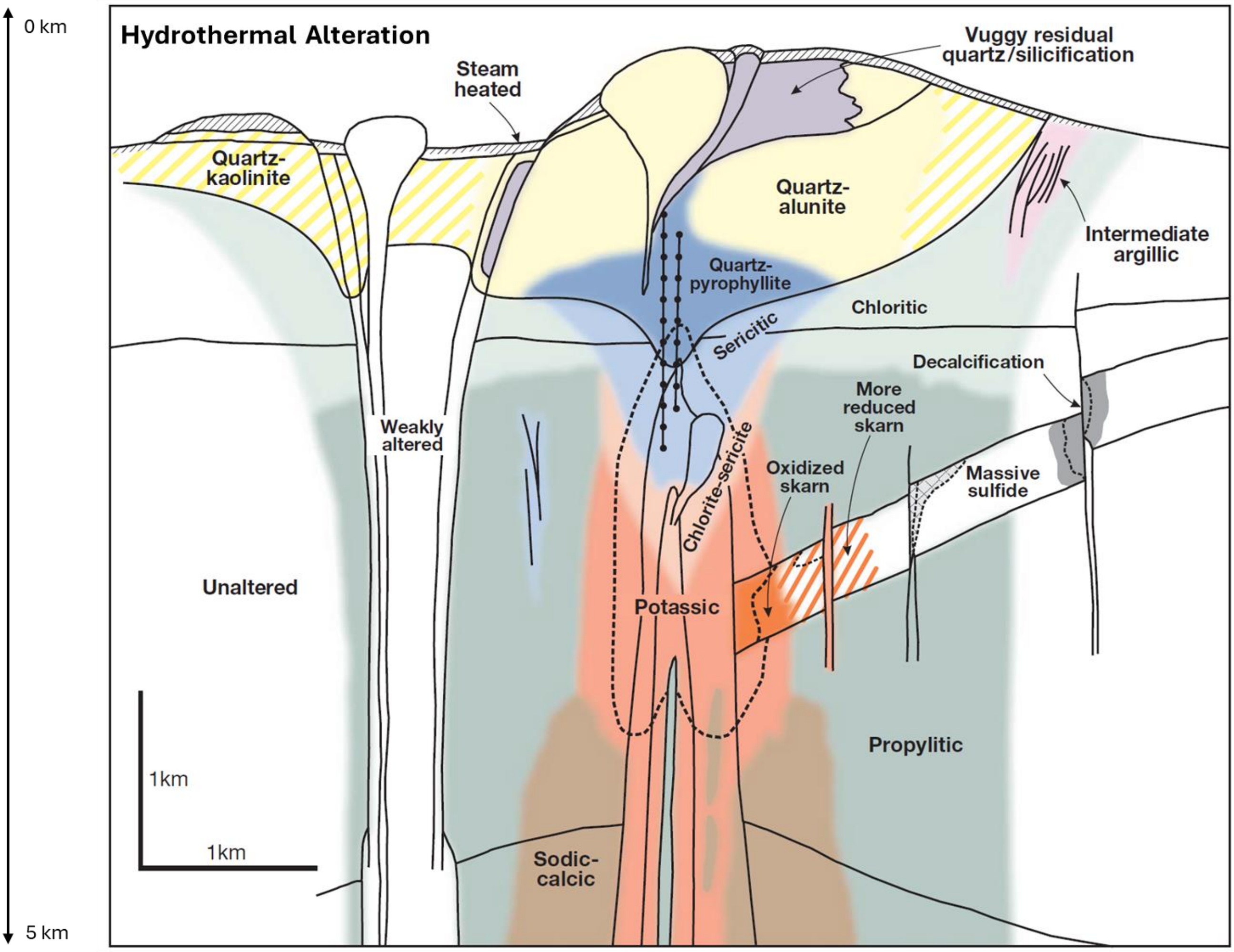


Figure 8.2 : Example Cross Section of Porphyry Systems in an Arc Setting, Showing Alteration Variations (Sillitoe, 2010)



9 Exploration

9.1 Grids and Surveys

9.1.1 Grid

The coordinate system in use for the deposits is UTM Zone 19S, WGS-84 datum.

Geodesia Topograpfia Exploraciones provided a coordinate transformation program that allows coordinate conversion in various systems, WGS84 <> PSAD56 and WGS84 <> LTM.

A global positioning system (GPS) network for the proposed Productora plant site was prepared including 20 survey monuments to be used for the next stage of engineering design. A topographic coordinate conversion program was provided to correlate data from one datum base to the other. All the survey restitution work was performed by GeoImage.

9.1.2 Surveying

At Productora and Alice, topographic control was from a detailed aerial survey of the proposed Productora plant site area using a scale of 1:1,000 and 1 m contour spacing, prepared by GeoImage for HCH in June 2021. Topography at 1:2,000 scale was used for other areas. The topography covers an area of approximately 16,000 ha for the planned Productora plant site and pipeline route. The supporting grid for the planned mine and plant area and the pipeline system consists of six main points and a secondary grid of 53 points.

At Cortadera, topographical data was supplied by Minera Fuego upon exercise of the purchase option agreement for the Project. This survey also covered the San Antonio project area. Topographical surveys were undertaken by contract survey company Geodesia Topograpfia Exploraciones in 2011 and 2012 with the following methodology utilised for the survey.

Base Station “SIRGAS VALLENAR at HM Cortadera 1 to 40” was selected as the control point and the survey was undertaken within a polygon area, followed by:

- Measurement with Dual Frequency GPS Ashtech Brand Model Z-Xtreme and Allegro CX Electronic Notebook
- HM Cortadera Base Station 1 to 40 Z-Xtreme GPS Equipment
- Rover or Mobile Equipment, Z-Xtreme with Allegro System Notebook (RTK) Real Time Kinematic
- The topographical data points were then used to create a digital elevation model (DEM). The software used to create the DEM is unknown.

9.2 Geological Mapping

Extensive geological mapping has been completed across the Costa Fuego Project by HCH since 2010, this is summarised in Table 9.1.

Table 9.1 : Summary of Geological Mapping Completed at the Costa Fuego Project by HCH

Year	Project Area	Activity	Result
2010	Productora	Data compilation and validation of historical data from several sources including hard copy reports, published TSX announcements, and both hard copy and digital maps. Ground reconnaissance was also completed.	This work showed that the mineralisation at Productora is hosted within relatively permeable units of a felsic-intermediate volcanic sequence. The mineralisation was evident in a series of permeable units and fault-controlled disseminations and breccia that trend north-south, east-west and northwest-southeast. Jogs and intersections between fault sets, as well as between faults and permeable volcanic units, appears to have assisted the mineralisation process.
2013	Productora	Extensive mapping and rock chip sampling studies were completed by Dr John Beeson and other HCH employees at Productora.	Mapping was completed at 1:2500 scale across ~4 000 ha which detailed the regional lithology, structure, strain, and alteration relationships. More than 1,000 rock chip samples were also taken, which contributed to a 3D geology model of Productora which assisted in developing a resource estimate and provided multiple exploration targets.
2018	San Antonio	Geological mapping at 1:500 scale to create detailed maps of the area, improve understanding of mineralisation controls, and construct a simple 3D view of mapping data to facilitate exploration targeting.	Major structural features identified included folding, doming, faulting, and variably penetrative fabrics. Visible copper mineralisation identified at San Antonio was observed to be strongly controlled by narrow (1-2 m width) fault zones striking north to northeast and dipping steeply northwest/southwest or east to west. These fault zones carried visible copper mineralisation.
2019	Cortadera	Detailed mapping and rock chip sampling studies were completed by Dr John Beeson and other HCH employees along the Cortadera Cuerpo 1, 2 and 3 trend, as well as immediately north of Cuerpo 3 and at Cortadera Norte, to increase geological understanding and assist in targeting extensions to mineralisation.	Mapping was completed at either 1:1000 or 1:2000 scales using the Anaconda method (lithology, structure, veining, alteration) and included compilation of a geological map as well as compilation of a table of mapping points detailing lithology, alteration, structural features, vein type and abundance, strain intensity and rock chip sampling details. Surface mapping procedures were developed by HCH's Chief Technical Advisor, Dr Steve Garwin.

Year	Project Area	Activity	Result
	Cortadera (Cuerpo1-3 Resource Window)	Detailed geological mapping of the Cuerpo 1 to Cuerpo 3 area was conducted at 1:1000 scale focusing upon surface outcrops and exposures along tracks and creeks.	Elements of at least three regional stratigraphic units were recognised: the Punta del Cobre, Nantoco, and Totoralillo Formations. These rocks are variably overprinted by hydrothermal alteration, forming extensive skarn and hornfels zones around the porphyry systems. Various dykes of the pre-mineral, mineral, and post-mineral porphyry stages were recognised, following criteria defined by Tobey (2012b). Surface mapping showed that the pre-mineral and inter-mineral copper ± molybdenum ± gold porphyries are structurally controlled by north-south, west-northwest, northwest and northeast faults. This structural control is also observed in the distribution of the A-, B- and D-type veins. It was also observed that the A- and B-type veining abundance within the intrusions varies between < 1% to 8-10%. Cuerpo 1: Four main intrusive units were mapped at Cuerpo 1 (PD1, PD3, PD4 and PT1) based upon early quartz-vein content, sulphide mineralogy and hydrothermal alteration, together with petrographic observations. Cuerpo 2: Two dominant mineralised porphyry phases were distinguishable in outcrop at Cuerpo 2 (PD3 and PD4) based upon mineralogical and textural characteristics, as well as variations in early quartz vein content. Cuerpo 3: Six intrusive phases were mapped at Cuerpo 3 (PD, PD1, PD3, PD4, PT1 and PT2) based upon variations in early quartz vein abundance, sulphide mineralogy, hydrothermal alteration, as well as textural and mineralogical characteristics.
2019	Cortadera (Cuerpo 3 North)	Reconnaissance geological mapping of Cuerpo 3 Norte (Beeson 2019) targeted a geophysical anomaly semi-coincident with the north-south	Mapping showed that the Cuerpo 3 Norte area is dominated by chlorite skarn with variable intensity epidote-garnet alteration, as well as silica-sericite-chlorite hornfels,

Year	Project Area	Activity	Result
		dyke corridor trending immediately north of Cuerpo 3.	with overlying limestone exposed to the west.
	Cortadera (Cuerpo 1 North)	Geological mapping of the Cortadera Norte target area (Beeson 2019) targeted a coincident geophysical and geochemical anomaly.	To the northeast, of Cuerpo 1 a domain of strong to intense vein density with a northwest trend was mapped (i.e. sub-parallel to the Cortadera valley). Pyrite-bearing late tonalitic dykes observe a lack of quartz veining and show an intense supergene argillic alteration overprint.
2022	San Antonio	Geological mapping at 1:200 scale for specific areas of the Project, supplementing drill campaigns, was undertaken to improve understanding of key lithologies, structures and mineralisation controls.	Mapping confirmed the occurrence of mineralisation along narrow (1 – 2 m wide) NW and NS striking fault zones associated with intense epidote skarn alteration. The host stratigraphy includes a series of volcanoclastic quartz-bearing sandstones, calcareous sandstones, with the presence of andesitic tuffs, breccias and lavas to the east of the major regional structure, the Agua de los Burros fault. Several diorite dykes and minor felsic and andesitic dykes were identified which intrude the volcano-sedimentary host rocks. At surface oxide mineralisation was also observed to be associated along the contacts of these diorite dykes.
2022	Cortadera (Las Cañas Quebrada, Cuerpo 4)	Detailed Surface mapping at 1:1000 scale of roads and outcrops of the N-S trending Las Cañas Quebrada between the Cuerpo 1 and Cortadera North areas.	Detailed mapping identified the presence of a series of granodioritic to tonalitic intrusions alike to the PD, PD3, PD4 and PT seen at Cuerpo's 1, 2, 3. Strong chlorite – sericite ± albite ± quartz alteration was observed proximal to the intrusions, zoning to weak epidote – chlorite ± calcite ± garnet in the surrounding volcano-sedimentary sequences. The orientation of dykes, veining and faulting is dominated by NNE-SSW, NW-SE, and NE-SW trends, consistent with that observed at Cortadera. A- and B-type quartz veins > 1% were mapped along a 650 m strike extent.
2023	San Antonio	Detailed field review of mineralisation, structure, lithology, and alteration over select areas of the San Antonio resource, where	The mineralised structures were clearly observed at surface with localised quartz – limonite – chlorite - epidote alteration and associated copper oxides (brochantite, chrysocolla, and black copper oxides).

Year	Project Area	Activity	Result
		the main structure was interpreted to extend to surface.	
2024	Productora	Detailed field mapping to the west and north of Alice and Productora, focusing on outcropping areas with geophysical anomalism and outcropping silica ridges (in the west) and beneath the planned location of the tailing's storage facility (TSF) in the north. Mapping included lithology, alteration, structure, and mineralisation.	Detailed mapping around the N-S striking silica ridges identified areas of intensely advanced argillic altered (quartz-pyrophyllite ± jarosite ± alunite ± dickite) felsic volcanics. Strong pyrophyllite and/or alunite alteration was recognised to be associated with vuggy silica textures. Beneath the TSF footprint mapping identified a sequence of andesitic and felsic volcanics intruded by a felsic hypabyssal stock (the Cachiyuyito intrusion) which are covered by colluvial and alluvial deposits in the creeks.

9.3 Geochemical Sampling

HCH has completed numerous geochemical sampling programs across the Costa Fuego Project, they are summarised in Table 9.2.

Geochemical samples are considered representative and free from bias. Soil sampling is completed on a grid pattern using a documented and repeatable approach. Rock chip sampling is completed on an ad-hoc basis to complement geological mapping and construction of geological models.

Table 9.2 : Summary of Geochemical Sampling Completed at the Costa Fuego Project by HCH

Year	Project Area	Activity	Result
2010-2015	Productora	2,764 soil samples taken during a large, systematic program across the entire Project area. Samples were collected on a 400 m x 400 m staggered grid across the tenement package, with infill sampling completed in high priority areas on a 400 m x 200 m spacing. Samples were nominally taken in an area of 30 cm x 30 cm, at a depth of 15 cm to 20 cm. A 500 - 800 g sample was put through a 2 mm sieve with all passing material collected for assay. The samples were assayed at ALS laboratories by four acid digestion and ICP-MS which provided a 48-element analysis of all samples.	Geochemical sampling demonstrated that significantly elevated copper-gold-molybdenum grades, together with other elevated pathfinder elements, were evident within soils. Molybdenum in soils defined an anomaly immediately above the Productora mineralisation. Where uranium assays were elevated, uranium showed an association with copper, silver, molybdenum, gold, and cobalt. Zones dominated by albite versus K-feldspar-sericite alteration were defined, with copper-gold being associated with the K-feldspar-sericite alteration and magnetite being associated with the albitic alteration zones. These results were consistent with earlier petrographic work completed by Fox (2000).
2012-2023	Cortadera	1,978 soil samples were taken using the standard HCH procedure as described above for Productora.	Geochemical sampling within the Cortadera project was used to inform exploration activities north of Cuerpo 1 including the

Year	Project Area	Activity	Result
			Cortadera North and Cuerpo 4 targets. Molybdenum, copper, bismuth and ratios of Cu:Zn, Pb:Zn and Mo:Mn were used in conjunction with geophysical and mapping data to generate exploration targets.
2018-2023	San Antonio	229 soil samples were taken predominately to the north of San Antonio. 1,128 rock samples were taken along trenches and supplementary to mapping activities.	The known locality of some of the mineralised horizons at San Antonio, based on ongoing mining activities, meant that large grid soil sampling campaigns were unnecessary. The completion of soil and rock sampling was used to help inform the geological model.
2024	Productora	73 soil samples were taken to the west of Productora, to infill the pre-existing soil grid.	The infill soil program was used in combination with detailed mapping and geophysical surveys to inform exploration targeting across the lithocap target area. In particular lead, zinc, bismuth, arsenic and ratios of As:Zn, Sr:Pb, La:Pb were used to interpret areas of higher temperature alteration which may represent zones of fluid up-flow.

9.4 Geophysics

Several geophysical surveys were completed across the Cortadera project area prior to acquisition by HCH, they are summarised in Table 9.3.

Table 9.3: Summary of Geophysical Surveys Completed at Cortadera prior to HCH acquisition

Year	Activity
2009	Ground Magnetics Survey undertaken by Argali Geofisica over the Cortadera project. 108 km of continuous profile magnetic data was acquired on 100 m spaced N-S lines, station spacing was approximately 0.5 m to 1.5 m
2011	Ground Magnetic Survey undertaken by Argali Geofisica over the broader region surrounding Cortadera. 1 126 km of continuous profile magnetic data was acquired on 50 m spaced N-S lines, station spacing was approximately 0.5 m to 1.5m
2011	Two MIMDAS profiles were surveyed at Cortadera by Geophysical Resources and Services (GRS) using pole-dipole IP / Resistivity and EMAP Magnetotellurics. The 500 m spaced profiles are oriented 070° - 250° E and pass through Cuerpos 1 and 2.
2011 - 2012	An Induced Polarisation (IP) survey was completed at Cortadera across two campaigns by Argali Geofisica. 19.2 km (4 lines) were surveyed in 2011, followed by 14.6 km (3 lines) in August 2012. IP data was collected with pole-dipole array with a dipole spacing of 150 m. The time-domain waveform frequency used was 0.125 Hz (2 seconds).
2012	Evaluation and re-processing of Airborne Magnetic Survey completed by Fugro Gravity and Magnetics Services across the Cortadera project. Original data was acquired by World Geoscience Corporation in 1993, on a nominal 400 m line spacing, with lines oriented 165° - 345°.

HCH has completed seven geophysical surveys across the Costa Fuego Project, they are summarised in Table 9.4.

Table 9.4: Summary of Geophysical Surveys Completed at the Costa Fuego Project by HCH

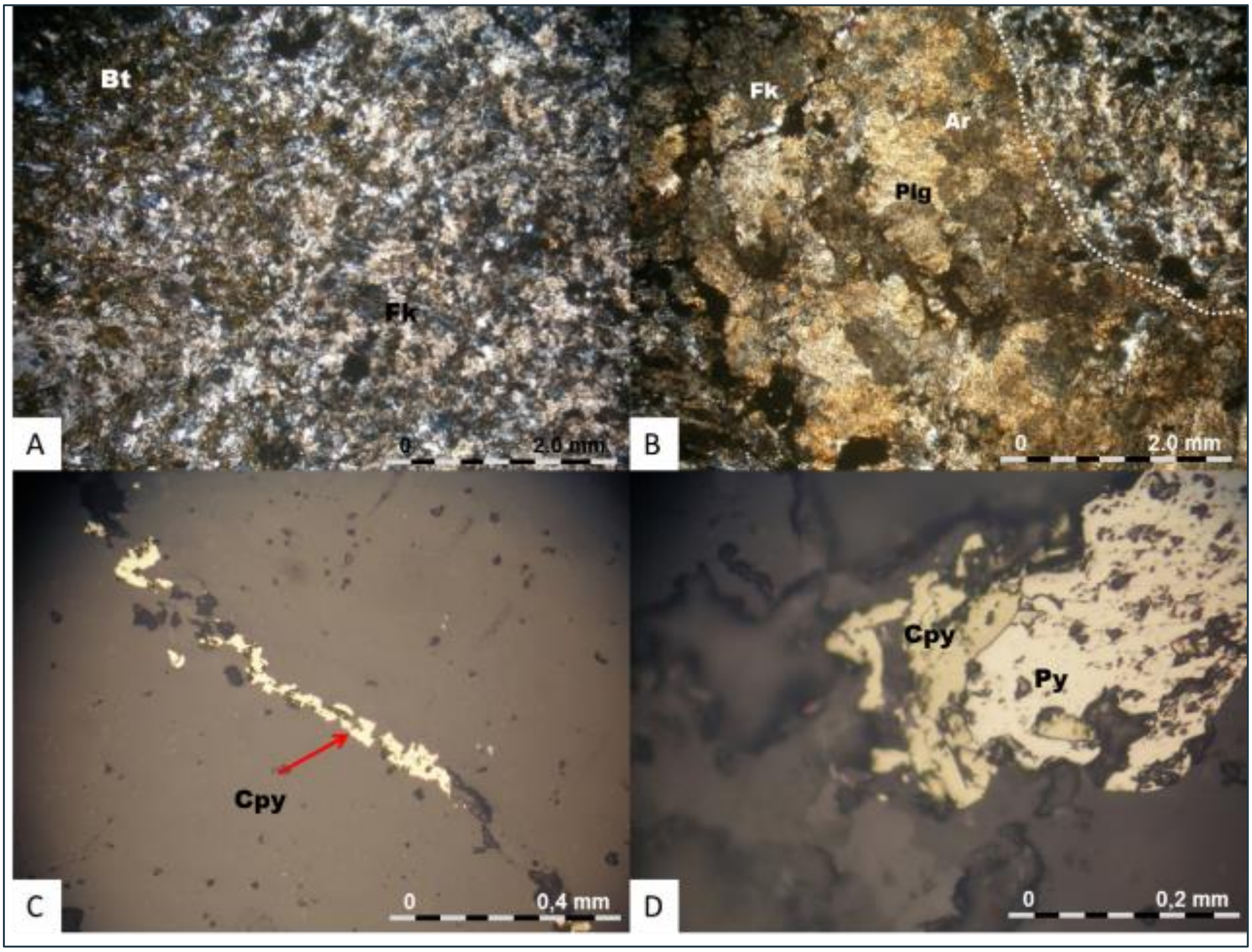
Year	Activity	Result
2010	An airborne geophysical survey was conducted at Productora by Geodatos and flown by helicopter with an average sensor height of about 145 m, on 100 m spaced east-west flight lines, and 1,000 m spaced north-south tie lines. Data collected included standard flight height, magnetic and radiometric data. This geophysical survey data was processed by geophysical consultants Southern Geoscience.	Magnetic and radiometric products, including a 3D magnetic inversion model, were produced to enable structural, lithological and alteration interpretation.
2015	An Induced Polarisation and Magnetotelluric (IP/MT) survey was completed at Productora in late August 2015. SouthernRock Geophysics completed a 26.7 km line, 150 m Pole-Dipole IP/MT survey at the Project. The survey was focused over the western part of the Project known as the Alice porphyry corridor.	This survey provided detailed 2D and pseudo 3D mapping of the resistivity and chargeability of the 6.5 km-long porphyry-style target area at the Alice prospect.
2021	A Ground Magnetic Survey was completed to the south of Cortadera, at the Santiago Z exploration prospect. Continuous profile magnetic data was collected for 470.6 km, along 100 m spaced N-S oriented lines. Station spacing was approximately 0.5 m to 1.5 m.	The central part of the survey identified a large magnetic low that may be indicative of magnetite destructive alteration.
2024	Four MIMDAS profiles were surveyed at Cortadera (17.2 Lkm) by GRS using pole-dipole IP / Resistivity and EMAP Magnetotellurics. Three cross-section profiles across Cuerpo 3 were oriented 070° - 250° E and one long-section profile was oriented NW-SE along the Serrano fault trend.	The survey identified the chargeable pyrite halo of the known porphyry system. Several chargeable and conductive anomalies were identified to the northwest of Cuerpo 1, and south west of Cuerpo 3.
2024	Three MIMDAS profiles were surveyed at Productora (10.2 Lkm) by GRS using pole-dipole IP / Resistivity and EMAP Magnetotellurics. The three 800m spaced profiles were oriented E-W (6822400mN, 6821600mN, 6820800mN) across the mineralised Productora breccia system, the Alice porphyry and the western lithocap target area.	The survey confirmed the presence of a shallow resistive anomaly beneath mapped silica ridges. And identified a deep resistive anomaly beneath Alice and the Productora breccia. A weak chargeability anomaly is associated with the Productora breccia, interpreted to relate to disseminated pyrite.
2024	An extension to the existing Ground Magnetics Survey was completed by Argali Geofisica at El Fuego in March 2024. Spacing of the N-S oriented lines varied between 50 and 100 m, to align with the grid of preceding surveys. Continuous profile magnetic data was collected for 105.3 km. Station spacing was approximately 0.5 m to 1.5 m.	The survey recognised the continuity of the regional Agua de los Burros fault as well as two low magnetic features; the first to north-west of San Antonio, and second to the west of the Valentina target area.
2024	A Multichannel Surface Wave Analysis (MASW) Seismic survey consisting of 14 profiles was completed by Geodatos across planned infrastructure at Productora.	The results of the survey were reviewed in combination with surface geological mapping to aid in the identification of various geoseismic units important for hydrogeological modelling beneath the planned tailing storage facility.

9.5 Petrology, Mineralogy, and Research Studies

HCH has an extensive collection of thin sections taken from diamond core which have accompanying imagery and thin section descriptions. These are available for future studies and data interrogation.

An example is shown in Figure 9.1. This sample is taken from PRP0420D (356.18 m) which was drilled into the Productora deposit.

Figure 9.1 : Example Thin Section Image M-92. Sample Taken from Productora Deposit Drilling, PRP0420D at 356.18 m depth



Note: A and B are in transmitted light, C and D are reflected light. Image A) provides an overview, principally composed of interstitial feldspar (Fk) with development of biotite (Bt). B) An aplitic, albitised plagioclase (Plg) texture being altered to K-feldspar (Fk) and clays (Ar). C) A micro veinlet filled with chalcopyrite (Cpy). D) Pyrite (Py) being replaced by chalcopyrite (Cpy) in texture decay.

10 Drilling

10.1 Introduction

Drilling across the Costa Fuego Project has been completed over the last 14 years, beginning with Productora and Alice in 2010, followed by San Antonio in 2018 and then Cortadera in 2019. A summary of all drilling that informs the 2024 Costa Fuego Project Mineral Resource Estimate is detailed in Table 1.2.

The majority of diamond (DD) and reverse circulation (RC) drilling and assay results used for the Productora and Alice MREs were generated by extensive HCH exploration and resource development drilling programs completed between 2010 and 2015. In 2021 an exploration program of 17 RC drillholes was completed across several regional targets at Productora, followed by a metallurgical campaign of four DD holes across Productora and Alice in 2023. In 2024, 14 drillholes were completed which are not included in MRE informing this PFS; seven DD for metallurgical testing, five RC drillholes for hydrological monitoring, and two RCDD for exploration.

The Cortadera deposit underwent considerable drilling between 2010 and 2013 while the Project was under the control of Minera Fuego Limited (Minera Fuego). A second large campaign of drilling was undertaken in 2019 and 2020 by HCH for verification, infill, and extensional purposes, which informed the maiden MRE in October 2020. From 2020 to late 2021, extensive in-fill and extensional drilling was undertaken at Cortadera for the March 2022 MRE update. Following this, HCH undertook a phase of resource extension and exploration drilling at Cortadera, which included six DD drillholes for utilised for geotechnical and metallurgical studies, as well as DD and RC drilling at depth at Cuerpo 1. This drilling informed the MRE used in this PFS. In 2024, a short exploration campaign was completed at Cortadera (two RC pre-collars and three DD tails) which are not in the current MRE.

DD and RC drilling was completed on the San Antonio deposit in 2018 by HCH, comprising 4,922 m over 41 RC holes to test the extent of mineralisation and underground workings. In 2022 an infill and extensional RC drilling campaign was undertaken at San Antonio, totalling 2,504 m over 16 RC and 3 DD holes.

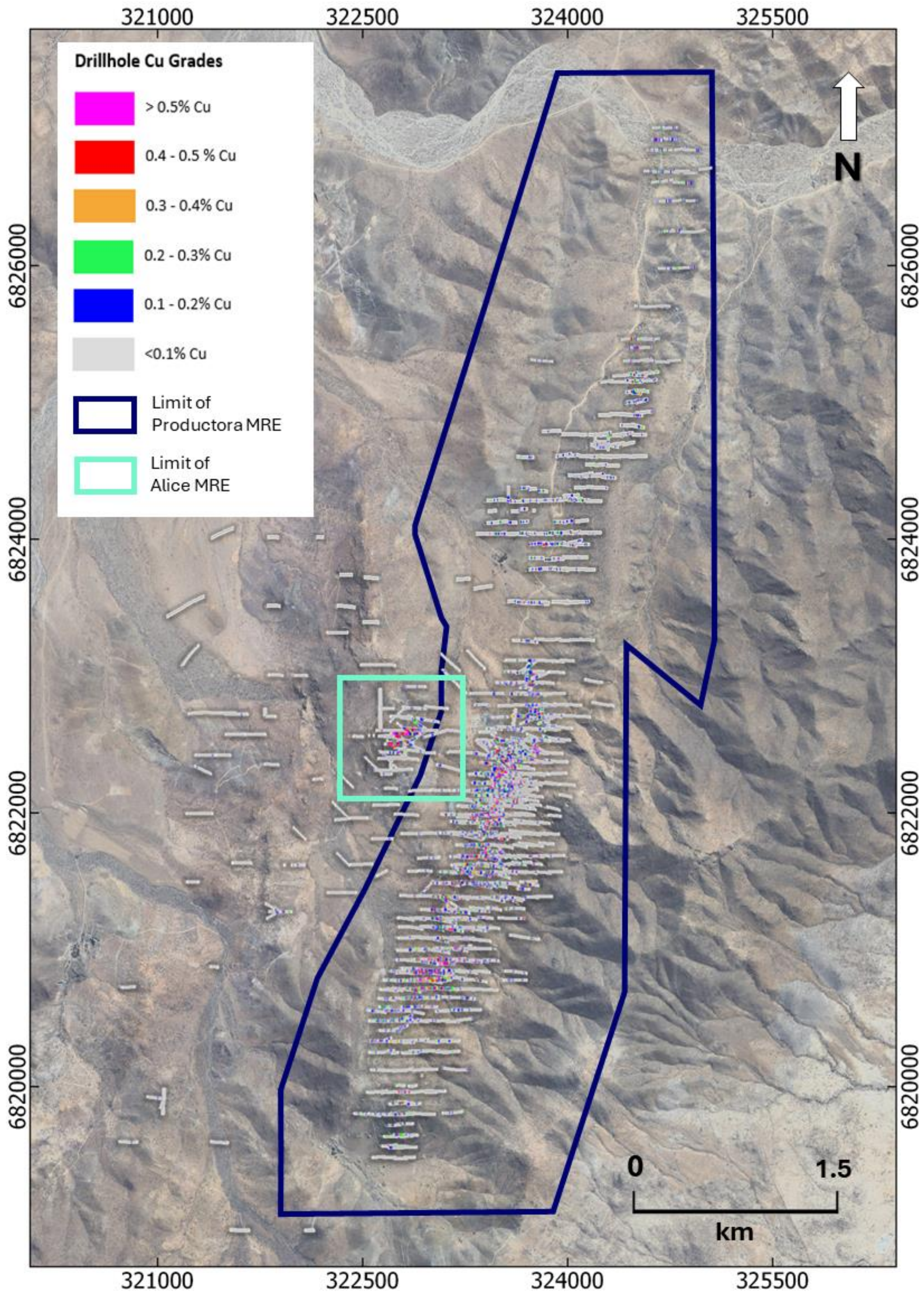
Table 10.1 : Breakdown of the Drilling Completed by HCH Across the Costa Fuego Project Mineral Resources

Project	Year	RC (m)	DD (m)
Productora	2010 - 2023	218,231	29,241
Alice	2010 - 2022	17,156	1,802
Cortadera	2019 - 2024	41,680	44,881
San Antonio	2018 - 2022	6,931	495

10.2 Productora

A plan view showing all drilling completed at Productora is shown in Figure 10.1.

Figure 10.1 : Plan View of Productora Showing All Completed Drilling Relative to the limits of the Productora and Alice MRE. Drill Holes Displayed by Copper Grade



10.2.1 Drilling Methods

The majority of drilling commissioned by HCH for the Productora and Alice Resource estimates was conducted by Blue Spec Sondajes (Bluespec). The rigs used were convertible between DD and RC drilling.

RC drilling used a face sampling down hole hammer with cyclone sample recovery. Drill hammers used were 542/543, PR 54 (Sandvik) and MR 120 (Mincon).

Diamond drilling was conducted using HQ diameter core (96 mm outside diameter and 63.5 mm inside diameter). Orientation of diamond core was monitored by the Reflex ACT III core orientation tool.

10.2.2 Hole Planning and Set Up

The Productora drilling consists of a nominal drill collar pattern of 80 m by 40 m and drilled to approximately 300 m vertical from surface. Drill holes are primarily oriented -60 to -80° degrees towards 090° azimuth along east-west sections. This allowed for optimal drill orientation for intersection of the north-northeast trending mineralisation. Drill holes oriented at an azimuth of 270° or plunging in or out of sections were completed to ensure geological representativeness, to test specific structural orientations, or due to limitations on drill position availability.

The Alice drilling consists of a nominal 80 m by 50 m grid spacing at orientations consistent with Productora drilling.

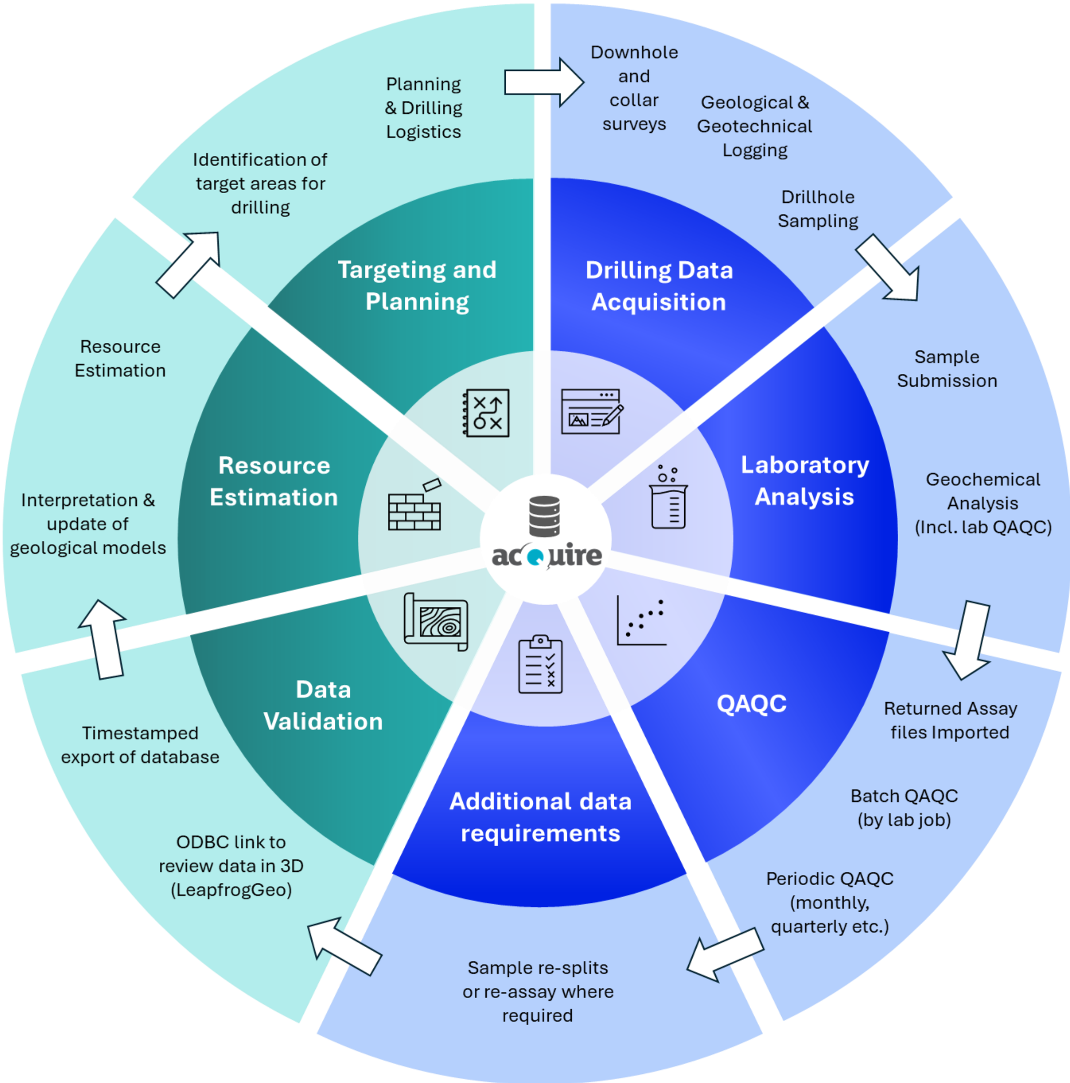
10.2.3 Geological Database

An internally managed and audited acQuire geological database was used to capture and manage geological drill data. Strict data flow procedures were followed during drilling. This ensured that all drill data was collected in a systematic, repeatable, and accurate manner allowing consistent and validated data for resource estimation. The various stages in drilling data capture completed by HCH are summarised in Figure 10.2.

An internally managed review of data management processes was undertaken in 2020, the results were deemed as satisfactory with no errors identified. An external database audit was completed by Expedio Services in November 2021. There were no major issues identified with the quality of the data and no material bias apparent from the check work.

Both internal and external audits have been reviewed by Elizabeth Haren and the data is deemed suitable for Mineral Resource estimation and other project development work.

Figure 10.2 : Workflow for Drilling Data from initial drill planning to Resource Estimation. Blue indicates stages of data capture and input into the AcQuire database. Green indicates stages reliant of data exported from the AcQuire database.



10.2.4 Collar Survey

Drill collars were routinely surveyed following completion of drilling campaigns by contract surveying company Topgeo Chile Spa. Validation of collar survey locations was completed by:

- Comparing planned coordinates with subsequent picked up coordinates to detect any material differences in location
- Visual checking of collar points against the topographical surface model
- Random GPS field checks of collar coordinates by both HCH employees, independent auditors, and the Mineral Resource QP during documented site visits

10.2.5 Down Hole Survey

The down hole surveying at Productora was completed by contracting companies Wellfield and North Tracer. Wellfield undertook down hole surveys from the beginning of HCH drilling until May 2013 using a mechanical gyroscope instrument (non-north-seeking), which provides accurate directional data (azimuth and dip) relative to an initial starting orientation provided by the supervising geologist. North Tracer undertook down hole surveys from May 2013 onwards using a north-seeking, high speed continuous gyroscopic camera.

Down-hole survey validation checks were completed by subsequent umpire survey checks as well as within-company resurveys. Results from resurveys and cross-company checks provided a good correlation between the two companies and provides confidence in the survey data for the drill holes used for resource estimation. Routine 3D review of the drilling database was also performed as part of drill hole survey validation, to detect any spurious survey measurements. Any errors are rectified directly into the acQuire database.

10.2.6 Geological Logging

Geological logging was recorded in a systematic and consistent manner such that the data was able to be interrogated accurately using geological modelling software programs. Collar, survey, sample register and sample condition, sample recovery data or comments (and commonly magnetic susceptibility) were recorded by a competent field technician.

All RC holes were logged by qualified and experienced geologists using a set of standardised logging codes. Geological attributes logged in RC chips include colour, weathering/oxidation, regolith, lithology, veining, alteration assemblages and their intensities, texture, sulphide mineralogy and their percentages. Geological attributes logged in diamond core include the same attributes as for RC drilling, and the following additional structural attributes:

- Linear structures (e.g. lineations)
- Planar structures (e.g. shears, veins, dykes, faults, cleavage, joints/fractures, schistosity)
- Preliminary geotechnical features such as Rock Quality Designation (RQD) and Fracture Frequency (FF).

Images of all diamond drill core was captured with digital photography, which is retained on HCH servers.

10.2.7 Bulk Density

A total of 2,164 bulk densities were available for the Productora Resource estimate, and 137 for the Alice deposit.

One in every five samples of diamond drill core was submitted for bulk density analysis as performed by ALS (ALS Code OA-GRA09). The testwork was undertaken on an approximate 10 cm piece of core taken from the designated sample bag and used to determine the bulk density for the 1 m interval. The bulk density test used the Archimedean water submersion method of analysis; a description of the ALS technique is outlined below.

Bulk density analysis on drill core samples is carried out by weighing the object, then slowly placing it into a bulk density apparatus filled with water. The displaced water is collected into a graduated cylinder and measured. The bulk density calculation considers the weight of the sample and the volume of water displaced;

the result is expressed as g/cm^3. The 10 cm length of core is then returned to the sample bag ensuring the 1 m interval is complete and intact prior to analysis for ME-ICP61.

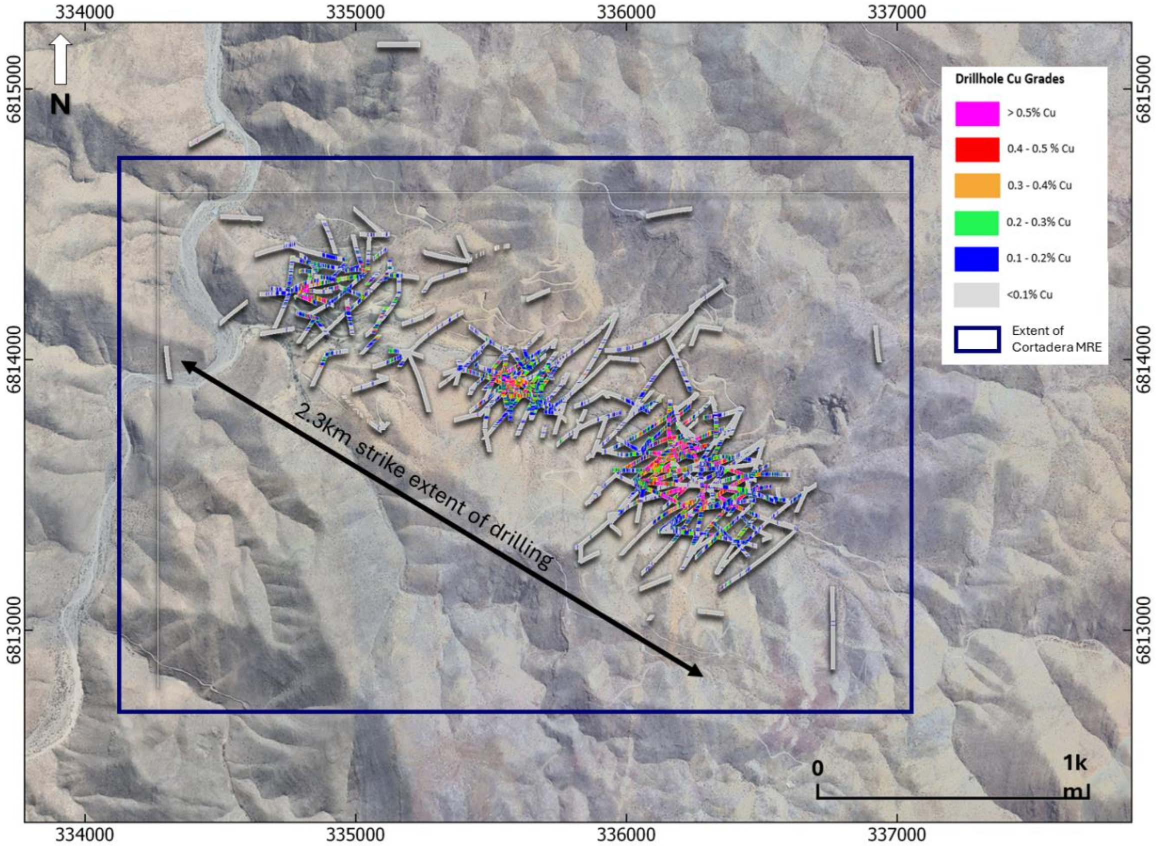
In addition to the bulk density analysis completed on drill core samples, pycnometer density tests by ALS (ALS Code OA-GRA08b) were also carried out on historic pulps for comparative purposes. Specific gravity (SG) analysis on pulps was carried out by placing 3 g of sample pulp into a pycnometer, which is then filled with a solvent to produce a slurry. The SG calculation was made comparing the weight of the sample and the weight of the displaced solvent. In practice, the pycnometer method returns elevated density results compared to drill core bulk density measurements, as it measures mineral density without accounting for natural rock porosity. The pulverisation process removes that porosity. The expected variance in the pycnometer results is proportional to the porosity.

Due to the likely issues with porosity across both Productora and Alice, only bulk density results from diamond drillholes were used to inform reported Mineral Resource Estimate density.

10.3 Cortadera

A plan view showing all drilling completed at Cortadera relative to the MRE extents is shown in Figure 10.3.

Figure 10.3 : Plan View of Cortadera Showing all Drilling Completed up to 2024 relative to the limits of the Cortadera MRE. Drill Holes Displayed by Copper Grade (HCH, 2024)



10.3.1 Drilling Methods

All drilling commissioned by HCH for the Cortadera Resource estimates was conducted by Blue Spec Sondajes (Bluespec). Drill rig configuration and methodology is as stated in Section 10.2.1.

Additionally, if diamond drilling was required from surface, a diameter of PQ3 (122.3 mm outside diameter and 83 mm inside diameter) was used before casing to HQ3 diameter core (96 mm outside diameter and 63.5 mm inside diameter) and NQ3 diameter core (75.7 mm outside diameter and 45 mm inside diameter).

Orientation of diamond core was monitored by the Reflex ACT III core orientation tool.

10.3.2 Hole Planning and Set Up

The spacing and location of much of the drilling at Cortadera is variable and averages approximately 80 m along strike and 150 m across strike. Drill holes dip 60 to 80° toward northeast or southwest. Additional orientations were used to ensure geological representivity and to optimise the use of available drill platforms.

10.3.3 Geologicial Database

A single acQuire database captures all HCH data, so information in Section 10.2.3 is also relevant for Cortadera.

Both internal and external audits have been reviewed by Elizabeth Haren and the data is deemed suitable for Mineral Resource estimation and other project development work.

10.3.4 Collar Survey

Drill collars were routinely surveyed following completion of drilling campaigns by contract surveying company Topgeo Chile Spa. Topographical equipment used was a CHCNAV model i80 Geodetic GPS, dual frequency, Real Time with 0.1 cm accuracy (mN, mE and mRL). Topographical collar surveys were supplied in both WGS84 Zone 19S and PSAD Zone 19S coordinate systems. Validation of collar survey locations was completed by:

- Comparing planned coordinates with subsequent picked up coordinates to detect any material differences in location
- Visual checking of collar points against the topographical elevation model
- Random GPS field checks of collar coordinates by both HCH employees, independent auditors, and the Mineral Resource QP during documented site visits
- Surveying of FJOD collar points by Geotopo for comparison against original files supplied by Minera Fuego.

10.3.5 Down Hole Survey

The down hole surveying at Cortadera for HCH drilling was completed by Bluespec drilling contractor using the Axis Champ Navigator north-seeking gyroscope tool.

Minera Fuego down hole surveys were also completed using a north-seeking gyroscope tool, although the details of the instrument type are not known.

Routine 3D review of the drilling database was performed as part of drill hole survey validation, to detect any spurious survey measurements. Any errors are rectified directly into the acQuire database.

Drill hole deviations and corresponding down hole sample locations are adequately measured; deviations are not considered to have any adverse impact on this Resource Estimate.

10.3.6 Geological Logging

Geological logging was recorded in a systematic and consistent manner such that the data was able to be interrogated accurately using modern mapping and geological modelling software programs. Geological attributes logged in RC chips includes colour, weathering/oxidation, regolith, lithology, veining, alteration assemblages and their intensities, texture, mineralogy and sulphides and their percentages.

Core reconstruction and orientation was attempted in every drilling run and completed where possible before marking up and logging of core. Geological attributes logged in diamond core includes the same attributes as for RC drilling, and additionally the following structural attributes:

- Linear structures e.g. lineations

- Planar structures e.g. shears, veins, dykes, faults, cleavage, joints/ fractures, schistosity
- Preliminary geotechnical features such as RQD and FPM
- Images of all DD core was captured with digital photography, and all core photos are retained on HCH's data servers.

Down hole structural measurements are considered during the interpretation of stratigraphy, understanding vein paragenesis and are incorporated into the geological modelling process.

All relevant procedures were taken from HCH's Productora project and adjusted where necessary.

10.3.7 Geotechnical Logging

Both RQD and recovery measurements were taken using the orientation and/or cutting line from the mark up of the core. The following procedure was used at Cortadera:

- Measurements were recorded mainly on a one-metre basis
- Recovery was a measurement of the total amount of core between each metre. This will be 100% unless the core-block indicates otherwise. For example, if a core block shows 0.13 m core-loss (CL) then the percent recovered is 87%
- RQD %: was calculated as all the pieces of core that were individually longer than 10 cm added up to give a percentage. For example, if there are three pieces of core in a one-metre interval that are over 10 cm - 19 cm, 17 cm and 16 cm - then the total RQD is 52%.

A review of the HCH geotechnical logging procedures was completed in 2021 by Ingeroc. The review found the data to be adequate for identifying different geotechnical units.

No RQD and fracture frequency data was captured by Minera Fuego on the historic dataset of 39 holes. In 2021 all available Minera Fuego core photos of drilling were supplied to Datarock Pty Ltd to extract the following information using their machine learning application:

- Cropped and depth registered imagery
- Fracture analysis (classification, frequency, spacing, joint set number, joint set roughness coefficient)
- RQD

10.3.8 Bulk Density

Cortadera has density data derived from diamond core. Minera Fuego and HCH conducted specific gravity testwork, with the techniques outlined in the section below.

A total of 1,304 bulk densities were available for the Cortadera Resource estimate.

One in every 15 diamond core samples was submitted for bulk density analysis as performed by ALS (ALS Code OA-GRA09). Those samples are then labelled 'Density test'.

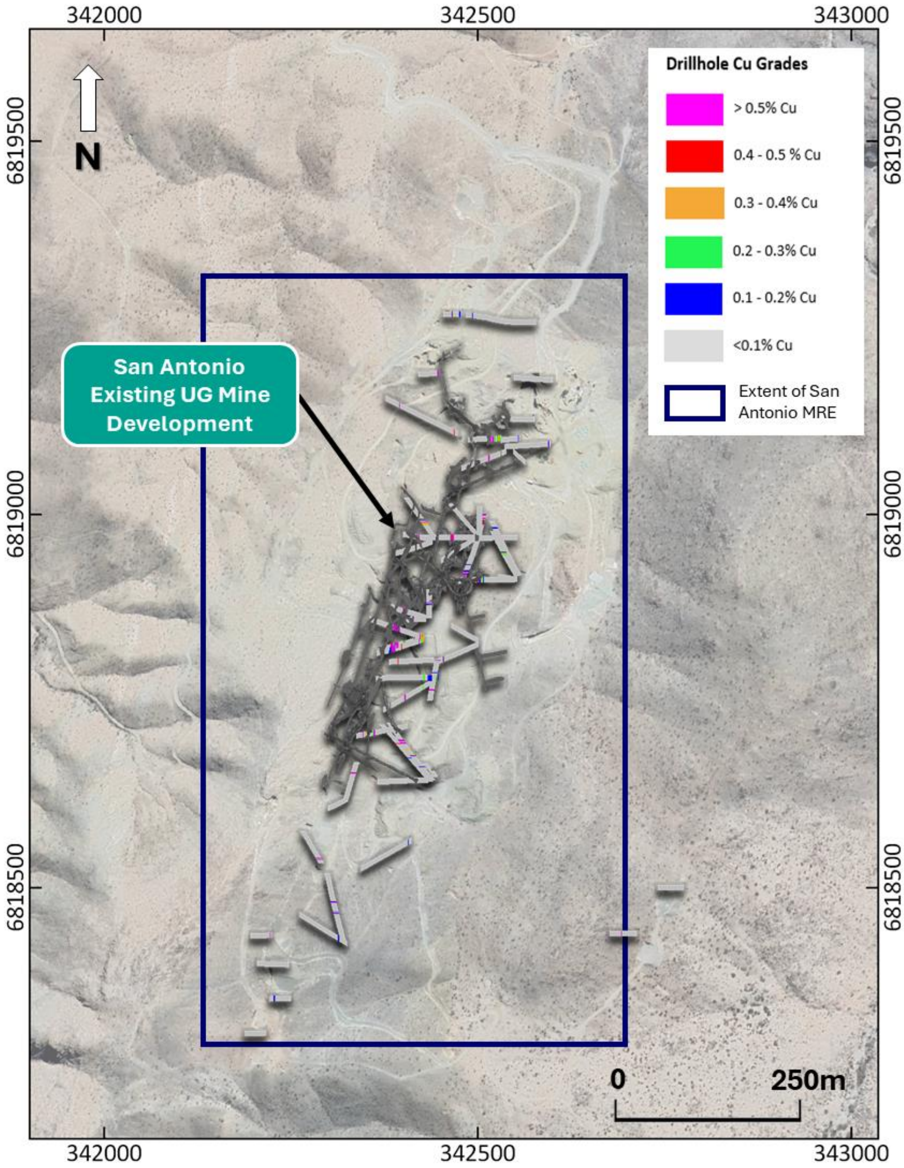
The testwork was undertaken on an approximate 10 cm piece of core taken from the designated sample bag and is used to determine the bulk density for the 1 m interval. The bulk density test consists of using the Archimedean water submersion method of analysis; a description of the ALS OA-GRA09 method is described in Section 10.2.7.

The 10 cm length of core is then returned to the sample bag ensuring the 1 m interval is complete and intact prior to analysis for ME-ICP61.

10.4 San Antonio

A plan view showing all drilling completed at San Antonio relative to the MRE extents is shown in Figure 10.4.

Figure 10.4 : San Antonio Drill Program Hole Locations Relative to the limits of the San Antonio MRE and Existing Underground Mine Development. Drill Holes Displayed by Copper Grade (HCH, 2024)



10.4.1 Drilling Methods

All drilling commissioned by HCH for the San Antonio Resource estimates was conducted by Blue Spec Sondajes (Bluespec). RC and DD drill rig configuration and methodology is as stated in Section 10.3.1.

10.4.2 Hole Planning and Set Up

The HCH RC drill program resulted in approximately 40 m x 40 m spacing along strike and between 40 m x 40 m and 80 m x 80 m spacing up and down dip of the mineralisation. Historic drilling includes underground channel and sludge drilling, providing localised drill spacing down to 20 m. Drill spacing has the highest density around the old underground workings. Broader spacing of approximately 300 m by 300 m covers the modelled extensions of mineralisation.

HCH DD drilling at San Antonio consisted of three drill holes designed to twin pre-existing RC holes for the purpose of metallurgical testing. DD holes are spaced approximately 150 - 200m along the N-S strike extent of the San Antonio mineralisation.

Topography plays a major role in hole planning at San Antonio, with steep slopes limiting access to much of the near-surface resource. Where drilling has not been possible, detailed mapping and outcrop sampling was completed.

10.4.3 Geological Database

A single acQuire database captures all HCH data, so information in Section 10.2.3 is also relevant for San Antonio.

Both internal and external audits have been reviewed by Elizabeth Haren and the data is deemed suitable for Mineral Resource estimation and other project development work.

10.4.4 Collar Survey

Collar surveys for drilling after 2019 were undertaken by contract survey company TopGeo Ingenieria Limitada. Collar surveys were reported in a table to HCH and imported to the acQuire database.

Historic drill collars and surface sample locations were provided to HCH as part of a data compilation and appear to have been provided in the PSAD56 UTM coordinate system. These were transformed by HCH to WGS84 UTM zone 19S via the following method (PSAD easting minus 184.13 m, PSAD northing minus 375.38 m). This shift is considered appropriate for the project location.

Validation of collar survey locations was completed by:

- Visual checking of collar points against the topographical elevation model
- Validated of some holes as part of a drone survey project completed in 2021

10.4.5 Down Hole Survey

Down hole surveying at San Antonio was completed by Bluespec drilling contractor using the Axis Champ Navigator north-seeking gyroscope tool post hole completion.

Routine 3D review of the drilling database was performed as part of drill hole survey validation, to detect any spurious survey measurements. Any errors are rectified directly into the AcQuire database.
Drill hole deviations and corresponding down hole sample locations are adequately measured; deviations are not considered to have any adverse impact on this Resource Estimate.

10.4.6 Geological Logging

Geological logging was recorded in a systematic and consistent manner such that the data was able to be interrogated accurately using modern mapping and geological modelling software programs. Geological attributes logged in RC chips includes colour, weathering/oxidation, regolith, lithology, veining, alteration assemblages and their intensities, texture, mineralogy and sulphides and their percentages.

Core reconstruction and orientation was attempted in every drilling run and completed where possible before marking up and logging of core. Geological attributes logged in diamond core includes the same attributes as for RC drilling, and additionally the following structural attributes:

- Linear structures e.g. lineations
- Planar structures e.g. shears, veins, dykes, faults, cleavage, joints/ fractures, schistosity
- Preliminary geotechnical features such as RQD and FPM
- Images of all DD core was captured with digital photography, and all core photos are retained on HCH's data servers.

Down hole structural measurements are considered during the interpretation of stratigraphy, understanding vein paragenesis and are incorporated into the geological modelling process.

All relevant procedures were taken from HCH's Productora project and adjusted where necessary.

10.4.7 Geotechnical Logging

Both RQD and recovery measurements were collected on the three DD holes, using the process described in Section 10.3.7.

10.4.8 Bulk Density

One sample in every 5 diamond core samples was submitted for bulk density analysis as performed by ALS (ALS Code OA-GRA09). Those samples are then labelled 'Density test'. A total of 74 bulk density results were returned on the HCH core.

The testwork was undertaken on an approximately 10 cm length piece of core taken from the designated sample bag and is used to determine the bulk density for the 1 m interval. The bulk density test consists of using the Archimedean water submersion method of analysis; a description of the ALS OA-GRA09 method is described in Section 10.2.7.

10.5 Comments on Section 10

During Elizabeth Haren's technical site visit in 2022, the drilling at the Costa Fuego Project was conducted in a professional manner using appropriate practices and produced sufficient quality and recovery to be used in a Mineral Resource Estimation. Elizabeth Haren is unaware of any material factors that would impact the accuracy and reliability of the sample results. The Company's Principal Resource Geologist and Resource Development Manager both visited site in 2023 as well, to review drilling and sampling operations and provide additional feedback to the QP.

11 Sample Preparation, Analyses and Security Sampling Methods

11.1 Sampling Methods

11.1.1 RC Sampling Methods

RC drill hole sampling at the Costa Fuego Project was executed at one metre intervals. Within logged mineralisation zones, the 1 m sample ("A" sample) was submitted. Outside the main mineralised zones (as determined by the logging geologist), 4 m composites were created from scoops of 1 m sample residues over this interval. The composited 4 m samples were analysed first and if required, the individual and original 1 m "A" samples comprising the 4 m interval were sent for analysis. This ensured that no mineralisation was missed while minimising analytical costs.

A fixed cone splitter was used to create two nominal 12.5% samples (Sample "A" and "B"), along with the large bulk reject sample. The "A" sample is always taken from the same sampling chute, and comprises the primary sample submitted to the laboratory. The "B" samples were retained for use as the field duplicate sample, residues were collected into large plastic bags and were retained on the ground near the drill hole collar, generally in rows of 50 bags.

11.1.2 DD Sampling Methods

Drill core sampling methods varied between projects based on mineralisation style, as seen in Table 11.1.

Whole core sampling was chosen as the preferred sampling method at Productora due to the porous and brecciated nature of mineralisation. This allowed for a larger and more representative sample. Some core has been half cored or completely retained for geological reference. Where mineralisation is more homogenous (Cortadera and Alice), half core samples were considered representative. The remaining half core was retained for review of lithology and mineralisation, and for further test work as required.

Geotechnical samples were taken for tests including triaxial (one sample per 250 m) and uniaxial tests (one sample per 50 m.

Table 11.1 : Summary of DD Sampling Length and Type

Project	Sample Lengths (m)	Sample Type
Productora	1	Whole Core
Alice	1	Half Core
Cortadera	2	Half Core

11.2 Metallurgical Sampling

Metallurgical sampling is discussed in Section 13.

11.3 Analytical and Test Laboratories

11.3.1 General

For all HCH samples, the primary sample preparation facility was ALS La Serena (Coquimbo), Chile. The primary analytical laboratories were ALS Santiago, Chile and Lima, Peru. These facilities have ISO 9001:2008 accreditation and La Serena also has ISO 17025 accreditation. Due to transport restrictions during the Covid-19 pandemic, Cortadera samples were sent to ALS Vancouver, Canada from March to April 2020. A small number of samples were also analysed in ALS Lulea, Sweden.

Periodic inspections of the laboratory facilities and meetings with ALS staff were conducted over the various drilling campaigns.

The check laboratory was Andes Analytical Assay Ltda. in Santiago, which also holds ISO 9001:2008 accreditations.

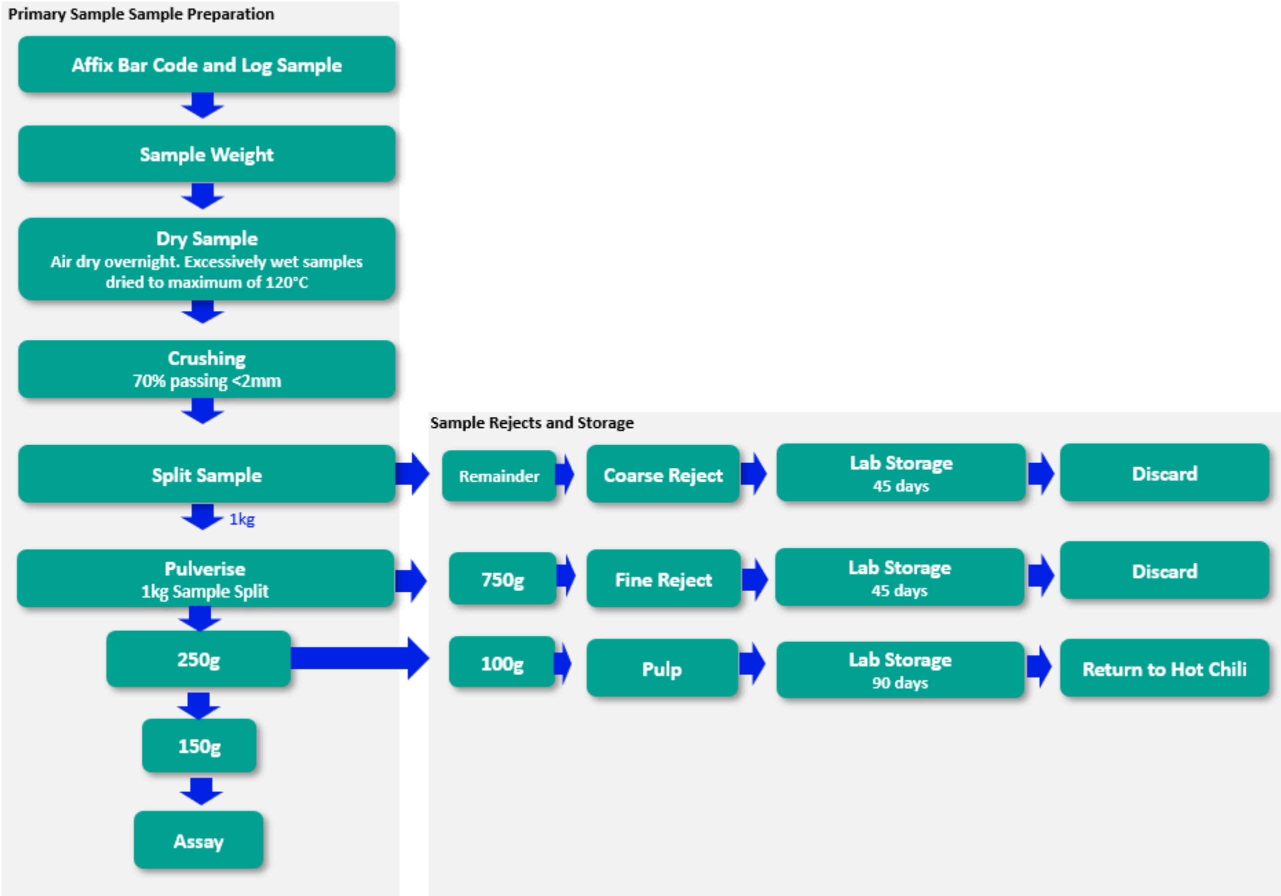
The Qualified Person, Elizabeth Haren, visited the primary sample preparation facility of ALS La Serena (Coquimbo), Chile and ALS Lima, Peru during 2022.

11.4 Sample Preparation

A summary of the sample preparation (ALS Code Prep 31B) procedures is provided in Figure 11.1 and described below:

- Crushing of RC, whole- or half-core samples such that a minimum of 70% is less than 2 mm.
- To obtain whole- or half-core sample duplicates, selected crushed core samples were split into two halves, with one half flagged as the original sample and the other half flagged as the duplicate sample. Empty numbered bags were inserted at the standard duplicate interval by HCH's geologists to indicate the requirement for a duplicate. (Note: RC duplicates were collected at a rate of approximately 1 in 50 m).
- Splitting out via a riffle splitter/rotary splitter of approximately 1 kg of the crushed product for pulverising such that a minimum of 85% passes 75 µm and the remainder was retained as coarse reject material.
- The analytical pulp was extracted from the finely pulverised and homogenised product.

Figure 11.1 : Flow Chart of Sample Preparation



11.5 Sample Analysis

A variety of methods have been used across each deposit, see Table 11.2 for details on these analytical methods. Across all deposits, initial analysis of RC and DD samples was by ALS Method ME-ICP61, with additional analytical stages triggered when results met certain criteria.

At Productora and San Antonio these criteria are:

- Au-ICP21 analysis for all samples with > 1,000 ppm Cu
- Cu-AA62 overlimit analysis for all samples with > 10,000 ppm Cu
- ME-MS61 analysis where trace level detection limits were required for exploration purposes.
- Cu-AA05 Soluble copper analysis for samples within the oxide and transitional zones (when determined by the logging geologist)

At Cortadera these criteria are:

- Au-AA23 analysis for all samples with > 1,000 ppm Cu
- Cu-AA62 overlimit analysis for all samples with > 10,000 ppm Cu

- Cu-AA05 Soluble copper analysis for samples within the oxide and transitional zones (when determined by the logging geologist)
- ME-MS61 analysis every 50th metre downhole or where trace level detection limits were required for exploration purposes.

Table 11.2 : Analytical Methods Used by HCH for Drill Hole Samples

Method Code	Detection limits of Cu and Au	Description
ME-ICP61	1 ppm - 1% Cu	Four-acid digestion (hydrochloric, nitric, perchloric, hydrofluoric) followed by ICP-OES determination of 34 elements (at intermediate level detection limits)
ME-MS61	0.02 ppm - 1% Cu	Four-acid digestion followed by ICP-MS determination of 48 elements (at trace level detection limits)
Cu-AA62	0.001 ppm - 50% Cu	Four-acid digestion, followed by AAS measurement
Cu-AA05	0.001 – 10 % Cu	Soluble copper analysis by sulphuric acid leach followed by AAS finish
Au-ICP21	0.001 -10 ppm Au	30 g lead-collection fire assay, followed by ICP-OES
Au-AA23	0.005- 10 ppm Au	30 g lead-collection fire assay, followed by AAS

11.5.1 Minera Fuego

Analytical work by Minera Fuego was conducted over three laboratories: Activation Labs (ACTLAB), ALS and Andes Analytical Assay (AAA) between 2011 and 2013. Several methods were used for analysis at each laboratory, as described in Table 11.3. Laboratory preparation methods employed by the three laboratories used by Minera Fuego was not documented.

Minera Fuego supplied a full assay spreadsheet outlining the sample details including QA/QC standards, blanks and duplicates with analytical result and laboratory batch details. Of the 10,949 primary assays received, 7 727 (70.6%) have original laboratory datafiles to verify the source spreadsheet.

No assessment of laboratory standards and practices was undertaken by HCH for the Minera Fuego drill results, but twinned drillholes were completed for verification purposes.

Table 11.3 : Analytical Methods on Minera Fuego Drilling at Cortadera			
Laboratory	**Drill holes analysed**	**Method Code**	**Method Description**
ACTLAB	FJOD-01 to FJOD-09	ARMS	3-acid digestion followed by a MS finish. Lower detection limit of 0.01 ppm Cu
		3ACID-AAS	3-acid digestion followed by AAS. Lower detection limit of 0.001% Cu
		FA-AAS	30-gram lead-collection fire assay, followed by AAS. Lower detection limit of 0.01 ppm Au
ALS	FJOD-10 to FJOD-24	AA61	4-acid digestion (hydrochloric-nitric-perchloric-hydrofluoric) with AAS finish to a detection limit of 1 ppm Cu
		AA62	4-acid digestion (hydrochloric-nitric-perchloric hydrofluoric), followed by AAS measurement. Lower detection limit of 0.001% Cu
		AA23	Fire assay followed by AAS. Lower detection limit of 0.005 ppm Au
AAA	FJOD-25 to FJOD-39	4AHF	4-acid near total digestion, followed by AAS. Lower detection limit of 0.01% Cu
		ICP-AES	4-acid digestion, followed by ICP-AES analysis. Lower detection limit of 1 ppm Cu.

Where analysis returned results below the detection limit, these have been set in the database to show the value as half the detection limit of that method. Results that are returned as over range are set as the upper detection limit. These database rules apply to both HCH and Minera Fuego samples.

11.6 Sample Security

HCH maintained strict chain of custody security procedures for all samples sent to and from the analytical laboratories.

Once assigned a sample number, individual samples to be sent to ALS laboratories were sealed using a staple gun and accompanied by three identical sample tickets (one stapled to plastic bag to identify any tampering/breakage of seal prior to opening at the laboratory in preparation, and another placed in the bag and a third retained by HCH). Any broken staple seals on samples were to be notified by ALS to HCH. No sealed bags were reported as being opened or broken by ALS.

For both RC and diamond samples, sample bags were placed inside larger plastic bags and delivered by a dedicated truck to the ALS analytical laboratory in La Serena (Coquimbo), Chile for sample preparation and routine analysis.

Following analysis at ALS, the RC and DD coarse rejects were returned to site and stored in sequence in plastic bags under shade cloth next to the Productora core facility. The laboratory pulps were returned and stored at the Productora core facility in organised, dry, and locked storage containers.

11.7 QAQC of Analytical Laboratories

11.7.1 Summary

A routine quality assurance and quality control ("QAQC") programme has been implemented by HCH to monitor the on-going quality of the analytical database. This programme involves Coarse and Certified Blanks, Certified Reference Material ("CRMs" or standards), Duplicates and inter-laboratory comparisons. Table 12.1 provides a summary of all quality control samples and their insertion rates.

HCH's QAQC programme reviews copper, molybdenum, silver, and gold for each returned laboratory batch. For the sake of brevity, a summary of copper performance is presented in this section. Each of the elements monitored has performed satisfactorily.

11.7.1.1 Certified Reference Materials Summary

Certified Reference Materials (CRMs) comprise 2% of the total Costa Fuego Project analytical database. A varied selection of CRM's is available with the CRM selected to correspond with the expected geochemical profile of the sample, matrix matched by rock and mineralisation type. CRM availability from suppliers is finite so multiple CRMs have been used over the last decade of drilling at the Costa Fuego Project. Current drilling programs use OREAS series CRMs; OREAS-501b, OREAS-502, and OREAS-503b combined represent approximately half of the total CRM population (Figure 11.2 and Figure 11.3).

Table 11.4 : Summary of QAQC Samples across the Cortadera, Productora, and San Antonio Projects.

QAQC Element	Total	(%)	Comment
Normal Samples	**251,194**	85%	
Certified Blanks	**1,473**	0%	
IN-BMF-172	200		
IN-BMF-260	1		
IN-BMF-333	106		
IN-M569-260	240		
IN-M615-285	13		
OREAS 22c	910		
OREAS-501b	1		
OREAS-504	1		
Coarse Blanks	**4,006**	1%	
BLANCO	328		
BLANK	622		
QtzBLK	3056		HCH generated quartz blank
Certified Reference Material	**4,922**	2%	
69	4		FJOD Series (Minera Fuego)
CDN-CGS-22	60		
CDN-CGS-23	61		
CDN-CGS-29	5		

Table 11.4 : Summary of QAQC Samples across the Cortadera, Productora, and San Antonio Projects.			
QAQC Element	**Total**	**(%)**	**Comment**
CDN-CM-12	94		
CDN-CM-16	42		
CDN-CM-21	19		
Cu 129	1		
Cu 153	6		
Cu 154	3		
Cu 168	6		
Cu 169	1		
Cu 172	4		
Cu 179	2		
ESTANDAR	15		
GBM399-6	397		
GBM908-7	389		
IN-BMF-172	17		
OREAS 22c	7		
OREAS 902	231		
OREAS-153a	10		
OREAS-290	136		
OREAS-501	193		
OREAS-501b	915		
OREAS-501d	31		
OREAS-502	808		
OREAS-502c	99		
OREAS-503	172		
OREAS-503b	751		
OREAS-504	231		
OREAS-505	211		
OX 69	1		
Field Duplicates	**3,695**	1%	Taken from the cone during RC drilling
Coarse Duplicates	**417**	0%	Taken by laboratory during sample preparation at HCH instruction
Duplicates	**2,803**	1%	Assorted collection methods including Quarter Core, Chip Scoop, Riffle split, Cone and Crush methods. Historic use code.
Pulp Duplicates	**0**	0%	
Lab Duplicates	**24,299**	8%	
Umpire Samples	**3,089**	1%	
Total QC Samples	**44,704**	15%	

Table 11.4 : Summary of QAQC Samples across the Cortadera, Productora, and San Antonio Projects.			
QAQC Element	**Total**	**(%)**	**Comment**
Total Samples	**295,898**	100%	

Figure 11.2 : Percentage distribution of CRMs for the Costa Fuego Project

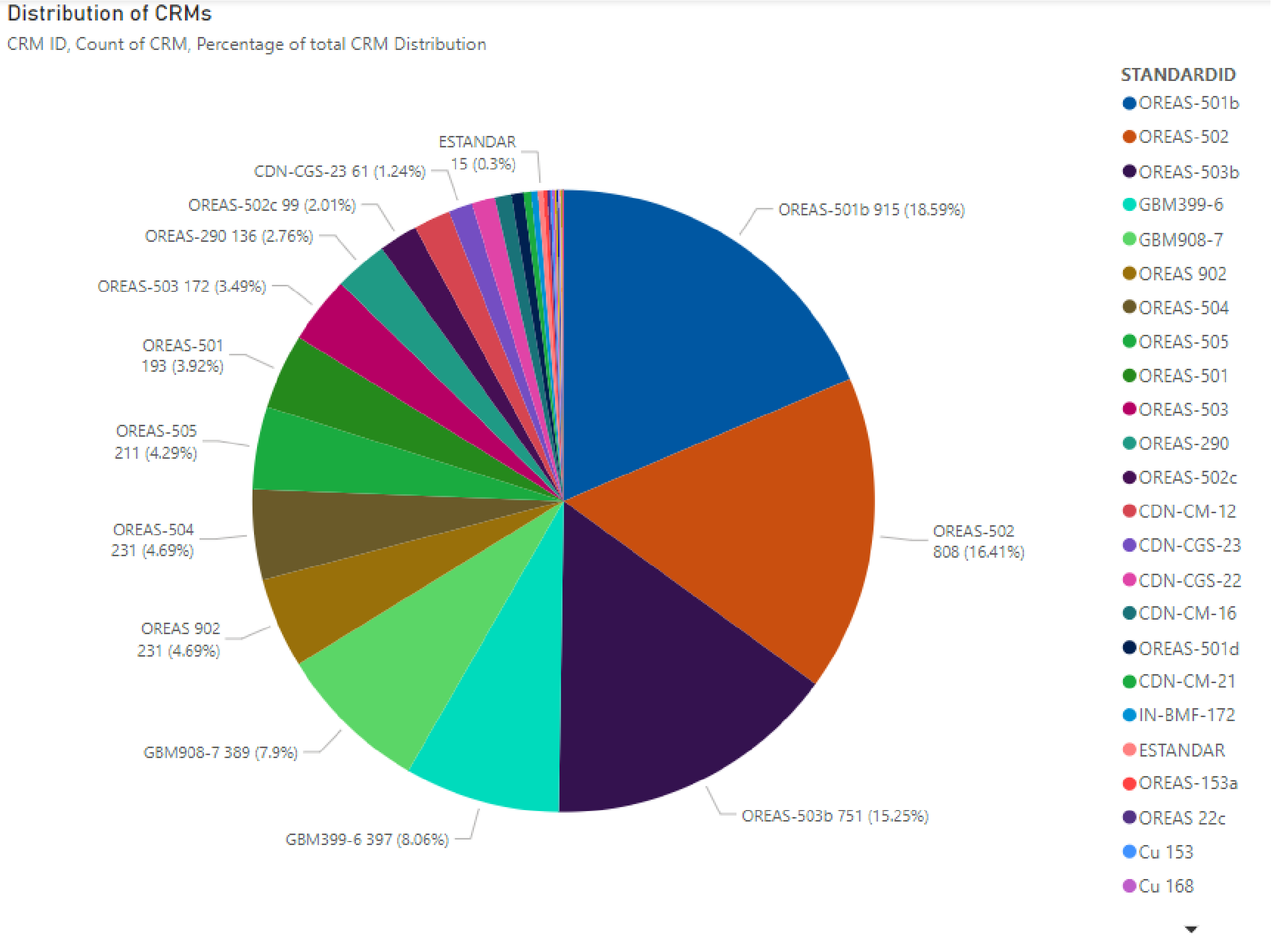
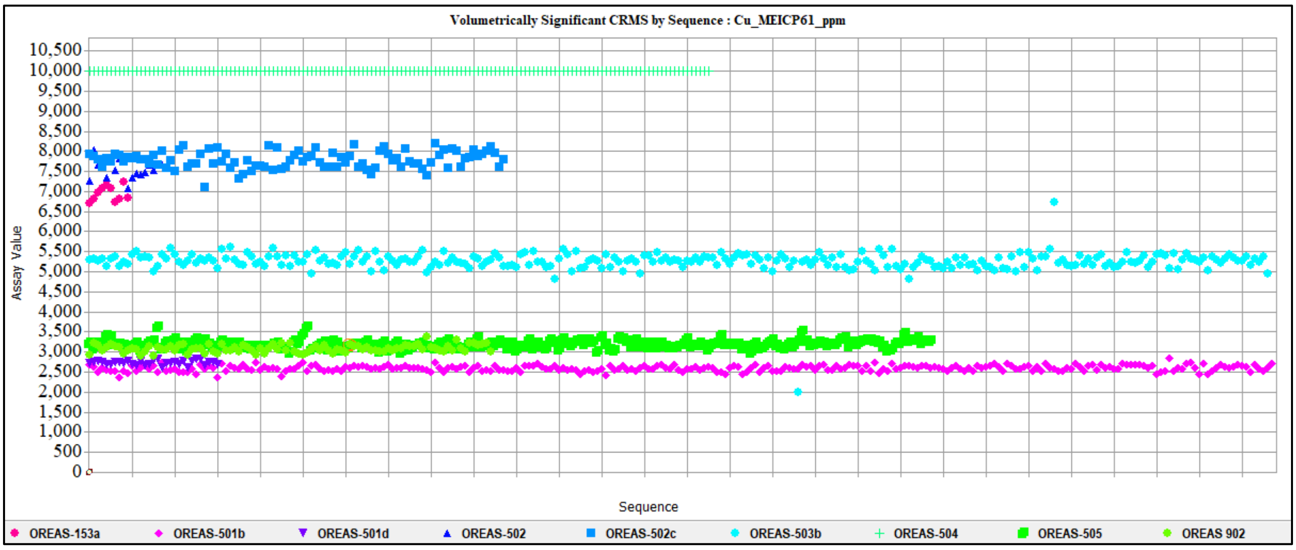


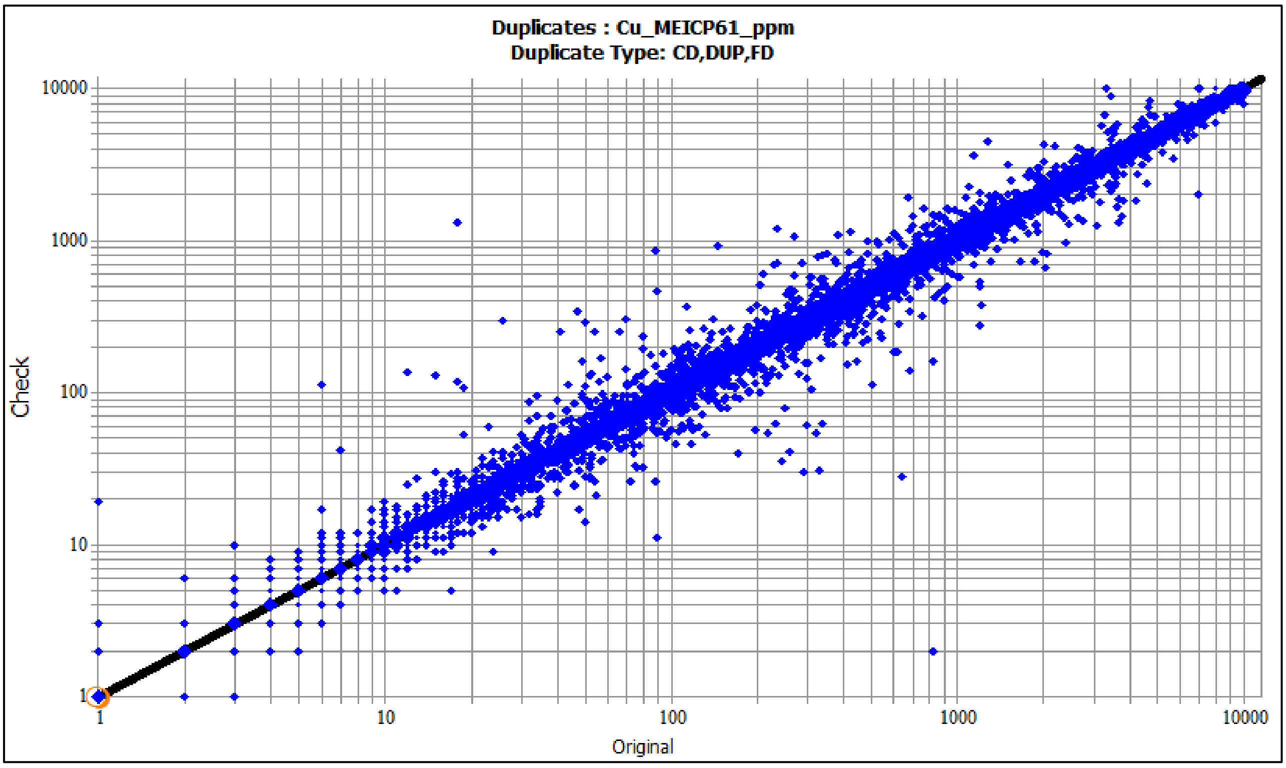
Figure 11.3 : CRM summary for all projects for most volumetrically significant CRMS showing coverage across a wide range of Cu ppm ranges.



11.7.1.2 Duplicate Pair Summary

The HCH analytical database stores information on the laboratory method used for each analysis, and as such there are a variety of analytical methods and results stored for each element. The most common laboratory method is reported in this section and is disclosed on each figure. The total number of duplicate pairs, across all methods, is 6,718. The most common method for copper is the analytical method MEICP61 which reports in Cu ppm and results of the duplicate pairs across the Costa Fuego Project are shown in Figure 11.4, a strong correlation across all copper value ranges.

Figure 11.4 : Duplicate pair log-log scatter plot for Cu ppm across the Costa Fuego Project



11.7.2 Certified Reference Materials

HCH utilised several multi-element mineralised “standards” (certified reference material or CRM) and certified “blank” samples supplied by Ore Research & Exploration Pty Ltd (OREAS). The standards used by HCH for QA/QC are packaged as 60 g pulp bags.

One mineralised standard was inserted every 50th metre into each batch of samples submitted for analysis. The reference material type and grade ranges for the CRM standards used for QA/QC correspond to the rock type and mineralisation grades (copper, gold and molybdenum) routinely encountered within each project. These standards enable checks on analytical accuracy.

The top three most used standards, and their relative percentage of the CRM population are discussed in this section (Table 11.5). Each of the top three CRM’s are from Ore Research Pty Ltd and are prepared from a porphyry copper-gold deposit located in central western New South Wales, Australia with the addition of a minor quantity of Cu-Mo concentrate.

Table 11.5 : Summary of common copper CRM proportions and expected values used at the Costa Fuego Project

Standard ID	Count	% of CRM Population	Certified Value (Cu ppm)	Acceptable Min (Cu ppm)	Acceptable Max (Cu ppm)
OREAS-501b	915	19%	2600	2270	2930
OREAS-502	808	17%	7550	6950	8150
OREAS-503b	731	15%	5310	4620	6000
Total Top 3	**2,454**	**51%**			
Total CRMS	**4,836**				

11.7.2.1 OREAS-501b

OREAS-501b performed consistently well with no samples returning outside of the error tolerance range and a statistically significant population of this CRM was used at each project site (Figure 11.5 to Figure 11.7).

Figure 11.5 : OREAS-501b results for Cu ppm for Cortadera

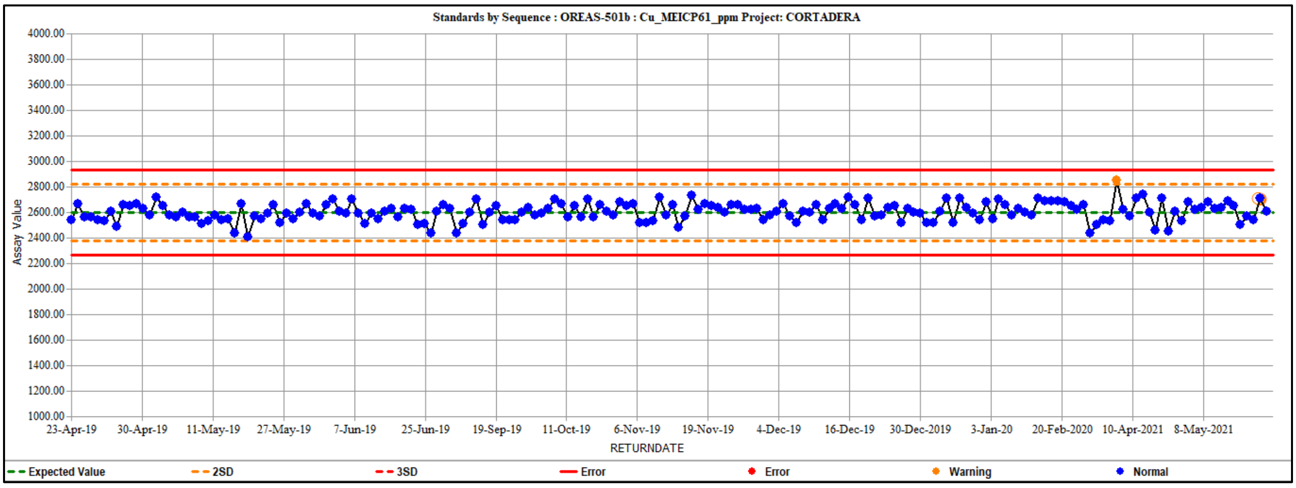


Figure 11.6 : OREAS-501b results for Cu ppm for Productora.

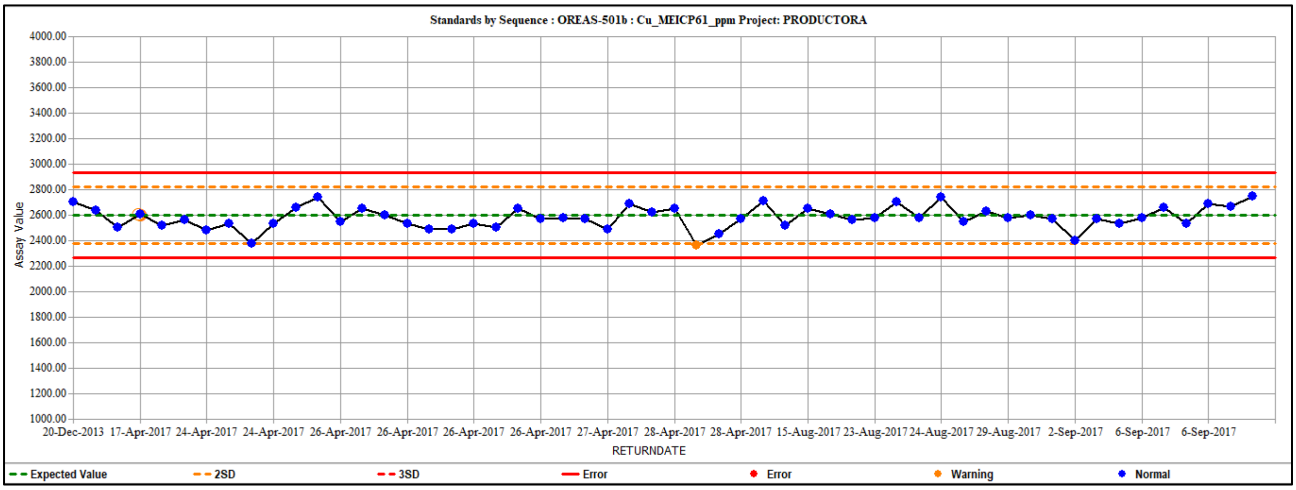
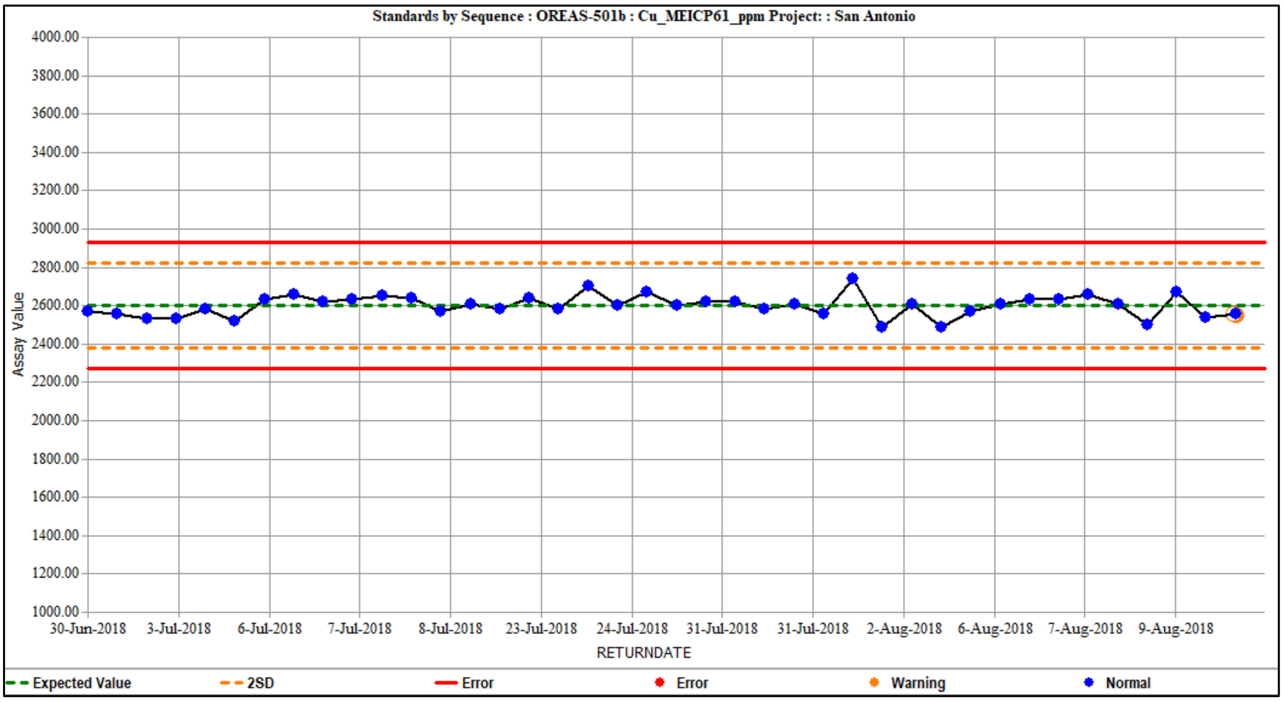


Figure 11.7 : OREAS-501b results for Cu ppm for El Fuego San Antonio



11.7.2.2 OREAS-502

Performance of this standard was excellent, with no results outside of the expected values at Cortadera and Productora (Figure 11.8 and Figure 11.9). No samples were used at El Fuego San Antonio.

Figure 11.8 : OREAS-502 results for Cu ppm for Cortadera.

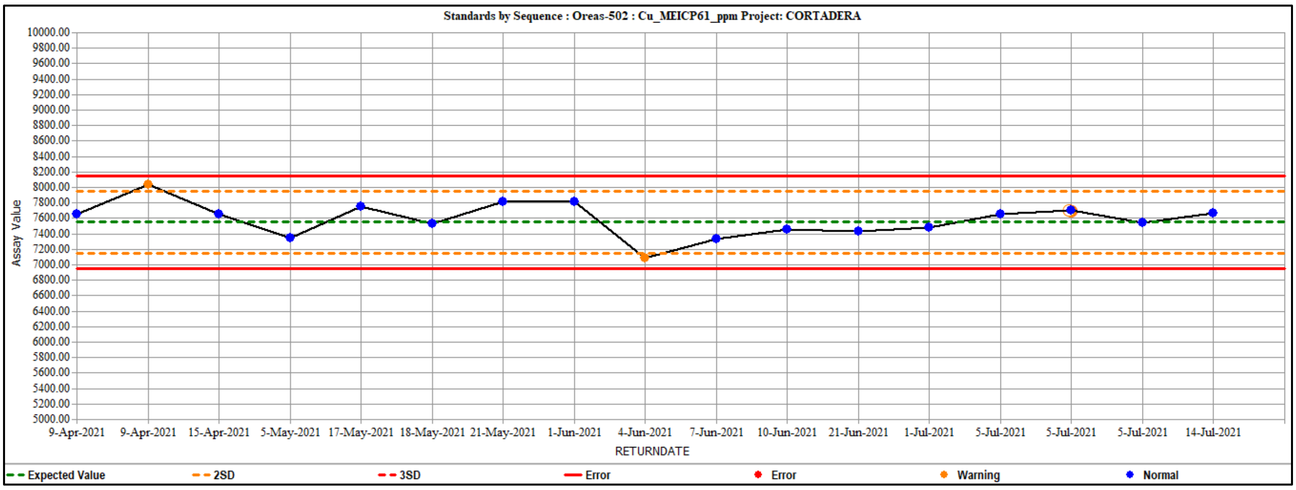
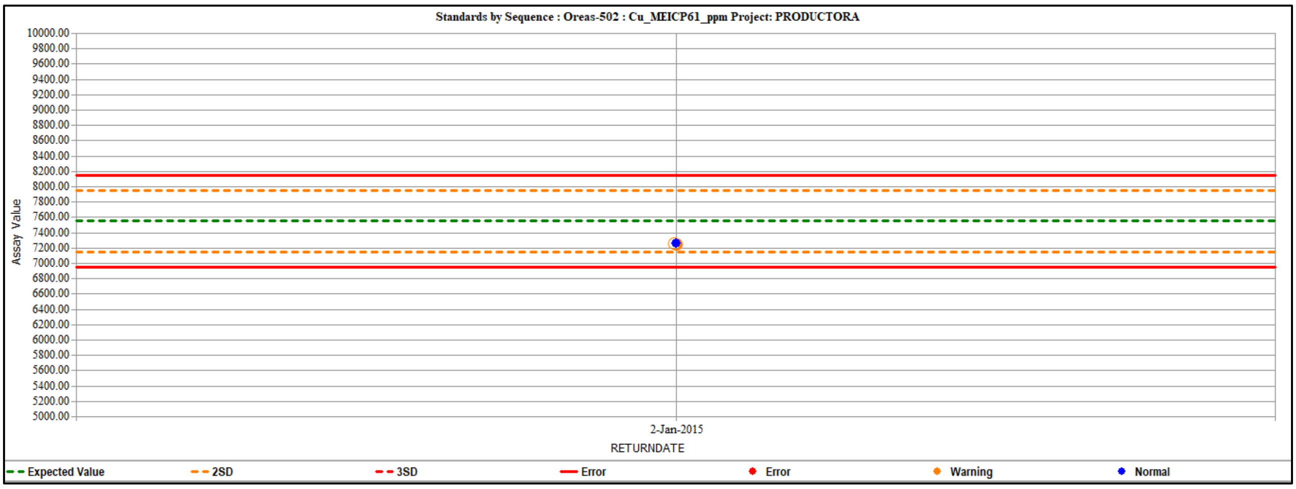


Figure 11.9 : OREAS-502 results for Cu ppm for Productora.



11.7.2.3 OREAS-503b

Performance for OREAS-503b is reasonable with only 2 samples falling outside of the error limits at Cortadera equating to a 99% pass rate (Figure 11.10 to Figure 11.12). All samples at Productora and El Fuego San Antonio also passed.

Figure 11.10 : OREAS-503b results for Cu ppm for Cortadera

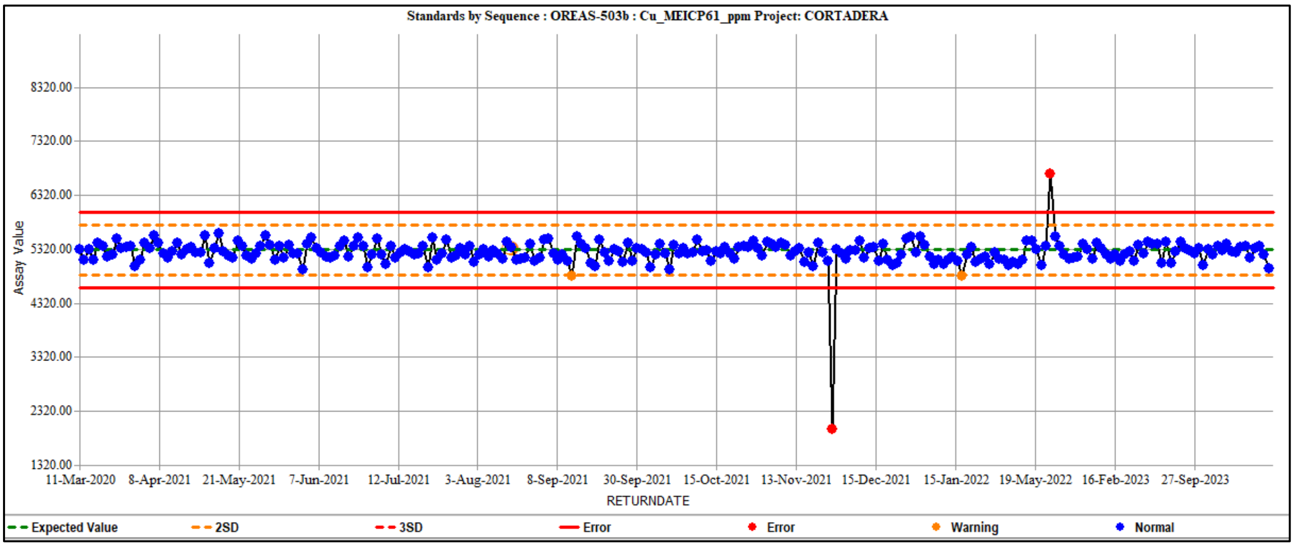


Figure 11.11 : OREAS-503b results for Cu ppm for Productora.

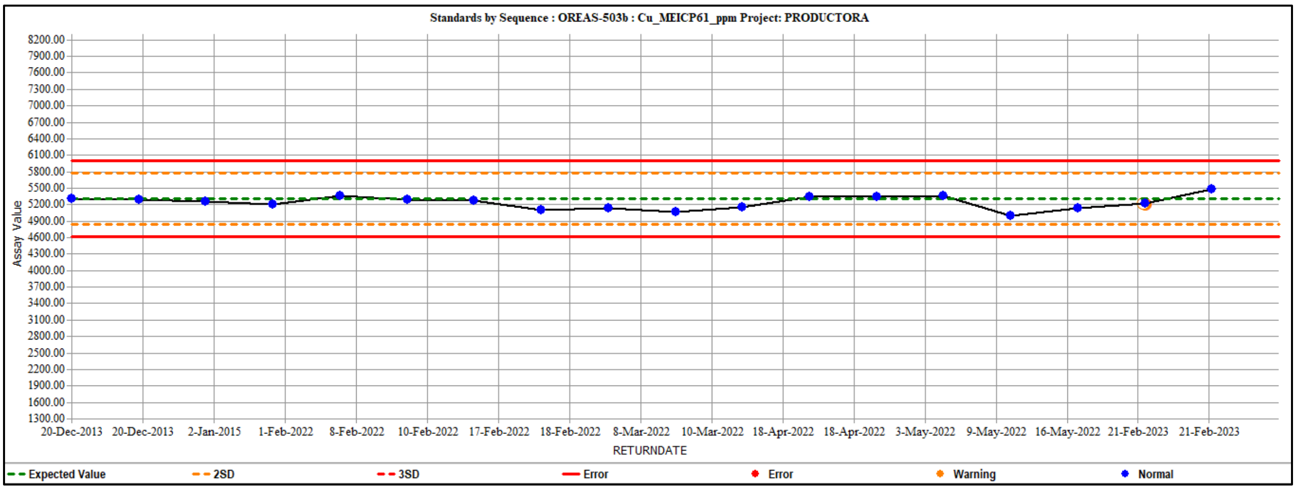
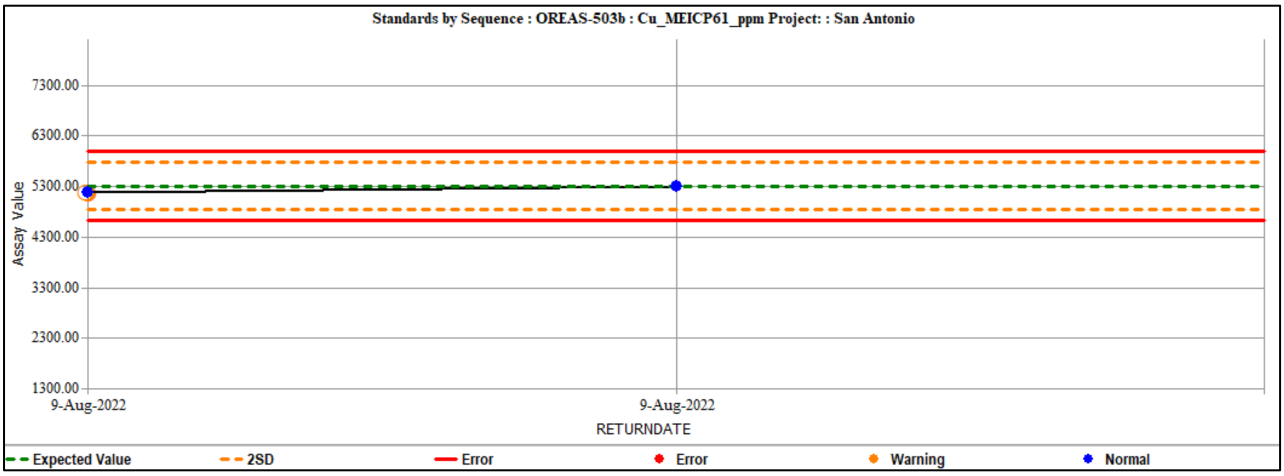


Figure 11.12 : OREAS-503b results for Cu ppm for El Fuego San Antonio



11.7.3 Blanks

HCH include both certified blanks and coarse quartz blanks in the QAQC programme. Certified blanks act as a CRM allowing for analysis of the accuracy of the analytical method, however, these bypass the sample preparation as they are delivered in pulverised form. One unmineralised "blank" was inserted every 100th sample submitted for analysis. Quartz blank samples undergo the full sample preparation workflow as the regular samples, thus highlight potential contamination resulting from the crushing and pulverising processes.

OREAS 22c is the most common certified blank in the database, representing 64% of the total certified blank population Figure 11.13 to Figure 11.15 show the performance by project area.

Any outliers identified are reviewed as the assays are returned and investigated if they are a standard swap or caused by preparation or analytical quality issues and whether the surrounding batch of samples needs to be re-assayed. The location of the outlier is taken into consideration as well, with CRM failures within the mineralised domains cut-off grades prioritised for review.

Figure 11.13 : OREAS 22c (Blank) results by Cu ppm for Cortadera.

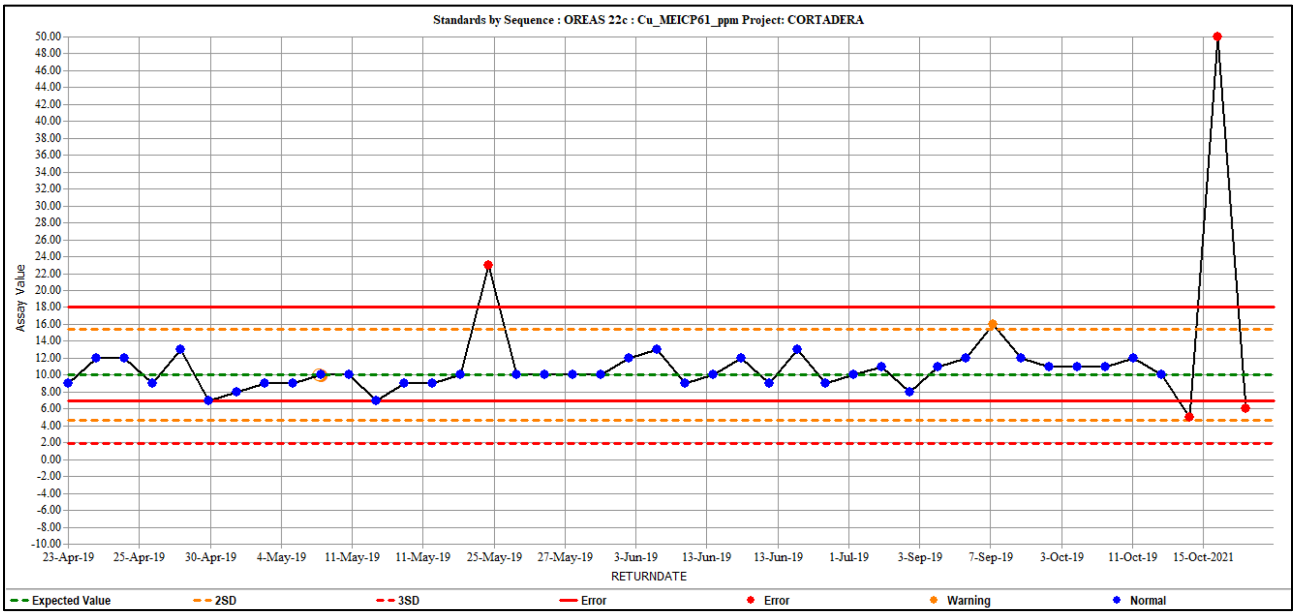


Figure 11.14 : OREAS 22c (Blank) results by Cu ppm for Productora.

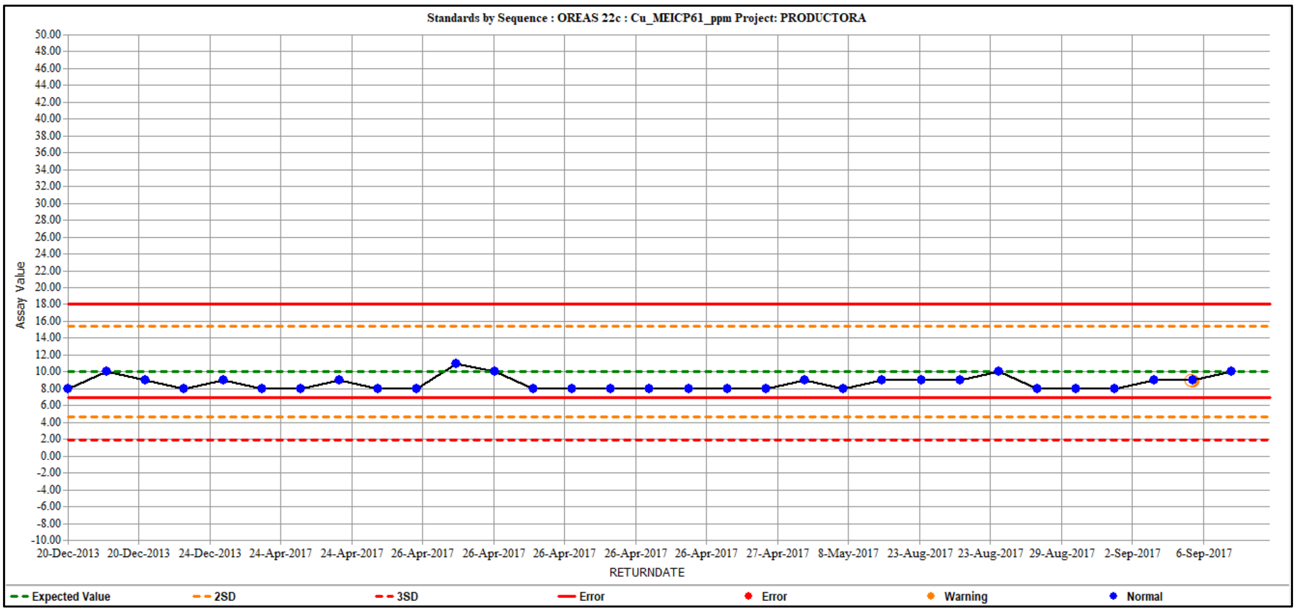
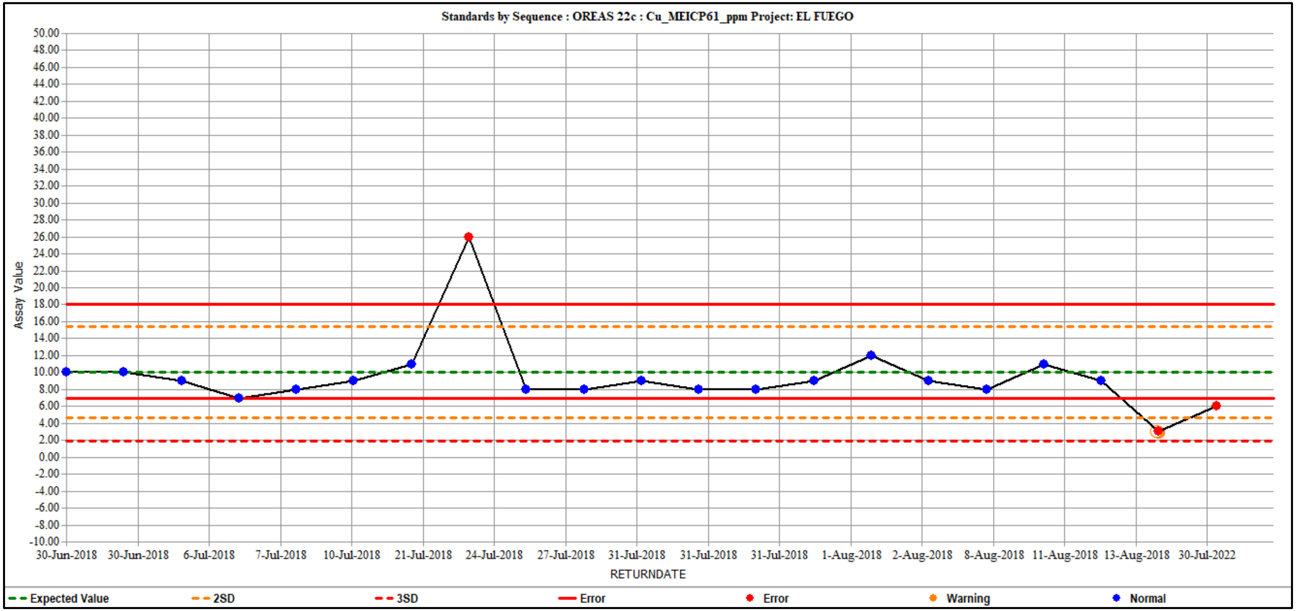


Figure 11.15 : OREAS 22c (Blank) results by Cu ppm for El Fuego San Antonio



HCH submits its own coarse quartz blanks (as distinct from the also HCH-submitted ‘blank’ pulp), as pulp CRMs bypass most of the initial sample preparation processes in the laboratory (Figure 11.16 to Figure 11.18). The submission of coarse blank material allows investigation of any potential crushing and milling bias. The coarse blank samples were inserted at the geologist’s discretion with an emphasis on placement in visual high-grade zones as well as at the start and end of a batch. Where samples return above the nominal 30 ppm Cu upper target, it is investigated to determine the relative percentage of the cross-sample contamination. Contamination up to 5% relative to the surrounding values is accepted.

Figure 11.16 : Quartz blank results for Cu ppm for Cortadera

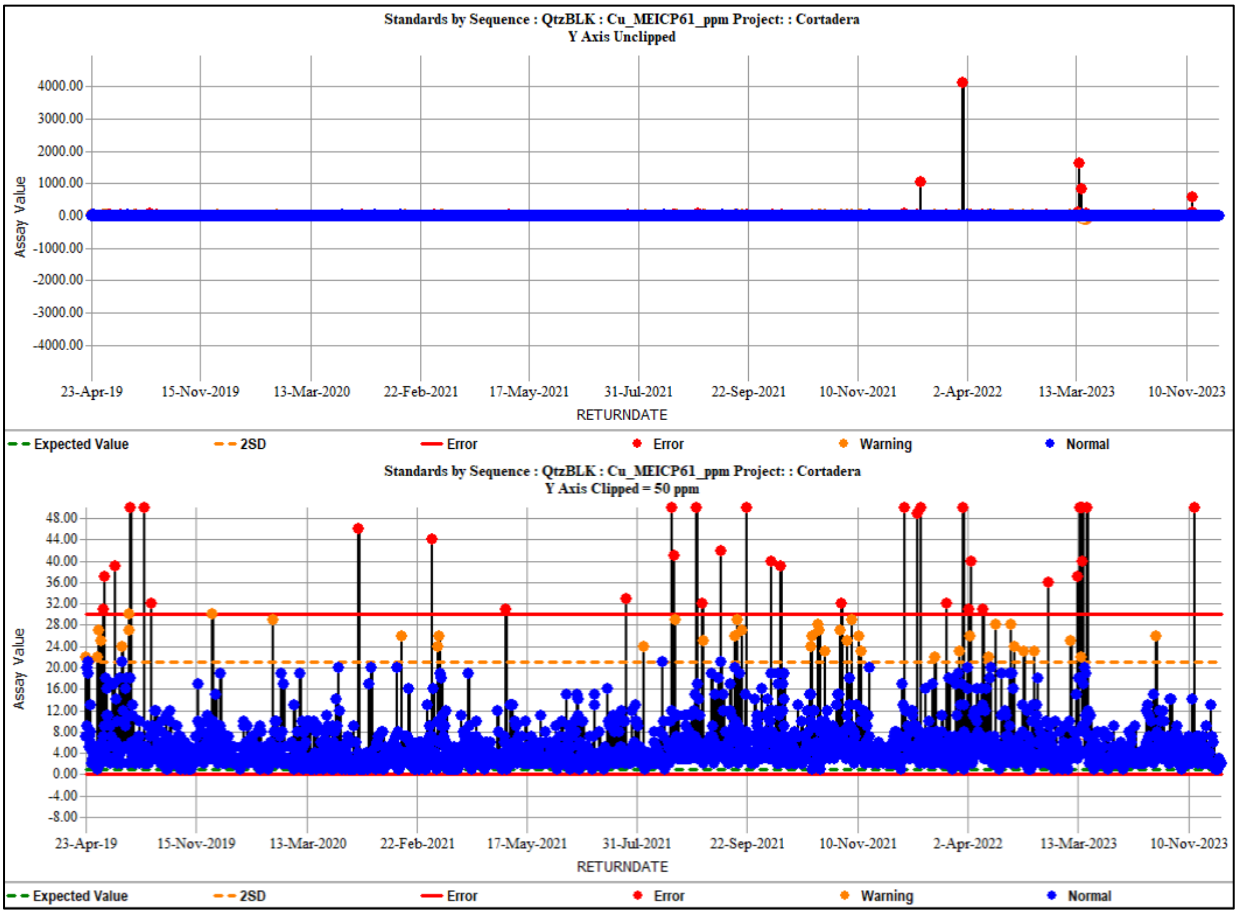


Figure 11.17 : Quartz blank results for Cu ppm for Productora

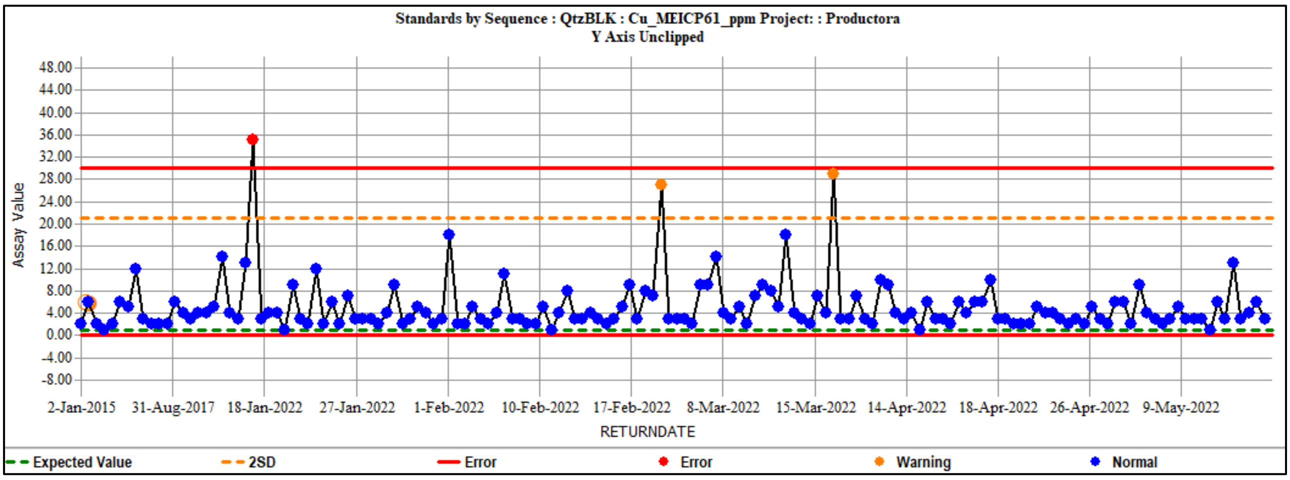
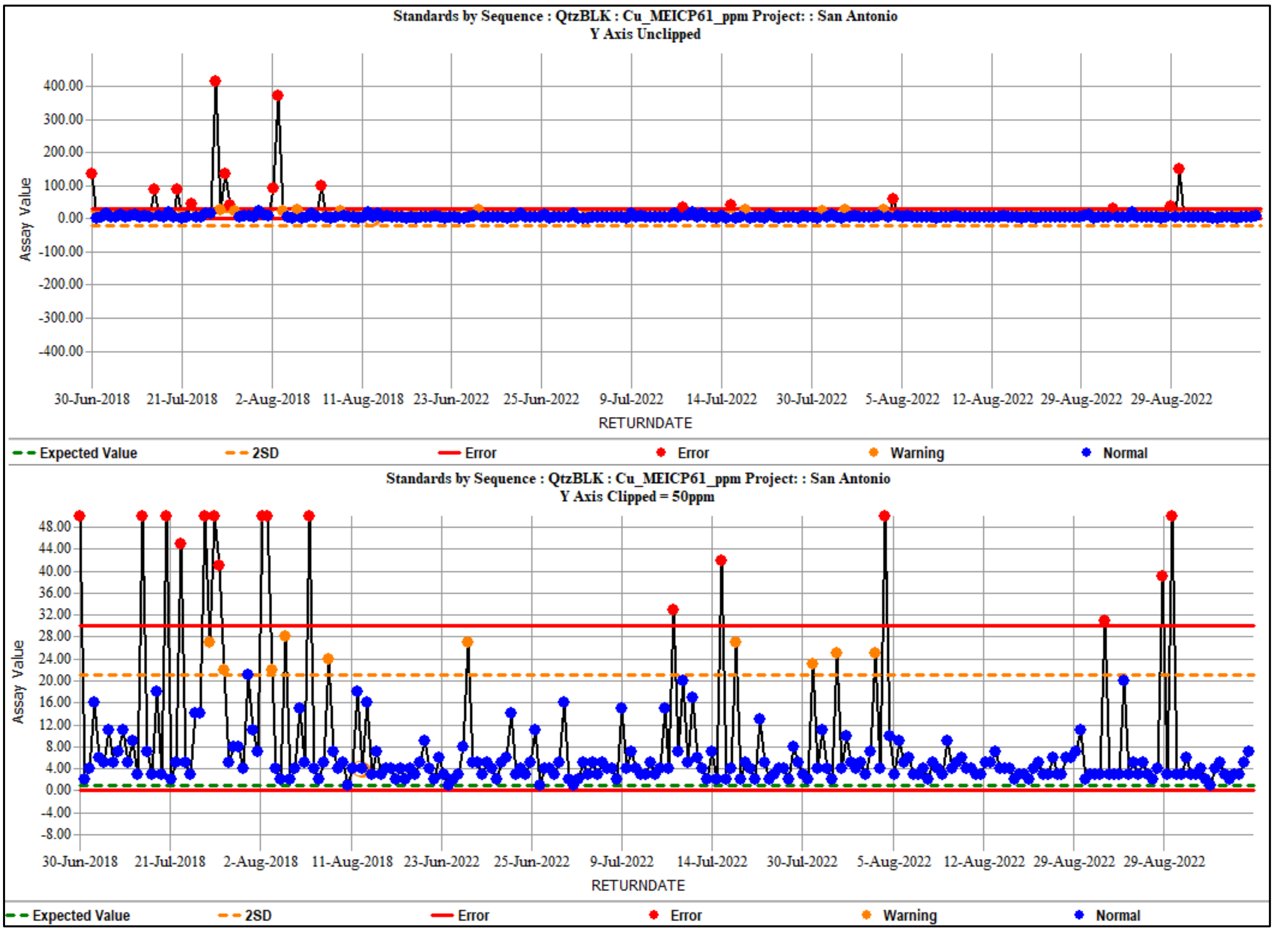


Figure 11.18 : Quartz blank results for Cu ppm for El Fuego San Antonio



11.7.4 Duplicate Pairs

All duplicates assigned by HCH to date are of the coarse sample type, where HCH geologists identify the need for a duplicated sample during the sampling procedure and the preparation laboratory takes a second sample to fill this duplicated sample following the crush and split stage of sample preparation. HCH sampling protocols implemented in 2022 target 5% as an insertion rate, with older sampling below this rate (Table 11.6).

Table 11.6 : Duplicate Pair Summary by Project

Project	Number of Duplicate Pairs	Insertion Rate
Cortadera	1,939	3%
Productora	4,634	2%
San Antonio	145	2%
Total	**6,718**	**2%**

There is a high correlation between the original and check samples with a statistically small number of outliers occurring within each project population.

Copper is routinely analysed by MEICP61 and reported as ppm. Values over 10,000 ppm are re-analysed using an ore grade method, creating an artificial ceiling to this method. The lower detection limit is 1ppm.
Strong correlation was noted across each project and copper value ranges, with Productora showing a wider dispersion of results (Figure 11.19 to Figure 11.21).

Figure 11.19 : Duplicate results for Cu ppm – Cortadera

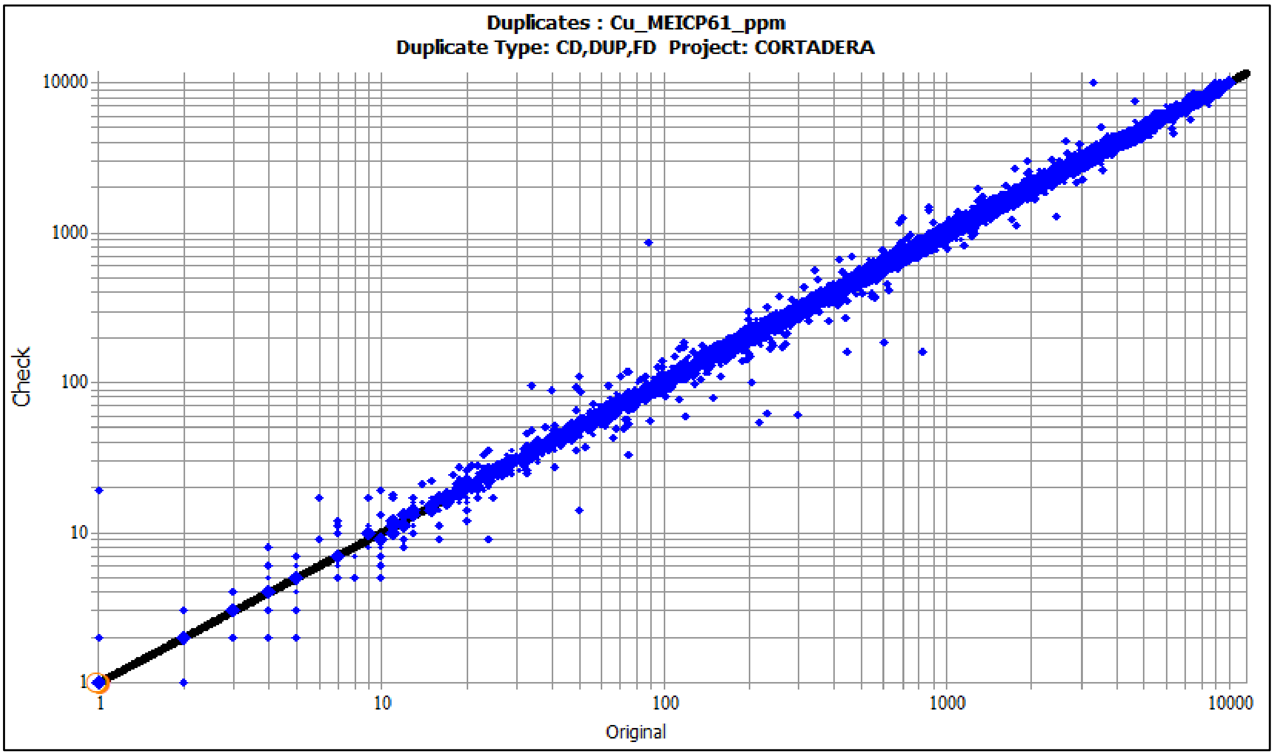


Figure 11.20 : Duplicate results for Cu ppm – Productora

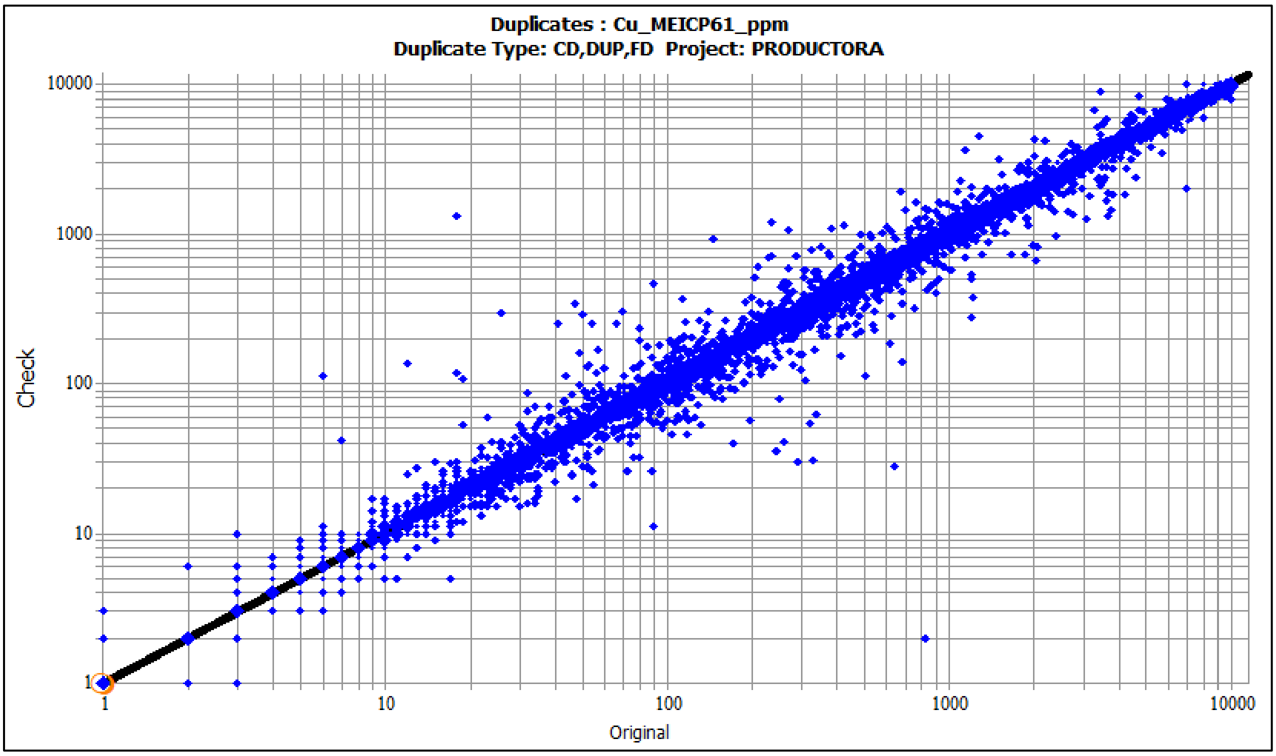
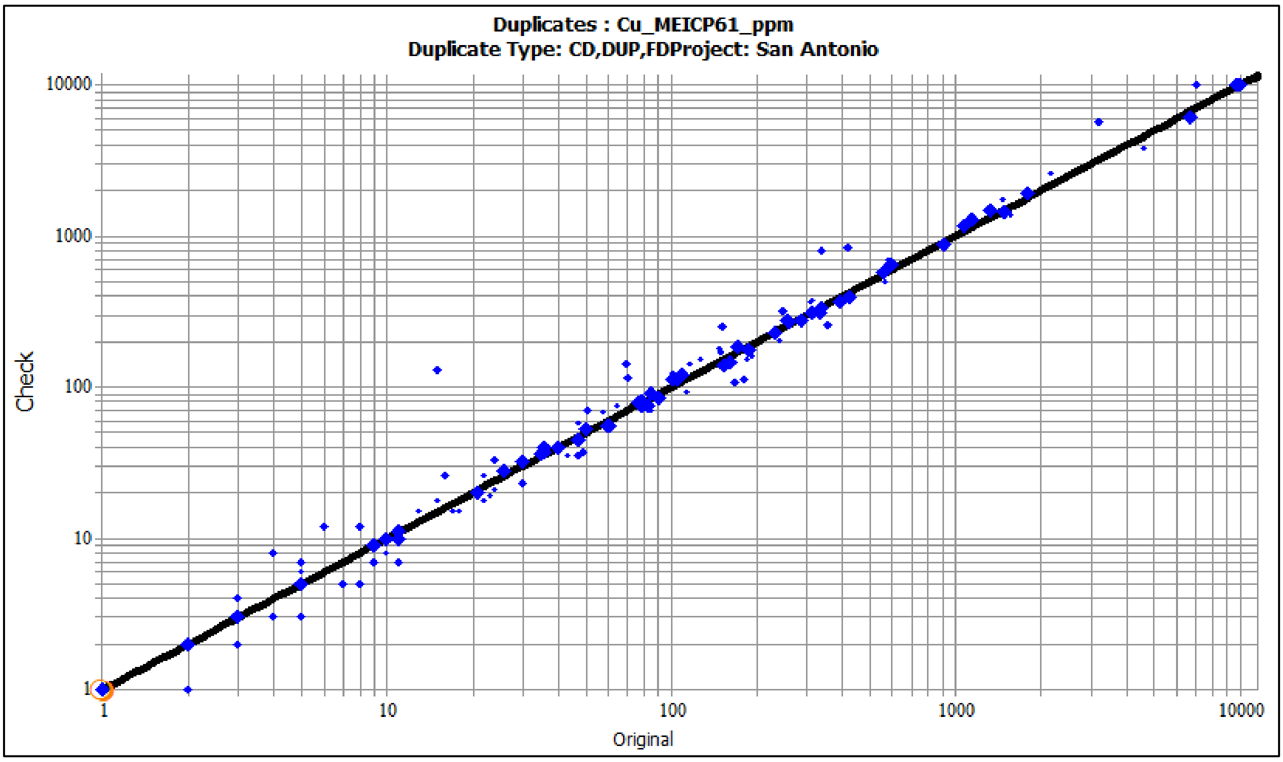


Figure 11.21 : Duplicate results for Cu ppm- EL Fuego San Antonio



11.7.5 Inter-Laboratory Comparisons

Inter-laboratory (umpire) programs are completed for each project. Umpire results are stored in the HCH AcQuire geological database as part of the QAQC program.

During the Productora 2014/2015 drilling campaign, pulp and coarse rejects were chosen from those submitted to ALS Global for umpire testing. These were submitted to Bureau Veritas in Santiago for assay as umpire checks on ALS's analysis. The analytical method is similar to that used by ALS (four-acid digestion, followed by ICP-AES determination). Umpire results for copper and molybdenum showed a fair correlation. Gold showed a noisier but overall fair correlation with no quantifiable bias relative to the original samples. This gives sufficient confidence in the relative accuracy and repeatability of ALS's analytical assay technique.

An umpire programme for Cortadera was conducted in 2021. In May 2021 108 samples, including 9 QAQC samples (7 standard/certified reference material, and 2 Blanks), were collected for the program. Sample types included percussion drilling rock chips, and NQ and HQ half core samples. Samples were sent to Bureau Veritas Laboratory in Santiago and results analysed in September 2021. The analysis found no consistent or material bias between the ALS and Bureau Veritas Laboratory across the analytes of Cu, Mo, Au and Ag. And no further recommendations were made related to this program.

Umpire samples for San Antonio were selected as part of a wider 2022 umpire program, which also contained samples from Cortadera and Productora drilled within 2022-2023. Samples were sent to Bureau Veritas Laboratory in Coquimbo (Chile) and results analysed in February 2024. The analysis found no consistent or material bias between the ALS and Bureau Veritas Laboratory across the analytes of Cu, Mo, Au and Ag. The program did omit a Cu overlimit method, which was triggered in multiple samples and a recommendation to conduct this additional analysis was made post review. Results from this analysis will be collated and reported.

11.8 Comments on Section 11

11.8.1 Minero Fuego Data

There is no record of procedures employed by Minera Fuego for their Cortadera drilling, sampling and logging. However, the remaining half-core has been obtained by HCH and utilised for validation, as well as subsequent twinned drilling and the initial 40,000 m resource drill program by HCH which confirmed the validity of the Minera Fuego drill holes. An additional 67,000 m resource infill and extensional drilling has been completed at Cortadera, since the maiden resource in October 2020.

11.8.2 Conclusions

The sample lengths, preparation and assay techniques are considered suitable for the styles of mineralisation and deposit types found at Productora, Alice, Cortadera and San Antonio. A site visit by QP Elizabeth Haren was completed in May 2022 which reviewed all current sampling protocols and observed drilling and sampling being conducted on site. Ms Haren visited the ALS sample preparation laboratory facilities in La Serena, Chile and the ALS analytical facilities in Lima, Peru. Ms Haren determined that the inclusion of these assays in the Costa Fuego Project Mineral Resource Estimates was appropriate.

A site visit was also completed by the Company's Principal Resource Geologist and Resource Development Manager in 2023, which included a detailed review of the RC sampling procedure at Cortadera.

12 Data Verification

12.1 Independent Qualified Person Review and Verifications

12.1.1 Pre-2013 Productora Independent Sample and Assay Verification

A limited number of verification samples were taken by Coffey Mining during a site visit in November 2012. A total of 17 samples from four drill holes were selected at random. Samples were taken by Coffey and delivered to the ALS analytical laboratory in Coquimbo, Chile. The results were directly sent to Coffey in Perth (Australia) and supported the original assays.

12.1.2 2014 Productora Resource Estimate Verification

AMC Consultants were engaged on a fee basis to conduct a peer review and external audit of the Productora Resource estimate.

A representative of AMC Consultants visited the Productora area in late September/early October 2014 as part of an independent site review, as well as completing an audit of the ALS preparation laboratory facilities in La Serena, Chile. AMC Consultants had access to the data, models and reports referred to in this Report and were involved in reviewing the Resource estimation process and inputs. As no significant additional drilling has been completed since the AMC audit, their findings remain current as of the date of this Report.

Subsequent verification of the Productora database was completed by the QP in 2021, during review of the Productora Mineral Resource Estimate.

12.1.3 2014 Productora Independent Sample and Assay Verification

AMC Consultants was engaged on a fee for service basis to undertake an independent review of site procedures and sampling.

Part of the review's scope of work was to collect independent check samples of the Productora mineralisation. These samples included:

- "B" field duplicates on current drill holes
- Five consecutive field duplicates from PRP0864
- Laboratory coarse rejects or "C" bulk sample rejects.

In total, 60 samples were collected from approximately 18 drill holes for various locations, styles of mineralisation, levels of mineralisation, and levels of weathering/oxidation.

The samples were taken by a representative of AMC Consultants and delivered to the ALS analytical laboratory in Coquimbo, Chile. The results were sent directly from ALS to AMC, which determined that they supported the original assays.

12.1.4 2024 Costa Fuego Project Independent MRE Audit

In February 2024, SD2 Consultants were engaged on a fee for service basis to review the February 2024 Mineral Resource Estimate, which included a review of data and the QAQC management program. The review did not identify any material errors or omissions, and the estimates were found to be of good quality, suitable for public reporting and for use in operational design and scheduling.

Recommendations made regarding the QAQC management program were minor and included suggestions to review:

- Routine (fortnightly – monthly) reporting to use a wider window to show trends at a three- to six-month scale
- Show the calculated standard deviation for performance and how this relates to the certified standard deviation for CRMs with a significant population in the database
- Use of pairs of similar CRMs where possible, to reduce the laboratories' ability to guess the expected value for repeated CRMs.
- Review the calculation of the +/- 20% error lines on duplicate plots in the routine reporting.

12.1.5 2024 Costa Fuego Project Independent Mineral Resource Estimate Assurance Review

In August 2024, Mark Noppe of the Sustainable Minerals Institute (SMI) were engaged by HCH to provide assurance, or otherwise, that the Mineral Resource estimation and reporting workstream for input into this Report has processes, methodologies, quality assurance/quality controls and documentation in place and completed by suitably qualified and experienced people that are fit for purpose for this Report and/or permitting and that any gaps in these items should not materially compromise the outcomes of this Report.

Over 120 documents were reviewed during the assurance process, including 10 QAQC documents, to ensure three levels of quality control and assurance were in place. The Reviewer was satisfied that HCH's reporting governance over the reliable generation and public reporting of its Exploration Information and Mineral Resources for the Costa Fuego Project deposit Mineral Resource is supported by appropriate assurance activities and controls and appropriate for both public reporting and input into this Report.

12.1.6 2024 Costa Fuego Independent Metallurgy Assurance Review

In September 2024, Sacanus and Met Engineering were engaged by HCH to provide assurance, or otherwise, that the metallurgical testwork and reporting workstreams for input into this Report has processes, methodologies, quality assurance/quality controls and documentation in place and completed by suitably qualified and experienced people that are fit for purpose for this Report and/or permitting and that any gaps in these items should not materially compromise the outcomes of this Report. Overall, the Metallurgical Assurance Report, by Sacanus and Met Engineering, considered that the work undertaken is fit for purpose, the proposed outcomes are achievable and support a PFS level of confidence.

Six key recommendations were made within the review:

- Construct a pilot plant to confirm the developed process flowsheet is viable at the project scale.
- Complete blend testwork to ensure materials from different domains/deposits can be co-processed without negatively impacting plant performance.

- Complete additional geometallurgical testwork to ensure sample representivity and update geometallurgical models
- Complete lock cycle testwork on Cortadera mineralised material
- Complete trade off studies for key aspects, including SAG vs HPGR grinding, flotation in fresh vs sea water, and grind optimisations with updated copper prices and operating costs.
- For leaching, complete closed loop column cell tests.

12.2 Verifications by Wood

12.2.1 Section 13 – Mineral Processing and Metallurgical Testing

As part of the data verification exercise, the metallurgical samples and procedures selected for the analytical testing were evaluated and found, for the most part, to be fit for purpose. The most material issue was raised by both the client and the metallurgical testing laboratory (Auralia Metallurgy Pty Ltd) was in regard to the accuracy of the molybdenum assays, especially for flotation tailings samples.

12.2.1.1 Molybdenum Contamination and Correction Procedure

A strong bias was demonstrated where the calculated head analyses were consistently higher than the measured head analyses for the same samples for all of the flotation tests performed before 2021. This dominantly affected Productora results from 2012 through to 2021 and it also affected some of the early work on Cortadera from 2018. The effect lessened and was then effectively eliminated as sources of Mo contamination were eliminated from the sample and testing preparation pathways.

The grinding media (rods) and the grinding mill itself were found to consist of high Mo stainless steel. Small amounts of steel are always liberated from the rods and the inside of the mill due to the grinding action and these will contaminate the sample. The main high value components of the stainless steel, Ni and Cr, are not payable metals in the Project, while Fe is present in the ore at such high levels that the contamination from grinding has no significant effect upon the iron balance. The Mo, however, is a payable metal and the Mo contamination is high enough to materially affect its balance.

Typical natural Mo levels in the ores range from 20 ppm to 300 ppm. Assuming the steel used for grinding is an alloy that contains 10% Mo, it only takes 0.1g of steel contamination to raise the Mo level of the feed by 10 ppm. Steel contamination is not recovered to concentrates because it does not float with the Cu and Mo minerals. Therefore, all of the steel and its Mo contamination effect report to the flotation tailings. As flotation concentrates typically represent less than 10% of the feed mass a 10 ppm Mo contamination of feed translates directly to an 11 ppm Mo contamination of tailings.

The amount of contamination, in grams of steel, is not constant as it depends upon the amount of grinding performed. The approximate Mo contamination levels in tailings was determined by subtracting the calculated Mo head assay from the measured Mo head assay for the same test sample. A clear trend of increasing contamination was evident for each of the test samples as the grind time increased. Increasing grind times are used to target finer flotation feed P_{80} values. These trends were clearly sample specific and all had similar sloped trends. The relationships were not perfect as they ignore the expected natural variation in uncontaminated Mo feed grade for any given sample. However, the trends were strong enough and consistent enough to generate a lookup table of Mo contamination for all affected samples and composites. All that is required to use the lookup table is the sample name and the target grind P_{80} in microns. The correction values in the lookup table range from only 1 ppm, when grinding to the coarse flotation feed size of 212 µm with three different samples, up to 142 ppm when grinding to the fine flotation feed size of 75 µm with composite sample "SDG".

The differing contamination levels for each sample reflect the varying abrasiveness and hardness of the ore minerals in the grinding environment. It is notable that an attempt to incorporate measurements of Bond's abrasion index into this analysis was unsuccessful.

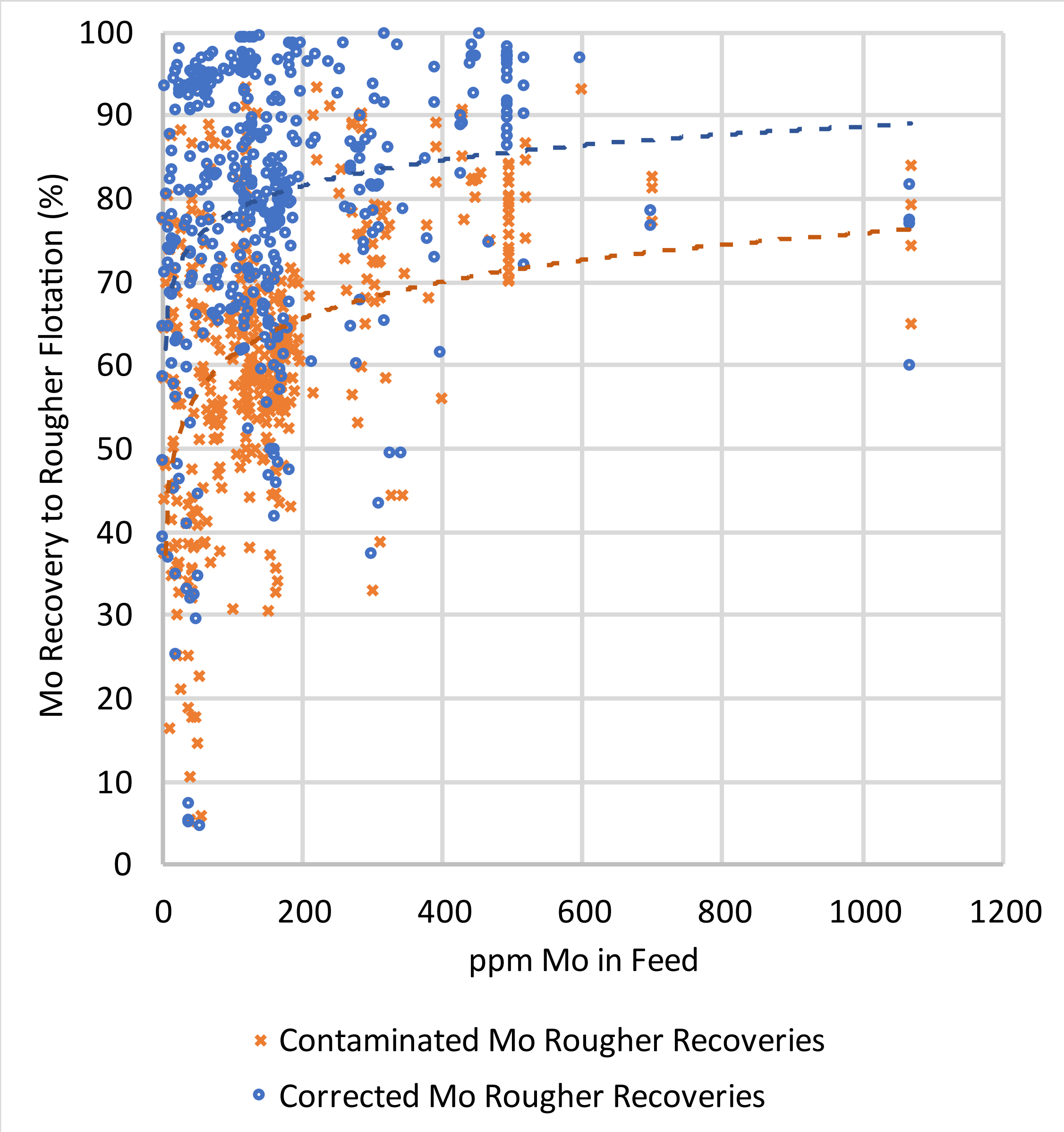
Using these relationships, each of the affected flotation test results (more than 300 tests) was modified by subtracting the Mo correction value for the test from the tailings assay. Note that the correction value was subtracted from the tailings rather than the head sample to incorporate some more conservatism into the correction. As a result of the correction the recovery of Mo in the test would increase. The increase amount varies because of varying head grades of Mo and the varying recovery levels of Mo in the test itself. Some samples would increase Mo recovery by only a few percent and others would increase recovery by 30% or more.

The impact of this correction was to derive new recovery estimation equations for Mo for each of the deposits. These equations have been incorporated in mine planning and flow through into revenue generation estimates. In addition, the mass balance in the Mo recovery area (Cu/Mo flotation and Mo roughing and cleaning) now reflect the higher recovery outcomes and the higher concentrate mass flows within the Mo circuit.

In a small number of cases it was necessary to decrease the Mo correction value such that it didn't result in a Mo recovery of >100%. This was typically performed with samples that provided >90% Mo recovery uncorrected.

The global effect of the correction can be seen in Figure 12.1. The global average Mo rougher recovery increased from 62% to 79% after correction and this is a relative increase of 27% recovery. Note that these averages represent all tests performed, including tests with poor results and using poor molybdenum collectors together with more recent tests for which there is no correction necessary and none was applied. To model Mo recovery for design and revenue purposes, specific subsets of tests were selected to determine appropriate Mo recovery equations for each of the deposits and their optimised circuit outcomes.

Figure 12.1 : Molybdenum Recovery Correction for Contamination -



12.2.1.2 Other Non-Material Data Issues

No other data issues having a material impact upon the Project were detected in the review of past and recent test results. Some less important issues were detected relating to the low metal grades, especially in tailings for Cu, Au, Ag and other minor elements. Some past judgements regarding comparisons between grind sizes, reagents and other flotation conditions were found to be misleading due to the reported precision of the assays and repeatability of the assays in tailings samples. For, example, if a change of the last significant figure in a tails assay (say from 0.05 to 0.04% Cu) results in a recovery difference of 5 or 10% for the test, then it is virtually impossible to compare the recovery effect of changing a flotation test condition. The effect was that some older optimisation comparisons have been reassessed as there being no difference between comparative test as opposed to the original assessment where one condition was considered superior to the rest.

Steps were taken in recent work to ensure that the precision of reporting, or where necessary the precision of the analytical method, was sufficient to detect recovery changes of the order of 1%, which is beyond the repeatability of sampling and testing.

The QP has verified the data by comparing it to the supporting documentation. As a result of the data verification and with the above allowances and corrections, the QP concludes that the Project data and inputs are suitable for the assessment of the concentrator testwork.

12.2.1.3 Leaching Testwork

During the 2024 leaching metallurgical tests carried out in columns and at a semi-pilot scale, the copper content of the PLS solutions in each test was rigorously controlled.

Indeed, the 1 m column leaching test program, using ore samples from Productora and Cortadera deposits, involved a total of 890 drainages, which were analysed for copper, iron, acid, and other metals.

Likewise, in the testing program in iso-containers and large-diameter columns, 270 chemical analyses of drainage solutions were performed.

In general, it can be said that – in the case of copper – all analyses involving concentrations above 10 g/l were first analysed by atomic absorption (AA), but the final result came from a volumetric analysis, since atomic absorption analysis required numerous sample dilutions, which often leads to analytical and human errors. Effluents with copper content below 10 g/l were analysed entirely by atomic absorption.

Chemical analyses were generally performed in a certified laboratory (Metalab) with internal and external controls. In addition, one in every 10 samples analysed for copper, both by atomic absorption and volumetric analysis, was sent to a second certified control laboratory (Geomet).

The results allow us to ensure that the analyses carried out by the Metalab laboratory are closely related to the control analyses carried out by Geomet, both in the case of analyses by volumetry (Figure 12.2) and those carried out by atomic absorption (Figure 12.3).

In all column and Iso-container tests apart from solution analysis, metallurgical balances were closed by analysing the heads and tails of the solid samples using atomic absorption.

Figure 12.2 : Chemical Analysis Results – Main Laboratory v/s Control Laboratory Analyses by Volumetry

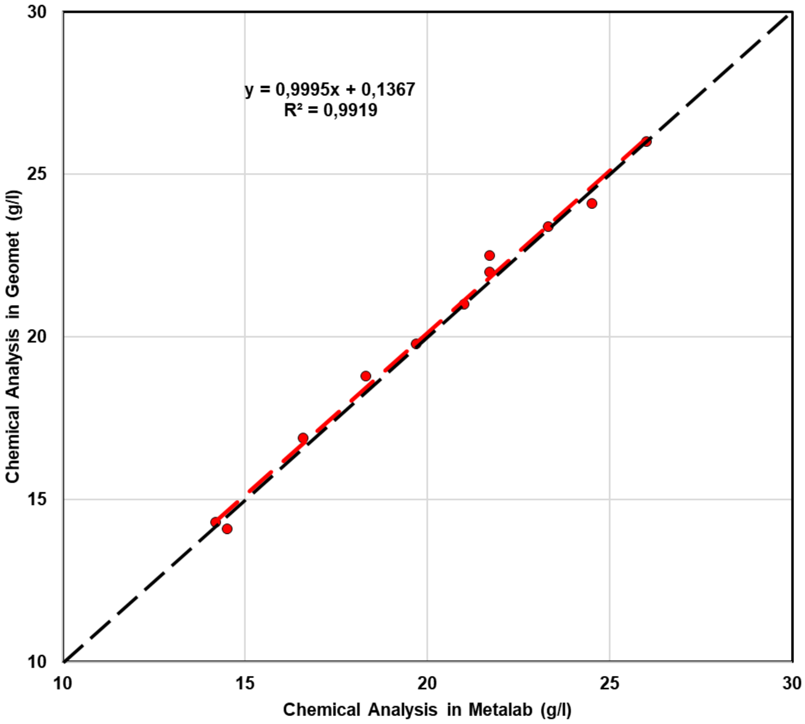
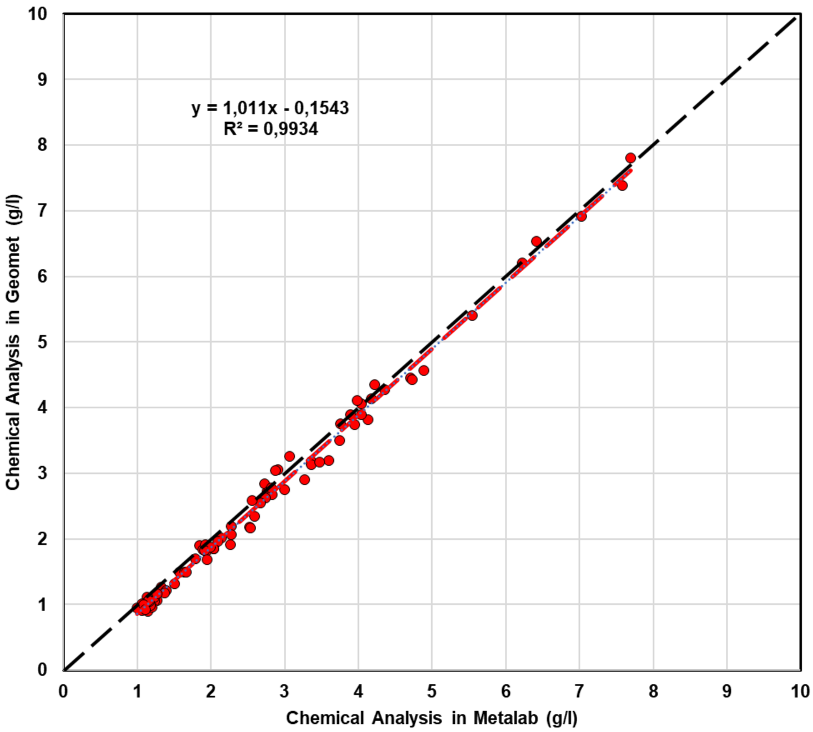


Figure 12.3 : Chemical Analysis Results – Main Laboratory v/s Control Laboratory Analyses by Atomic Absorption



12.2.2 Section 17 – Recovery Methods

12.2.2.1 Concentrator

The QP has verified the data by checking original sources, comparing it to the supporting documentation and by benchmarking. As a result of the data verification, the QP concludes that the Project data and inputs are adequate for developing the recovery methods applied in the PFS.

12.2.2.2 Leaching

The QP has verified the data by checking original sources, comparing it to the supporting documentation. As a result of the data verification, the QP concludes that the Project data and inputs are adequate for developing the recovery methods applied in the PFS.

12.2.3 Section 18 – Project Infrastructure

As part of data verification exercise, the infrastructure QP has verified the data by comparing it to the supporting documentation. As a result of the data verification, the QP concludes that the Project data and inputs are adequate for the purposes used in the Project infrastructure.

12.2.4 Section 19 – Market Studies and Contracts

As part of data verification exercise, the market studies and contracts QP has verified the data by comparing it to the supporting documentation. As a result of the data verification, the QP concludes that the Project data and inputs are adequate for the purposes used in the market study and contracts.

12.2.5 Section 21 – Capital and Operating Costs

As part of data verification exercise, the capital and operating costs QPs has verified the data by comparing it to the supporting documentation to confirm the cost estimates. The review considered escalation, scaling due to increase in production capacity and a review of power and diesel pricing. As a result of the data verification, the QPs concludes that the data and inputs are adequate for the purposes used in the cost estimates.

12.2.6 Section 22 – Economic Analysis

As part of data verification exercise, the economic analysis QP has verified data inputs to the financial model being consistent with the relevant content in the other sections of this Report. As a result of the data verification, the QP concludes that the inputs are suitable for use in the economic analysis.

The economic analysis QP reviewed supporting documentation on taxes provided by Asesorias Bindu SpA that supports the use of the tax information in the financial model.

12.3 Verifications by ABGM

Two third-party audits were completed on the Costa Fuego Project data, block models and Mineral Resource Estimates in 2024. The Qualified Person (Elizabeth Haren) is a third party to HCH who has visited site and completed extensive independent data verification prior to estimation being completed.

The mine optimisation and mine planning parameters were obtained through benchmarking exercises and a comprehensive mining parameters database developed over a 15-year period (ABGM PTY LTD Planning and mining database). The geotechnical parameters used in this study, were based on the 2016 Study and a preliminary slope stability study developed by third party companies (Ingeroc) who visited site in 2022 and Geomechanics Mining and Technology (GMT) who completed a conceptual study for the block cave in 2021.

12.4 Verifications by Elizabeth Haren

The internal data verification and quality assurance programs were deemed appropriate by the Qualified Person (Haren Consulting) for Mineral Resource Estimation. Haren Consulting is a third party to HCH and completed extensive independent data verification of collar and downhole surveys, assays, geological logging and mapping, structural measurements and drilling metadata prior to estimation being completed.

Haren Consulting was engaged on a fee basis to conduct a peer review and external audit of the Costa Fuego Project Mineral Resource Estimate and visited site in May 2022. An audit of the ALS sample preparation laboratory facilities in La Serena, Chile and ALS analytical facilities in Lima, Peru was also completed at this time.

12.5 Verifications by High River Services

High River Services had access to the data, models and reports referred to in this Report, and was involved in reviewing the existing baselines, audits, management plans and other relevant environmental documents on the Project.

The EIA of the Project is currently being prepared and no significant additional environmental information has been completed as of the effective date of this Report; therefore, High River Service’s findings as presented in this Report are current as of the date of this Report.

12.6 Verifications by GMT

Geotechnical data verification for the Costa Fuego Project was completed by GMT in 2024 under the supervision of the Qualified Person, Mr. David Cuello. This process aimed to ensure the accuracy, representativeness, and completeness of the geotechnical information used in support of the open pit and underground design criteria.

Verification activities included:

- Field Inspections: Three site visits were undertaken to Cortadera, Productora, and San Antonio to verify the consistency between observed conditions, core logs, and interpreted structural models. This included review of selected drill cores, geological and geotechnical logs, and surface mapping.
- Data Audit and QA/QC: A centralized geotechnical database was constructed integrating drillhole logs, mapping records, and laboratory results. QA/QC checks were conducted to assess alignment between mapped and calculated parameters, including RQD, FF/m, GSI, RMR, and Q' values. Statistical correlation analysis was applied to identify and correct anomalies or bias.

- Laboratory Test Review: The UCS, triaxial, Brazilian tensile strength, and direct shear test datasets were reviewed for test quality, spatial distribution, and failure mode validity. Non-representative results were excluded to preserve dataset integrity.

The Qualified Person has reviewed the verification program and considers the geotechnical dataset sufficient for its intended use in mine design and resource classification at the PFS stage. No material errors or omissions were identified that would affect the interpretations or conclusions presented in this Report.

12.7 Comments on Section 12

Haren are satisfied with the exploration, sampling, security, and QA/QC procedures employed by HCH for the Costa Fuego Project and that their results are sufficient to produce data suitable for the purposes described in this Report.

Wood is of the opinion that metallurgical samples and procedures selected for the analytical testing carried out as part of this study are of sufficient quality for inclusion in a PFS.

ABGM is of the opinion that the data pertaining to the mine design and schedule is sufficient to support this PFS.

13 Mineral Processing and Metallurgical Testing

13.1 Introduction

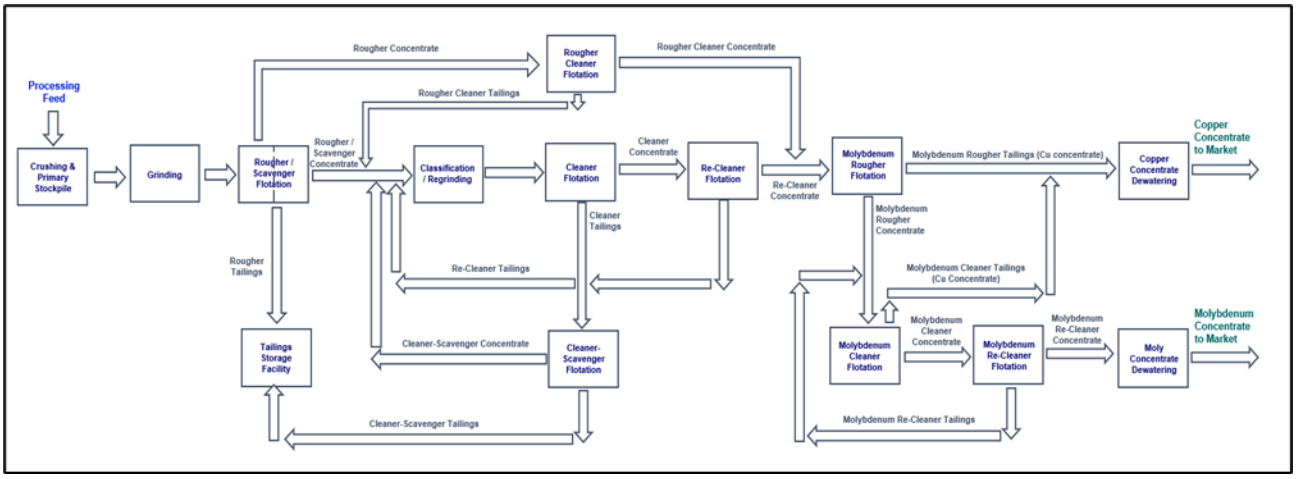
This section summarises the results from metallurgical testwork programs completed by HCH for the Costa Fuego Project from 2012 to the present.

Mineralisation is moderately complex and varied across four different deposits at the Costa Fuego Project. The geology and mining studies have arrived at a combined open-pit and underground mining operation. The metallurgical testwork programs have resulted in process facilities consisting of:

- a concentrator treating sulphide ore and producing separate copper concentrate (containing gold and silver credits) and molybdenum concentrate
- heap leach and dump leach facilities treating oxide ore and low-grade sulphide ore respectively, which produces copper cathode. The hypersaline process using concentrated brine from electrodialysis is utilised.

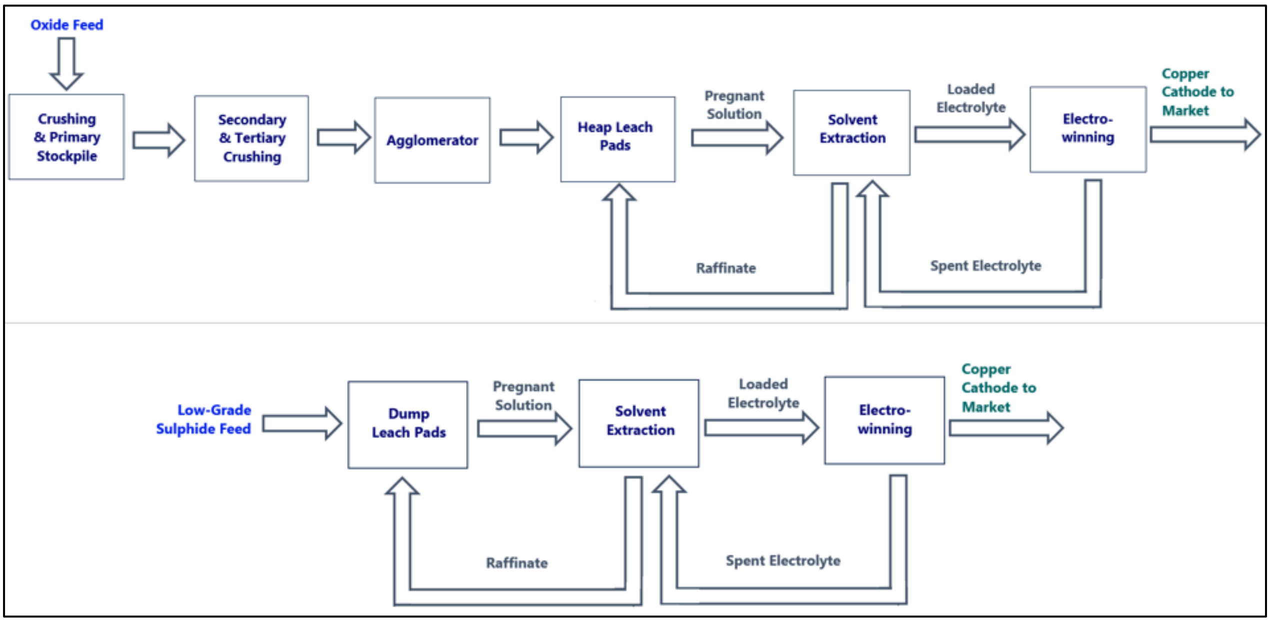
For guidance with interpreting the testwork, simplified Costa Fuego Project concentrator and leaching block flow diagrams are shown in Figure 13.1 and Figure 13.2 respectively.

Figure 13.1 : Costa Fuego Project Concentrator Process Plant Block Flow Diagram



Source: Process Plant Block Flow Diagram from Wood (2025)

Figure 13.2 : Costa Fuego Project Leaching Process Plant Block Flow Diagram



Top: Heap leaching. Bottom: Dump leaching. Both stand-alone operations with common SX-EW facilities
Source: PMC, 2025

The metallurgical testwork programs completed at the Costa Fuego Project across the history of the Project are outlined in Table 13.1.

Table 13.1: Costa Fuego Project Metallurgical Testwork Programs

Processing Type	Project(s)	Year	Purpose	Number of Samples	Laboratory (Job)
Heap Leach of Oxide	Productora / Alice	2014 - 2015	Preliminary Heap Leach Characterisation (QEMScan, Sequential Cu, Cu distribution by size, bottle roll)	14	Mintrex (1425) ALS (SC100)
			Comminution testwork (Ai, CWi, UCS)	3	
			Heap Leach Stacking, Agglomeration, and Hydrodynamic Testing	3	HydroGeoSense
			Column leach test (1 m)	5	ALS (SC100)
		2021 - 2022	Heap leach testwork – percolation tests, bottle roll, and column leach tests (4.5 m)	2	IMO (6312)
		2024	Variability Study - column leach tests (1 m), including comparison of hypersaline and conventional acid leaching	29	Nova Mineralis
	Cortadera	2020	Bottle Roll Testing	3	Auralia (AM063)

Table 13.1: Costa Fuego Project Metallurgical Testwork Programs

Processing Type	Project(s)	Year	Purpose	Number of Samples	Laboratory (Job)
	Cortadera	2024	Variability Study- column leach tests (1 m), including comparison of hypersaline and conventional acid leaching	8	Nova Mineralis
Dump Leach of Sulphide	Productora	2021 - 2022	Conceptual Study – 30 cm micro-column leach tests	4	Nova Mineralis
		2024	Variability Study - 1 m column leach tests	8	Nova Mineralis
			Scaling Study – IBC test stacked equiv. 9 m	1	Nova Mineralis / Aminpro
	Cortadera	2021 - 2022	Conceptual Study – 30 cm micro-column leach tests	3	Nova Mineralis
		2024	Variability Study – 1 m column leach tests	14	Nova Mineralis
Flotation	Productora/ Alice	2012 - 2013	Comminution testwork	24	Mintrex/ALS (SC93/SC98)
		2013-2014	Comminution, mineralogical and flotation testwork	3	Ausenco/ALS (A14273)
		2014 - 2015	Comminution testwork	5	Mintrex/ALS (A16082)
		2014 - 2015	Flotation testwork - flowsheet development	13	Mintrex/ALS (A16082)
			Flotation testwork - variability composites	15	
			Flotation testwork and development of molybdenum flotation scheme	2	
		2020 - 2022	Flotation – Productora transition, Productora Fresh and Alice	9	Auralia (AM063) Auralia (AM146)
			Check of Productora material against Cortadera flotation scheme	2	
		2024	Flotation – Sea Water vs. Tap Water – Productora	1	Auralia (AM248)
			Flotation – Sea Water vs. Tap Water – Alice	1	
		2025	Molybdenum Flotation Sighter Testwork	1	Auralia (AM263)
	Cortadera / San Antonio	2020 - 2022	Flotation flowsheet development – Cortadera Open Pit	24	Auralia (AM063) Auralia (AM099)
			Flotation flowsheet development – Cortadera Underground	21	
		2020 - 2022	Comminution testwork	22	Auralia (AM063) JKTech (22011) Auralia (AM099)
		2022	Flotation and Mineralogy – Cortadera Open Pit and Underground	2	Auralia (AM099)
		2024	Flotation – Seawater vs. Tap Water – Cortadera Open Pit	1	Auralia (AM248)

Table 13.1: Costa Fuego Project Metallurgical Testwork Programs

Processing Type	Project(s)	Year	Purpose	Number of Samples	Laboratory (Job)
			Flotation – Seawater vs. Tap Water – Cortadera Underground	1	
			Flotation – Seawater vs. Tap Water – San Antonio	1	
		2025	Molybdenum Flotation Sighter Testwork	2	Auralia (AM263)

13.2 Mineralogical Analysis

Mineralogical examination was conducted at ALS Metallurgy (Perth) on Productora and Alice feed samples used for flotation Productora PFS testwork (various reports). Mineralogy included x-ray diffraction (XRD) and QEMScan (Quantitative Evaluation of Minerals by Scanning Electron Microscopy).

The samples examined by XRF for each deposit included:

- Productora: FD-1 to FD-14 (excluding FD-7), V-1 to V-6, V-11 to V-12 (21 samples)
- Alice: V-13, V-14 and V-18 (3 samples).

13.2.1 Mineralogy by X-ray Diffraction (XRD)

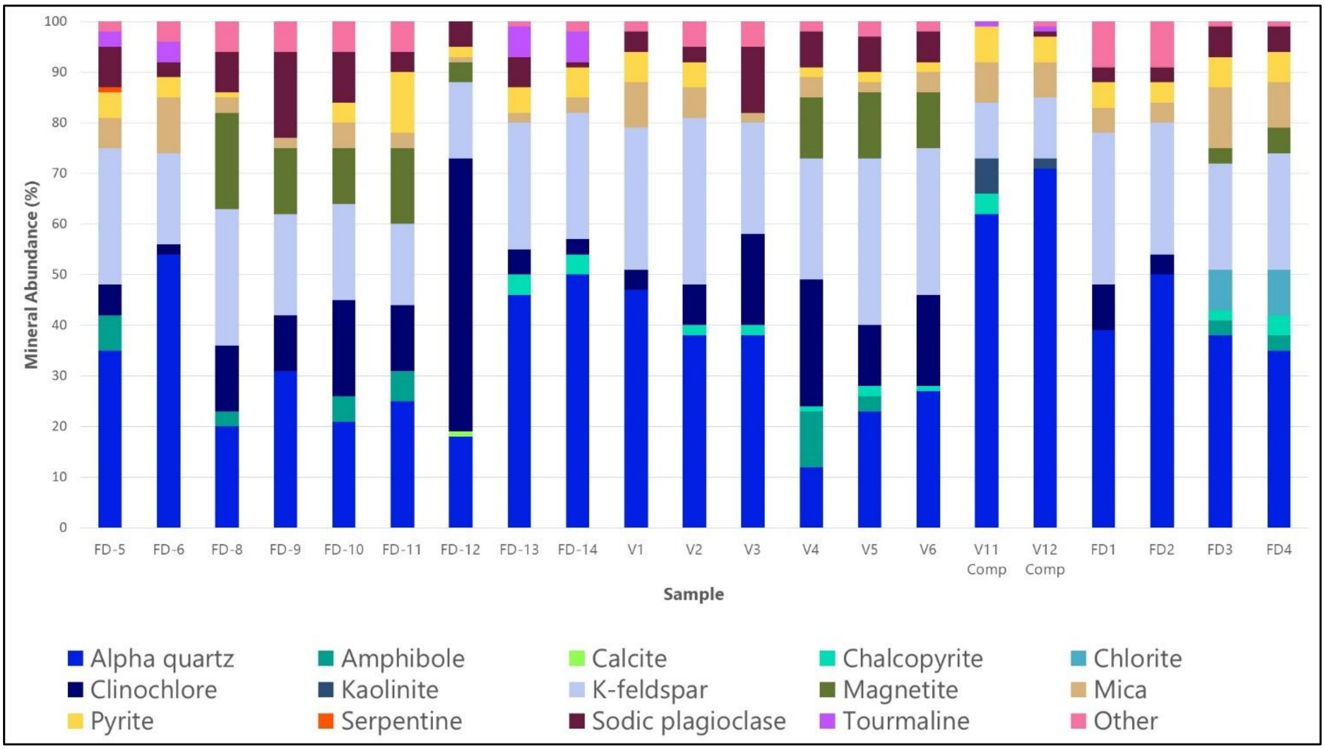
The XRD mineral deportments of the fresh Productora flotation test samples are shown in Figure 13.3.

Alpha quartz is the dominant mineral in Productora followed by potassium feldspar and clinochlore. Note that Alpha quartz is the room temperature crystal structure form. It is only identified as Alpha quartz because XRD identification relies on the crystal structure. The beta form of quartz only occurs at temperatures above 570°C. All future quartz identification in this Section will simply be referred to as quartz.

There is about 5% sodium feldspar (sodic plagioclase) on average. The two sulphides identifiable by this technique are chalcopyrite and pyrite. No other copper bearing minerals were identified within the accuracy of the method, which is typically down to about 1% by volume of a well-defined species.

As the span between Maximum and Minimum values demonstrate, the mineral contents are variable across the samples.

Figure 13.3 : XRD Analysis of 21 Productora Flotation Feed Samples

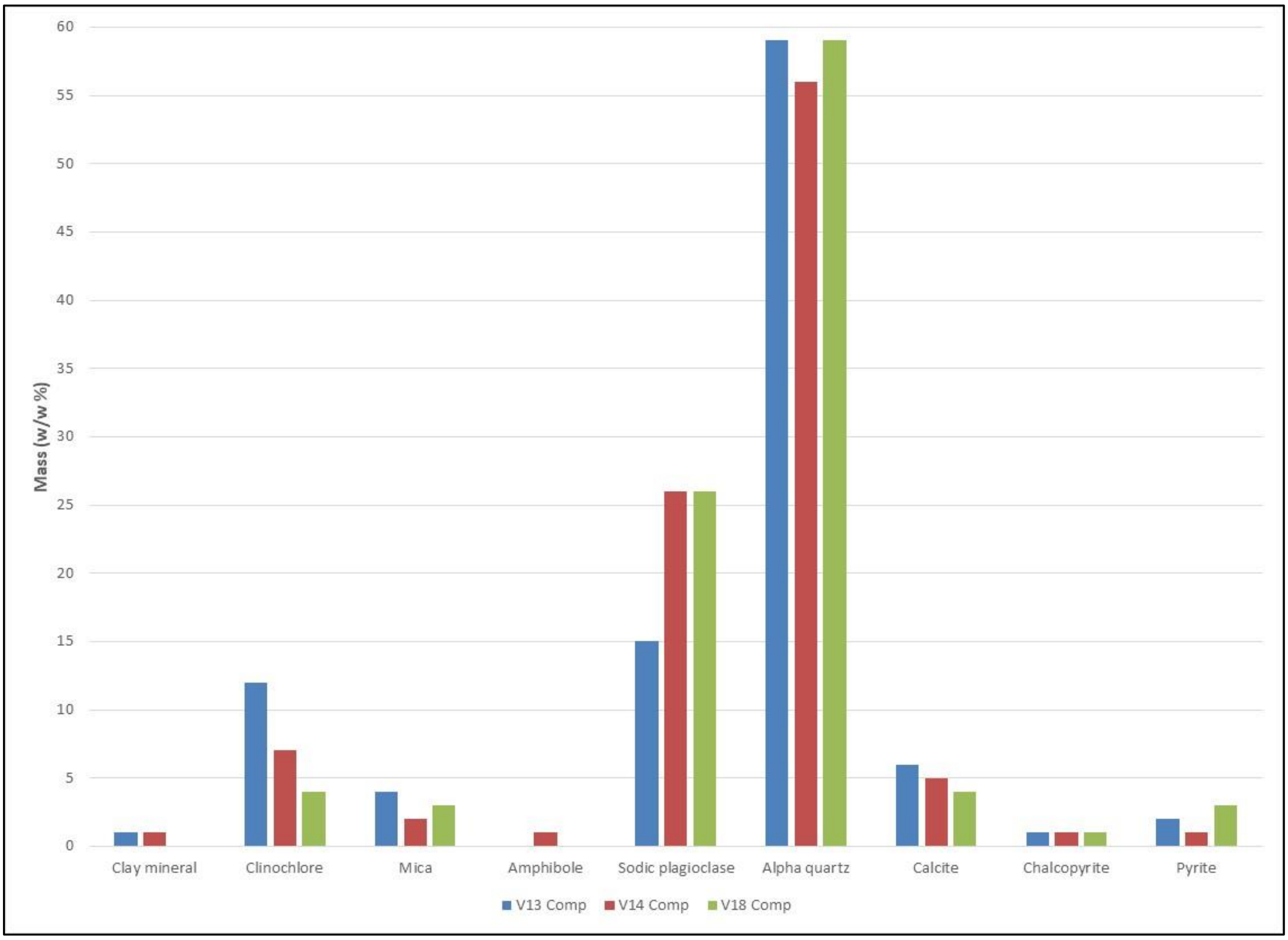


Minerals common to all samples are quartz, K-feldspar and mica. The vast majority of samples also have pyrite, chalcopyrite, clinochlore and sodic plagioclase (Na-feldspar). About half the samples have magnetite with proportions ranging from 2% to 20%.

The mineral mass deportments for the Alice XRD analysis results are shown in Figure 13.4.

Alice is geologically different to Productora and is dominated by quartz and sodic plagioclase. Alice also contains more calcite and, as with Productora, the only two identified sulphides are chalcopyrite and pyrite. The compositions are relatively consistent given that there are only three samples.

Figure 13.4 : XRD Analysis of Alice Flotation Feed Samples



Note: No XRD studies were conducted on Cortadera samples this time.

13.2.2 QEMScan

QEMScan is a more precise method for determining mineralogical composition than XRD. The main benefits compared to XRD are that QEMScan always provides a mineral allocation when conducting measurements and QEMScan provides information about each particle's composition.

The data gathered by QEMScan is grouped into mineral phases and connected mineral phases are arranged into particles. It is then possible to analyse properties of the particles in the sample, such as the liberation of one mineral from all the other minerals, or the sizes of the mineral phases and the particles.

13.2.2.1 Modal Mineralogy

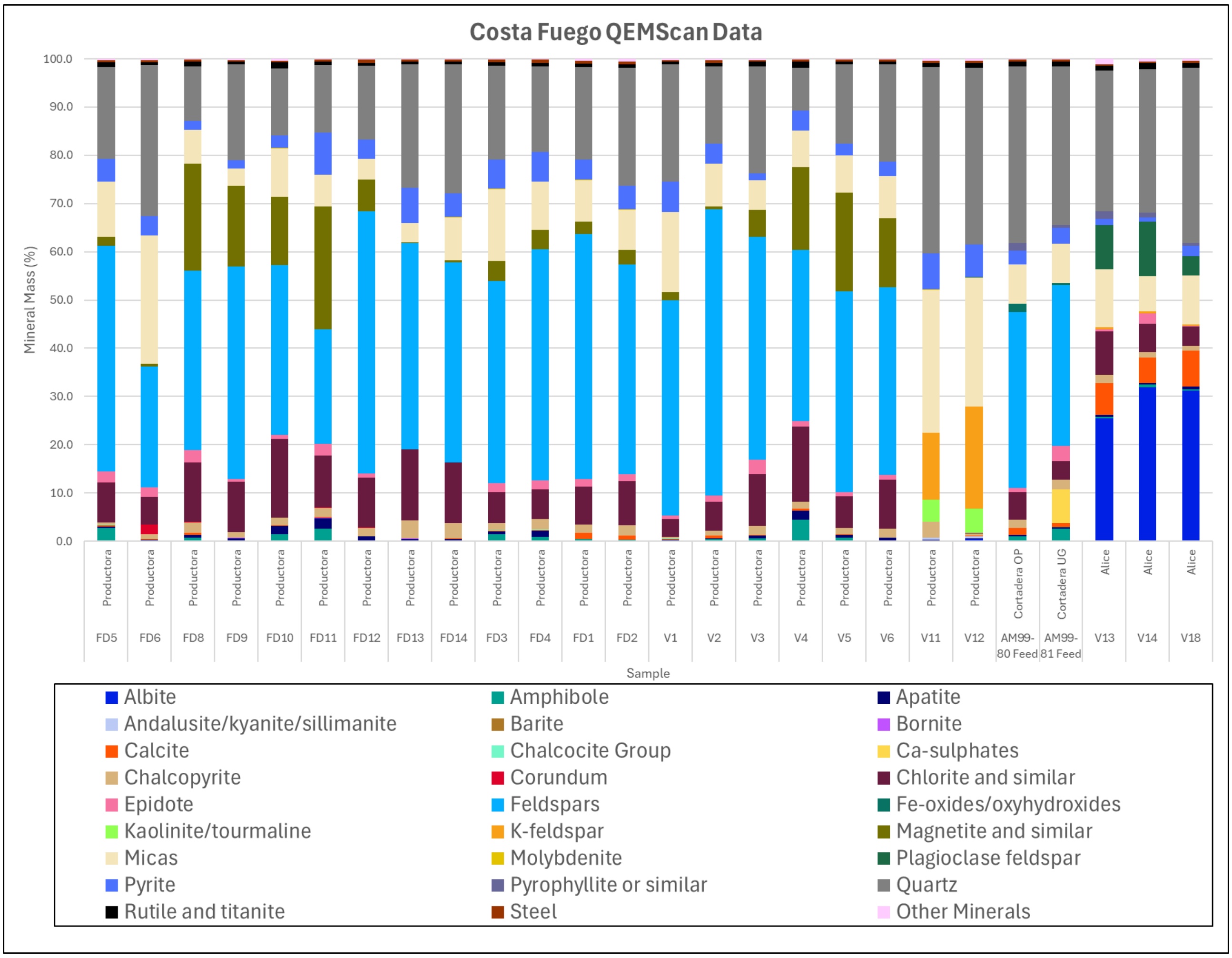
The simplest QEMScan output for a sample is the modal mineralogy, the mass percentages of each mineral present.

The following samples were analysed by QEMScan for each deposit and their modal mineralogy are compared in Figure 13.5:

- Productora: FD-1 to FD-14 (excluding FD-7), V-1 to V-6, V-11 to V-12 (21 samples)
- Alice: V-13, V-14 and V-18 (3 samples)
- Cortadera OP: Cortadera OP Master Composite flotation test fractions
- Cortadera UG: Cortadera UG Master Composite flotation test fractions.

Note that while the Cortadera feed analyses are shown, these feeds were not measured directly but are aggregated from analyses that were performed on flotation concentrates and tailings. Analysis of mineral behaviour across a flotation test greatly improves understanding of why particles and values are recovered or lost.

Figure 13.5 : Comparison of QEMScan Modal Mineralogies for Productora, Alice and Cortadera Samples



The only minerals present in similar magnitudes across all four deposits are pyrite, chalcopyrite, quartz and micas. Chalcopyrite is confirmed as the only significant carrier of Cu in the Costa Fuego deposits and it can be concluded that the properties of chalcopyrite determine copper grade and recovery behaviour for the Project.

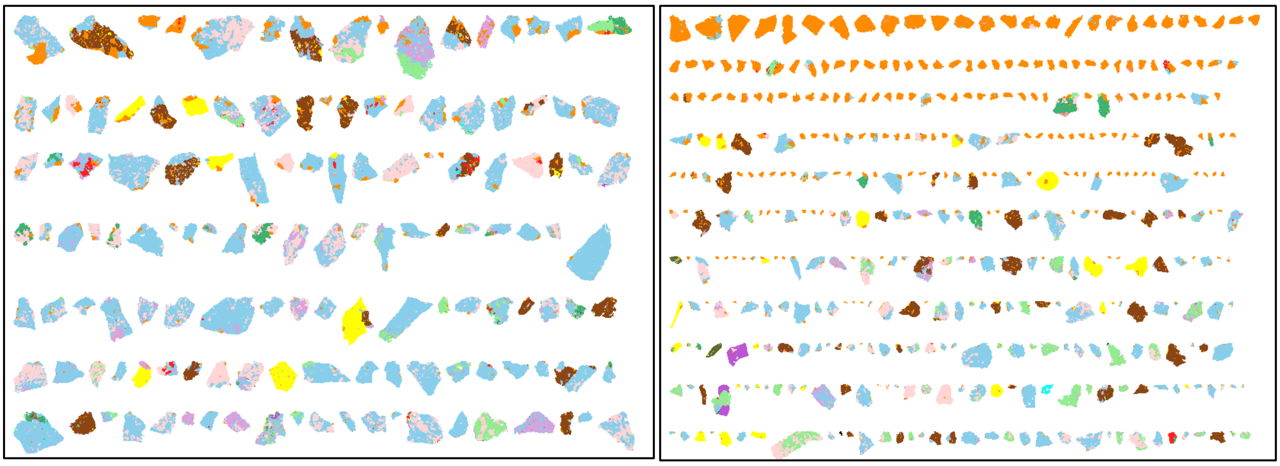
An unusual aspect of the Cortadera UG samples is that the mineral anhydrite ($CaSO_4$, which is chemically the same as gypsum and only lacking water of crystallisation) is consistently present in these samples at an average of about ten weight percent.

13.2.2.2 Chalcopyrite Liberation - Productora

The liberation of chalcopyrite at different size fractions for Productora sample FD1 is demonstrated in Figure 13.6. The QEMScan images compare the +125 µm fraction, where only a small proportion of the chalcopyrite (orange) is in liberated particles, to the -125 +38 µm fraction, where most of the chalcopyrite is in liberated particles.

These images support the test results which consistently show increased copper recovery as the grind P_{80} is reduced across the range 180 µm to 75 µm.

Figure 13.6 : FD1 Particles Containing Chalcopyrite: +125 µm Fraction Left, -125 +38 µm Right



13.2.2.3 Lost Copper - Cortadera OP Rougher Tails

A rougher tailings sample from the definitive open circuit flotation test on the Cortadera OP master composite was submitted for QEMScan analysis (test AM99-80).

The distribution of lost Cu amongst liberation and size classes in the rougher tails is shown in Table 13.2.

Table 13.2 : Liberation of Copper Minerals in AM99-80 Rougher Tail

Liberation Classes	Combined Cu Minerals (mass% in fraction)			
	+53 µm	-53 µm/+25 µm	-25 µm	Combined
Well liberated	0.0	0.9	15.8	19.5
High grade middlings	0.0	0.1	2.8	3.3
Medium grade middlings	3.1	2.3	3.0	8.4
Low grade middlings	17.2	3.7	4.0	24.2
Locked	38.5	6.8	1.9	44.5
Total	58.7	13.8	27.5	100.0

The table shows:

- 59% of the lost Cu is in the +53 µm fraction. Locked and low-grade chalcopyrite in the +53 µm fraction (see at right) is responsible for the majority of the losses and the orange chalcopyrite grains have an average size of only 14 µm. The lost chalcopyrite in coarse particles is unavailable for recovery as it would require the feed to be ground to an extremely fine and uneconomical size.
- Approximately 27.5% of the lost Cu is in the -25 µm fraction with ~15.8% in the well liberated class (see at left). This lost Cu may be available for recovery. Typically, fine liberated values lost to tailings can be recovered with additional collector and additional flotation time.
- The size range -53 to+25 µm is ideal for flotation and the losses in this range are low.

13.2.2.4 Pyrite Flotation - Copper

An opportunity exists to include a pyrite flotation circuit in the Costa Fuego flowsheet and this is described in Section 24. This circuit would recover gold and other elements into a pyrite concentrate that could potentially be sold to market or it could be treated on-site to recover values. Pyrite flotation is non-selective in that it recovers all remaining sulphides and it also recovers copper sulphide minerals that were not recovered in the Cu/Mo flotation circuit, or were rejected in cleaning, either due to lack of liberation from pyrite or gangue or because it was ultra-fine and slow floating.

QEMScan analysis was completed on pyrite concentrates generated from both Cortadera OP and Cortadera UG samples. QEMScan showed there is a significant proportion of chalcocite in the Cortadera OP pyrite concentrate. Tests of small amounts of a less selective Cu collector have failed to selectively recover Cu minerals without also recovering a large amount of pyrite. There are also fine chalcopyrite phases encapsulated within pyrite grains which are undesirable to recover in a Cu flotation stage. These copper minerals, which are effectively lost from the existing Cu flotation circuit, can be recovered in a non-selective pyrite flotation stage. Copper sulphides associated with silicates will also be recovered, provided some sulphide mineral is exposed at the particle surface. However, fine copper minerals locked within silicate grains are unlikely to be recovered in either a copper concentrate or a pyrite concentrate.

The liberation table for the Cu minerals in the Cortadera OP pyrite concentrate is in Table 13.22. The table has been calculated from the reported data so that the first two columns add to 100%. Copper recovery to the correct location, the Cu/Mo concentrate, was 88% and 4.5% of the copper reported to the pyrite concentrate. Almost 8% of the Cu remained un-floated and reported to pyrite flotation tails.

Table 13.3 : Liberation of Copper Minerals in AM99-80 Cortadera OP Pyrite Concentrate

Liberation Classes	Size		
	+25 µm	-25 µm	Combined
	Combined Cu minerals (mass% in fraction)		
Well liberated	14.9	23.7	38.6
High grade middlings	4.7	4.3	8.92
Medium grade middlings	10.8	3.4	14.3
Low grade middlings	21.6	2.8	24.3
Locked	12.6	1.3	13.9
Total	64.6	35.4	100.0

The table shows that about 50% of the copper in the pyrite concentrate has good liberation and may be able to be recovered in the copper concentrate with either additional collector or residence time in the scavenger flotation stage. Note, the +25 µm material will typically float better than the -25 µm material. While there may be an opportunity to improve primary Cu recovery, it is also a possibility that the lost Cu is difficult to float using RTD2086 and will continue to report to pyrite concentrate under most circumstances.

For the medium grade and locked materials, additional regrinding is required to liberate values from waste before flotation. It may be more technically and economically feasible to recover this copper directly from a pyrite concentrate, rather than attempt to regrind to fine sizes to recover it to Cu concentrate.

13.2.2.5 Concentrate Contamination

The main contaminants in the Cu/Mo concentrate are silicates and pyrite. The open pit Cu concentrate (Test AM099 80) is 85% chalcopyrite with 3.5% pyrite and about 8% silicates. The concentrate grade was 27.4% Cu, 30% S and 9.5% SiO_2. Similarly, the Cortadera UG Cu/Mo concentrate (Test AM099 81) was 27.6% Cu, 30.9% S and 7.7% SiO_2. Both are relatively clean concentrates and do not require further upgrading.

If flotation grade improvement had to be performed, then froth washing is one of the most effective methods. Much of the pyrite and silicates in the concentrate is liberated and is present through entrainment within the water accompanying the froth. Typically, froth washing can raise the copper grade by removing free silica and pyrite, but it does this with minimal to no loss of liberated copper. In the Cortadera concentrates, both pyrite and silicate liberation values are in the range 60 to 70% This means that it will be possible to improve concentrate grade with froth washing. Froth washing is difficult to test in the laboratory but has previously been confirmed by Wood at pilot scale on a benchmark copper flotation project where pyrite was a significant concentrate contaminant) and is also standard plant practice when cleaning or recleaning in column, Jameson or similar cells.

13.2.2.6 Other Mineralogical Findings

Significant factors that are apparent from the mineralogical analyses are that:

- The majority of the Mo (~90%) in feed and copper concentrate is fully liberated and there is no liberated Mo in tailings.
- Cleaner scavenger tailings are very fine and the main constituents have a high degree of liberation.

- All species in the rougher tailings have poor liberation.

Specific mineralogical question can be answered by referencing the individual QEMScan reports.

13.3 Sulphide Ore Testwork

This section summarises the results of metallurgical testwork programs completed on sulphide samples from the Costa Fuego Project. The results confirmed the technical and economic viability for flotation of sulphide materials and provided input for pre-feasibility level design.

13.3.1 Comminution Testwork for Sulphide Process Plant

Comminution testwork has been completed on samples from all deposits present in the Costa Fuego Project. The comminution testwork has provided the design basis for the concentrator's crushing and grinding circuits which will process ores from all deposits included in the PFS. Testwork was also conducted to determine the variability of comminution parameters within the deposits to ensure adequate flexibility and catch-up capability is included in the design.

Figure 13.7 : Samples used for Comminution testwork at Cortadera (Open Pit and Block Cave) - plan view (above) and long-section view (below)

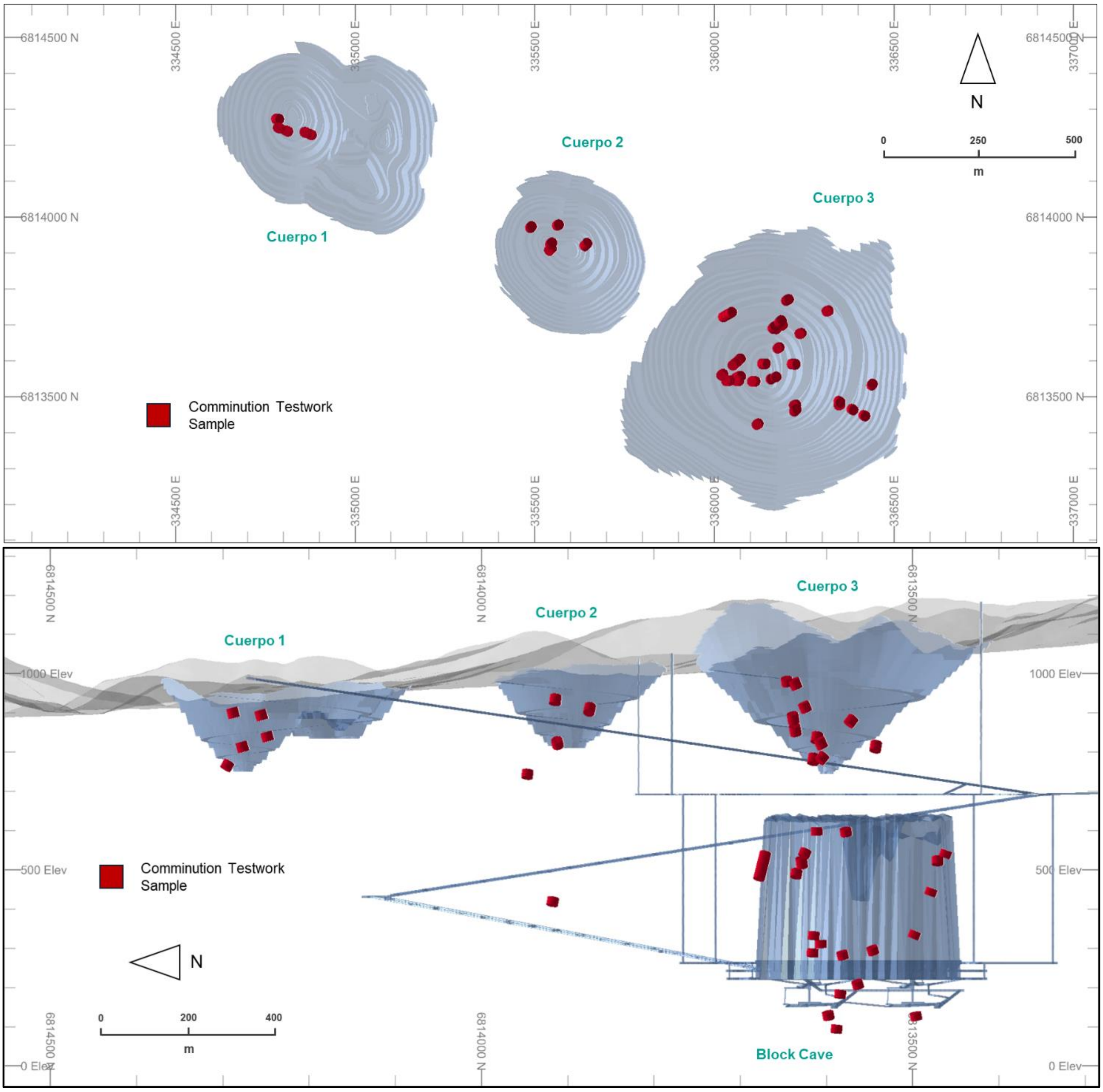
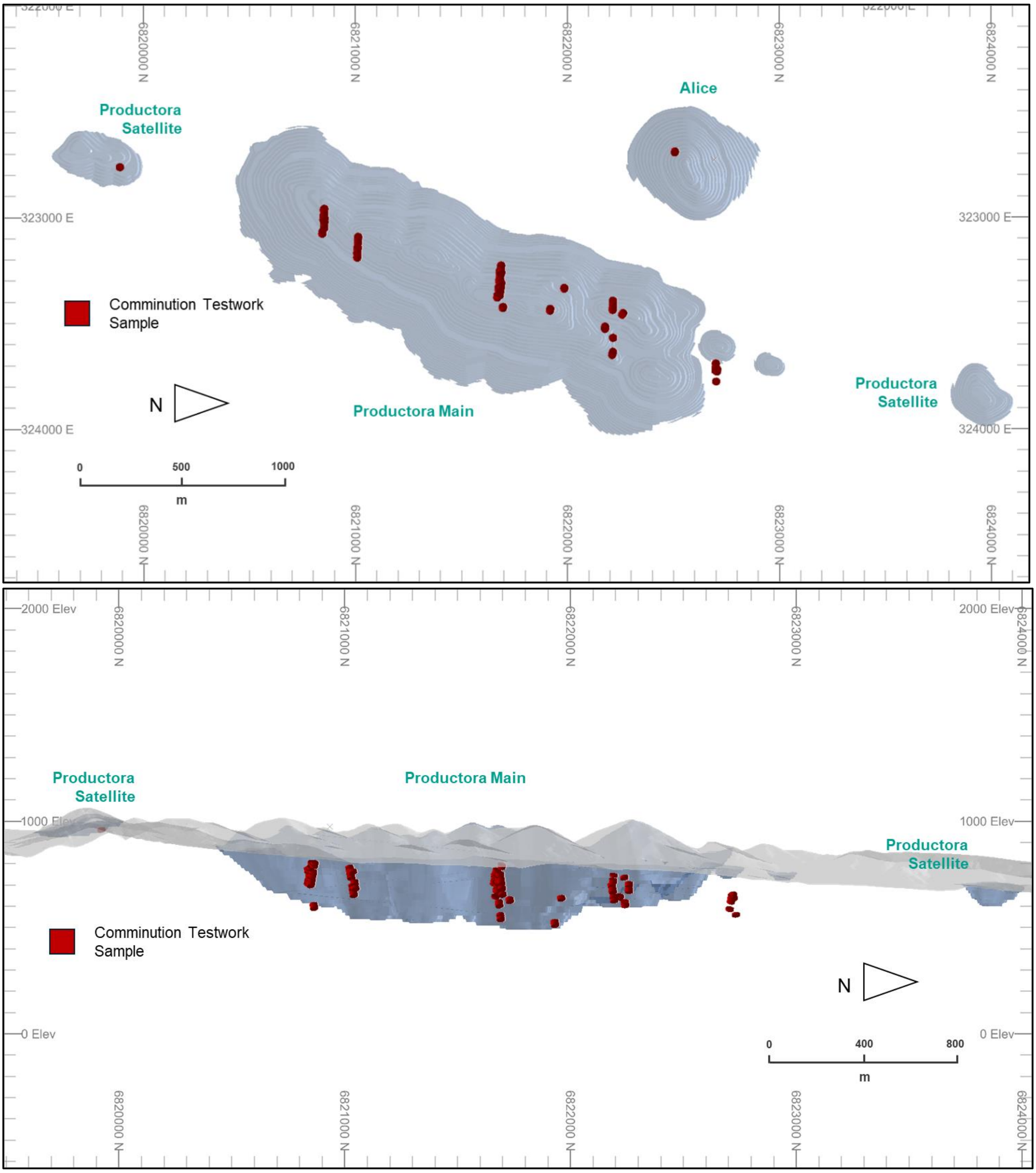


Figure 13.8 : Samples used for Comminution testwork at Productora and Alice - plan view (above) and long-section view (below)



The tests conducted provided measurements of Unconfined Compressive Strength (UCS), Bond Crushing Work Index (CWi), Bond Rod Mill Work Index (RWi), Bond Ball Mill Work Index (BWi), Abrasion Index (Ai), Drop Weight Index (DWi) parameters and SAG Mill Comminution (SMC) parameters.

13.3.1.1 Summary of Testwork Results

The results of all available comminution tests have been compiled to provide a comparison of the two main deposits (Productora, Cortadera OP and Cortadera UG) and the smaller Alice deposit. The comparative comminution properties are summarised in Table 1.3. Note that only samples designated as 'fresh rock' have been included.

Table 13.4: Summary of Average Comminution Properties for Costa Fuego Project Deposits

Item	Productora	Alice	Cortadera OP	Cortadera UG	DMCC Design[1]
Samples	22	2	20	21	-
DWi (kWh/m^3)	7.6	7.0	6.6	8.7	7.51
BWi (kWh/t)	18.5	17.4	14.2	16.9	18.3
Ai	0.28	0.27	0.14	0.17	0.37

[1]Note that Daniel and Morrell Comminution Consulting (DMCC) conducted a comminution circuit design for Mintrex as a component of the Productora PFS in 2014. Consequently, the DMCC parameters all relate to Productora and are included for historical reference.

The data in Table 13.5 provides guidance as to the variability of the comminution properties within each deposit.

Table 13.5 : Detailed Comminution Testwork Properties for Costa Fuego Project Deposits

Item		Productora	Alice	Cortadera OP	Cortadera UG
Samples	N	22	3	20	21
DWi (kWh/m3)	Max	9.6	7.5	9.0	10.7
	80th Percentile	8.4	7.3	7.9	9.4
	Average	7.6	7.0	6.6	8.7
	Minimum	4.6	6.5	3.6	7.4
Comment: Costa Fuego deposits have high competence (Drop Weight Index or DWi), with the highest being Cortadera UG. Note that the relative differences between DWi values are closely correlated to the respective SAG mill specific energy requirements. Therefore, Cortadera UG would have the highest specific energy requirement, and this gives it the lowest SAG mill throughput rate.					
Item		**Productora**	**Alice**	**Cortadera OP**	**Cortadera UG**
Mia (kWh/t)	Max	24.3	21.6	24.2	26.7
	Average	21.5	20.5	19.3	23.3
	Minimum	14.5	19.4	12.3	20.9
Comment: The Morrell Mia factor is used in equations that predict SAG mill, AG mill or rod mill energy requirements from the mill feed size down to about 750 µm. It is closely related to the DWi and follows a similar pattern. All Costa Fuego deposits have high Mia values, with the highest being Cortadera UG.					
Item		**Productora**	**Alice**	**Cortadera OP**	**Cortadera UG**
Mic (kWh/t)	Max	9.9	8.4	9.7	11.1
	Average	8.4	7.9	7.4	9.4
	Minimum	5.2	7.4	4.2	8.2
Comment: The Morell Mic factor is used in equations that predict conventional crushing energy requirements such as cone, jaw and gyratory crushers. Like Mia, it is closely related to the DWi and follows a similar pattern.					
Item		**Productora**	**Alice**	**Cortadera OP**	**Cortadera UG**
BWi (kWh/t)	Max	23.4	18.9	15.6	20.7
	80th Percentile	20.7	N/A	14.9	18.3
	Average	18.5	17.4	14.2	16.9
	Minimum	14.8	15.5	12.7	13.8
Comment: BWi is used in equations that predict ball mill grinding energy from about 2 mm top size down to about 50 µm (using the Bond method) and Mib is used across the range from 0.75 mm down to about 50 µm in the Morrell Method. The Mib, the Morrell ball milling index, is related to BWi as it is an alternative interpretation of the same Bond BWi Test data. The grinding work indices are consistent with each other but follow a different pattern to the competence (DWi) related factors. Where the highest competence and coarse breakage parameters (DWI, Mia, Mic and Mih) are to be found in The Cortadera UG deposit, the hardest grindability values (BWi and Mib) exist in the Productora deposit.					
Item		**Productora**	**Alice**	**Cortadera OP**	**Cortadera UG**
Ai	Max	0.43	0.33	0.30	0.28
	80th Percentile	0.37		0.17	0.19
	Average	0.28	0.27	0.14	0.17
	Minimum	0.11	0.20	0.04	0.06

Table 13.5 : Detailed Comminution Testwork Properties for Costa Fuego Project Deposits

Comment: Ai (Bond abrasion index) is used in equations that predict steel grinding media consumption together with consumption of steel liners in mills and crushers. The measure is related to how quickly steel is worn away when subjected to tumbling and sliding action from the test rock. Ai is a key factor in operating cost calculations and will influence the annual utilisation of crushers and mills. All the average Ai results are low to moderate values, meaning that abrasion related costs will also be relatively low. The Productora and Alice deposits are twice as abrasive as the Cortadera deposits.					
Item		**Productora**	**Alice**	**Cortadera OP**	**Cortadera UG**
SG	Max	3.01	2.70	3.13	3.11
	Average	2.69	2.69	2.68	2.79
	Minimum	2.58	2.66	2.52	2.66
The highest SG is for the Cortadera underground mineralisation, and this is likely due to the elevated presence of iron and copper sulphides (SG = 4.0 to 5.2) and anhydrite (SG = 3.0).					

The two main deposits (Productora and Cortadera) have distinct comminution properties, and these differences have been recognised when developing plant throughput estimation methods and selecting comminution equipment for this PFS.

13.3.1.2 Comminution Testing Implications

The comminution property differences are significant enough to warrant the calculation of comminution circuit design factors and limitations for each deposit individually. This is necessary to ensure that the design is flexible enough to process feed at close to design rates across the life of the Project as the feed blend changes.

The average comminution results have been used to predict specific power consumption values for SABC (SAG, ball mill and pebble crushing circuit) operation for the Costa Fuego Project deposits. These are summarised in Table 13.6.

Table 13.6 : Design (80th Percentile) Specific Energy by Equipment for Costa Fuego Project Deposits

Specific Energy (kWh/t)	Productora	Alice	Cortadera OP	Cortadera UG
Primary Crusher	0.1	0.1	0.1	0.2
SAG Mill	8.7	8.4	9.4	10.1
Pebble Crusher	0.4	0.4	0.4	0.5
Ball Mill	14.9	12.4	9.1	11.8
Total	24.1	21.3	19.0	22.6

The average ore properties are also used to predict milling rates and these are summarised in Table 13.7. The average properties are also used for power consumption OPEX calculations.

Table 13.7 : Nominal (average) Specific Energy by Equipment for Costa Fuego Project Deposits

Specific Energy (kWh/t)	Productora	Alice	Cortadera OP	Cortadera UG
Primary Crusher	0.1	0.1	0.1	0.2
SAG Mill	8.0	8.1	7.8	9.4

Table 13.7 : Nominal (average) Specific Energy by Equipment for Costa Fuego Project Deposits

Specific Energy (kWh/t)	Productora	Alice	Cortadera OP	Cortadera UG
Pebble Crusher	0.4	0.4	0.3	0.4
Ball Mill	12.9	11.2	8.3	10.5
Total	21.4	19.8	16.5	20.5

The specific energy requirements will be applied in the plant design and this work is summarised in Section 17 of this Report.

Although the overall power requirements are similar for Productora and Cortadera UG, the operational power split between the two mills would be different in each case. When milling Productora ore, only 36% of the comminution power will be consumed in SAG milling. Depending on the mill sizes selected, a bottleneck in the ball mill circuit would be expected to occur well before a SAG mill power limit was reached. For Cortadera UG ore, SAG mill power is 45% of operational power. Again, depending on mill size selections, Cortadera UG ore is much more likely to result in a SAG mill bottleneck than Productora ore.

13.3.1.3 Geometallurgical Prediction of Comminution Properties

With the differences in comminution properties between the Costa Fuego Project deposits, variable plant throughputs have been calculated in cases where sufficient data is available.

All available comminution data was reviewed in 2024, with a focus on spatial location of samples and opportunity to domain based on known rock properties. BWi and DWi data was loaded into IoGAS to check for relationships between the comminution test result and the head assays on those samples using the least squares algorithm to determine the regression equations.

13.3.1.3.1 Productora/Alice

A total of 22 comminution tests have been undertaken at the Productora project, with good coverage over the mining inventory, and within each of the metallurgical domains. Three tests have been completed at Alice, all within the primary Alice porphyry resource.

Univariate and multivariate regressions were trialled for each of BWi and DWi at Productora, with the strongest statistical correlations (R^2) selected for calculation purposes, with support from knowledge of the underlying mineralogy. For Alice, average values were used as regressions could not be created with the limited data set. Results are detailed in Table 13.8 below, as well as comments on the use of each variable in the regression.

Table 13.8 : Productora/Alice DWi and BWi Calculations

Productora	
DWi (kWh/m³)	DWi = 10((0.2669 * LOG10(NA_PCT)) + (-0.06764 * LOG10(CA_PCT)) + 0.872) Where: NA_PCT = Sodium head assay (%) CA_PCT = Calcium head assay (%)
BWi (kWh/t)	BWi = (-5.2724 * NA_PCT) + 23.3575 Where: NA_PCT = Sodium head assay (%)
Alice	
DWi (kWh/m³)	DWi = 8.1
	Comment: Average value of three data points used. Not enough data to create regression.
BWi (kWh/t)	BWi = 17.4
	Comment: Average value of three data points used. Not enough data to create regression.

13.3.1.3.2 Cortadera

Comminution variability has been measured for the two groups of Cortadera samples, 21 from the underground mining area and 20 from the open pit mining area. Sampling is spread over multiple lithologies and alteration domains, providing good representativity across the mining inventory.

An attempt was made to determine correlations between all BWi/DWi and assay head results at Cortadera, but these were generally low making it difficult to construct a reliable regression model.

The Cortadera damage zone model (discussed in Section 14.1.1) has been used to differentiate between geometallurgical domains at Cortadera. This is due to noted variation in BWi, with high BWi samples trending towards the anhydrite node on a Ca-Fe-S ternary diagram which indicates that samples with a lower BWi have little to no anhydrite present. Most of these low-anhydrite samples sit within the damage zone model.

When considering only the samples outside of the damage zone, BWi correlation factors improve to above 0.5 for manganese (Mn), calcium (Ca), and cobalt (Co). Similarly, when only samples inside of the damage zone are considered, DWi correlation factors also improve, with values above 0.5 for a number of elements including Cu:S., chromium (Cr), Na, magnesium (Mg), and scandium (Sc).

For consistency, DWi also uses the damage zone domaining, returning good correlations for data both inside and outside of the constraint.

Results are detailed in Table 13.9 below, as well as comments on the use of each variable in the regression.

Table 13.9 : Cortadera DWi and BWi Calculations

Cortadera – Inside of Damage Zone	
DWi (kWh/m³)	DWi = 10^(-0.7723 * LOG10(FE_PCT)) + 1.225 Where: FE_PCT = Iron head assay (%)
BWi (kWh/t)	BWi = 10^(0.1203 * LOG10(CA_PCT)) + (1.0299 * LOG10(AL_PCT)) + 0.265 Where: CA_PCT = Calcium head assay (%) AL_PCT = Aluminium head assay (%)
Cortadera – Outside of Damage Zone	
DWi (kWh/m³)	DWi = (1.594 * K_PCT) + (-2.3272 * CU_PCT) + 8.2956 Where: K_PCT = Potassium head assay (%) CU_PCT = Copper head assay (%)
BWi (kWh/t)	BWi = (-8.7998 * CU:S) + (2.4674 * NA_PCT) + 12.9068 Where: CU:S = Ratio of Copper head assay (%) to Sulphur head assay (%) NA_PCT = Sodium head assay (%)

The equations provide predictions of BWi and DWi values for ore in each block in the mine plan. The parameters are used to determine the plant bottleneck (SAG mill or Ball mill) and the block bottleneck throughput rate.

By this method the mine plan aggregates its own quarterly throughput predictions based on both geology and on metallurgical test results.

13.3.2 Sulphide Flotation Testwork

13.3.2.1 Introduction

Sulphide flotation testwork has been completed on samples from all deposits across the Costa Fuego Project. These historical testwork programs have delivered the flotation schemes (including reagent schemes, grind sizes and flowsheets) that are required for each of the ore types and have provided the information required for flotation design in this PFS. Two San Antonio test samples have been assessed over the history of the Project.

The objectives of the flotation testwork were to establish optimal flotation conditions for recovery of copper, molybdenum and gold into saleable products, model variations in recovery with feed grade and provide flotation circuit design data. The program also generated samples of concentrate and tailings for settling, pumping and filtration testwork.

For the purposes of this PFS, the treatment of ore blends will be considered on a theoretical basis as limited blended flotation testing has been performed. More comprehensive blend testing will be conducted in preparation for Feasibility Study (FS).

The blend scenarios that would be used to drive any definitive blend testwork will be an output of mine planning in the form of an Ore Delivery Schedule. Ore delivery schedules have been developed for the PFS but no preliminary schedule was available at the time the flotation testwork was planned and performed. The development of any reasonable mine schedule is highly dependent upon the metallurgical response of the deposits and the methods by which those deposits are to be mined, stockpiled and processed. The testwork reported here provides the metallurgical context for the deposits and is a key input to mine planning, rather than a representing testwork aligned to a given mine plan.

Figure 13.9 : Samples used for Flotation Testwork at Cortadera (Open Pit and Block Cave) - Plan View (Above) and Long-section View (Below)

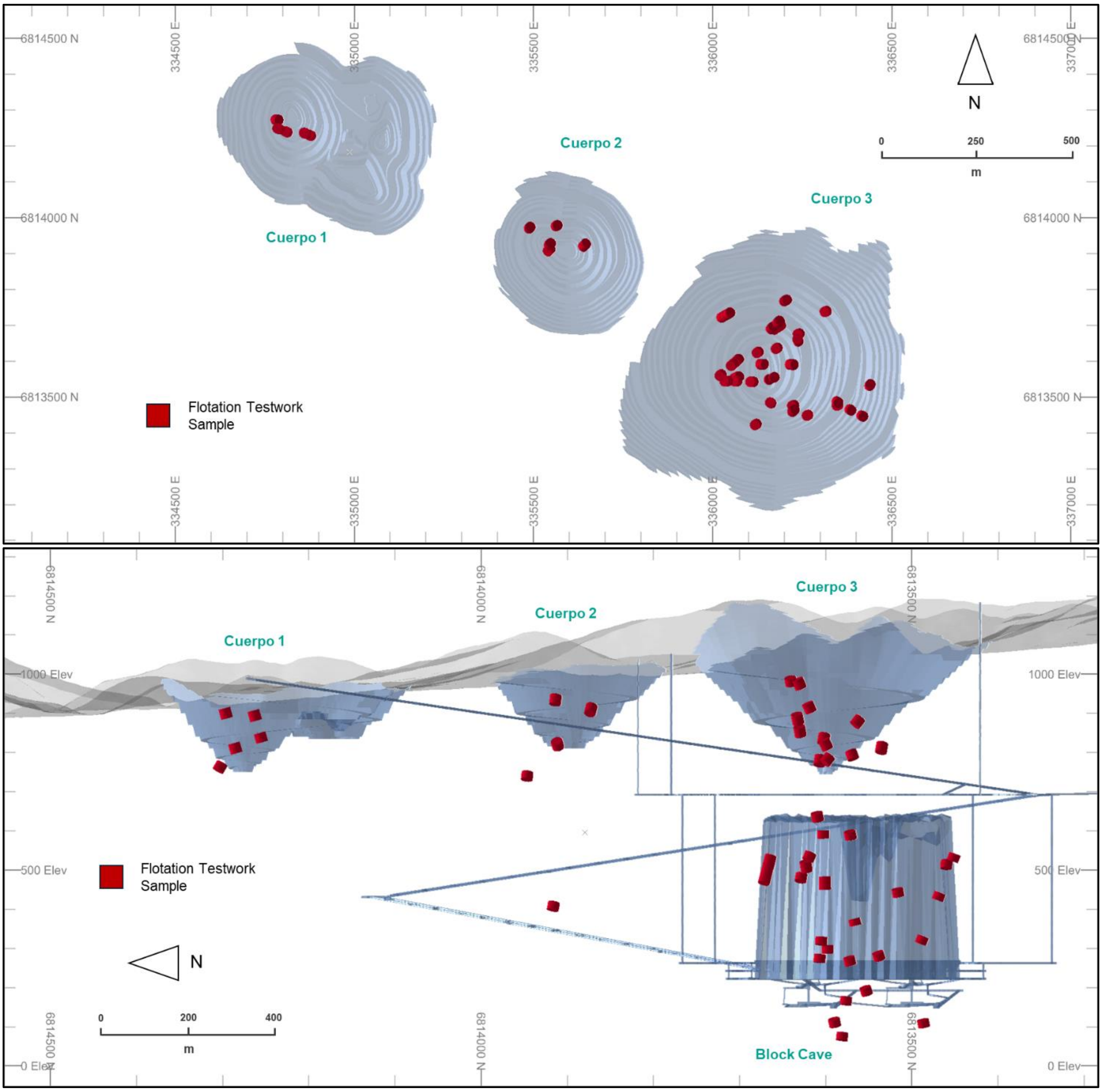
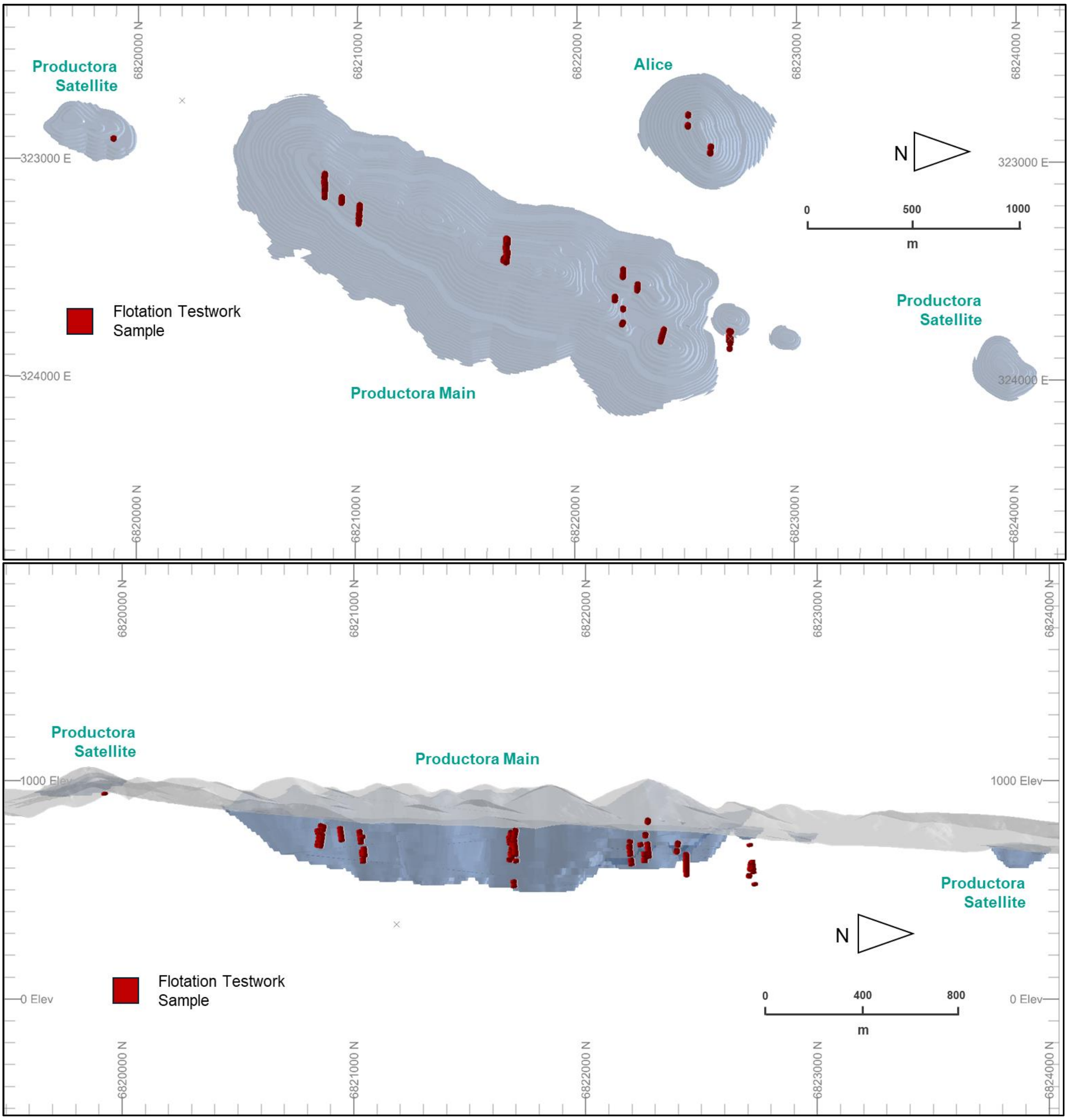


Figure 13.10 : Samples used for Flotation testwork at Productora and Alice - Plan View (Above) and Long-section View (Below)



13.3.2.2 Productora and Alice Deposits

13.3.2.2.1 2014-2015 – Productora/Alice Flotation - Work Program Mintrex & ALS A16082

In the A16082 program completed by ALS, flotation testwork was conducted on samples from the Productora and Alice deposits.

This testwork program showed that the RTD2086 collector (a xanthate ester) was extremely selective and highly suited for flotation in sea water. As RTD2086 is used at natural pH its use does not result in any problematic buffering (excess use of acid or alkaline to attain a pH target) that occurs in seawater when adjusting pH.

It was identified that samples from different zones of the deposit gave different flotation responses. Generally, transitional and oxide samples gave poorer, but not unexpected, performance when subjected to the standardised reagent scheme.

13.3.2.2.2 Molybdenum Concentrate Testwork

Within A16082 a testwork program was conducted to determine if a saleable molybdenum concentrate (~50% molybdenum grade) could be produced.

The most important sources of molybdenum in the Project are the Productora ores (average Mo head grade of 151 ppm, with a recovery of 79% to Cu/Mo concentrate and accounting for 67% of Project Mo) and the Cortadera UG ores (average head grade of 93 ppm Mo, also with a recovery of 79% accounting for 29% of project Mo). Cortadera OP contains only 30 ppm Mo and has low recovery of 64% and only contributes 3% of project Mo. Alice contributes less than 1% and San Antonio effectively has no contribution.

The A16082 test program was completed on a high-grade Mo sample (PRP0812). Initially a bulk Cu/Mo rougher concentrate was produced from the PRP0812 sample and 71% of the Mo was recovered in this concentrate. The recovery was relatively low and this can be attributed to the sample consisting of RC chips and also containing transition intervals. In addition, as the Cu-Mo rougher stage was conducted in open circuit the reported Mo recoveries to concentrate are conservative compared to what would be achievable in locked cycle.

This program included Mo rougher flotation and Mo cleaner work on the Cu/Mo bulk concentrate. In Mo rougher flotation, NaSH was used to depress Cu and RTD2086 was used as the collector. Kerosene was added to aid molybdenum recovery. Frother was only required sparingly in the Mo cleaner work.

Test MN1494 achieved 86.7% Mo stage recovery (with respect to the test feed, the Cu/Mo bulk concentrate) to produce a saleable grade Mo concentrate of 49.8% after only a single stage of molybdenum cleaning. This translated to a total Mo recovery of 61.7% (with respect to plant feed) as shown in the high-level Mo mass balance from this test in Table 13.10.

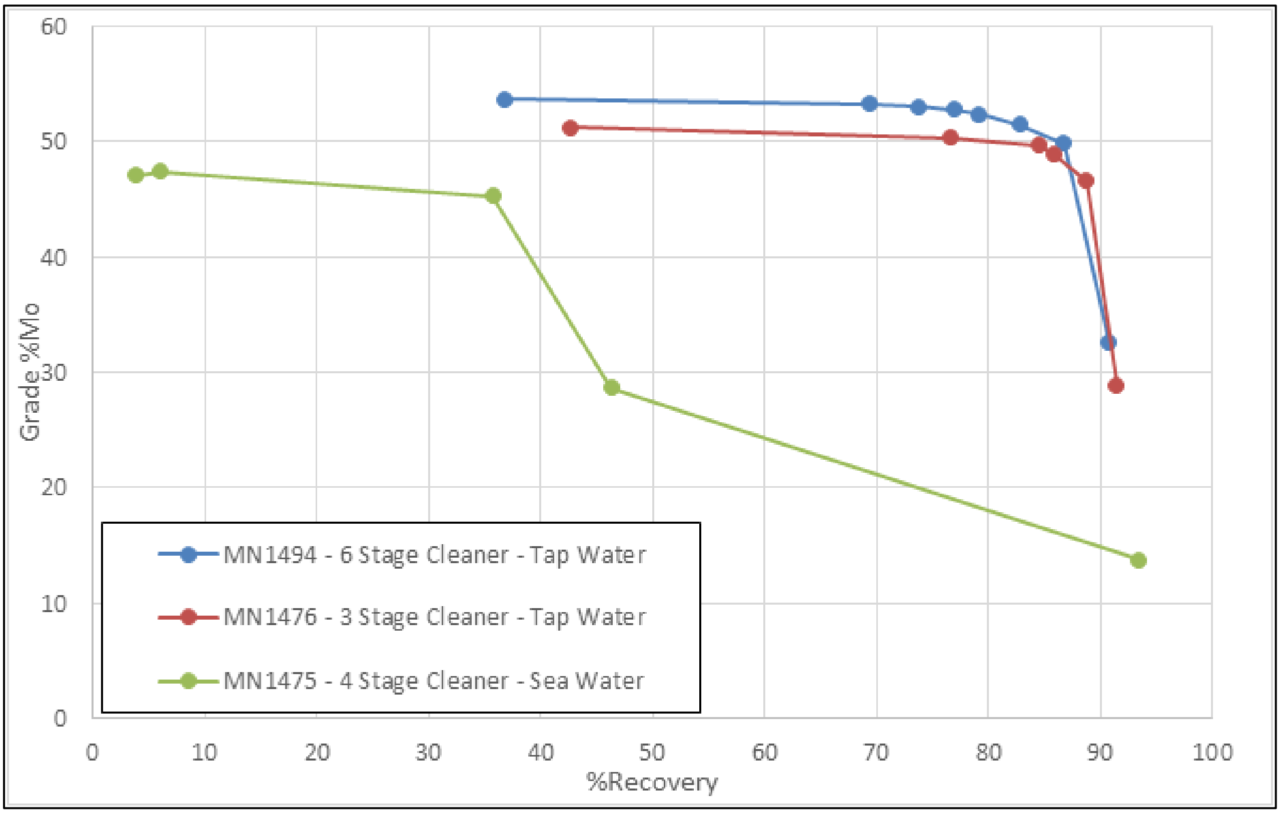
Table 13.10 : Molybdenum Mass Balance for Test MN1494 from sample PRP0812

Stream	Mo Grade (%)	Total Moly Recovery (w.r.t. Plant Feed)	Flotation Stage Recovery (w.r.t. Bulk Mo/Cu Rougher Concentrate)
Molybdenum 3rd Cleaner Concentrate	52.4%	56.3%	79.1%
Molybdenum 2nd Cleaner Concentrate	51.5%	58.9%	82.8%
Molybdenum 1st Cleaner Concentrate	49.8%	61.7%	86.7%
Molybdenum Rougher Concentrate	32.5% (325,000 ppm)	64.6%	90.7%
Bulk Mo/Cu Rougher Concentrate	1.87% (18,700 ppm)	71.2%	100%
Ore/Plant Feed	0.05% (500 ppm)	100%	

These molybdenum results are encouraging especially as the sample can be considered compromised as it consisted of RC drill chips and incorporated transition intervals, rather than fresh diamond drill core. The QP notes that reverse circulation chips do not provide adequately representative material for flotation testwork and that future molybdenum testwork must use diamond drill core. Additional testwork will be completed on core as part of the FS.

The initial molybdenum recovery tests were all conducted in sea water and the froth during these floats was very poor and unstable. The final tests used Perth tap water and a significant change was seen in the flotation performance, as shown in Figure 13.11.

Figure 13.11 : Molybdenum Cleaner Flotation Tests – Sea Water vs. Tap Water



This work confirmed that the use of sea water is not a viable option for the molybdenum circuit. A thickening/washing stage, using desalinated or low saline water, is required before attempting to separate molybdenum from the copper in the Cu/Mo concentrate. The washing stage is not perfect and results in the molybdenum rougher stage being conducted in a low-saline water environment, in the region of 1000 ppm TDS. Subsequent molybdenum cleaning stages will be conducted after dilution with desalinated (reverse osmosis or RO) water, meaning that the TDS level should be less than 500 ppm. Further testwork is required during FS to determine the upper acceptable TDS levels in the molybdenum roughing and cleaning stages.

Note that each of the Mo rougher flotation stages were conducted using nitrogen rather than air as this minimises (sometimes reduces by 50%) the consumption of NaSH. It is unclear if nitrogen is necessary throughout the Mo circuit or is only required for some of the Mo flotation stages. Again, this will be the subject of testwork during the FS.

13.3.2.2.3 Productora – Flotation Test Results

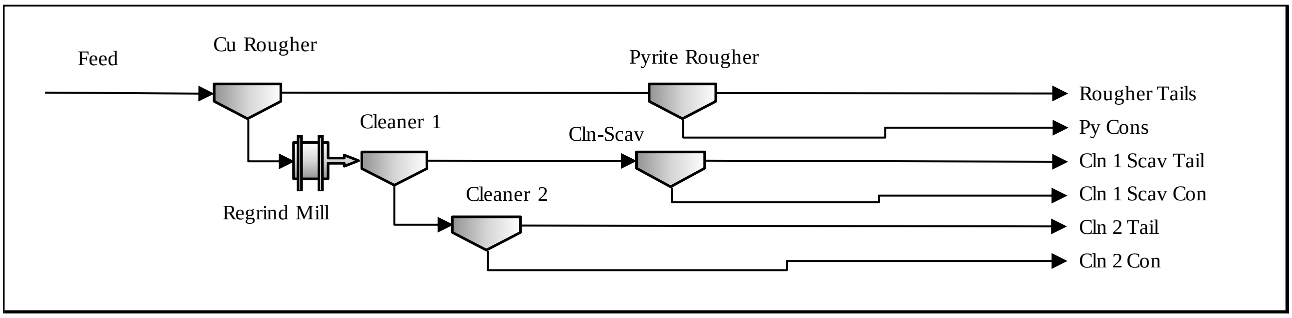
The standardised flowsheet and reagent scheme for Productora and Alice, developed for the Productora PFS, uses RTD2086 at a natural pH and a small frother addition. The flowsheet consists of a Cu/Mo rougher using RTD2086, followed by a pyrite rougher using PAX (potassium amyl xanthate). Cu/Mo rougher concentrate is reground to nominally 25 µm P_{80} and is cleaned followed by cleaner-scavenging to produce a throw-away cleaner scavenger tail. The cleaner concentrate is recleaned to produce the final Cu/Mo concentrate. The two middlings streams in the open circuit test are the cleaner scavenger concentrate and the recleaner tailing. In a locked-cycle test, the middling streams are recycled back to cleaner feed. In many tests the pyrite rougher was not included, and it is not incorporated in the concentrator design for this PFS.

The standardised scheme is presented in Table 13.11 and the Productora standard test flowsheet is shown in Figure 13.12.

Table 13.11 : Productora Standardised Flotation Scheme

Stage	Conditioning Time (minutes)	RTD208 Dosage (g/t)	PAX (g/t)	Flotation Time (minutes)
Cu Rougher	4	25	-	13
Pyrite Rougher	2	-	75	8
Regrind Cu Rougher Concentrate to P_{80} 25µm in ceramic bead mill	-	-	-	-
Cu Cleaner 1	2	10	-	11
Cu Cleaner-Scavenger	1	2	-	7
Cu Re-Cleaner (Cleaner 2)	1	5	-	12

Figure 13.12 : Productora Standard Flowsheet



Optimisation testwork underpinning this flowsheet was conducted on a series of samples but was mainly focused on the FD1 composite sample because of its larger available mass and the representative nature of the FD1 blend.

Once optimised conditions were being tested on the variability samples, it was clear that different zones of the deposit were giving different flotation outcomes. As expected, low recoveries were associated with samples identified as containing transitional sulphide (designated as containing greater than 10% acid soluble copper, ASCu) and oxide material (>30% ASCu).

Samples were grouped based on the weathering zone, and by location. Oxide samples were excluded from concentrator flowsheet development testwork and transition samples were considered but did not drive design.

Flotation testwork optimised grind size, flotation collector type, and flotation collector addition rates. It also explored regrinding, cleaning and scavenging stages to deliver concentrates with grades >25% Cu.

Optimised flotation schemes for selected samples were used to develop locked cycle test (LCT) flowsheets.

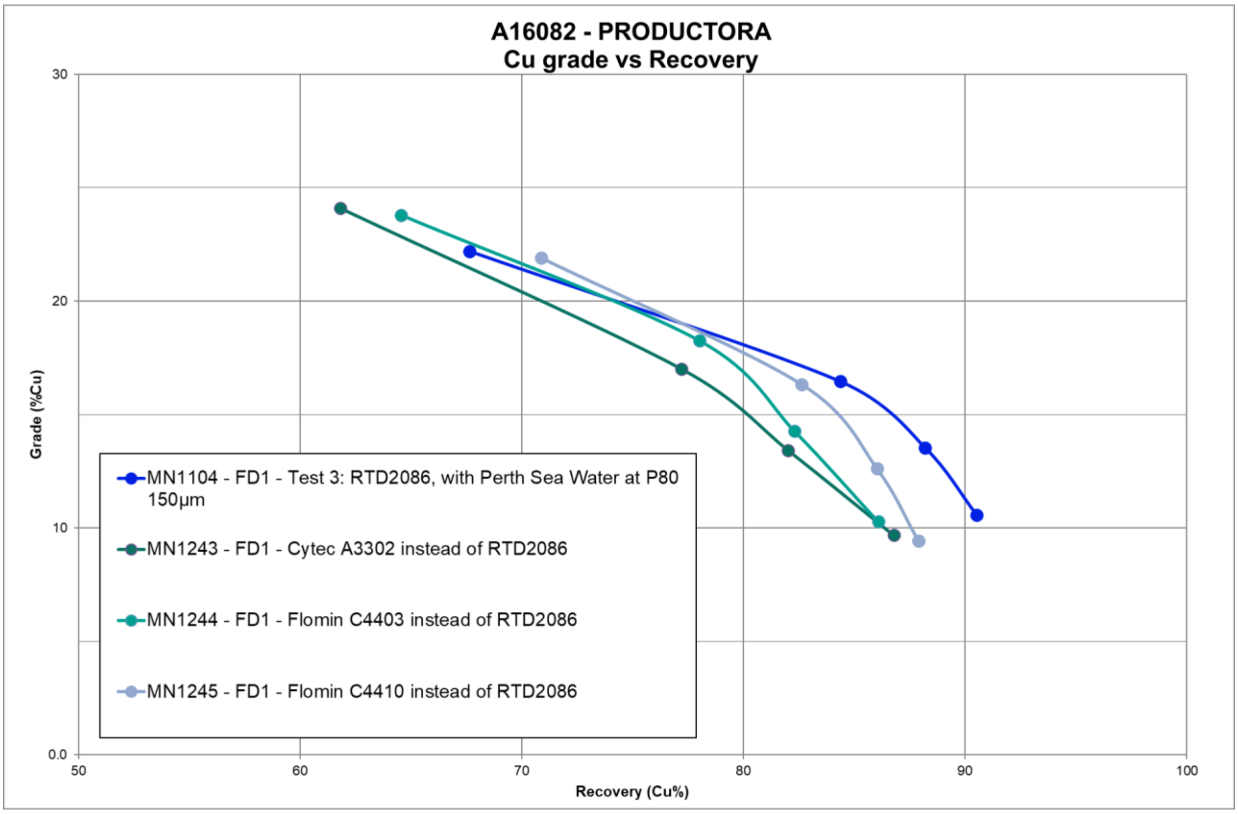
Primary grind sizes ranging from P_{80} 75 µm to 180 µm were tested on selected samples to ascertain the impact of grind P_{80} on rougher copper recovery. In general, the testwork demonstrated that rougher copper recovery increased as the grind P_{80} size decreased.

Copper recovery from Productora transitional samples was much lower than from fresh samples. Further testwork will be completed in the FS to identify the root cause and identify solutions.

13.3.2.2.4 Productora - Reagent Scheme (A16082)

As part of test program A16082, three collectors similar to RTD2086 were tested in rougher flotation to see if it was the appropriate collector for Productora ores. The Cu grade-recovery curves for these tests are shown in Figure 13.13.

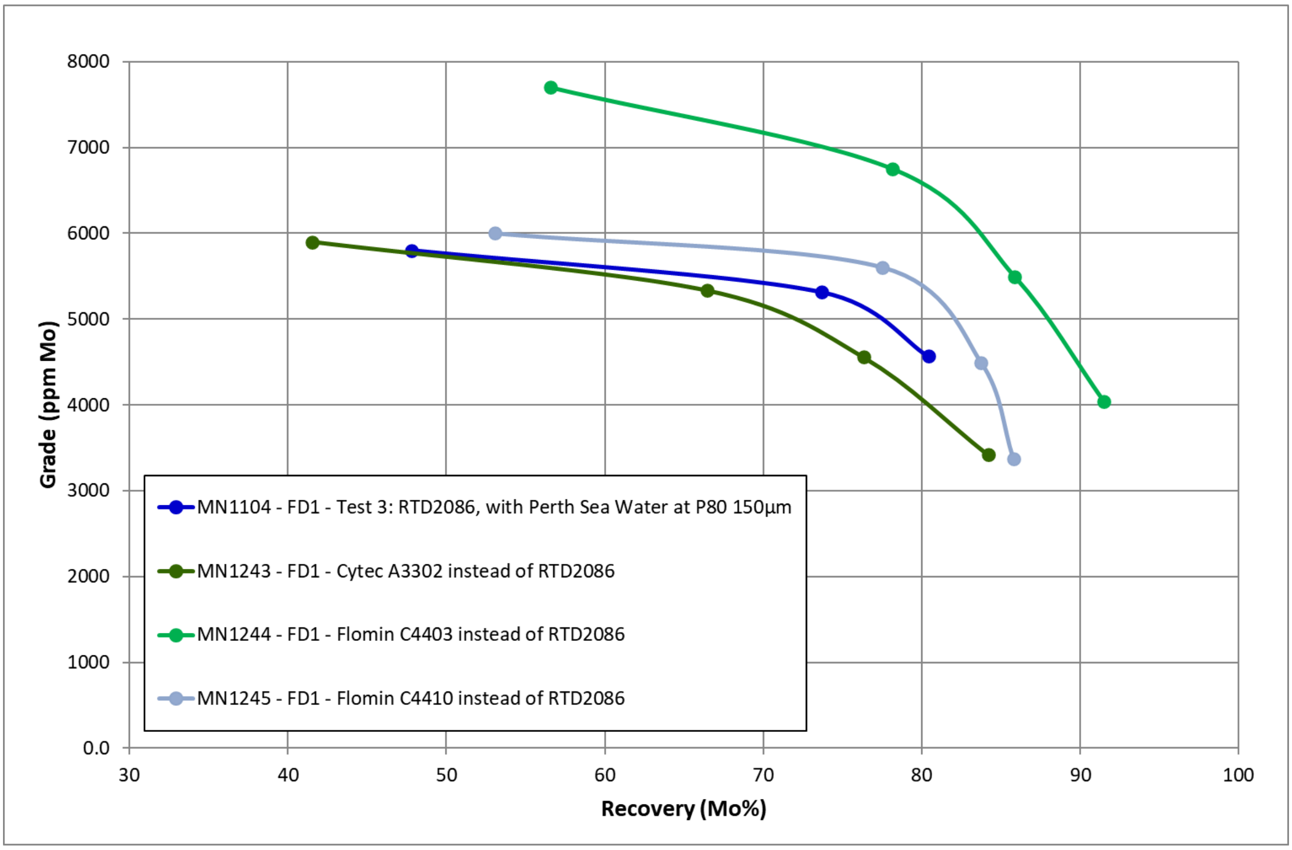
Figure 13.13 : Comparison of Xanthate Ester Reagents – Cu Recovery and Grade



At a fixed rougher Cu concentrate grade of around 10%, RTD2086 has the highest Cu recovery compared to the other three reagents.

The Mo recovery and grade performance is shown in Figure 13.14.

Figure 13.14 : Comparison of Xanthate Ester Reagents – Mo Recovery and Grade



At the same rougher concentrate molybdenum grade of 4000 ppm, collector C4403 had the highest Mo recovery by about 7%. However, the superior performance of RTD2086 for Cu recovery is more economically relevant than the Mo recovery difference. Therefore, RTD2086 is the selected Cu/Mo collector for Productora.

In summary, the use of RTD2086 alone as a collector at natural pH delivers excellent copper recovery and acceptably high recovery of molybdenum.

13.3.2.2.5 Alice Flotation Testwork Results (A10682)

As part of the A10682 program, three Alice samples underwent flotation testwork.

All Alice samples tested were classed as Fresh and they have been tested using the standard Productora flowsheet.

Alice deposit rougher flotation Cu recoveries are all very high with all but one test exceeding 90%. However, as Alice has a relatively small ore tonnage and will be processed with blends of ores from other deposits, it is not critical to characterise all metallurgical aspects of Alice to the same degree as Productora.

13.3.2.3 Cortadera and San Antonio Deposits

13.3.2.3.1 2020 – Cortadera and San Antonio Comminution and Flotation - Work Program Auralia AM063

A flotation testwork program was completed in 2020 at the Auralia Metallurgy Laboratory in Perth (Program AM063). The samples tested included:

- CRP0011D and CRP0013D (Cortadera UG sulphide samples)
- SAG-01 (San Antonio sample).

The Cortadera samples responded well to the Productora flotation scheme, with the four initial tests delivering in the vicinity of 25% copper concentrates at 90% copper recovery or higher.

13.3.2.3.2 2020 – Cortadera Variability Testwork - Work Program Auralia AM099

Auralia completed a variability testwork program (AM099) on samples from both the Cortadera Open Pit (OP) and Cortadera Underground (UG) mining areas. This program culminated in the development of separate flowsheets for treating OP and UG ore.

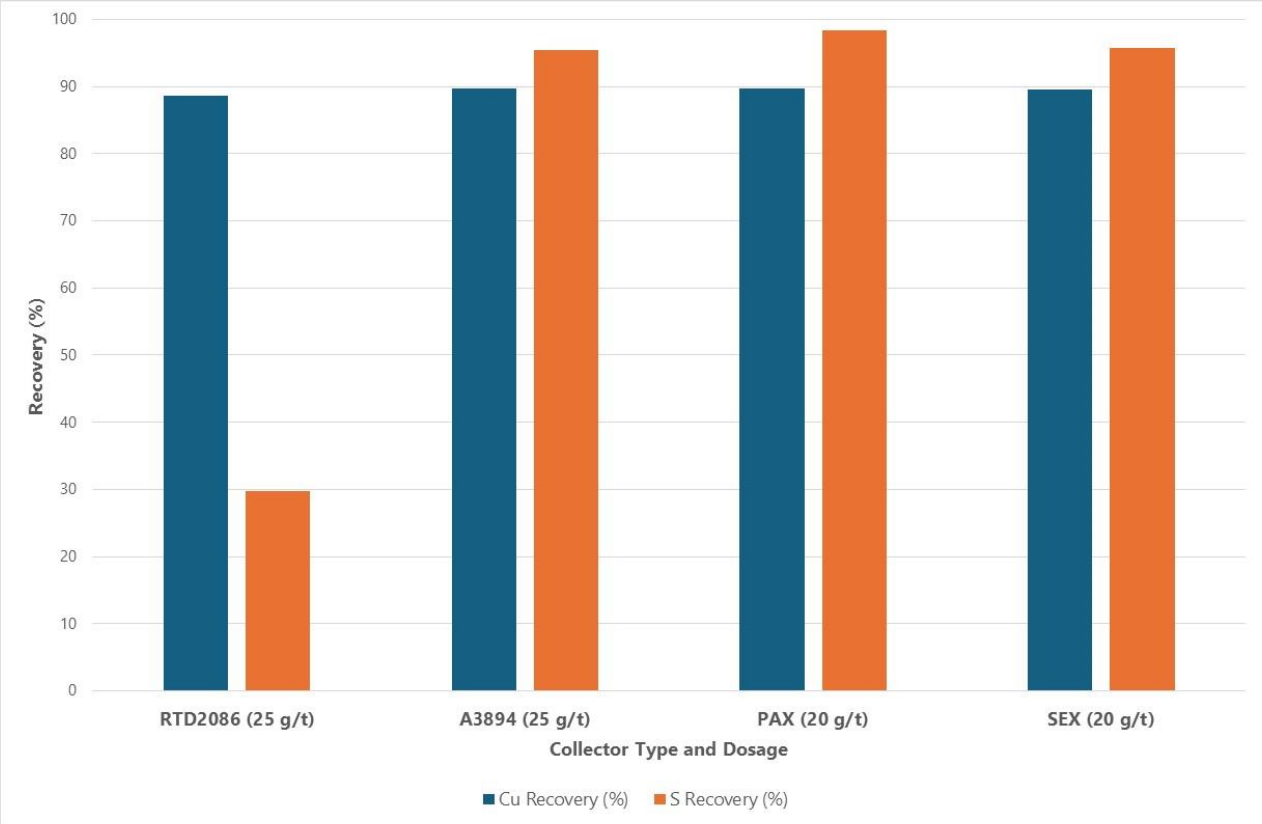
For the Cortadera open pit deposit (OP) samples, testwork commenced on the Open Pit MC composite (made up from 20 variability samples) and after optimisation the "Cortadera OP Flowsheet" was developed. This flowsheet was then tested on each of the variability samples.

Testwork was also conducted on a second OP composite which omitted two of the transition variability samples (Open Pit MC2).

Grind size optimisation rougher tests were performed on the Open Pit MC composite. This testwork showed that Cu recovery increased consistently at finer grind sizes. A grind P_{80} of 106 µm was adopted for all Cortadera testwork.

As part of AM0099, four different collectors were compared on the Cortadera Open Pit MC composite. The copper and sulfur recoveries are shown in Figure 13.15. Sulfur recovery is an indicator of pyrite recovery when it is greater than 30%.

Figure 13.15 Comparison of Collectors for OPC Sample



All collectors broadly had the same Cu recoveries, whilst RTD2086 was the only collector that showed selectivity for copper against sulfur (pyrite). Subsequently, RTD2086 was confirmed as the preferred collector for the Cortadera Open Pit test series.

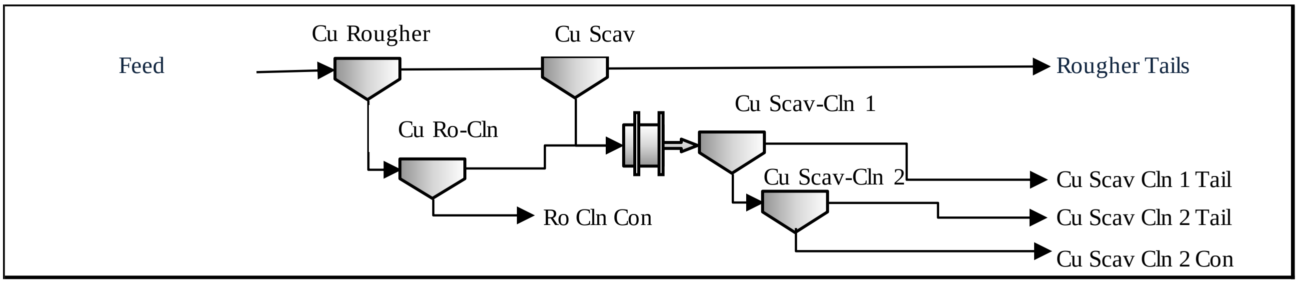
The standard test scheme for the Cortadera OP Flowsheet is shown in Table 13.12.

Table 13.12 : Cortadera OP Standardised Flotation Test Scheme

Stage	Conditioning Time (minutes)	pH	RTD208 Dosage (g/t)	Frother	Flotation Time (minutes)
Mill P_{80} 106 µm	-	Natural pH	-	As required	-
Copper Rougher	1		5		2
Copper Rougher-Cleaner	1				1
Copper Scavenger	1		3		5
Regrind Scavenger Concentrate and Rougher Cleaner Tails to P_{80} 25 µm	-		-		-
Copper Scavenger-Cleaner 1	2		5		7

The schematic OP standard flowsheet is shown in Figure 13.16. The rougher concentrate reports to one stage of Rougher-Cleaning and does not require regrinding. The Rougher Cleaner tail joins the scavenger concentrate for regrinding and reports to two Scavenger-Cleaner stages. The final Cu concentrate is the combination of the Rougher-Cleaner Concentrate and the Scavenger-Cleaner 2 Concentrate.

Figure 13.16 : Cortadera Open Pit Standard Flowsheet



Pyrite flotation was conducted on the rougher tails in much of the Cortadera test program, but these results do not contribute to the PFS flowsheet design.

Testwork was also completed on Cortadera underground samples, which are typically from depths of more than 400 m downhole. Testwork commenced on a composite (made up from 24 variability samples) and was used for developing the "Cortadera UG Flowsheet". This flowsheet was then tested on each of the variability samples.

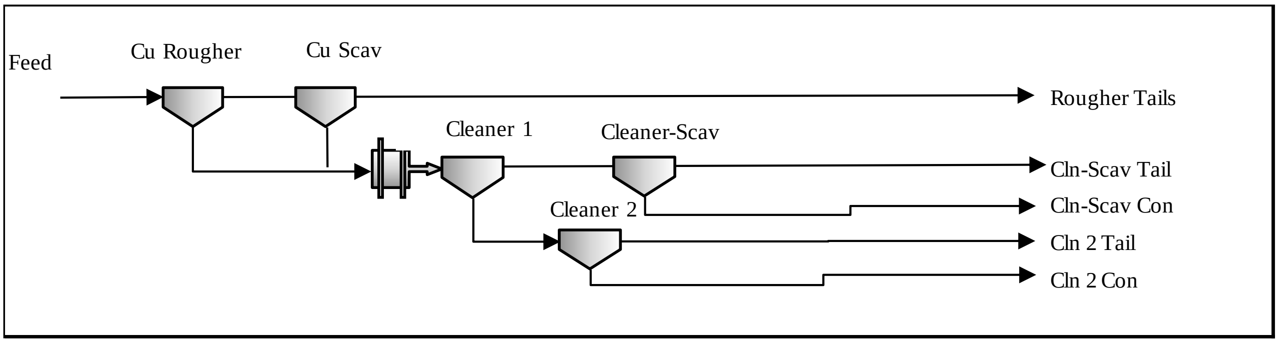
The test conditions for the Cortadera underground (UG) flowsheet is shown in Table 13.13.

Table 13.13 Cortadera UG Standardised Flotation Test Scheme

Operation	**Conditioning time (minutes)**	**pH**	**Collector RTD2086 (g/t)**	**Flotation time (minutes)**
Mill P_{80} 106 µm		Natural pH		
Copper Rougher Concentrate	1		5	2
Copper Scavenger Concentrate	1		3	5
Regrind Combined Rougher and Scavenger Concentrates to P_{80} 25µm				
Copper Cleaner Concentrate 1	1		7	6
Copper Cleaner-Scavenger Cleaner Concentrate	1		3	4
Copper Cleaner 2 Concentrate				4.5

The Cortadera underground (UG) flowsheet is slightly different from the Cortadera OP flowsheet as shown in Figure 13.17. In this flowsheet combined rougher and scavenger concentrates report to regrinding and the reground material is subject to two stages of cleaning and a Cleaner-Scavenger stage. The final Cu concentrate is only comprised of the Cleaner 2 Concentrate and the cleaner scavenger tail is sent to waste.

Figure 13.17 : Cortadera UG Standard Flowsheet



Unlike the Cortadera open pit ore, Cortadera UG cleaning stages (at least in the case of the flowsheet development composite) were not able to achieve target 25% Cu concentrate grade unless rougher concentrate had been reground.

The three ore types (Productora, Cortadera OP and Cortadera UG) have varying process flowsheet configurations, collector additions and flotation times. Cleaning of unground rougher concentrate (as per Figure 13.16), could well be applicable to other ore types and blends and not just for Cortadera OP. The QP recommends it is always an option available in the plant and can be chosen or deselected as the ore performance demands.

The advantages associated with making part of the final concentrate without regrinding are:

- less regrinding power consumption
- less regrinding media consumption
- faster settling in the CCDs and Cu concentrate thickener
- lower moisture in filtered concentrate
- reduced loading on the recleaner flotation circuit.

This study has shown that, although there are differences amongst the schemes for the different deposit types, they can all be configured with one set of flotation cells. To allow this flexibility in the flowsheet, adequate regrind power, sufficient residence times per stage and flexible concentrate destinations have been incorporated.

Cortadera OP and UG rougher / scavenger flotation requires much less collector (8 g/t RTD2086) compared to Productora ore (25 g/t). This raises the possibility that it will be difficult to choose the correct collector dose when processing ore blends. When excess collector is added in any of the three schemes described above the normal result is for pyrite to begin floating with the Cu and Mo. It will be important to determine the influence of collector addition rate on Cu/Pyrite selectivity for blends of Cortadera and Productora ore sources in the FS testwork program. Excess pyrite reporting to the final concentrate will prevent attainment of 25% copper grade in final concentrate.

In general, Cortadera has lower copper recoveries, lower gold recoveries and more pyrite than Productora. Mineralogical analysis has revealed that Cortadera has poorer chalcopyrite liberation from pyrite compared to Productora.

13.3.2.3.3 2020 - San Antonio - Flotation Testwork Results

Only two San Antonio test samples have been assessed over the history of the Project, the SAG-01 composite (a fresh, underground sample from the current San Antonio mine workings) and the San Antonio (Sapmet) composite.

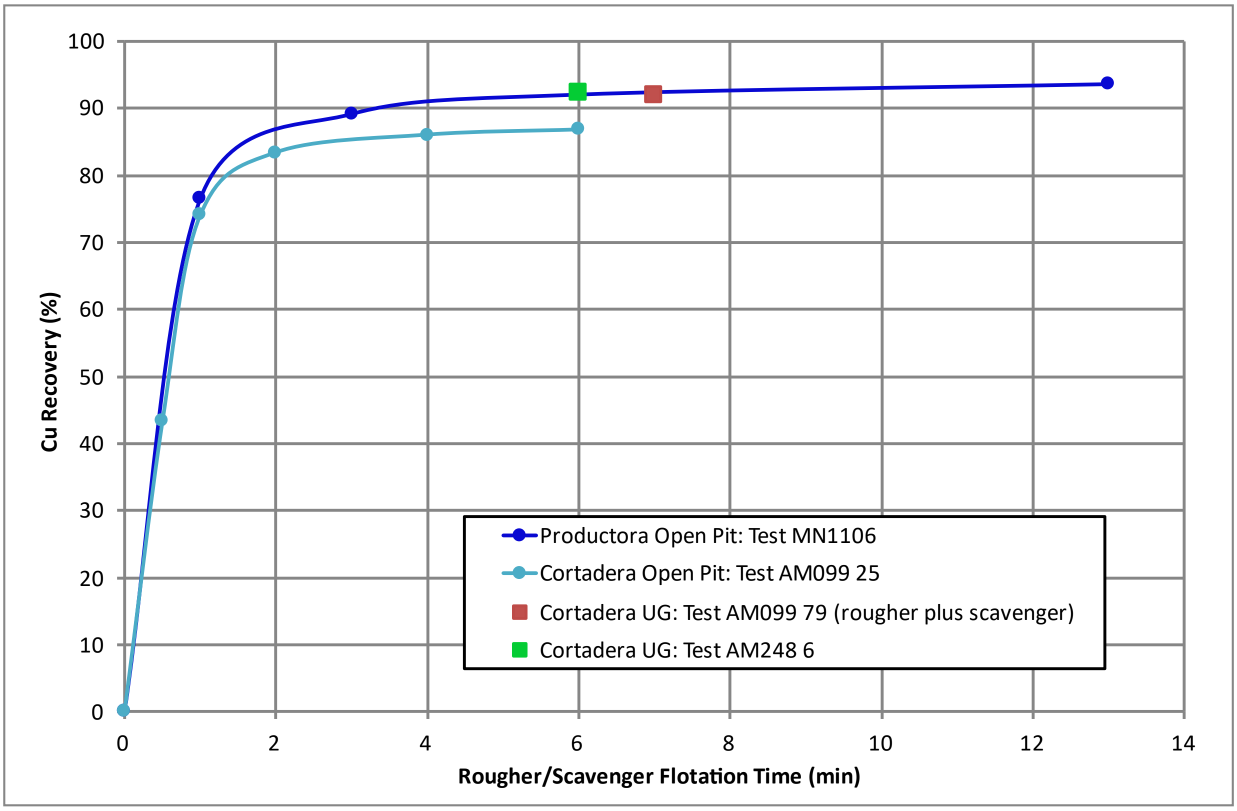
As part of program AM0063, tests were completed on the SAG-01 composite. Both Cu and Au recoveries increase strongly as the P_{80} decreases. A grind of 106 µm P_{80} is appropriate for San Antonio ore, giving high recoveries of Cu (90%), Au (73%) and Ag (70%).

As part of AM0248, testwork was completed on the Sapmet Composite which was formed with core reserves from the SAPMET-002 and SAPMET-003 drill holes. This sample contained a large proportion of San Antonio transitional ore and gave low Cu recoveries ranging from 54% to 72% across several tests. The sample was stored for many years prior to testing and the results have not been relied upon for design purposes. These tests were conducted with the primary objective of providing tailings samples for geochemical evaluation.

13.3.2.4 Flotation Kinetics (using results from 2014 to 2024)

Examples of the kinetics of rougher Cu/Mo flotation for the three deposits are compared in Figure 13.18.

Figure 13.18 : Typical Flotation Kinetics for Samples from the Main Deposits



No rougher/scavenger only flotation tests were performed for the Cortadera UG ores so the recalculated combined rougher/scavenger concentrate is plotted against the total rougher/scavenger flotation time for two separate tests on different UG composites.

The flotation of chalcopyrite is rapid and similar in all cases. The Cortadera OP sample has a lower ultimate recovery due to its finer grained nature, poorer liberation of chalcopyrite from pyrite and the tendency for some oxidation to exist within the fresh intervals used to prepare the composite.

Similar rapid sulphide flotation was observed in cleaning and in the Mo flotation stages. These kinetics and the locked cycle testing stage times have been used in selecting the residence times for the stages.

13.3.2.5 Copper Concentrate Quality

Detailed concentrate analyses have been performed on copper/molybdenum concentrates produced by locked cycle testwork on ore samples from the three major deposits to determine diluent elements and potential level of penalty elements in the Costa Fuego Project ores. Note that although these concentrates did not have the Mo removed the mass of Mo concentrate removed is small and the effect on penalty elements is insignificant. Table 13.14 shows the assays on the concentrates for typical penalty elements, against a benchmarked example used for this study. The table demonstrates that acceptable contamination levels are readily achievable from all deposits and for all elements of concern.

Table 13.14 : Costa Fuego Concentrate Assays compared to Indicative Smelter Penalty Limits[1]

Element	Penalty Limit[1] (ppm)	Productora (ppm)	Cortadera OP (ppm)	Cortadera UG (ppm)
As	2,000	200	33	40
Sb	1,000	400	6	8
Bi	500	30	1.6	3.0
Cl	500	360	100	150
Pb	10,000	4,200	197	95
Zn	30,000	1,800	560	470
Ni+Co	5,000	2,200	50	240
Al_2O_3 + MgO	50,000	12,800	24,000	26,000
F	330	140	<200	<200
Hg	10	2.0	0.4	0.8
Se	300	80	85	95

[1]See Table 19.8, Indicative Penalty scale B.

13.3.2.6 Use of Sea Water for Flotation at Costa Fuego

Given the Project location near to the coastline and in Chile's Atacama Desert, there is a scarcity of fresh water and an abundance of sea water. Many mines in the Atacama (such as Escondida) successfully use sea water for processing.

Testwork comparing the flotation performance in seawater with tapwater commenced in at least 2014 and perhaps earlier, but no locked cycle comparisons had been performed until 2024. Comparative Locked Cycle testwork has been conducted to definitively confirm whether sea water has any negative impact upon revenue generation in the Project.

13.3.2.6.1 2024 Water Quality Testwork (AM248)

In the AM248 program completed by Auralia Metallurgy in 2024, tests were conducted on different samples comparing flotation response in fresh water (Perth tap water) and sea water. After checking the recovery responses in open circuit tests, a set of six locked-cycle tests was completed on Cortadera and Productora composites testing sea water against tap water. These locked-cycle tests include a copper recovery stage and a pyrite rougher stage.

Results from these tests are shown in Table 13.15. Note, the recoveries (distributions) refer to the final copper concentrates only and not the additional recoveries to the pyrite concentrates.

Table 13.15 : Sea Water vs. Fresh Water Locked Cycle Test Results in Copper Concentrate

Identification	Water Type	Copper			Gold			Silver		
		Head Grade (%)	Conc. Grade (%)	% Dist.	Head Grade (g/t)	Conc. Grade (g/t)	% Dist.	Head Grade (g/t)	Conc. Grade (g/t)	% Dist.
Productora OP	Sea	0.51	24.6	60.3	0.07	3.83	44.5	0.31	11.3	43.3
	Tap	0.51	21.5	59.5	0.07	3.65	46.6	0.31	11.0	42.8
Cortadera OP	Sea	0.42	26.0	82.8	0.13	4.25	41.6	0.84	24.9	44.7
	Tap	0.42	24.8	82.3	0.13	3.80	35.3	0.84	28.7	49.5
Cortadera UG	Sea	0.51	26.8	91.0	0.18	5.22	58.0	1.00	35.0	61.2
	Tap	0.51	25.9	87.3	0.18	5.30	51.7	1.00	32.0	57.4

For all composite samples, the tests with sea water gave better copper grades and recoveries in concentrates than tap water. For gold and silver, sea water had superior results to tap water in two of the three tests.

Based on these copper results, the QP is satisfied that seawater was and is the appropriate choice for process water in the comminution and Cu/Mo flotation section of the Costa Fuego Project concentrator.

13.3.2.6.2 2023 - Washing Chlorides from Cu Concentrate

Chlorides are undesirable in the Cu concentrate, but in a sea water circuit they will naturally be present unless removed by washing. Washing tests were conducted on the two Cortadera concentrates in the AM099 test program to demonstrate chloride removal. The results are shown in Table 13.16.

Table 13.16 : Results of Washing of Chloride for Copper Concentrate

Identification	Type	Cl (mg/L)
Cortadera OP Concentrate	Unwashed	320
	Washed	100
Cortadera UG Cu/Mo Concentrate	Unwashed	480
	Washed	150

In both tests, washing reduced the chloride contents further below the 500 mg/L penalty rate as defined earlier in Table 13.14.

13.3.2.7 Locked-cycle Tests (2015 to 2023)

A summary of the results for the locked-cycle flotation tests is shown in Table 13.17. It includes the Cu Concentrate grades and Cu recovery from the average 4th-6th cycles, along with the comparative Cu recoveries if Cu Concentrate grades are artificially fixed at 25%.

Table 13.17 : Locked-cycle Tests

Test Number	Sample	Test Flowsheet	Cu Head Grade (%)	Average 4th-6th Cycles		
				Cu Conc. Grade (%)	Cu Recovery (%)	Estimated Cu Recovery at Fixed Cu. Conc Grade of 25% (%)
AM099-82	Open Pit MC2	Cu Cleaner/Cu Scav-Cleaner/Cu Rougher/Cu Cleaner/Pyrite Rougher/Pyrite Cleaner	0.48	27.5	80.3	84
AM099-83	Open Pit MC2	Cu Cleaner/Cu Scav-Cleaner/Cu Rougher/Cu Cleaner/Pyrite Rougher/Pyrite Cleaner	0.48	27.5	76.8	82
AM99-85	UG Comp	Cu Cleaner/Cu Scav-Cleaner/Cu Rougher/Cu Cleaner/Pyrite Rougher/Pyrite Cleaner	0.46	25.7	89.6	90
MN1249	FD1	Rougher/Regrind/3 Stage Cleaners	0.44	28.9	89.2	91
MN1262	FD1	Rougher/Regrind/2 Stage Cleaners	0.44	28.2	89.4	90
MN1268	FD4	Rougher/Regrind/2 Stage Cleaners	0.44	28.8	68.9	71
MN1269	FD2	Rougher/Regrind/2 Stage Cleaners	0.59	28.8	85.1	86
MN1276	FD12	Rougher/Regrind/2 Stage Cleaners	0.51	31.3	90.6	92
MN1431	FD12	Rougher/Regrind/1 Cleaner Stage	0.51	20.0	95.4	N/A
MN1474	FD2	Rougher/Regrind/2 Stage Cleaners	0.59	27.2	90.7	92
MN1478	FD13	Rougher/Regrind/2 Stage Cleaners	0.89	27.4	91.9	92
MN1488	V13	Rougher/Regrind/2 Stage Cleaners	0.50	28.7	92.6	93

Table 13.17 : Locked-cycle Tests

Test Number	Sample	Test Flowsheet	Cu Head Grade (%)	Average 4th-6th Cycles		
				Cu Conc. Grade (%)	Cu Recovery (%)	Estimated Cu Recovery at Fixed Cu. Conc Grade of 25% (%)
MN1529	V18	Rougher/Regrind/2 Stage Cleaners	0.24	25.1	96.3	96.3
MN1530	FD3	Rougher/Regrind/2 Stage Cleaners	0.54	30.7	82.9	85
MN1531	V7	Rougher/Regrind/2 Stage Cleaners	0.34	29.7	85.9	88
MN1536	V13	Rougher/Regrind/2 Stage Cleaners	0.50	29.8	89.2	90
MN1537	FD6	Rougher/Regrind/2 Stage Cleaners	0.22	28.1	84.1	86
MN1538	V11	Rougher/Regrind/2 Stage Cleaners	1.01	31.7	94.6	95

All but one of tests achieved concentrate grades of 25% Cu or higher and most of the recoveries were higher than 85%. When the Cu recoveries were normalised at a Cu concentrate grade of 25%, it showed a slight increase in average recovery.

The comparison between the two LCTs with sample FD12 (MN1276 and MN 1341) demonstrates the effect of using excess RTD2086 collector and removing a cleaning stage. The concentrate grade fell by more than 11% and the recovery increased by about 5%.

13.3.2.8 Final Cu Concentrate Assay

Final Cu concentrate samples obtained from Locked Cycle Tests (Cycle 4-6) performed using Productora, Cortadera Open Pit and Cortadera Underground representative samples in 2024, were assayed for main elements of interest, and impurities. The results[1,2] reported copper concentrate grade above 25%, with no penalty levels for any of the elements assayed (Refer to Section 19). The detailed list of elements is provided in Table 13.18.

Table 13.18 : Final Cu Concentrate for the Costa Fuego Project

Element	Unit	Final Cu Concentrate Costa Fuego Project
Cu	%	25.6
Mo	ppm	586
Au	ppm	3.82
Ag	ppm	23.1
Al2O3	%	2.66

Element	Unit	Final Cu Concentrate Costa Fuego Project
As	ppm	18.9
Ba	ppm	96
Bi	ppm	2.6
CaO	%	0.59
Cd	ppm	2.0
Cl	ppm	200
Co	ppm	323
F	ppm	238
Fe	%	28.1
Hg	ppm	0.78
K	ppm	4568
MgO	ppm	3599
Mn	ppm	122
Na	ppm	2,611
Ni	ppm	178
P	ppm	134
Pb	ppm	45
S	%	32.6
Sb	ppm	9
Se	ppm	69
SiO2	%	9.5
Sn	ppm	6
Sr	ppm	33
Te	ppm	3.0
Th	ppm	3.9
Ti	ppm	563
Zn	ppm	301
Zr	ppm	125

[1] Weighted average by copper metal produced by deposit on a LOM basis

[2] Final Cu concentrate stream includes the two streams reporting to the Final Cu concentrate (Molybdenum rougher flotation tail and Molybdenum cleaner flotation tail, as per Figure 17.6).

13.3.2.9 Grind Size Correction – Laboratory to Plant

Preferential grinding of the copper minerals, when compared to grinding of the host rock, is a physical process that cannot occur in the batch laboratory rod mill. This means that once ground and ready for the flotation test, all mineral species in flotation feed have essentially the same P_{80}, typically 106 µm. However, in an industrial plant using hydrocyclone classification for ball milling, the high copper-mineral specific gravity (SG) results in a finer P_{80} for copper than for the overall ground product. Consequently, the laboratory optimal P_{80} arising from the testwork program is appropriate for copper minerals such as chalcopyrite with SG 4.2 and pyrite with SG 5.0. However, the lower SG gangue minerals (with SGs of 2.5 to 3.0) dominate the plant feed and report to plant flotation feed at a coarser P_{80} than the sulphide minerals. This occurs because cyclone cut point is unique for each mineral and is inversely proportional to the particle SG.

Consequently, the overall grind size for plant design is set to be one standard sieve size coarser than the optimal testwork P_{80} for Cu recovery. Practically, this means that a 106 µm P_{80} grind size in testwork translates to a 125 µm P_{80} grind target in the plant.

13.3.2.10 Dewatering Testwork

13.3.2.10.1 Introduction

Tailing and concentrate samples from the Productora sulphide flotation test program underwent physical testing. The tests conducted included:

- Concentrate thickening and filtration
- Tailings rheology, thickening and filtration.

The tailings samples underwent rheology testwork at ALS Perth and Newpark, tailings evaluation at Knight Piésold (KP) and thickening and filtration testwork at Outotec.

The concentrate testwork was conducted on a bulk flotation concentrate (from the PRP0812 sample). The concentrate tested was at P_{80}= 30 µm. This concentrate sample used in the concentrate filtration and thickening work was not ideal, as RC chip sample was used to produce the concentrate. More thickening and filtration testwork is recommended to be conducted on concentrate generated from diamond drill samples.

Samples for tailings testwork were composited from the rougher tailings from multiple flowsheet development flotation tests. Two composites were made up, one at a P_{80} grind size of 106 µm and the other at a P_{80} of 150 µm. Furthermore, to simulate cleaner tailings, approximately 4% of the dry sample mass was ground to ≈25 µm, then added back to each composite.

13.3.2.10.2 Copper Concentrate and Tailings Thickening Testwork Results, Outotec 2015

Testing showed that the flotation tailings could achieve an underflow density of 70% solids (w/w) at a flux of 0.5 t/m^{2}h and using 10 g/t of flocculant. Under these conditions, the overflow solids content was less than 150 mg/L. Testwork outcomes are given in Table 13.19.

For the Cu concentrate, an underflow density of 65% solids (w/w) was achievable at a flux of 0.25 t/m^{2}h and using 10 g/t of flocculant. Overflow solids under these conditions was less than 150 mg/L.

Outotec high-rate thickeners with Outotec Vane feedwell were recommended for the Project.

Table 13.19 Copper Concentrate and Tailings Thickening Testwork Results

Flotation Tails	Results
P_{80} (µm)	177
Solids loading (t/m^{2}h)	0.5
Liquor Rise Rate (m/h)	1.59
Feed slurry density (% w/w solids)	35
Slurry pH	8.25
Flocculant dosage (g/t)	10
Underflow density (% w/w solids)	69.6
Overflow clarity (ppm)	<150

Table 13.19 Copper Concentrate and Tailings Thickening Testwork Results

Flotation Tails	**Results**
Copper Concentrate	**Results**
P_{80} (µm)	25
Solids loading (t/m²h)	0.25
Feed slurry density (% w/w solids)	18
Slurry pH	6.0
Flocculant dosage (g/t)	10
Underflow density (% w/w solids)	66.9
Overflow clarity (ppm)	<150

It must be recognised that these samples represented Productora ores and that the tailings sample was significantly coarser than the expected tailings for this PFS. The concentrate sample size distribution is appropriate for this PFS.

13.3.2.10.3 Tailings Thickening Testwork – Fremantle Metallurgy 2024

Table 13.20 Tailings Thickening Testwork Results

Productora Flotation Tails	
P_{80} (µm)	106
Solids loading (t/m²h)	0.6
Liquor Rise Rate (m/h)	4.47
Feed slurry density (% w/w solids)	12
Flocculant dosage (g/t)	20
Underflow density (% w/w solids)	57.5
Overflow clarity (ppm)	243
Cortadera OP Flotation Tails	
P_{80} (µm)	106
Solids loading (t/m²h)	0.6
Liquor Rise Rate (m/h)	4.32
Feed slurry density (% w/w solids)	12.5
Flocculant dosage (g/t)	15
Underflow density (% w/w solids)	59.1
Overflow clarity (ppm)	296
Cortadera UG Flotation Tails	
P_{80} (µm)	106
Solids loading (t/m²h)	0.6
Liquor Rise Rate (m/h)	3.49
Feed slurry density (% w/w solids)	15
Flocculant dosage (g/t)	15
Underflow density (% w/w solids)	66.4

Table 13.20 Tailings Thickening Testwork Results

Overflow clarity (ppm)	86

These results are poorer than the earlier results for Productora and this is partly due to the finer feed and is also due to the presence of transitional materials in the two open pit samples. The flotation test samples were made up from available materials and achieved their purpose to provide definitive intercomparative flotation testing in both seawater and freshwater. Each sample is not intended as a definitive representation of the ore sources from which they were derived.

It is recommended that new representative samples of the ore sources and blends of ore sources be prepared during the FS and are subject to thickening testwork.

13.3.2.10.4 Copper Concentrate Filtration Testwork Results

The same Productora concentrate sample that was used for thickening testwork was also used for filtration testwork. Initially, testwork focused on membrane filter press technology. Pressure filtration technology was then tested as an alternative, aimed at achieving reduced wash times. A summary of the major outcomes from the Outotec report are described below.

13.3.2.10.5 Membrane Filter Press (MFP)

In a membrane filter press, pumping and pressing stages are relatively short with air drying achieving further dewatering. Introducing a wash stage significantly increases the total cycle time.

- Using high pressure (10 bar) air for drying reduces cake moisture content. Introducing a wash stage increases cake moisture content. Target moisture of 8% was achieved in the 25 mm chamber using 10 bar air for drying. There is a direct relationship between increasing filtration rate and increasing cake moisture content. Introducing a wash stage significantly decreases filtration rate by increasing total cycle time.
- Using a wash volume of 0.34 m^3/t, it is possible to achieve a test filtration rate of 56 kg D.S./m^2h with a cake moisture content of 9.7%.
- Using a wash volume of 0.34 m^3/t achieved a total filter efficiency of 99%.
- Results showed that using a wash volume of 0.34 m^3/t results in 355 ppm Cl remaining in the cake (target of <500 ppm) and 625 ppm Cl in the entrained liquor (target 6,000 ppm).

13.3.2.10.6 Pressure Filtration (PF)

Cycle time with washing is significantly less for pressure filtration technology than it is for membrane filter press technology.

- The target moisture of 8% was not achieved. During testing, the particle size distribution (PSD) was measured, and the grind size was deemed to be finer than anticipated. With such a fine grind size, the transportable moisture level (TML) would be higher. Testing using MFP technology showed that higher pressure air (10 bar) would lower cake moisture content by approximately 1%.
- There is a direct relationship between increasing filtration rate and increasing cake moisture content. Introducing a wash stage significantly decreases filtration rate by increasing cycle time.

- Using a wash volume of 0.29 m^3/t, it is possible to achieve a test filtration rate of 179 kg D.S./m^2h with a cake moisture content of 9.9%.
- Using a wash volume of 0.29 m^3/t achieved a total filter efficiency of 98.6%.
- Results show that using a wash volume of 0.29 m^3/t results in 405 ppm Cl remaining in the cake (target of <500 ppm) and 628 ppm Cl in the entrained liquor (target 6,000 ppm).

It must be remembered that the concentrate used in this work was prepared from Productora ore only and the particular test sample was derived from RC chips for the purpose of molybdenum flotation testwork.

It is recommended that concentrates derived from all main ore sources and some from blended ore sources be produced for filtration testwork during the FS.

13.3.2.10.7 Copper Tailings Rheology Testwork Results

Two Productora tailing samples (at P_{80}=106 and 150 µm) were also used for rheology testwork at ALS Perth and Newpark. The objective of this testwork was to confirm whether each of the tailings samples would be amenable to centrifugal pumping. Results from the Newpark testwork are shown in Table 13.21.

Table 13.21 : Flotation Tailings Rheology Results

FD tailings (P_{80}=106 µm)		FD tailings (P_{80}=150 µm)	
Solids (%)	**Yield (Pa)**	**Solids (%)**	**Yield (Pa)**
74	31	74	23
65	2	65	2
57	1	57	0
50	0	50	0
45	0	45	0

Based on this rheology testwork data, it was concluded that centrifugal pumping can be used for tailings.

13.3.2.11 Pyrite Flotation Circuit Opportunity

An opportunity exists to include a pyrite flotation circuit in the Costa Fuego Project flowsheet. This circuit would recover gold and other elements bearing pyrite concentrate that could potentially be sold to market or it could be treated on-site to recover values. Pyrite flotation is non-selective in that it recovers all remaining sulphides, and it also recovers copper sulphide minerals that were not recovered in the Cu/Mo flotation circuit or were rejected in cleaning due to lack of liberation from pyrite or gangue.

Significant testwork has been completed to date which demonstrates that a pyrite flotation circuit is robust and is able to produce concentrates grading between 35% (rougher) and 50% (cleaner) sulphur from the major orebodies.

QEMScan analysis was completed on the pyrite concentrates from both Cortadera OP and Cortadera UG samples. QEMScan showed there is a significant proportion of chalcocite in the Cortadera OP pyrite concentrate. Tests of small amounts of a less selective Cu collector have not demonstrated that these minor Cu minerals can be recovered without also recovering a large amount of pyrite. There are also fine chalcopyrite phases encapsulated within pyrite grains which are undesirable to recover in a Cu flotation stage. These copper minerals, which are effectively lost from the existing Cu flotation circuit, can be recovered in a pyrite flotation stage. Copper sulphides associated with silicates will also be recovered, provided some sulphide surface is exposed at the particle surface. However,fine copper minerals locked within silicate grains are unlikely to be recovered in either a copper concentrate or a pyrite concentrate.

The liberation table for the Cu minerals in the Cortadera OP pyrite concentrate is in Table 13.22. Copper recovery to Cu concentrate was 88% and 4.5% of the copper reported to the pyrite concentrate. Almost 8% of the Cu remained un-floated and reported to pyrite flotation tails.

Table 13.22 : Liberation of Copper Minerals in AM99-80 Cortadera OP Pyrite Concentrate

Liberation Classes	Size		
	+25 µm	-25 µm	Combined
	Combined Cu minerals (mass% in fraction)		
Well liberated	14.9	23.7	38.6
High grade middlings	4.7	4.3	8.92
Medium grade middlings	10.8	3.4	14.3
Low grade middlings	21.6	2.8	24.3
Locked	12.6	1.3	13.9
Total	64.6	35.4	100.0

The table shows that about 50% of the copper in the pyrite concentrate has good liberation and may be able to be recovered in the copper concentrate with either additional collector or residence time in the scavenger flotation stage. Note, the +25 µm material will typically float better than the -25 µm material. Although this appears to be an opportunity for improving primary Cu recovery, it is also a strong possibility that the lost Cu is simply difficult to float using RTD2086 and will continue to report to pyrite concentrate under most circumstances.

For the medium grade and locked materials, additional regrinding is required to liberate values from waste before flotation. It may be more technically and economically feasible to recover this copper in a pyrite concentrate, than attempt to regrind to fine sizes to recover it to Cu concentrate.

13.3.2.12 Impact of Rougher Tailings Assay Method on Copper Recovery

The magnitude and variability of the Cu recovery losses to rougher tailings is partly due to the difficulty in assaying low Cu grades (which the rougher tailings generally are). With the chosen assay method, currently the assay the detection limit is 0.02% and tailings assays generally vary by quanta of 0.01% Cu from there. Thus, each increase in the tailings assay results in a significant step change in the Cu losses.

An opportunity exists to use a more precise method for assaying the Cu in rougher tailings and reduce the step change magnitude that exists in the results with respect to recovery losses.

13.3.2.13 Flotation Recovery Predictive Models

Flotation programs have tested ores from all four deposits and the results have been examined to determined recovery models for the four revenue-generating elements. Different models have been developed for each deposit and the deposit responses have been further delineated for fresh and transitional ore types. Metal recovery models have been developed based on the head grade elements that are both in the geological model and the metallurgical testwork results. This arrangement allows the use of geological assay data for calculating predicted recoveries as a component of mine planning. All model equations are included in Table 13.23. Where head grade-recovery models could not be suitably developed, due to insufficient data or excessive scatter in results, fixed recoveries have been assigned.

Table 13.23 : Modelled Sulphide Recoveries to Concentrate

Item	Productora	Alice	Cortadera OP	Cortadera UG	San Antonio
Cu Recovery (Fresh)	Rec = 9.072 x CuF% + 83.66; Max=95%	RecCu = 0.4951 x CuF% + 91.0	Rec = 17.016 x Ln(CuF%) + 96.378; Max=90%, Min=18%	Rec = 8.615 x Ln(CuF%) + 96.122; Max=95% at CuF>0.88%	93% Fixed
Cu Recovery (Transitional)	RecCu = 19.609 x CuF% + 63.443; Max=90%	Not applicable	Rec = 17.016 x Ln(CuF%) + 86.378; Max=80%, Min=8%	Not applicable	83% Fixed
Cu Recovery (Oxide)	56% Fixed	46% Fixed	50% Fixed	Not applicable	70% Fixed
Au Recovery (Fresh)	Rec = 145.4 x AuFppm + 38.549; Max=80%	RecAu = 145.4 x AuFppm + 46.692; Max=80%	Rec = 104.74 x AuF(g/t) + 29.42	Rec = 30.368 x AuF(g/t) + 51.637	70% Fixed
Au Recovery (Transitional)	Rec = 145.4 x AuFppm + 38.549; Max=80%	Not applicable	Rec = 104.74 x AuF(g/t) + 29.42	Not applicable	70% Fixed
Mo Recovery (Fresh)	Rec = 0.9 x [5.676 x Ln(MoFppm) + 51.191]; Max=95%	Rec = 0.9 x [5.676 x Ln(MoFppm) + 51.191]; Max=95%	Rec = 0.9 x [12.563 x Ln(MoFppm) + 21.88]; Max=92%	Rec = 0.9 x [3.2359 x Ln(MoFppm) + 70.35]; Max=95%	50% Fixed
Mo Recovery (Transitional)	Rec = 0.9 x 62.2% = 56.0%	Not applicable	Rec = 0.9 x 24.3% = 21.8%	Not applicable	50% Fixed
Ag Recovery (Fresh)	40% Fixed	40% Fixed	27% Fixed	38% Fixed	65% Fixed
Ag Recovery (Transitional)	40% Fixed	Not applicable	27% Fixed	Not applicable	65% Fixed

13.4 Leaching Testwork

13.4.1 Introduction

This section summarises the metallurgical leaching test work completed on samples from the Costa Fuego Project, considering the two main deposits, Productora and Cortadera, which represent more than 90% of the processing feed considered for leaching technology. The results confirmed the technical and economic viability for oxide and low-grade sulphide to heap leaching and dump leaching technology, respectively, as well as provided input for pre-feasibility level design.

13.4.2 Heap Leaching

Samples from the Oxide and the Transitional zone were selected for the heap leaching program from Productora and Cortadera mines.

13.4.2.1 Mineralogy

The mineralogy of the samples from the oxide zone consisted of various oxide copper minerals, with chrysocolla being the main oxide mineral. The mineralogy of samples from the transition zone included secondary sulphides such as chalcocite and exotic copper compounds.

13.4.2.2 Metallurgical Testwork

13.4.2.2.1 2014-2015 Preliminary Heap Leaching Characterisation

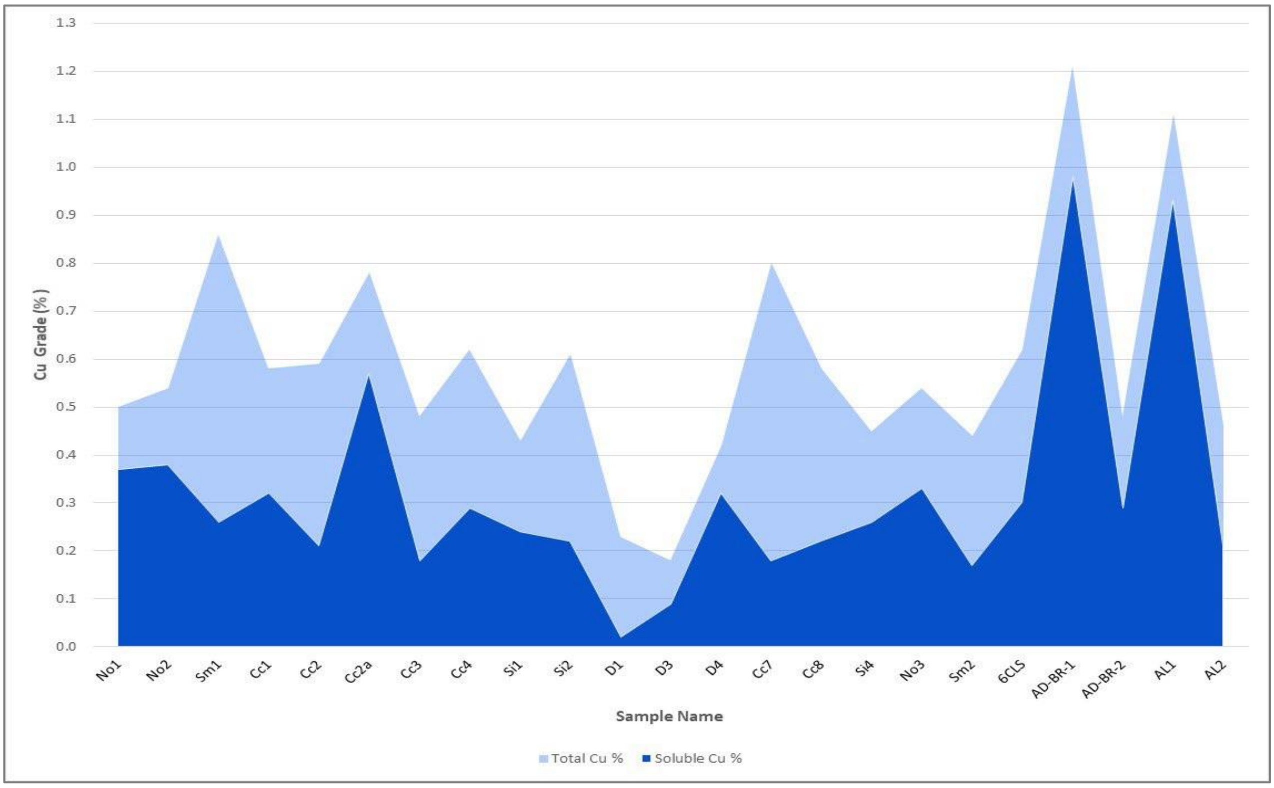
In 2014-2015, preliminary heap leaching characterisation testwork was conducted on Productora samples, the main deposit discovered at the time. The initial testwork focused on conventional sulfuric acid heap leaching methods. Later in the Project, the focus changed to NOVAMINORE® hypersaline technology.

The preliminary testwork was managed and designed by Mintrex Pty Ltd, and consisted of the following:

- ALS-SC100 testwork program completed by ALS Metalurgia Santiago: Mineralogy, abrasion index testwork, bond crushing work index testwork, bottle roll leach testwork and open-circuit column leach testwork.
- ALS-SC192 testwork program completed by ALS Metalurgia Santiago: Bottle roll leach testwork.
- ALS-A16182 testwork program completed by ALS Patagonia S.A.: UCS testwork.

Drill core samples were selected to encompass the broad range of copper oxide types and oxidation levels at Productora. Elemental characterisation completed on the samples used is presented in Figure 13.19.

Figure 13.19 Total Copper and Soluble Copper Head Grade



13.4.2.2.2 Hardness Characterisation

Comminution testwork included Abrasion index tests for three samples (SC100), CWi tests for 18 samples (SC100) and UCS tests for 25 samples (A16182).

A summary of the results of the comminution testwork is shown in Table 13.24. These results indicate a moderately hard processing feed.

Table 13.24 Hardness Characterisation Test Results

Bond Crusher Work Index (CWi)	Number of samples	18
	80th Percentile	9.3 kWh/t
	Minimum	1.3 kWh/t
	Maximum	29.7 kWh/t
Unconfined Compressive Strength (UCS)	Number of samples	25
	80th Percentile	60 MPa
	Minimum	6 MPa
	Maximum	145 MPa
Abrasion Index (Ai)	Number of samples	3
	80th Percentile	0.290

13.4.2.2.3 Bottle Roll Leaching Test

In the initial testwork program (ALS-SC100), bottle roll leach tests were completed on 19 samples. There were three variable leaching conditions for these tests – pH 2 in tap water, pH 2 in seawater and pH 1 in seawater. Subsequent bottle roll tests were completed on four additional samples (program ALS-SC192) and were completed with the conditions of pH 2 in seawater and pH 1 in seawater.

The results of the testwork completed reported highest copper recovery at pH 1 with seawater, but the acid consumption values were on average six times higher than pH 2 seawater. Copper recovery results were comparable between pH 2 tap water and pH 2 seawater, but with average acid consumption two times higher in tap water.

The tests show that by leaching in seawater, the copper recoveries were often higher than the acid soluble copper deportments, which are often considered a good guide as to the likely maximum copper recovery.

However, the bottle roll tests test produced relatively high average acid consumption of 70 kg/t likely due to conditions explored to expediate leaching kinetics.

13.4.2.2.4 Open-Circuit Column Leaching

As part of program ALS-SC100 testwork program, column leach testwork was conducted on five agglomerated composites for durations varying between 30 and 50 days.

Acid strengths in the leach solutions were selected based on the initial bottle roll test results and acid agglomeration values were based on preliminary agglomeration tests. These initial column leach tests were small scale (80 mm column diameter, 1 m height). The samples were crushed to -12.5 mm, the irrigation rate was 10 L/h/m^2 and the leaching solution acid strength was either 5 g/L or 10 g/L.

A summary of the conditions and results from the open-circuit column leach tests is shown in Table 13.25.

Table 13.25 Conditions and Results from Open-Circuit Column Leach tests

Composite Sample	Leach Time (days)	Acid Agglomeration Amount (kg/t)	Leach Solution Acid Strength (g/L)	Final Copper Recovery (%)	Final Gross Acid Consumption (kg/t)
Cc7	40	10	5	49.4	21.9
Cc8	40	5	5	60.1	11.9
Si4	30	5	5	60.4	10.6
No3	30	25	10	79.9	58.4
Sm2	50	10	5	48.3	27.1

In this case, acid consumptions were moderate, although relatively high for sample No3 likely due to the host rock mineralogy of this sample.

13.4.2.2.5 Stacking, Agglomeration and Hydrodynamic Testing

Also in 2014-2015, a testwork program was carried out by HydroGeoSense Laboratory to define the physical and hydraulic responses of Productora samples. Three Productora composite samples were tested at two crush sizes (Sample 'A' at -25mm and Sample 'B' -12.7mm) and at a variety of levels of agglomeration quality. Key findings of this study included:

- Hydrodynamic properties of all three composites were somewhat negatively impacted when the feed material was crushed to -12.7 mm. However, improved agglomeration mitigated this impact.
- The results from the Stacking Tests showed that Productora was a good candidate for percolation heap leaching. All three composites crushed to 12.7 mm can be considered suitable for a multi-lift (permanent) heap process with a final heap height of 32 m.
- The hydraulic conductivity profile of all the samples suggested conductivity will not be an issue.
- Partitioning of porosity into micro and macro categories occurs at a 50:50 ratio. This ratio allows bulk solution movement and intimate contact between solution and feed material.
- Results from the Hydrodynamic Column Tests (HCT) conducted for each of the B samples confirmed the Productora composites will likely support percolation leaching of a 16 m heap.
- Almost no slumping was noted for the duration of the hydraulic conductivity tests, indicating a robust porous structure for all samples.
- The stacking and hydrodynamic column testing confirmed that the agglomerated feed material tested will likely support percolation heap leaching at the stack heights and application rates as used at the time. The tests also indicated that the feed material likely has the necessary geotechnical stability to allow for multiple lift stacking of the heaps.
- Tested operational saturation values indicated that all these samples will likely be able to support the percolation process at the heap up to 16 m without difficulty.

13.4.2.2.6 2021-2022 Percolation, Bottle roll and 4.5 m Column Leaching

In 2021-2022, additional oxide leach testwork was completed on two Productora samples to complement the existing oxide metallurgical testwork. The work was completed by Independent Metallurgical Operations Pty Ltd (Project IMO 6312) on two composite samples (Composite A and Composite B) which were sourced from drill core. The scope of this testwork was:

- Agglomeration and percolation testwork
- Intermittent bottle roll leach testwork
- Column leach test.

Agglomeration and Percolation

Percolation testwork was used to establish the agglomerate stability and percolation rate. The findings from this test program were:

- Agglomerates reported an acceptable <10% slump after an acid curing dosage of 10 kg/t.

- There was a measurable breakdown of the material when it became wet at acid dosage rates of 30 kg/t used in agglomeration which resulted in poor percolation rates. This appeared to be resolved by lowering the acid strength in the agglomeration step prior to loading the material into the column leach.

Bottle Roll Leaching

Intermittent bottle roll tests were conducted on both composites to determine the acid consumptions and copper recoveries. The tests were conducted in seawater and the test duration was 168 hours. The results from these tests reported final copper recoveries after 168 hours of 65.7% for composite A and 70.7% for composite B. These results are summarised in Table 13.26.

Table 13.26 Bottle Roll Test Results for IMO 6312 Program

Sample	Gross Acid Consumption (kg/t)	Cu Recovery (%)
Composite A (IBR-01)	13	65.7%
Composite B (IBR-02)	15	70.7%

Column Leach Tests

1 m column leach tests were conducted on both composites. The columns were operated for 42 days, irrigated with 10 g/L sulfuric acid in seawater at an irrigation rate of 10 $L/h/m^2$. The results reported a copper recovery of 87% for Composite A and 84% for Composite B. Acid consumptions reported 17 kg/t for Composite A and 14 kg/t for Composite B.

13.4.2.2.7 2020-2021 – Cortadera Bottle Roll Leaching

The Cortadera deposit was acquired and included lately as part of the Costa Fuego Project. In August 2020, 72-hour bottle roll leach tests were completed as part of program AM0063 conducted by Auralia Metallurgy. This test program was completed on two transitional samples (FJOD-01 and FJOD-02) and an oxide composite sample. The objective of these tests was to ascertain the impact on copper recovery and acid consumption by varying pH. For each sample, the leach tests were completed in seawater at three different pH: 1.5, 1.8 and 2.0. A summary of the results from these tests is shown in Table 13.27.

Table 13.27 Results for bottle roll tests from program AM0063

Sample	Target Test pH	Acid Consumption (kg/t)	Cu Recovery (%)
FJOD-01 (Transitional)	2.0	13	38
	1.8	17	52
	1.5	25	63
FJOD-02 (Transitional)	2.0	15	48
	1.8	19	58
	1.5	26	66
	2.0	10	29

Sample	Target Test pH	Acid Consumption (kg/t)	Cu Recovery (%)
Oxide Composite	1.8	12	34
	1.5	20	38

As expected, acid consumption and Cu recovery increased as pH decreased. For all three samples, the acid consumptions at pH 1.5 were almost double those at pH 2.

13.4.2.2.8 2024 NOVAMINORE® Leaching Technology

NOVAMINORE® (NOVAMINORE) technology is a processing method which is an alternative to conventional copper acid heap leaching and copper dump leaching. NOVAMINORE is a hypersaline technology developed by Nova Mineralis company, which applies high concentrations of chloride ions (via salt addition in the cured stage and use of hypersaline water) and regulated irrigation-rest cycles, so there is rock fracturing which directly targets chalcopyrite particles. The process benefits from saline water where there is only low-quality water source available, such as seawater. This reduces dependency on freshwater resources, which is a significant benefit for the Costa Fuego Project, located in an area of freshwater scarcity.

The major differences between the conventional leaching method and the NOVAMINORE® method are:

- In the NOVAMINORE technology, heap leaching must be completed in hypersaline water, whereas conventional heap leaching is typically conducted in freshwater sources. While conventional leaching uses salt water, either seawater or brine, in certain applications, and even salt in the curing stage, NOVAMINORE technology is distinguished by a specific irrigation and resting cycle and the option of air blowing for sulphide ore.
- In the NOVAMINORE technology, heap leaching irrigation cycles are completed intermittently (with rest periods); whereas in conventional heap leaching irrigation is continuous.
- In the NOVAMINORE technology, temperature is controlled (air blow and blanket irrigation) only when chalcopyrite is the target. In the other cases, as conventional heap leaching, NOVAMINORE technology is conducted at ambient temperature.

In 2024, a testwork program was conducted by Nova Mineralis (at Geomet laboratory) and Aminpro laboratory, both based in Santiago, Chile, to determine the effectiveness of NOVAMINORE technology on oxide and transitional samples from Productora and Cortadera.

For the Productora deposit, 35 1 m column leaching tests were conducted simultaneously as a part of this program - 29 tests using the NOVAMINORE methodology and six tests using conventional leaching techniques. Samples were selected to represent the mineralogical variability of the various alteration domains in both oxide and transitional material. The six duplicated NOVAMINORE and conventional leaching tests were run using oxide material.

For the Cortadera deposit, ten 1 m column leaching tests were conducted simultaneously - eight tests using the NOVAMINORE methodology and two tests using conventional leaching techniques. Samples were also selected to represent mineralogical oxide and transitional material. The two duplicated NOVAMINORE and conventional leaching tests were run using oxide and transitional material.

The conditions of these tests are shown in Table 13.28, while pictures represent the mini columns, with conventional technology and NOVAMINORE technology (Figure 13.20) and the NOVAMINORE 1 m Column Leaching Test (Figure 13.21).

The conventional technology used in the PEA study's backup tests, which used seawater and acid curing, was compared to the hypersaline NOVAMINORE technology applied in the PFS study.

Figures 13.21 and 13.22 show the Cu extraction and net acid consumption of the comparative mini-column tests.

Table 13.28 Tests Conditions Used in 2024 1-m Columns for NOVAMINORE and Conventional Methods

Condition	Units	NOVAMINORE Technology	Conventional Technology
General Conditions			
Column Size	-	1 m height x 20.3 cm (8 inches) diameter	1 m height x 20.3 cm (8 inches) diameter
Sample Particle Size Distribution	-	P_{100} -1.27 cm (0.5 inches) 65%-70% -0.95 cm (0.38 inches)	P_{100} 1.27 cm (0.5 inches) 65%-70% 0.95 cm (0.38 inches)
Sample Mass Requirement per Test	kg	45	45
Curing Stage Conditions			
Acid Addition (Oxide Samples)	kg/t	10-20	10-20
Acid Addition (Transitional Samples)	kg/t	15-25	15-25
Sodium Chloride (Salt)	kg/t	15	-
Artificial RF Solution	kg/t	70-90	70-90
Resting Time	days	10	3
Operational (Irrigation) Conditions			
Irrigation Regime	-	Six intermittent cycles	Continuous
Process Time	days	90	90
Irrigation Rate	L/hr*m^2	10	10
Leaching Rate	m^3/t	3-4	14-15
Temperature	°C	Room	Room
Irrigation Solution			
H^+	g/l	10	5
Cu^{2+}	g/l	0.5	0.5
Fe^{2+}	g/l	5	5
Cl^-	g/l	80-120	20
Mg^{2+}	g/l	5	5
Al^{3+}	g/l	5	5

Note: Nova Mineralis' ore leaching technology considers various temperature ranges during operation. For copper concentrates, the preferred temperature is around 45°C, although for chalcopyrite-rich ores, the most appropriate temperature is only 35°C.

The situation is different when processing oxide and transitional minerals, as the kinetics of copper dissolution in this case depend on the leaching practices, not the temperature. This is why all column leaching tests (Nova technology and conventional leaching tests) were conducted at room temperature, which meant a temperature between 15°C and 25°C inside the place where the tests were performed.

Furthermore, all columns (Nova technology tests and conventional leaching tests) were protected with thermal insulation material, isolating the ambient conditions from what would occur inside the columns. This meant that in all cases, the temperatures inside the columns ranged between 20 and 25°C.

It should be noted that the environmental conditions of the project site allow for higher temperatures than those indicated to be expected, so, if there is any influence on the results, it can be considered that such results can be classified as conservative from the point of view of the temperatures that may be reached.

Figure 13.20 From Left to Right: Conventional Leach Column and NOVAMINORE Column

Source: PMC, 2024.

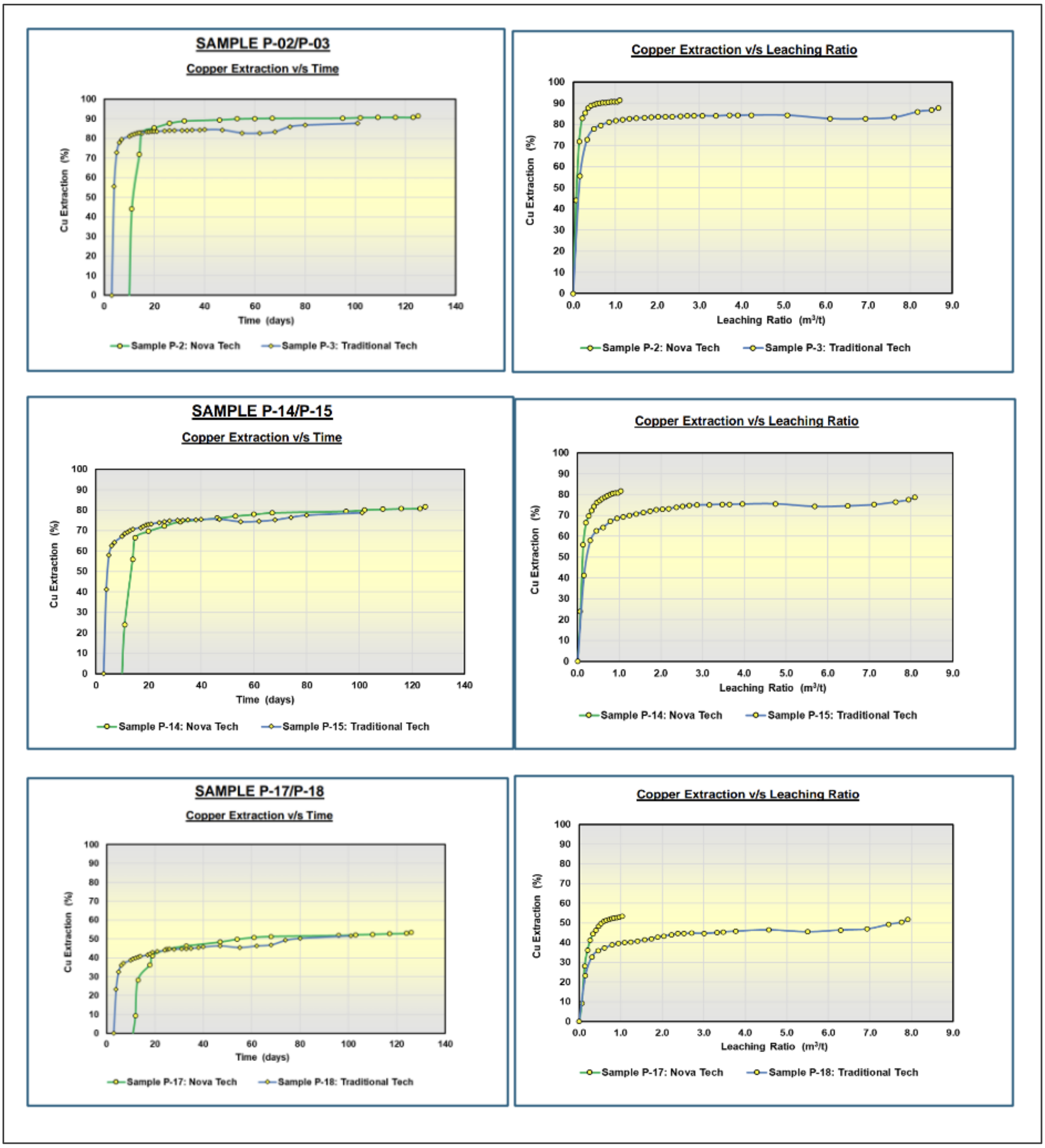
Figure 13.21 NOVAMINORE 1 m Column Leaching Test

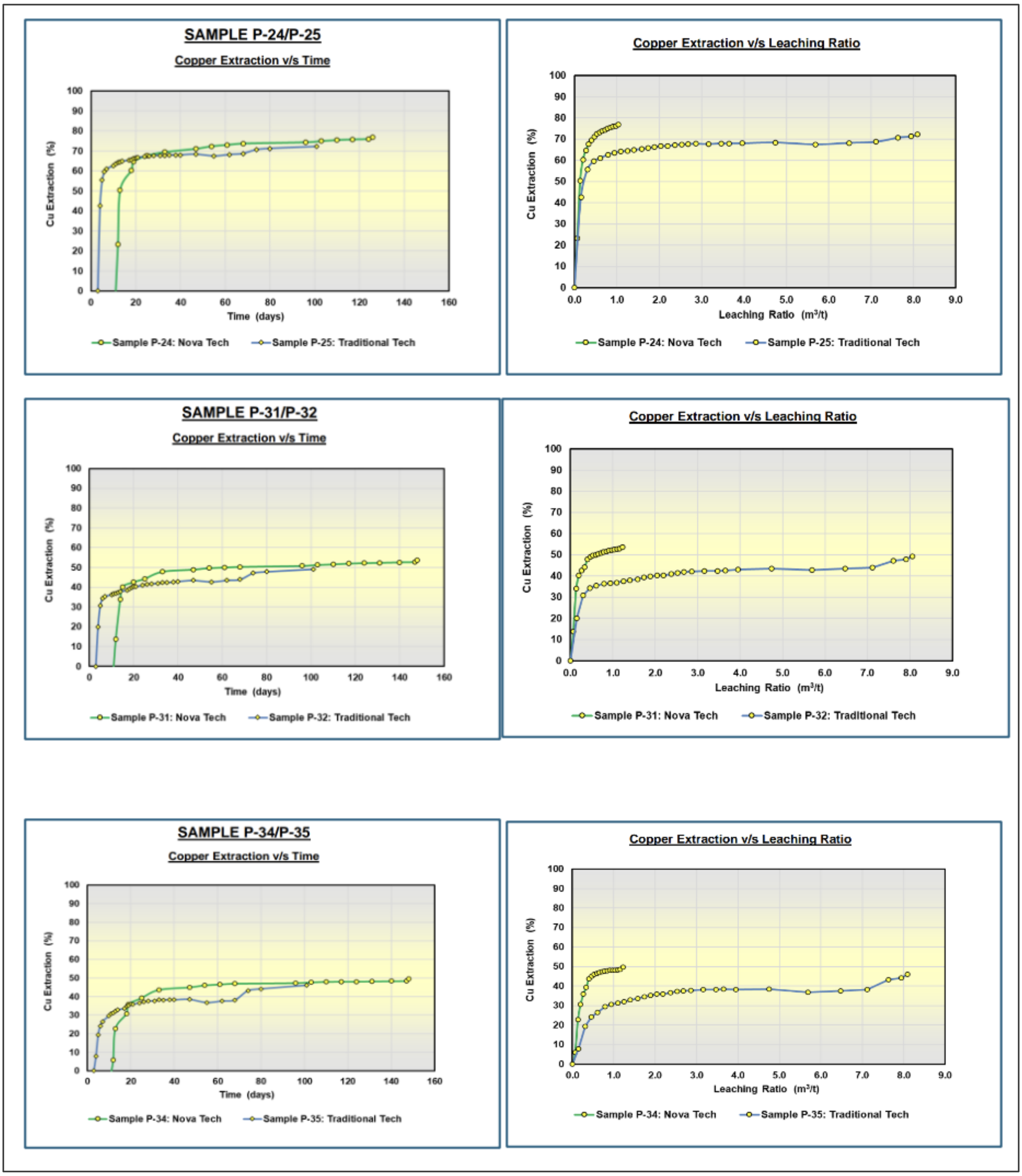
Source: PMC, 2024.

The results of comparing the NOVAMINORE leaching program and conventional leaching are shown below for Productora and Cortadera.

Productora:

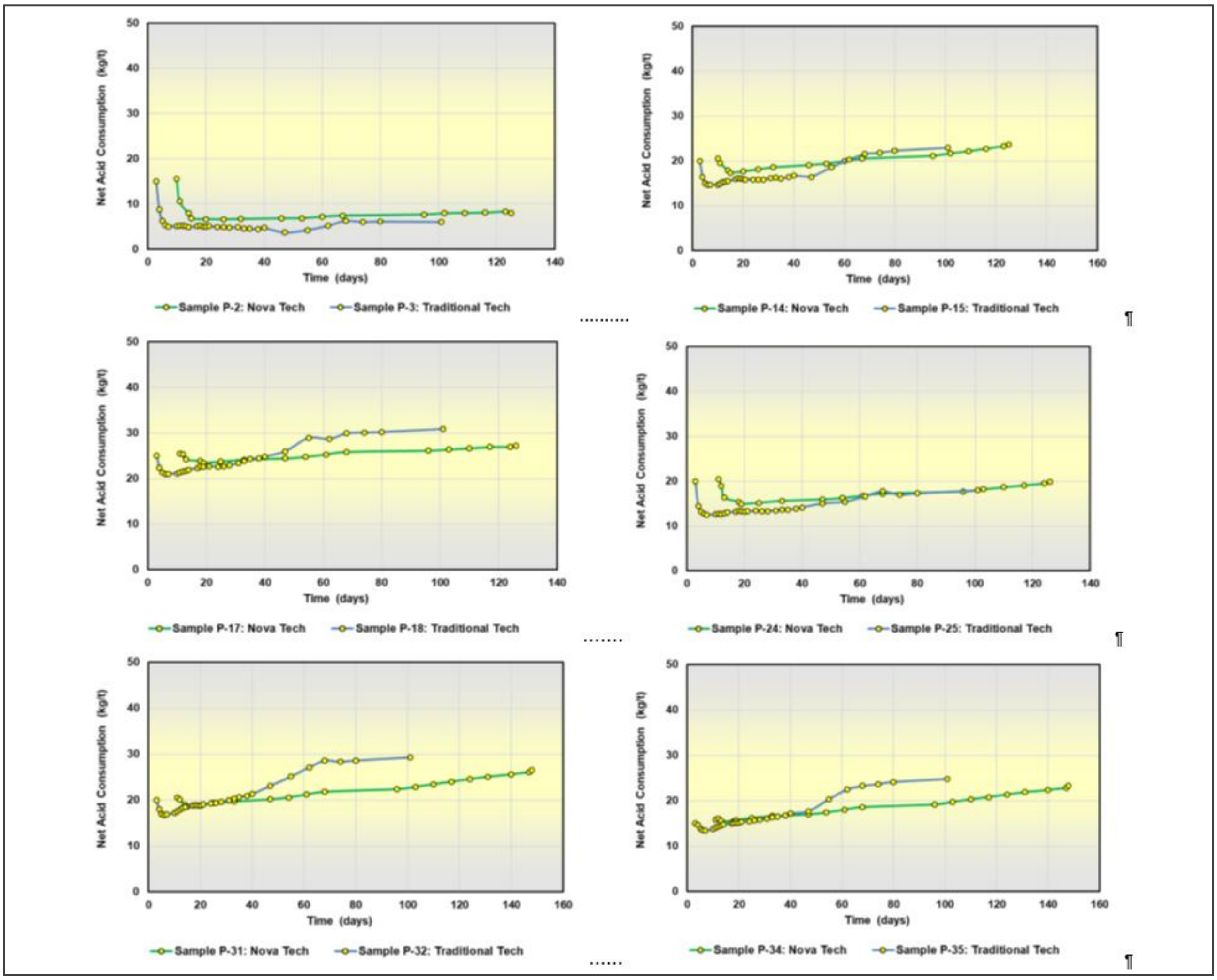
Figure 13.22 Productora Copper Extraction for Simultaneous Duplicate NOVAMINORE and Conventional Column Leaching Test





According to the results of the testwork completed on Productora samples, copper recovery appears equal or higher when NOVAMINORE technology was used versus conventional. Regarding the leaching ratio (m3/t), there is a more efficient use of the irrigation solution, resulting in higher PLS Cu concentration with less solution consumption. There is also an opportunity for other elements to be leached, such as cobalt (more information in Section 24).

Figure 13.23 Productora Net Acid Consumption for Simultaneous Duplicate NOVAMINORE and Conventional Column Leaching Test



According to the results of the testwork completed on Productora samples, acid consumption appears to be similar for initial leaching time when NOVAMINORE technology was used versus conventional, but it appears higher for conventional after initial leaching time.

Cu extraction by tails assay balance.

The summary Table 13.29 presents the Cu recovery obtained by tails assay after the column discharge and the recovery by solution presented in the previous graphs.

Table 13.29 Summary Productora column tests

Sample Name	Technology Hypersaline or Conventional	Column Test Number	Processing Days	Head Cu	Head CuS	Copper Recovery (%) by tails	Net Acid Consumption (kg/t)	Copper Recovery (%) by solution
P-01	Hypersaline	COL-50	125	0.31	0.24	83	17	75
P-02	Hypersaline	COL-51	125	0.55	0.52	91	8	93
P-03	Conventional	COL-82	101	0.48	0.46	88	6	78
P-04	Hypersaline	COL-52	172	1.73	1.64	93	-1	90
P-05	Hypersaline	COL-53	172	1.54	1.49	94	10	94
P-06	Hypersaline	COL-54	125	0.39	0.22	75	23	74
P-07	Hypersaline	COL-55	172	1.01	0.79	81	22	77
P-08	Hypersaline	COL-56	125	0.32	0.24	88	18	85
P-09	Hypersaline	COL-57	125	0.77	0.11	94	7	90
P-10	Hypersaline	COL-58	148	0.97	0.16	92	14	95
P-11	Hypersaline	COL-59	125	0.19	0.06	61	27	57
P-12	Hypersaline	COL-60	148	0.63	0.35	77	17	74
P-13	Hypersaline	COL-61	148	0.61	0.36	85	22	80
P-14	Hypersaline	COL-62	125	0.40	0.19	82	24	81
P-15	Conventional	COL-83	101	0.42	0.20	79	23	82
P-16	Hypersaline	COL-63	125	0.54	0.32	65	28	66
P-17	Hypersaline	COL-64	126	0.43	0.17	53	27	51
P-18	Conventional	COL-84	101	0.47	0.18	52	31	54
P-19	Hypersaline	COL-65	126	0.19	0.05	45	27	44
P-20	Hypersaline	COL-66	126	0.12	0.03	34	28	31
P-21	Hypersaline	COL-67	126	0.29	0.09	42	34	40
P-22	Hypersaline	COL-68	148	0.25	0.13	43	36	39
P-23	Hypersaline	COL-69	148	0.78	0.14	82	19	80
P-24	Hypersaline	COL-70	126	0.58	0.36	77	20	74
P-25	Conventional	COL-85	101	0.61	0.38	72	18	73
P-26	Hypersaline	COL-71	126	0.54	0.19	59	24	58
P-27	Hypersaline	COL-72	126	0.15	0.08	49	21	45
P-28	Hypersaline	COL-73	126	0.29	0.17	37	30	35
P-29	Hypersaline	COL-74	148	0.47	0.27	62	26	59
P-30	Hypersaline	COL-75	126	0.64	0.45	37	23	37
P-31	Hypersaline	COL-76	148	0.46	0.25	54	26	49
P-32	Conventional	COL-86	101	0.51	0.27	49	29	49
P-33	Hypersaline	COL-77	148	0.47	0.29	39	23	40
P-34	Hypersaline	COL-78	148	0.30	0.17	50	23	47
P-35	Conventional	COL-87	101	0.34	0.19	46	25	49

Table 13.30 presents the summary between hypersaline leach versus conventional leach at the top, and summary of Cu recovery (extraction) for Hypersaline technology (NOVAMINORE), calculated by tails versus solutions, at the bottom. Net acid consumption is included for reference.

Table 13.30 Summary Productora comparative column leach tests

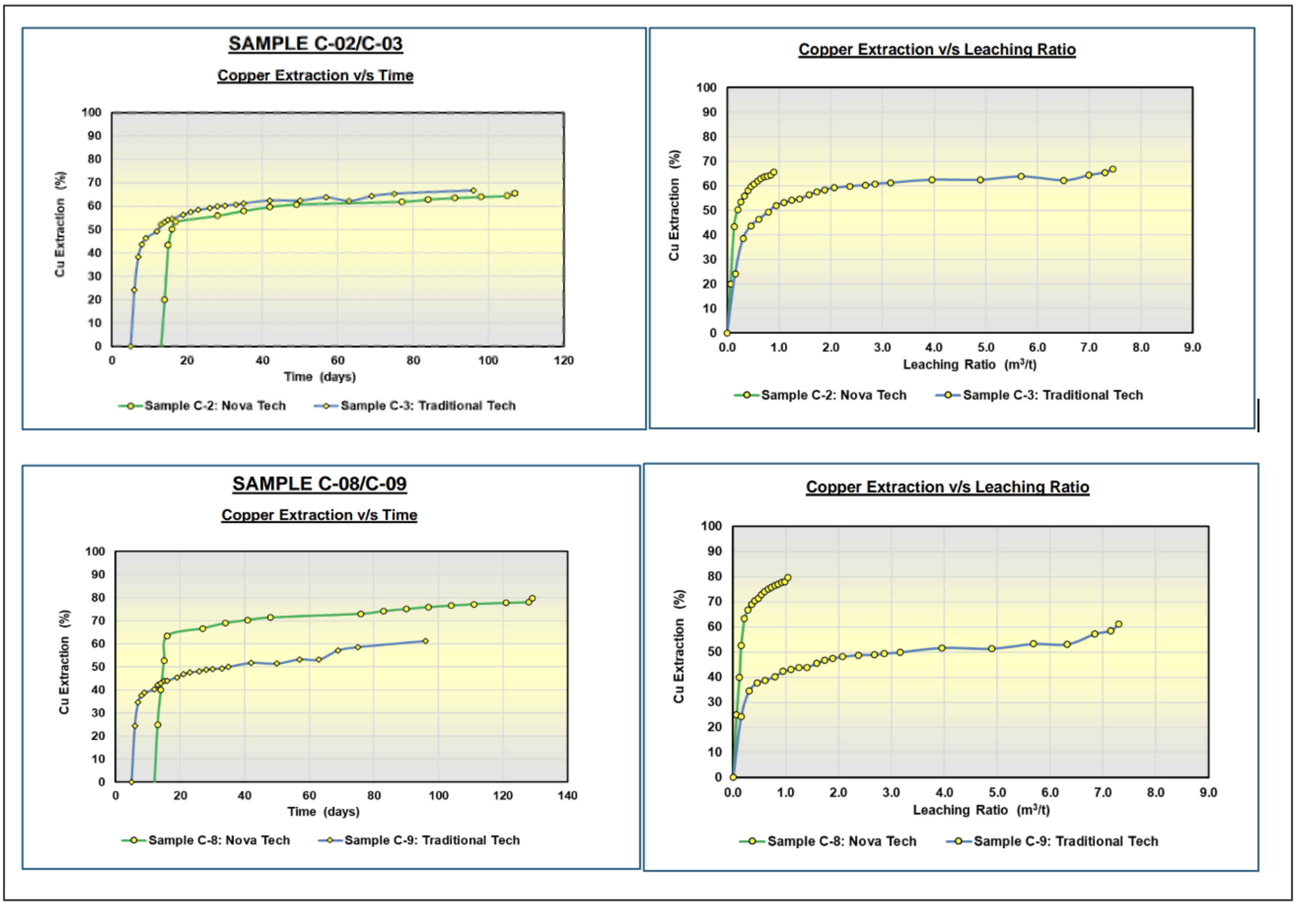
Summary Hypersaline vs Conventional Leaching	Cu Rec.(%)	Net acid kg/t
Average Hypersaline leach by tailings balance (6 samples)	68	21
Average Conventional leach by tailings balance (6 samples)	64	22

Summary Hypersaline by Tails vs solutions	Cu Rec.(%)	Net acid kg/t
Average recovery by tails balance (29 samples)	66	22
Average recovery by solutions balance (29 samples)	64	22

The data presented show better recovery and acid consumption performance for the hypersaline process. Also, the Cu recovery by tails in the column test shows better results than the preliminary results obtained by solution balance. These results positively affect 4% of the recovery model developed preliminarily. This behaviour is similar to the column test for the low-grade ore tests presented ahead in the dump leaching tests section.

Cortadera:

Figure 13.24: Cortadera Copper Extraction for Simultaneous Duplicate NOVAMINORE and Conventional Column Leaching Test



According to the results of the testwork completed on Cortadera samples, copper recovery appears to be equal or higher when NOVAMINORE technology was used versus conventional. Again, in this case, in terms of the leaching ratio (m^3/t), there is a more efficient use of the irrigation solution, and there also seems to be an opportunity for other elements to be leached (See Section 24)

Acid consumption also appears to be similar in the initial leaching time when NOVAMINORE technology was used versus conventional. Still, it is very high for the conventional process after the tests continued for longer.

Cu Extraction by Tails Assay Balance

The summary Table 13.31 presents the Cu recovery obtained by tails assay after the column discharge. It also includes recovery by solution as presented in the previous graphs.

Table 13.31 Summary Cortadera column tests

Sample Name	Technology Hypersaline or Conventional	Column Test Number	Processing Days	Head Cu	Head CuS	Copper Recovery (%) by tails	Net Acid Consumption (kg/t)	Copper Recovery (%) by solution
C-1 Oxide	Hypersaline	COL-24	107	0.24	0.11	8	34	8
C-02 NM	Hypersaline	COL-25	107	0.36	0.21	68	25	67
C-03 LT	Conventional	COL-88	96	0.38	0.22	67	40	69
C-4 Oxide	Hypersaline	COL-26	130	0.29	0.11	74	22	76
C-5 Transitional	Hypersaline	COL-27	130	0.27	0.21	15	49	13
C-6 Transitional	Hypersaline	COL-28	130	0.29	0.12	77	27	72
C-7 Transitional	Hypersaline	COL-29	154	0.26	0.06	72	22	72
C-8 NM	Hypersaline	COL-32	129	0.25	0.11	80	21	81
C-9 LT	Conventional	COL-89	96	0.26	0.11	61	22	63
C-10 Transitional	Hypersaline	COL-33	129	0.27	0.11	71	27	68

Table 13.32 presents the summary between hypersaline leach versus conventional leach at the top, and summary of Cu recovery (extraction) for Hypersaline technology (NOVAMINORE), calculated by tails versus solutions, at the bottom. Net acid consumption is included for reference.

Table 13.32 Summary Cortadera comparative column leach tests

Summary Hypersaline vs Conventional	Cu Rec.(%)	Net acid kg/t
Average Hypersaline leach	73.6	23
Average Conventional leach	63.9	31

Summary Recovery by tails vs solutions	Cu Rec.(%)	Net acid kg/t
Average recovery by tails balance	59	29
Average recovery by solutions balance	59	29

These results confirm the hypersaline process's better performance, with higher recovery and less acid consumption. Regarding the Cu recovery, the tails and solution methods were the same, although across a small number of samples to compare.

Mineralogy of tested samples

Figure 13.25 and Figure 13.26 below present the mineralogy of two representative samples of the oxide and transition minerals of the Productora deposit.

- The main minerals in the oxide sector are chrysocolla and malachite, while gangue is quartz, feldspar and plagioclase.

- The main minerals of the transition sector are chrysocolla, malachite, limonite and copper wad, while the gangue is equal to the oxide zone.

Likewise, Figure 13.27 Figure 13.27 and Figure 13.28 present the mineralogy of two representative transition and sulphide samples from the Cortadera deposit.

- The main minerals of the transition sector are Cu Pitch, limonite, chlorite, and Cu Wad, although the MO-13 sample was initially classified as oxide. The gangue consists of quartz, muscovite, and plagioclase.
- The main mineral of the sulphide sector is chalcopyrite, while the gangue is similar to the transition zone.

Figure 13.25 Oxide Productora sample

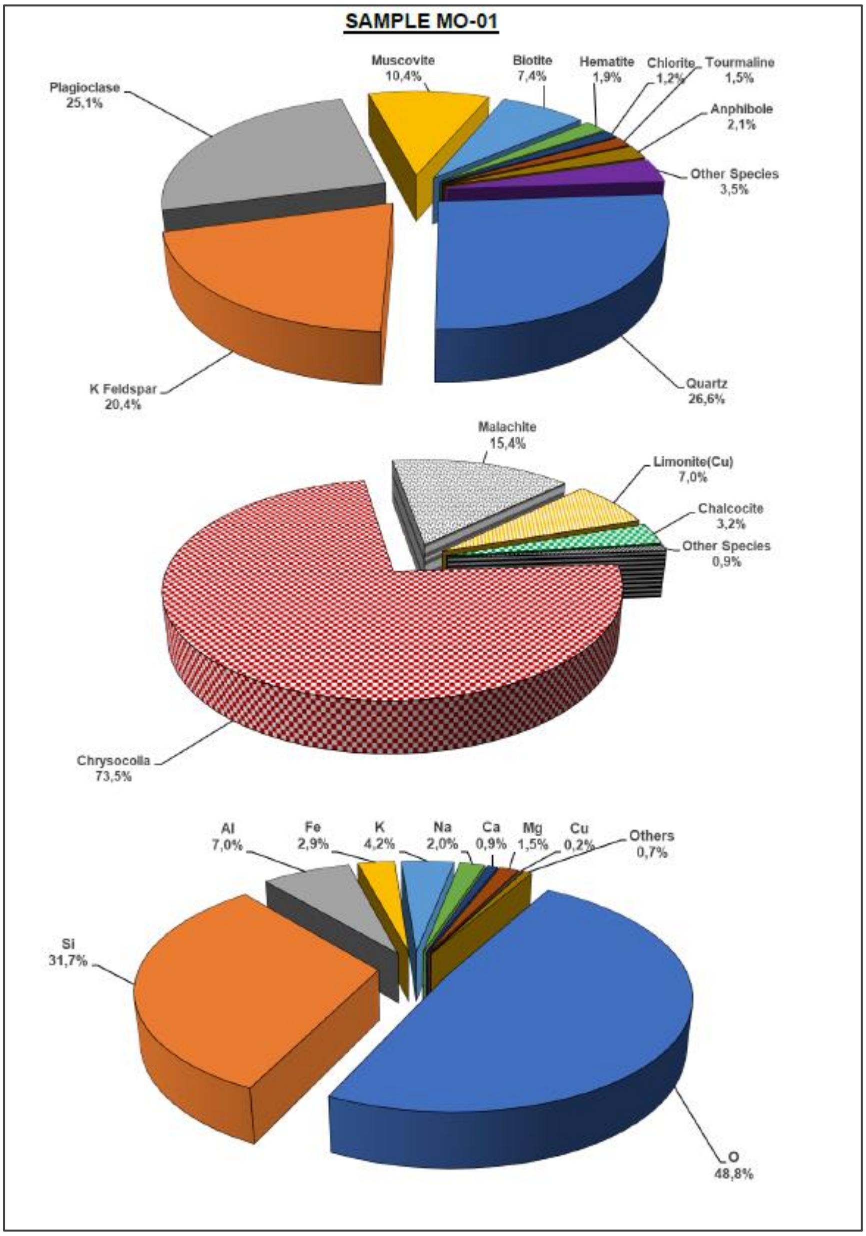


Figure 13.26 Transition Productora sample

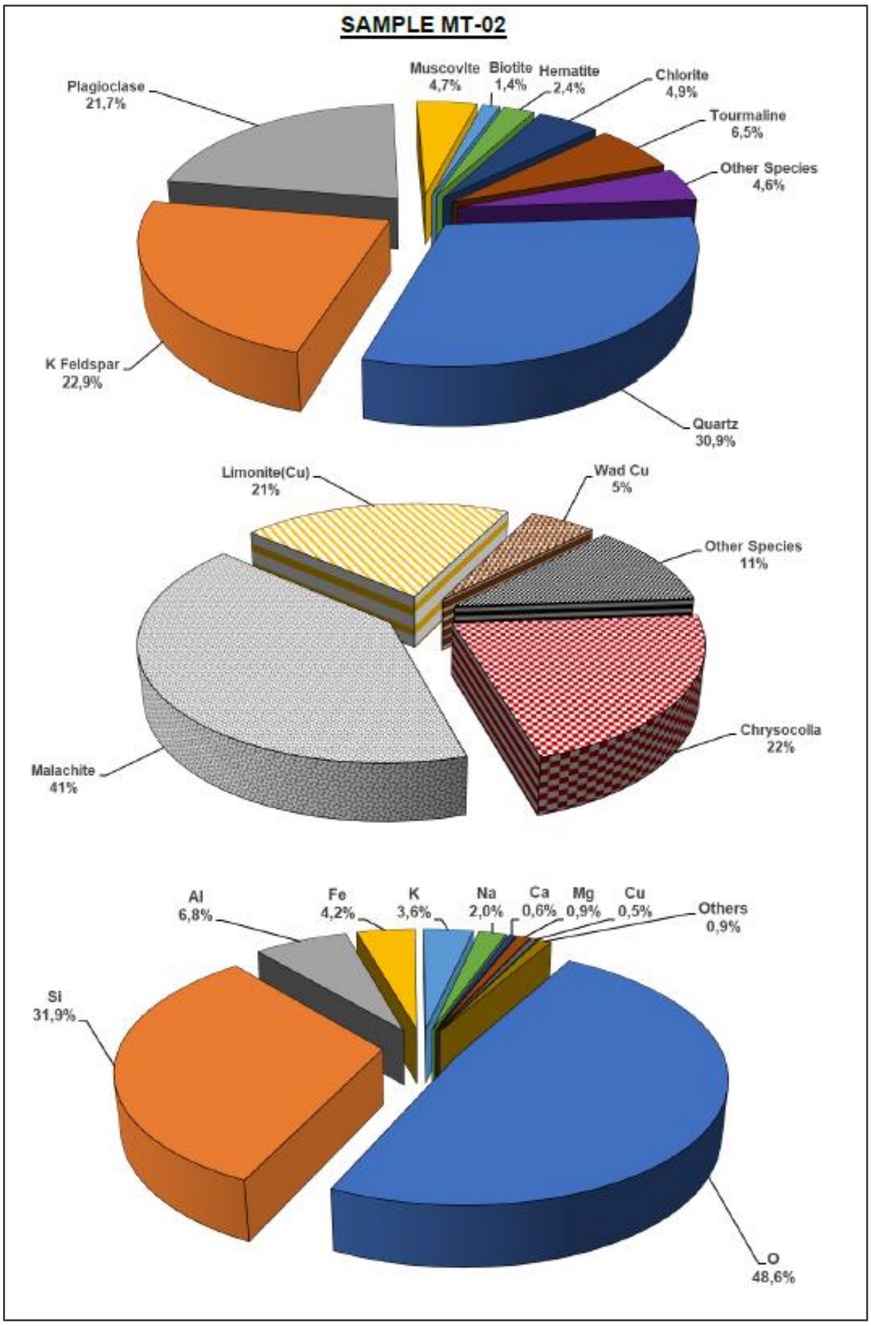


Figure 13.27 Transition Cortadera sample

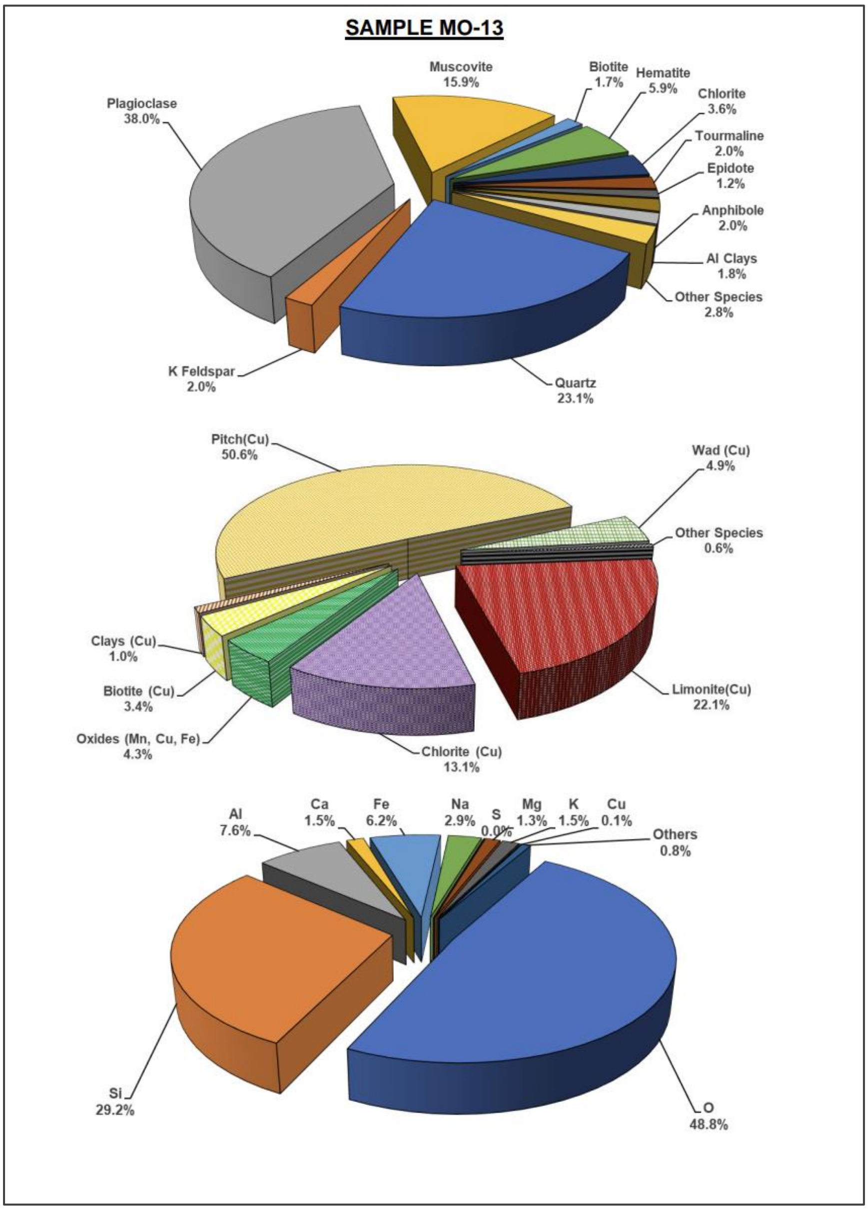
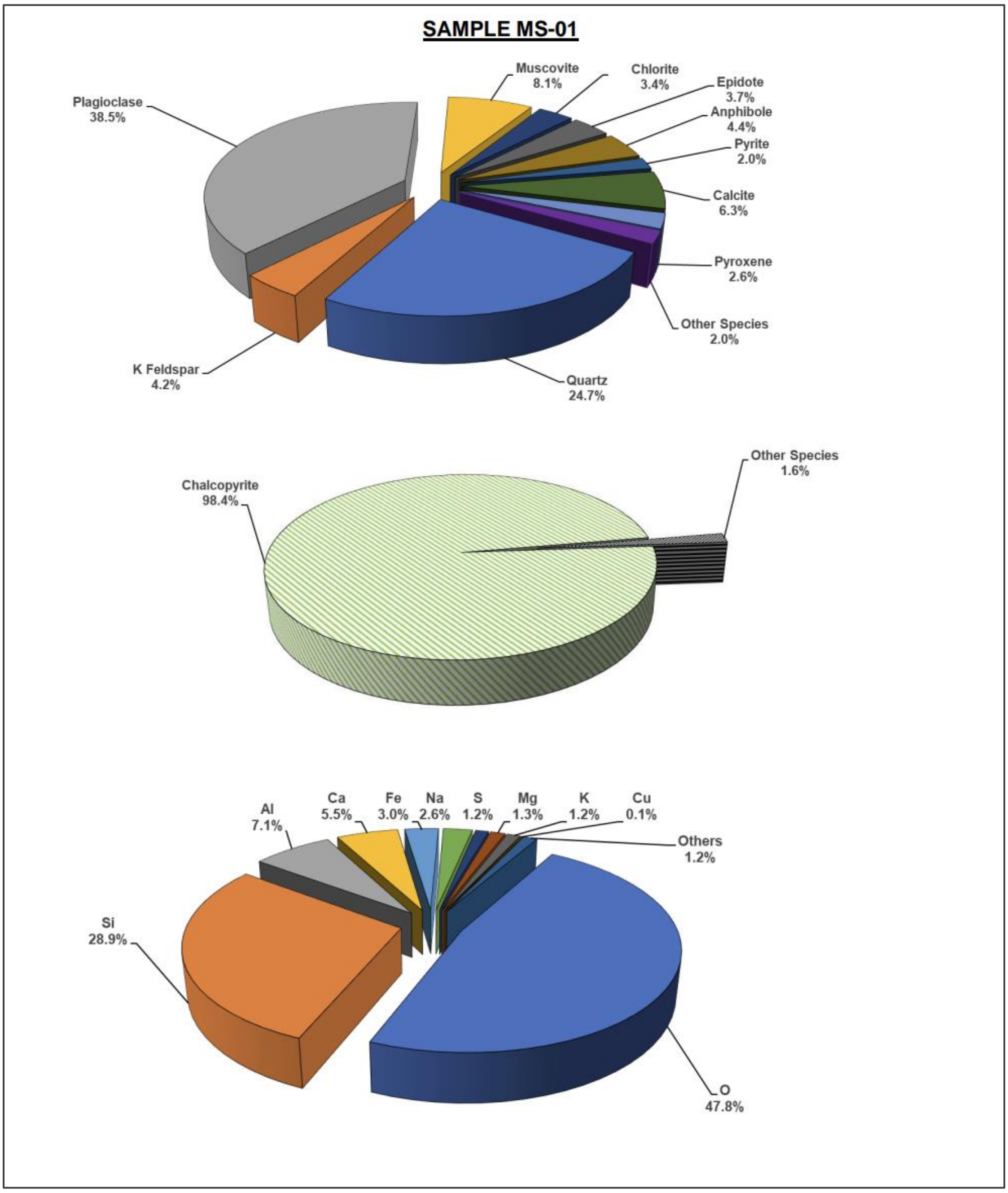


Figure 13.28 Sulphide Cortadera sample



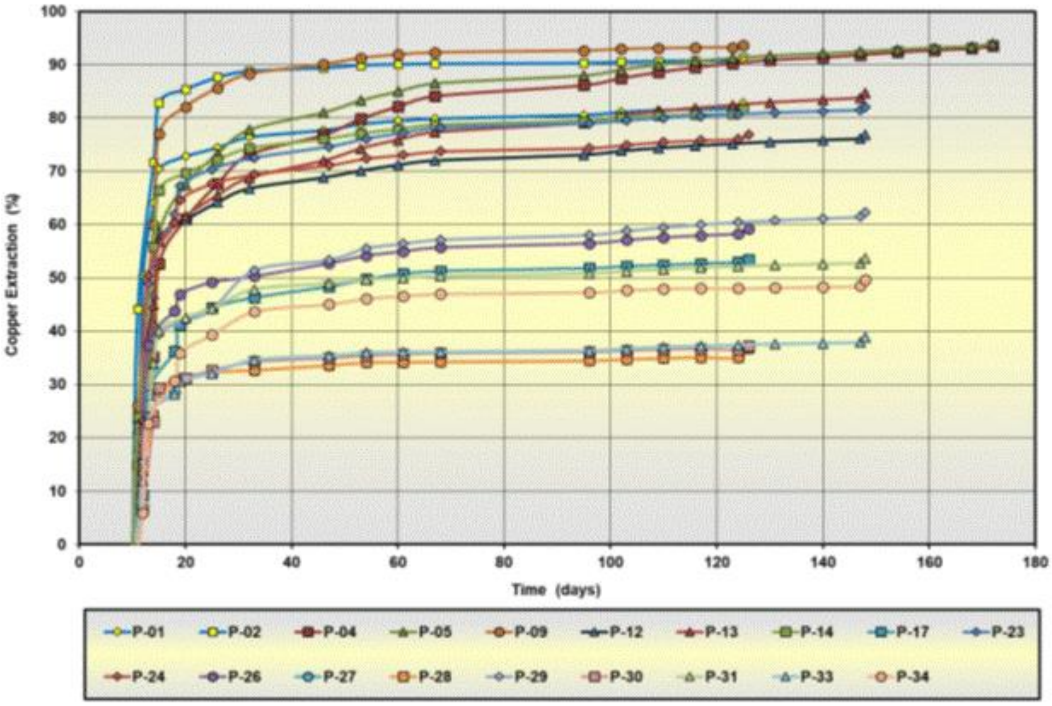
Variability column tests

The full 1 m NOVAMINORE variability leaching program results for Productora and Cortadera are shown below.

Figure 13.29 and Figure 13.30 present the kinetic Cu recovery curves for 120 leaching days for oxide and transitional Productora samples.

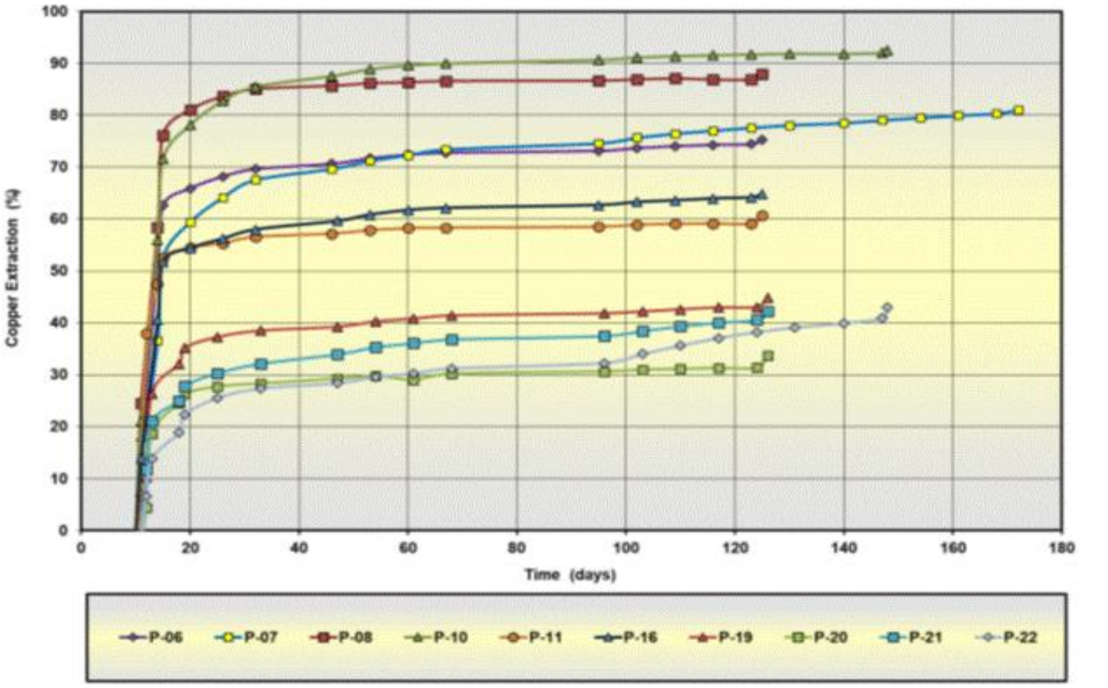
Productora: Oxide mineral samples.

Figure 13.29 Productora Copper Extraction for NOVAMINORE Oxide Column Leaching Test



Productora: Transitional mineral samples

Figure 13.30 : Productora Copper Extraction for NOVAMINORE Transitional Column Leaching Test

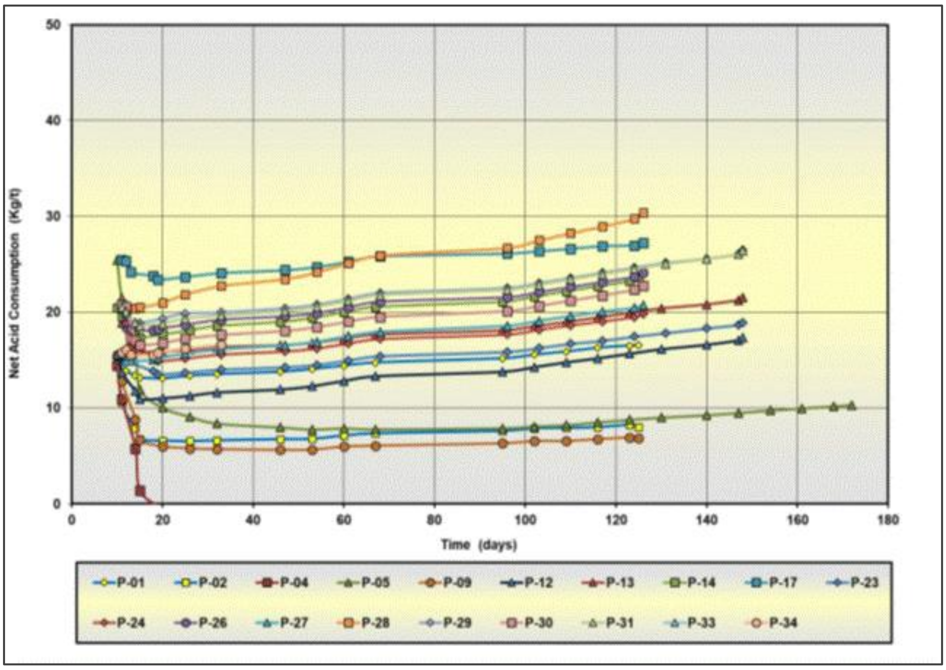


According to the results of the testwork completed on Productora samples considering oxide and transitional mineral samples, copper extraction varied from about 10% to more than 95% after about 120 days of leaching time. These results represent the variability of the deposit and were used to develop copper recovery predictive models for the Productora mine.

Figure 13.31 and Figure 13.32 present the acid consumption curves for 120 leaching days for Productora samples.

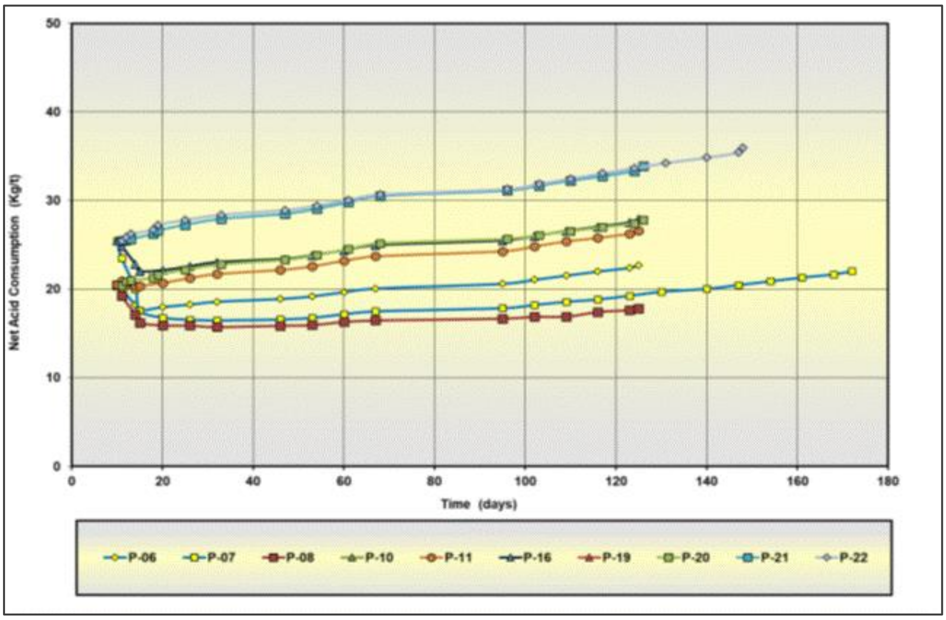
Productora: Oxide mineral samples.

Figure 13.31 Productora Net Acid Consumption for NOVAMINORE Oxide Column Leaching Test



Productora: Transitional mineral samples

Figure 13.32: Productora Net Acid Consumption for NOVAMINORE Transitional Column Leaching Test

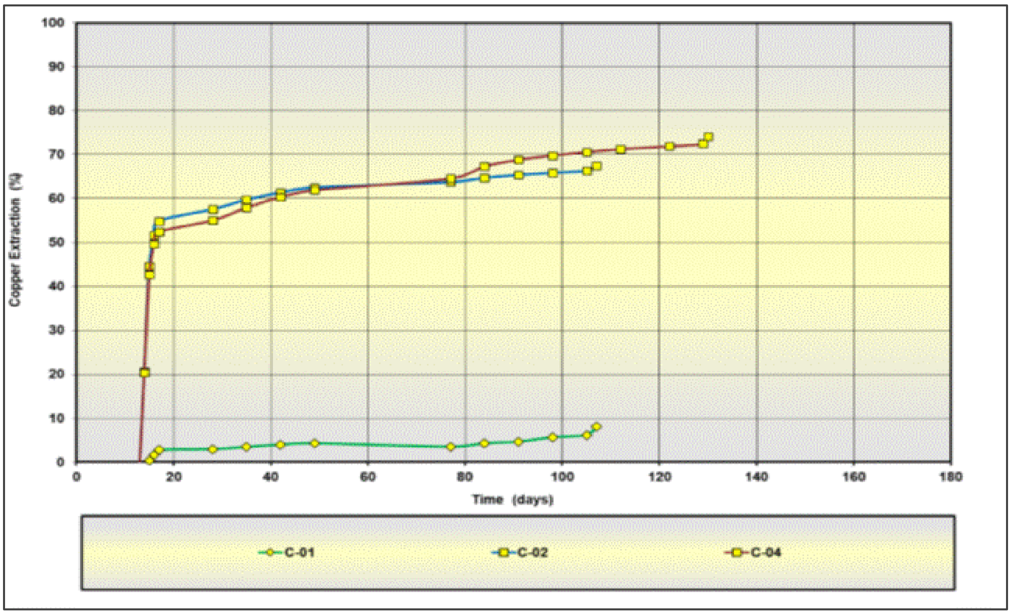


According to the results of the testwork completed on Productora samples considering oxide and transitional mineral samples, net acid consumption varied from less than 10 kg/t to less than 35 g/t after about 120 days of leaching time. Again, these results represent the variability of the deposit and were used to develop net acid consumption predictive models for the Productora deposit.

Figure 13.33 and Figure 13.34 present the Cu recovery curves for 120 leaching days for Cortadera samples.

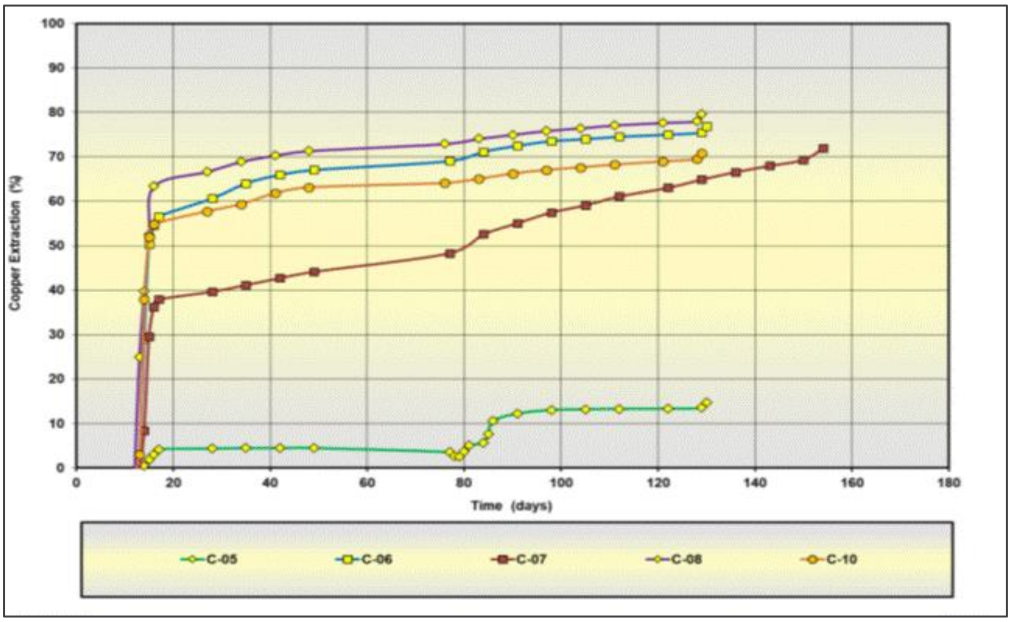
Cortadera: Oxide mineral samples

Figure 13.33 : Cortadera Copper Extraction for NOVAMINORE Oxide Column Leaching Test



Cortadera: Transitional mineral samples

Figure 13.34 : Cortadera Copper Extraction for NOVAMINORE Transitional Column Leaching Test

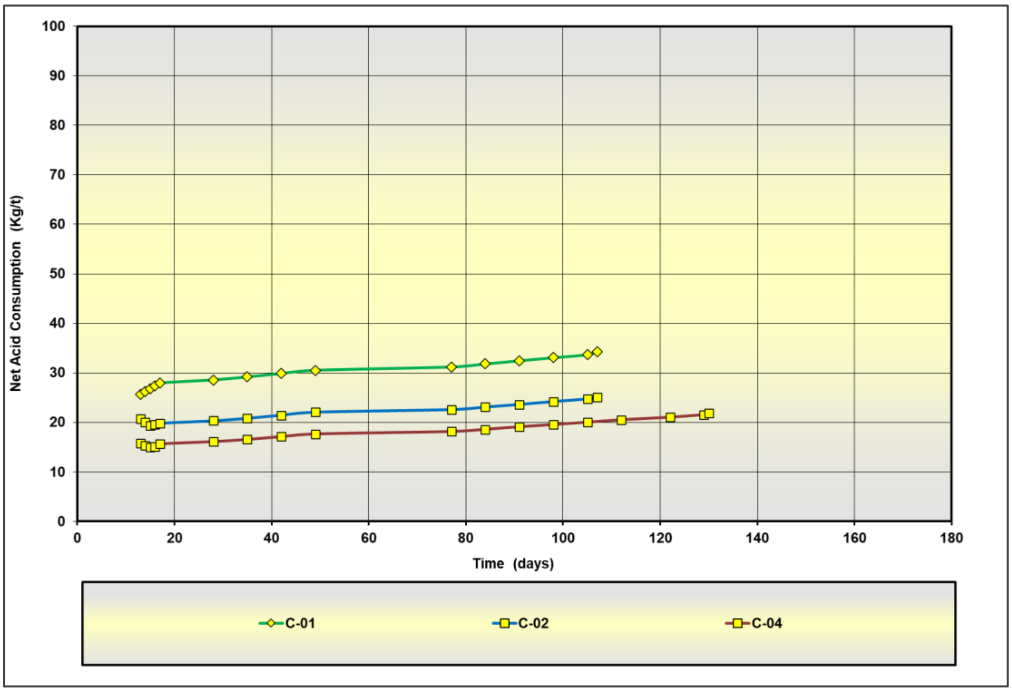


According to the results of the testwork completed on Cortadera samples, considering oxide and transitional mineral samples, copper extraction varied from about 5% to more than 70% after about 90 days of leaching time. These results represent the variability of the deposit and were used to develop copper recovery predictive models for the Cortadera deposit.

Figure 13.35 and Figure 13.36 present the acid consumption curves for 120 leaching days for Cortadera.

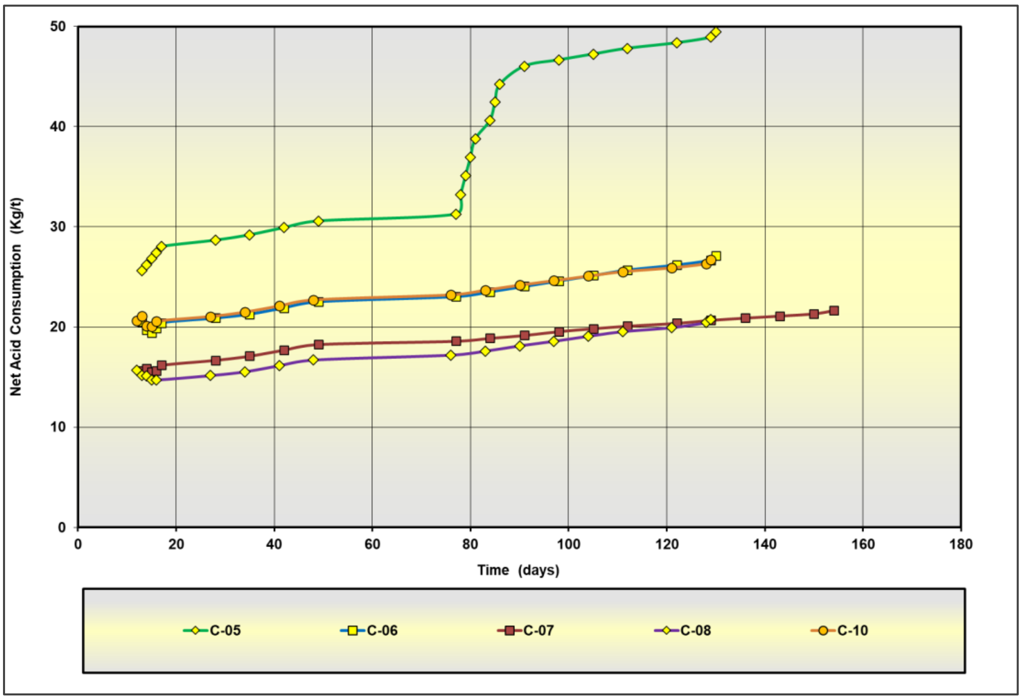
Cortadera: Oxide mineral samples.

Figure 13.35 Cortadera Net Acid Consumption for NOVAMINORE Oxide Column Leaching Test



Cortadera: Transitional mineral samples.

Figure 13.36 Cortadera Net Acid Consumption for NOVAMINORE Transitional Column Leaching Test



According to the results of the testwork completed on Cortadera samples, considering oxide and transitional mineral samples, net acid consumption varied from about 20 kg/t to about 40 kg/t after about 90 days of leaching time. Again, these results represent the variability of the deposit and were used to develop net acid consumption predictive models for the Cortadera deposit.

13.4.2.3 Predictive Models

The following sections show the predictive leach models for both Cortadera and Productora. They also include metal recovery and acid consumption.

Table 13.33 Predictive Models used for Copper Recovery and Acid Consumption

	Productora	Cortadera
Copper Recovery	Predicted Cu Recovery % = (58.209 * Cu %) + (-7.5857 * Na %) + 51.5024	Predicted Copper Recovery % = (-2.2209 * Co ppm) + 93.58

	Productora	Cortadera	
Acid Consumption	Predicted Net Acid consumption (kg/t) = $10^{((0.267 * LOG_{10}(Ca\ \%)) + 1.3755)}$	Geometallurgical Domain	Acid Consumption Value (kg/t)
		Andesitic Volcaniclastic	21
		Apatite Calcsilicate	20
		Apatite Carbonate	60
		Calcsilicate	36
		Carbonate	56
		Porphyry Intrusion	25

Preliminary results of the ongoing 2024 heap leach columns test work were supplied to HCH by PMC on the 24th of November 2024. At the time of this review, the Productora columns had been leached for ~60 days and Cortadera columns for ~42 days. PMC analysed the data to determine the number of days for projecting the leaching period. For each column, the copper recovery and acid consumption were projected to 240 days by extrapolating the initial test results using a series of equations. For columns where Cu extraction occurred quickly, PMC recommended reducing the leaching time for some samples. Therefore, PMC suggested a variable leaching period of up to 240 days.

To facilitate a comparison of the Nova Mineralis and traditional heap leach technologies, several duplicate samples have undergone column leaching by both methods (two at Cortadera and six at Productora). Due to the different techniques, the regressions below have not included columns using the traditional leach methodology.

The memo referenced presents for review a set of new models for predicting copper recovery (Productora and Cortadera) and acid consumption (Productora only). Acid consumption values have been assigned at Cortadera by the metallurgical domain.

The metal recovery and, therefore, acid consumption are for Cortadera open-pit material. There is no oxide material in Cortadera underground.

13.4.3 Dump Leach Testwork

13.4.3.1 Introduction

This section summarises the testwork and results that were used to identify the technical and economic evaluation of the Costa Fuego Project samples to NOVAMINORE dump leaching processes, as well as inputs for the design of the dump leach facility.

In 2022, a conceptual study into the dump leaching of primary low-grade production material was conducted. The study aimed to assess the amenability of acid dump leaching via the application of NOVAMINORE technology to low-grade material.

Following the initial conceptual study, variability and scaling studies were completed at both Productora and Cortadera.

13.4.3.2 2022 Conceptual Study Micro-column Leach Tests – Nova Mineralis

In 2022, a conceptual study testwork program was conducted by Nova Mineralis to evaluate the amenability of low-grade primary feed materials from the Productora and Cortadera deposits to NOVAMINORE technology. Micro column leach tests were completed on both Productora and Cortadera samples. There were four Productora samples, two Cortadera samples and one blended Cortadera/Productora sample. The locations of the samples used in the conceptual study are displayed in Figure 13.39. Samples were obtained from both drill core and underground.

Samples were crushed to P_{100} of 1.27 cm (0.5 inches), the micro columns were 30.5 cm in height by 10.2 cm in diameter, and the testwork was completed over 140 days with 15 irrigation cycles (each cycle was followed by a 7-day rest period).

Copper recovery and acid consumption kinetic data are displayed below (Figure 13.37 and Figure 13.38). In these figures, circles are used for Productora data points, and squares are used for Cortadera data points and triangles are used for mixed Productora/Cortadera data points.

Figure 13.37 Copper Recovery Kinetics for Nova Mineralis Conceptual Study

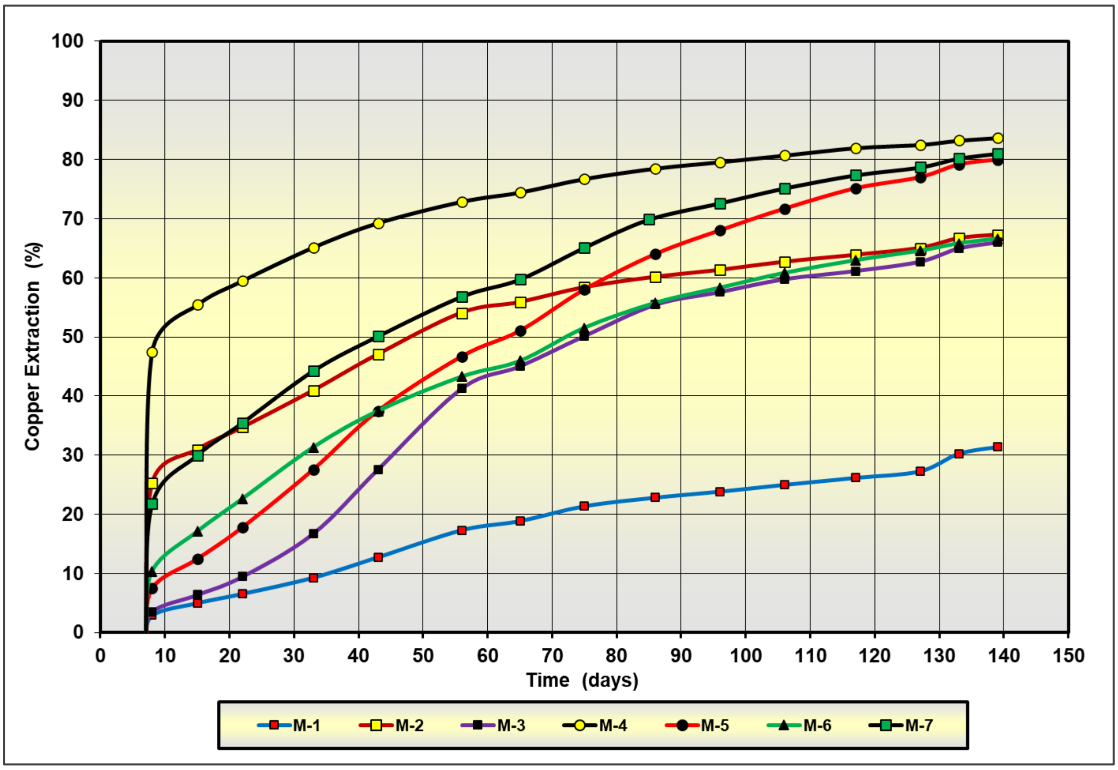


Figure 13.38 Acid Consumption Kinetic for Nova Mineralis Conceptual Study

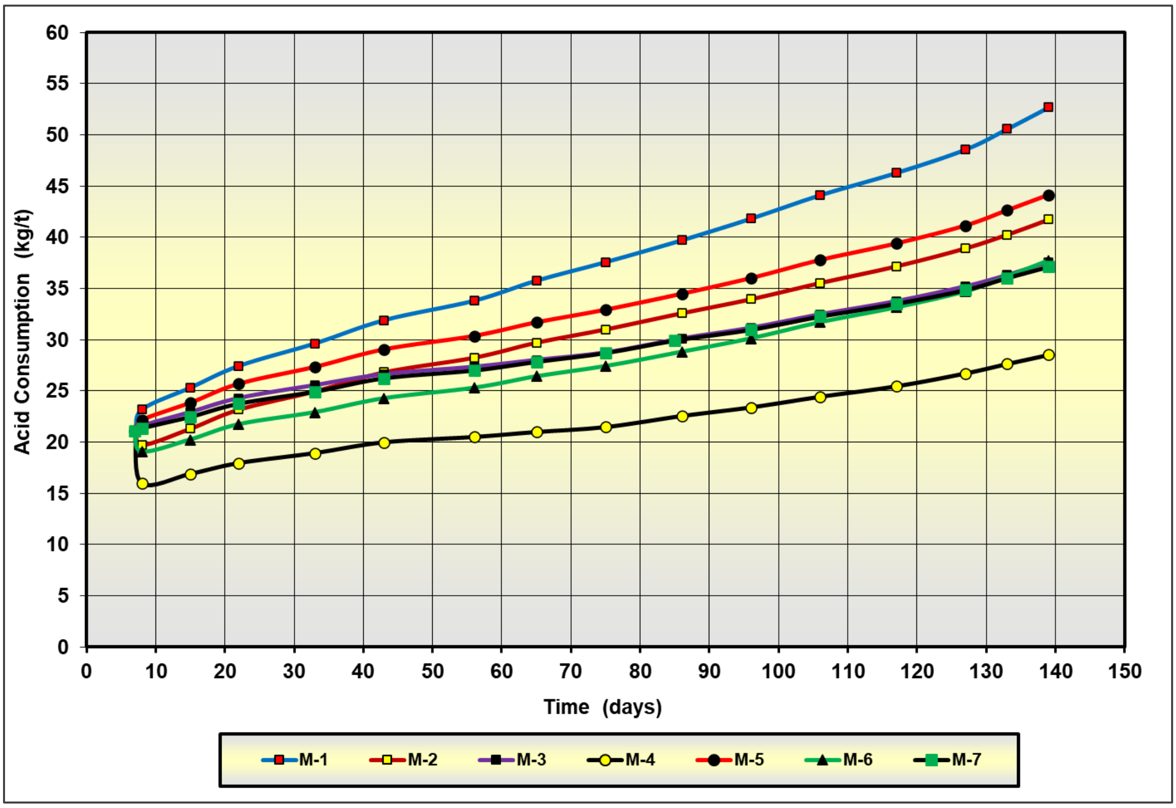


Table 13.34 shows the Cu recovery from the Nova Mineralis micro-column tests by solution and tails methods.

Table 13.34 Results from Nova Mineralis Micro Column Tests

Sample Name	Deposit	Column Test Number	Copper Recovery (%)		Net Acid Consumption (kg/t)
			Analysed Head Basis by solution (%)	Calculated Head Basis by tails (%)	
M-1	Cortadera	MCol 1	32.0	31.4	53
M-2	Cortadera/Productora blend	MCol 2	68.5	67.3	42
M-3	Cortadera	MCol 3	74.6	66.0	38
M-4	Productora	MCol 4	80.2	83.7	29
M-6	Productora	MCol 6	83.1	80.0	44
M-7	Productora	MCol 7	72.0	66.6	38
M-8	Productora	MCol 8	86.6	81.0	37

13.4.3.2.1 Comments on Microcolumns Amenability Tests

These tests demonstrated different behaviours between the minerals from both mines. Productora performed superior to Cortadera, registering greater copper recovery and lower acid consumption. At this micro-column lab on a small scale, the Cu recovery by solution is higher than recovery by tails.

13.4.3.3 2024 Variability Study Column Leach Tests – Nova Mineralis

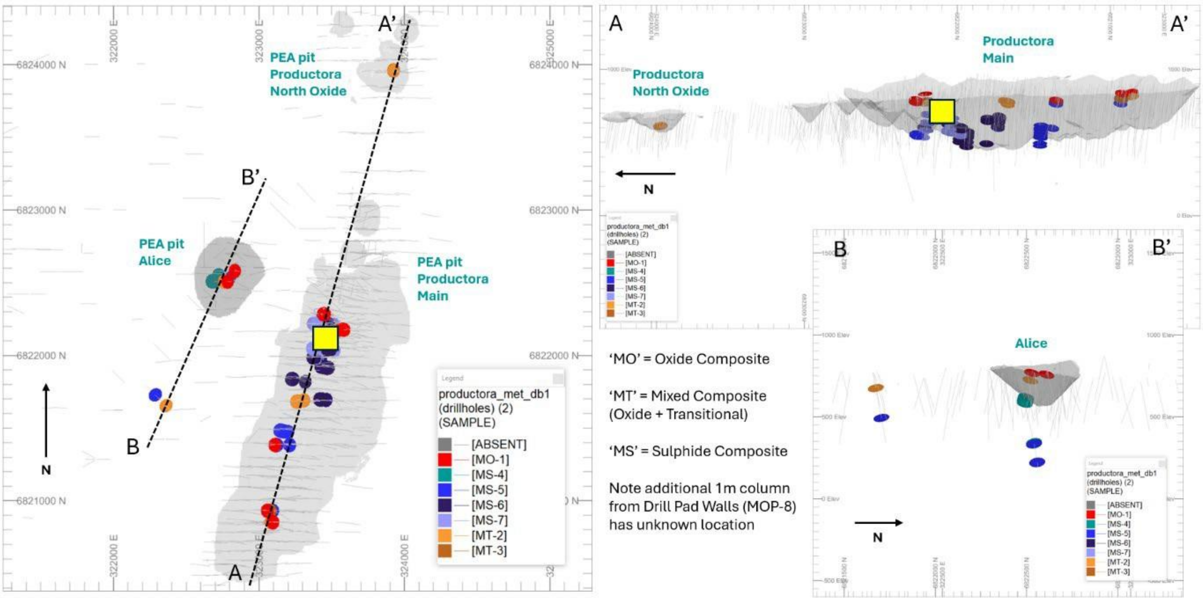
In 2024, Nova Mineralis conducted variability testwork and scaling study to further test the Costa Fuego Project feed materials to NOVAMINORE dump leach technology. This was completed on larger scale tests compared to the concept study.

13.4.3.3.1 Productora

The locations for the Productora samples used in these tests are displayed in Figure 13.39. For the IBC tests, samples were taken from existing low-grade stockpiles on the Santa Innes ROM pad (yellow square in Figure 13.39 and from existing underground mine workings from Santa Innes. Prior to sample collection, HCH Geologists ensured samples were representative of the mining inventory.

For the column test samples, nine of these samples were sourced from diamond drill core, two from drill pads and one was a duplicate of the IBC sample.

Figure 13.39 Location of Samples Used in Nova Mineralis Dump Leach Tests at Productora



For the column leach tests, the sample particle size was a P_{100} of 1.3 cm (0.5 inches) and the tests were completed for durations varying from 106 days to 240 days.

Copper recovery data and acid consumption kinetic data are displayed below. Table 13.35 shows the results from the Nova Mineralis mini-column tests using the tails assay after the column discharge.

Table 13.35 Results from Nova Mineralis Dump Leach Column Tests

Sample Name	Sample Type	Column Test Number	Leach Duration (days)	Copper Recovery by tails(%)	Acid Consumption (kg/t)
MO-1	Oxide	Col 21	106	65.4	32
MT-2	Transitional	Col-22	106	83.5	29
MT-2	Transitional	Col-23	106	85.4	29
MT-3	Transitional	Col-24	106	79.8	31
MS-4	Sulphide	Col-25	240	68.8	38
MS-4	Sulphide	Col-26	240	71.1	38
MS-5	Sulphide	Col-27	240	68.2	38
MS-6	Sulphide	Col-28	240	57.0	39
MS-7	Sulphide	Col-29	240	68.6	38
MOP-8	Sulphide /Transitional	Col-30	106	67.8	32
MOP-8	Sulphide/Transitional	Col-31	106	68.2	31
Duplicate from IBC Test	Sulphide /Transitional	Col-48	240	73.6	19

In 106 leaching days, the Cu recovery ranged from 65% to 85%. Higher recovery belongs to samples from oxidised and transitional zones, formed mainly with chrysocolla and green oxide compounds, while lower Cu recovery belongs to some transition zones formed by Cu sulphide and exotic insoluble Cu.

In extended 240 leaching days samples from sulphide zones achieved 57% to 71% Cu recovery, showing the efficiency of the hypersaline process to leach conventional insoluble compounds like chalcopyrite.

Acid consumption in column tests range from 20 to 40 kg/t. The scaling to industrial level is lower because in the column tests an over dosage was applied to assure tests continuity.

A dump leaching test was completed using nine 1 m^3 Intermediate Bulk Containers (IBCs). The IBCs were connected in series to simulate a 9 m high dump. In parallel to the IBC test, a duplicate sample was also loaded into a 1 m column as a part of the variability study (Col-48).

The results of the IBC leaching test and the duplicate column leaching test were extrapolated to estimate the expected copper recovery and acid consumption at 730 days for the IBC test and 240 days for the column test. These projected values were then used to calculate a scaling factor to apply to all the variability tests to obtain a database at an industrial scale for 730 days.

13.4.3.3.2 Cortadera

Fifteen 1 m columns tests were performed on low grade sulphide samples from Cortadera. The samples were sourced from drill core, including a duplicate sample for the stacked 4.5 m columns as a part of the scaling study. Test parameters were consistent with the column tests on Productora samples.

The locations of the sample are shown below (Figure 13.40).

Figure 13.40 Location of Samples Used in Nova Mineralis Dump Leach Tests at Cortadera

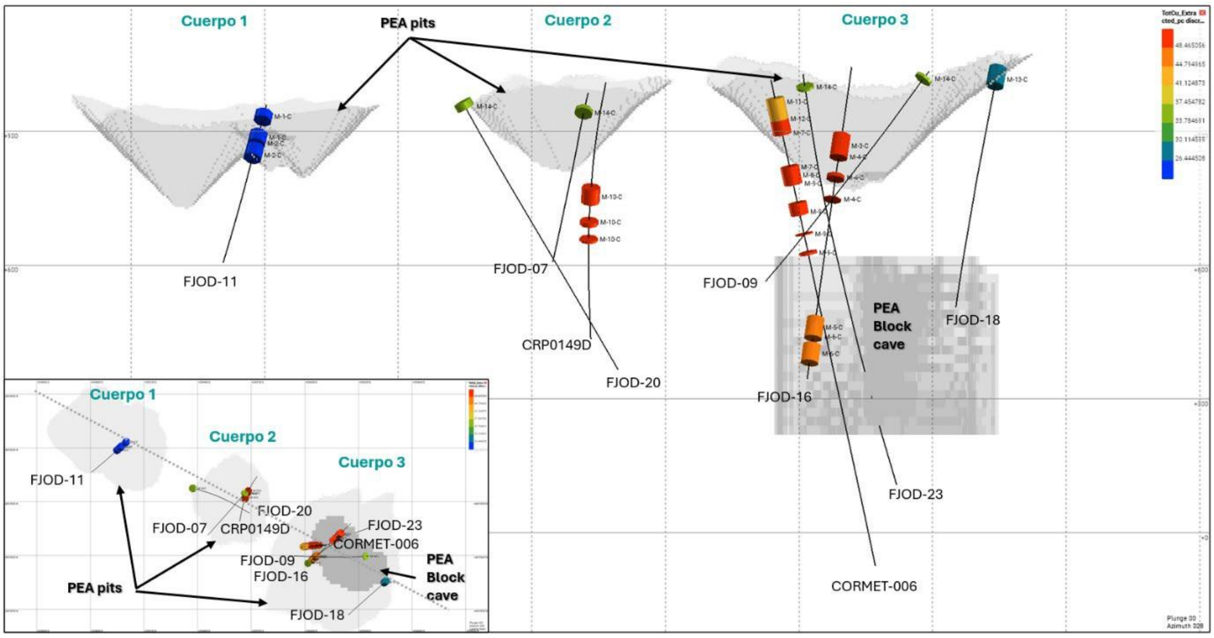


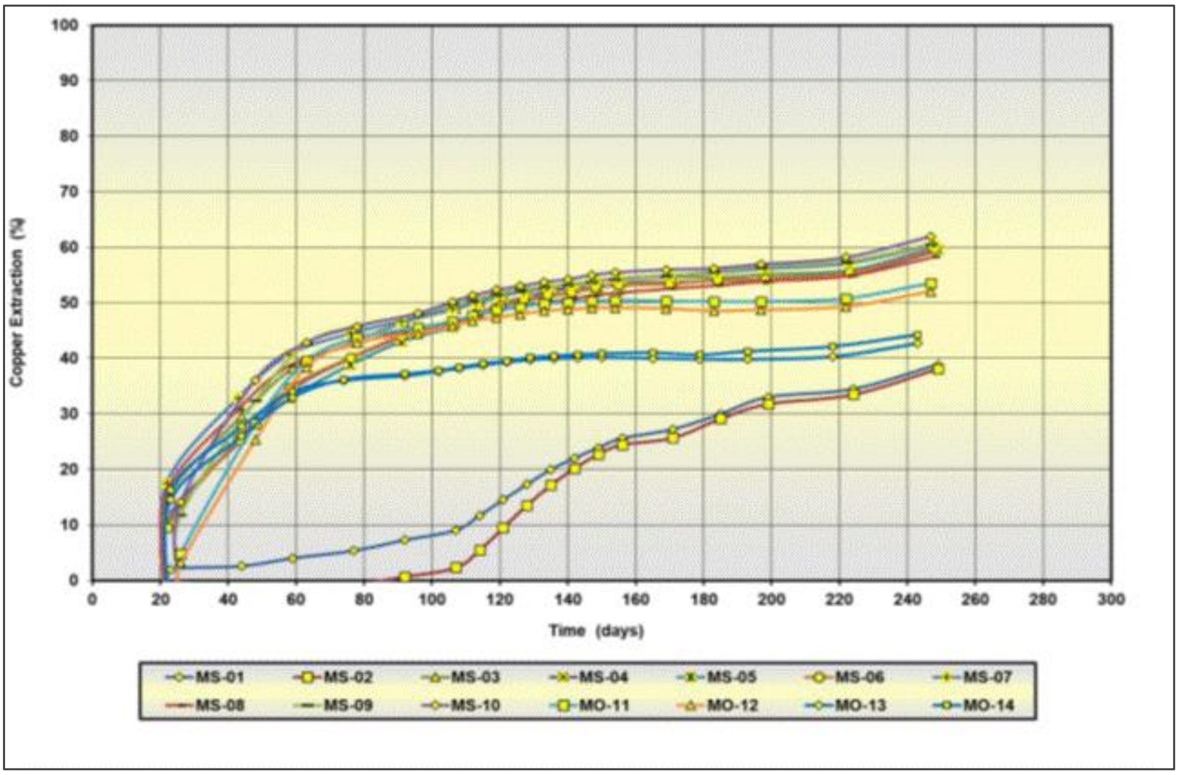
Table 13.36 shows the 1 m Cortadera column test results. Cu recovery is calculated after the column discharge using the tails assay.

Table 13.36 Final results for Cortadera 1 metre column tests

Column Number and Sample	Leaching Days	Final Copper Recovery by tails (%)	Final Acid Consumption (kg/t)
Col 32 (M-1-C)	224	32.8	67
Col 33 (M-2-C)	224	29.7	68
Col 34 (M-3-C)	224	52.8	47
Col 35 (M-4-C)	224	50.8	48
Col 36 (M-5-C)	223	50.2	49
Col 37 (M-6-C)	223	49.5	49
Col 38 (M-7-C)	223	54.0	48
Col 39 (M-8-C)	223	50.7	48
Col 40 (M-9-C)	222	54.6	47
Col 41 (M-10-C)	222	54.0	47
Col 42 (M-11-C)	222	43.9	45
Col 43 (M-12-C)	222	43.1	46
Col 46 (M-13-C)	218	27.0	46
Col 47 (M-14-C)	218	37.5	46
Col 49 (Duplicate pilot)	254	39.6	31

Copper recovery and acid consumption kinetics are displayed below in Figure 13.41.

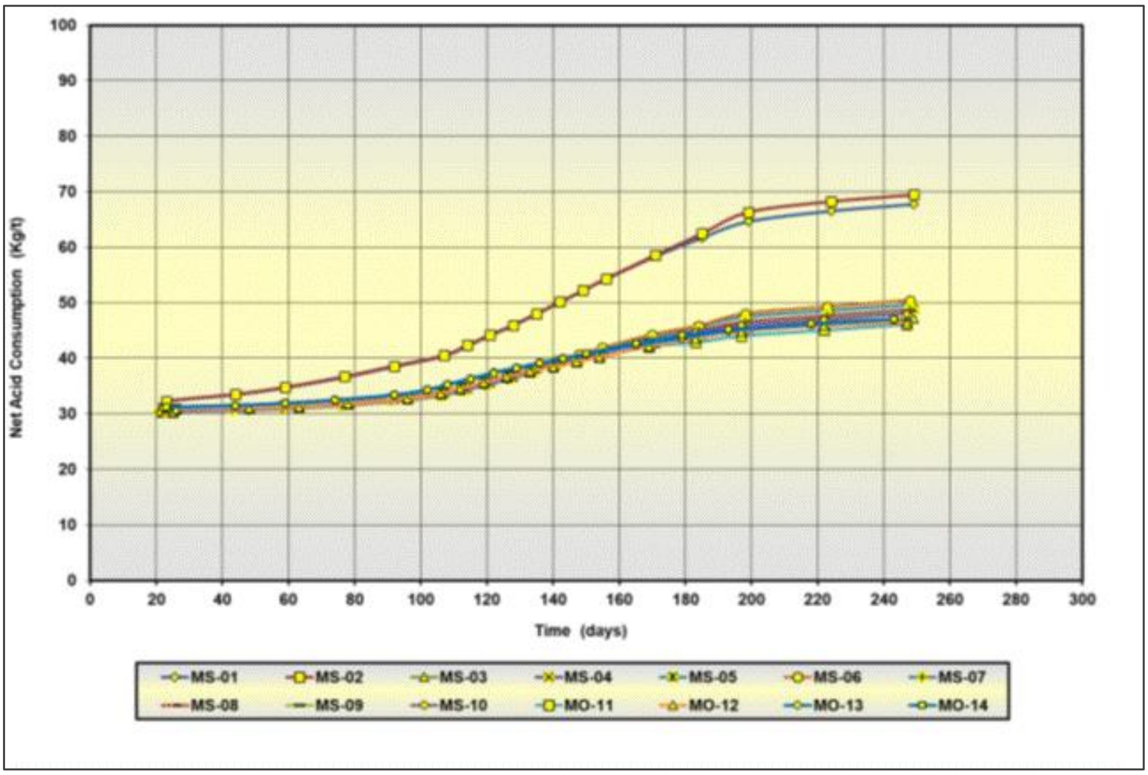
Figure 13.41 Copper Recovery Kinetics for 1 m Columns at Cortadera



In 240 leaching days the Cu recovery for Cortadera samples ranged from 30% to 55%. Higher recovery belongs to samples from oxidized zones, with chrysocolla and green oxide compounds, while lower Cu recovery belongs to some transition zones formed by Cu sulphide and exotic insoluble Cu.

Samples from Cortadera show wide variability in acid consumption with some over 50 kg/t that mean it is not economical mineral for dump leaching (Figure 13.42).

Figure 13.42 Acid Consumption Kinetics for 1 m Columns at Cortadera



The scaling study testwork for Cortadera was comprised of two stacked 4.5 m columns leached over a 1-year test period. A 1 m column with duplicate material was run at the same time over a 240-day test period. Intermittent irrigation was conducted using a synthetic solution made up of seawater and with acid strength, pH, Eh and impurity content consistent with recirculating solutions (RF) from an industrial dump leaching and solvent extraction/electrowinning process. During the rest periods, air was injected through the base of the columns by blowers, to ensure that the temperature of the processing feed bed was maintained at an acceptable level of approximately 25 to 30°C (in the case of an industrial ROM pile, temperature is maintained by air injection through the base of the leaching pile, using the drainage pipes, during the time when the pile is not under an irrigation condition, but at rest).

The results of the stacked test and duplicate 1 m column leaching tests were extrapolated to estimate the expected copper recovery and acid consumption at both 730 days and 240 days. This data was then used to calculate a scaling factor to apply to the variability tests to obtain a database at an industrial scale for 730 days.

13.4.3.4 Pilot IBC container Leach Tests – Nova Mineralis

13.4.3.4.1 Samples selection

Hot Chili Limited geology staff selected the available ROM samples on-site, including a pre-classification to select the most representative lots. Subsequently, the chosen batches were sampled and sent to the laboratory to verify their grades. It was defined to select the samples with grades above 0.15% Cu and less than 0.30% Cu.

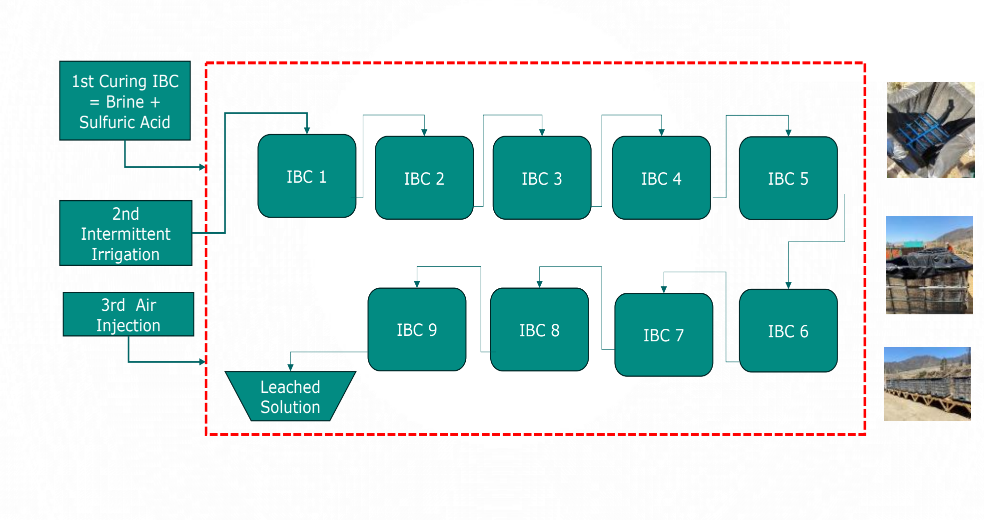
13.4.3.4.2 Sample preparation

For the test development, it was estimated that a single mineral sample weighing approximately 19 tons in ROM granulometry, below 6" was required to develop leaching tests in nine IBCs (Intermediate Bulk Containers) of 1 m^3 each, connected in series. Thus, a 9 m high material pile was simulated. Each IBC was loaded with approximately 1.5 tonnes, equivalent to 1 m^3 of mineral.

These IBCs were organised so that the effluent from the first was the feed for the second, the effluent from the second was the feed for the third, and so on until reaching the ninth IBC, which effluent was the rich solution of the entire leaching process (see Figure 13.43).

To start the process, these IBCs were cured with a solution made up of the equivalent brine from a reverse osmosis plant plus sulfuric acid. The brine was estimated to have around 40-50 g/l of chloride.
Once the curing process was completed, the mineral was allowed to settle. After that, intermittent irrigation started, considering a solution based on seawater with characteristics of acidity, pH, Eh and impurity content consistent with recirculating solutions from a leaching and solvent extraction process. During the resting periods, air was injected through by blowers from the base of the IBC, and the temperature was maintained at 30 C°. The leaching process, which included periods of irrigation and resting, lasted for a year.

Figure 13.43 IBC arrangement in Pilot plant



13.4.3.4.3 Sample characterisation

The test was evaluated in three sections (upper, medium and lower) for sample characterisation and recoveries calculation. The results indicate that all sections reported similar behaviour, thus validating the methodology used. Differences between calculated and analysed copper head grades are considered as to be within acceptable levels, especially for the lower values. Summarised sample information is presented in Table 13.37. Mineralogical characterisation is presented in Figure 13.45.

Table 13.37 Sample Cu Head Grade

Element	Upper section (IBC 1-2-3)		Medium Section (IBC 4-5-6)		Lower section (IBC 7-8-9)	
	Head Grade (%)		Head Grade (%)		Head Grade (%)	
	Calculated	Analysed	Calculated	Analysed	Calculated	Analysed
$Cu_{Tot.}$:	0.335	0.281	0.289	0.281	0.293	0.281
$Cu_{Sol\ H+}$:	0.019	0.016	0.016	0.016	0.017	0.016
$Cu_{Sol\ CN-}$:	0.035	0.029	0.030	0.029	0.030	0.029
Cu_{Insol}:	0.281	0.236	0.243	0.236	0.246	0.236
Fe_{Tot}:	-	2.73	-	2.73	-	2.73

Figure 13.44 ROM Sample Mineralogy

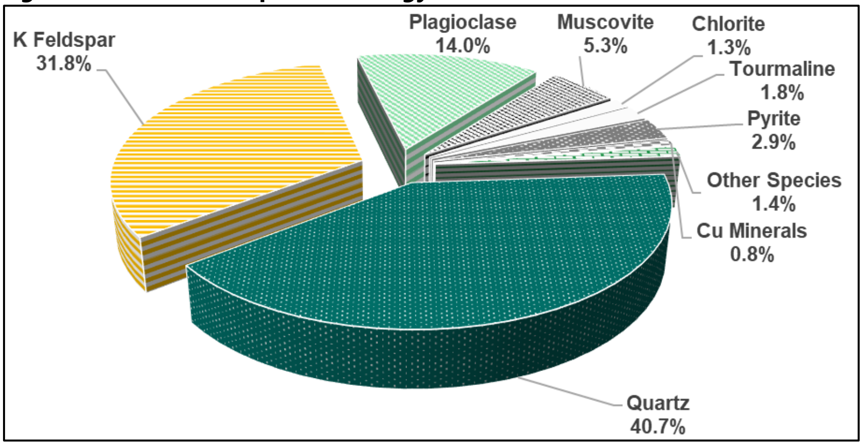
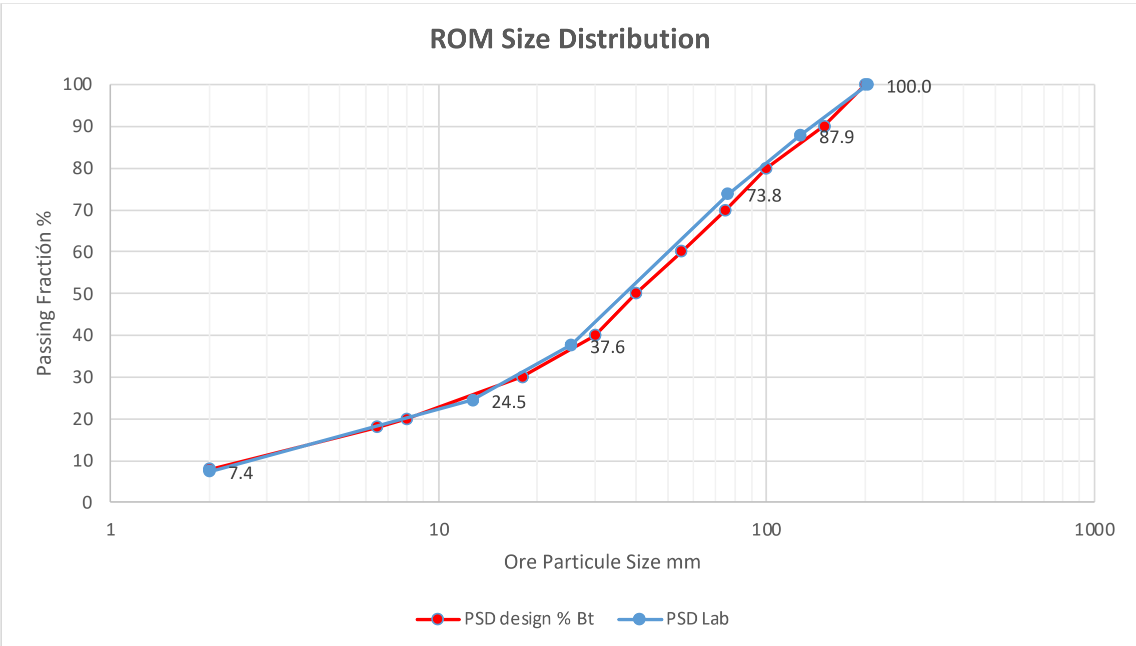


Figure 13.45 ROM Particle size distribution for Pilot test



Particle size corresponds to standard blasting fragmentation with an average 50% less 50 mm (2"). Optimised blasting may achieve 65% less than 50mm, potentially improving the copper recovery further.

13.4.3.4.4 Pilot test performance

After 365 operating days, the ROM leaching performance in acid consumption and Cu Extraction is presented in Figure 13.46 to Figure 13.49

Figure 13.46 Cu Extraction ROM versus Leaching time

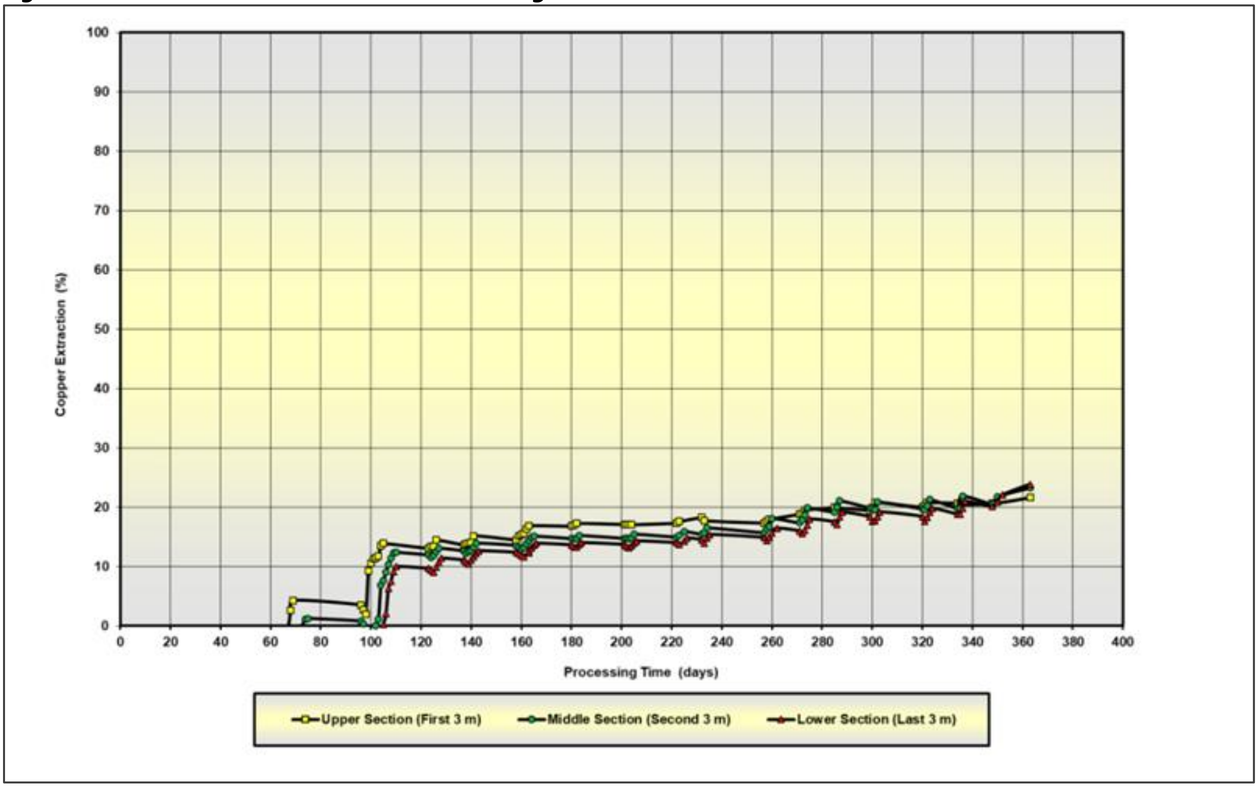
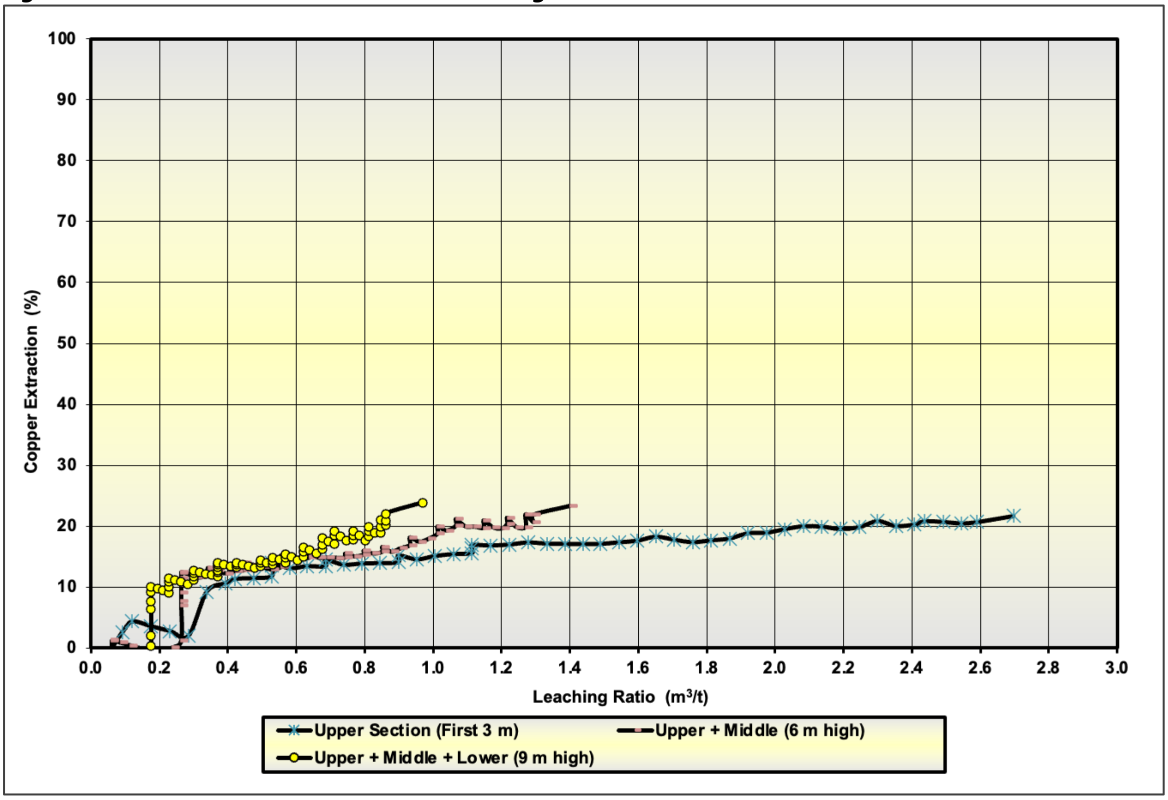


Figure 13.47 Cu Extraction ROM versus Leaching ratio



The Cu extraction at 365 days of the complete nine IBC containers obtained through the discharge of the PLS solution balance is 21%. The trend of the curves at 365 days, extrapolated using the logarithmic method, indicates that the forecast for Cu extraction (recovery) at 730 days is approximately 38% Cu, while acid consumption stabilises at 20 kg/t. Additionally, the leaching ratio (m^3/t) positively affects Cu extraction.

Figure 13.48 Net acid consumption ROM versus Leaching time

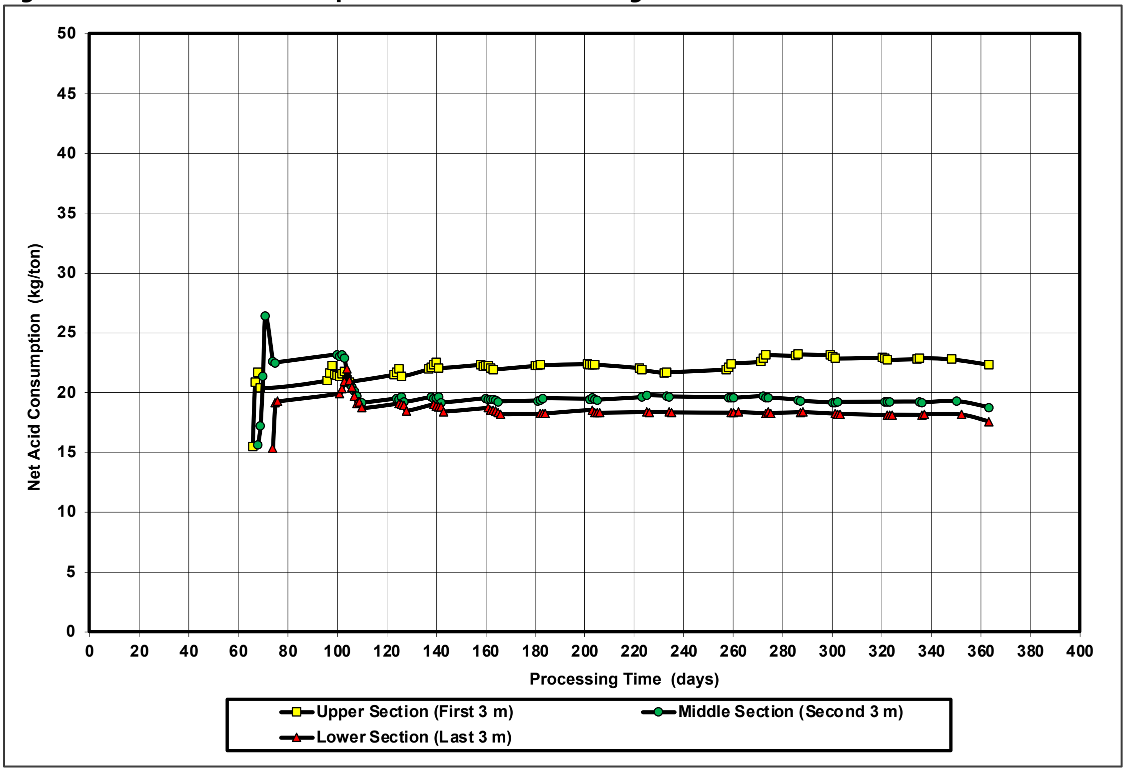
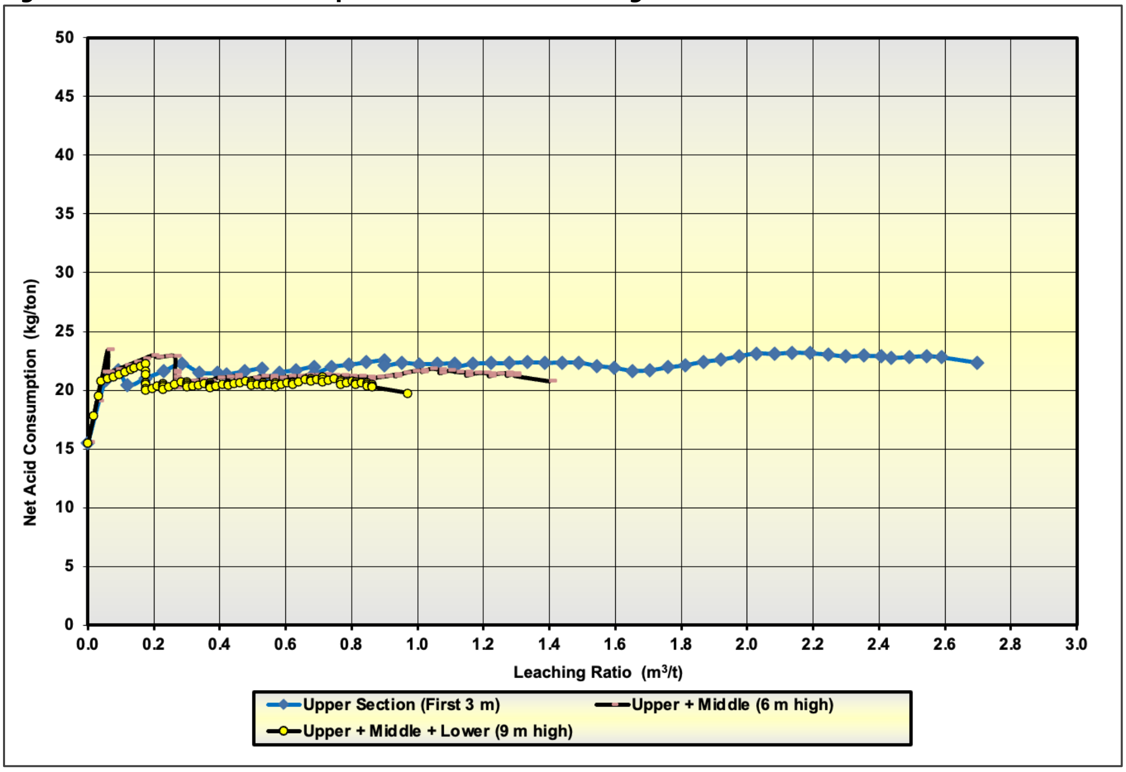


Figure 13.49 Net acid consumption ROM versus Leaching ratio



The trend curves obtained at 365 days indicate that, by logarithmic extrapolation, the net acid consumption forecast for 730 days will be 20 kg/t. Also, the trends versus leaching rate show the stabilisation of acid consumption.

Cu extraction by tails assay balance

At the end of the pilot test, the IBC containers from the three sectors (upper, middle, and lower) were unloaded for particle size and chemical analysis by size fraction. Using this tails grade information and the head grades registered at the start of the test, copper recoveries were calculated by sector and size fraction.

The average extraction obtained by tails assay after 365 days of leaching was 24%. Therefore, these results indicate a 3% increase in copper recovery (24 vs. 21% Cu) by tails analysis compared to solutions, as presented in Figure 13.47.

This result positively impacts the projected final recovery of 38% based on solution analysis and logarithmic scaling. The tails results obtained are shown in the following Table 13.38:

Table 13.38 Copper extraction by tails assay

	Head	**Tails Upper 3 m**	**Tails Medium 3 m**	**Tails Lower 3 m**	**Particle size distribution (%)**			
	Head Grade (%)	Tail Grades (%) Line 1 Upper 3 m	Line 2 Medium 3 m	Line 3 Lower 3 m	**Partial Size Fractions (%)** Head	Tails Upper 3 m	Tails Medium 3 m	Tails Lower 3 m
+5"	0.268	0.332	0.254	0.248	12.1	11.6	11.8	11.3
-5" / +3"	0.274	0.259	0.262	0.210	14.1	13.2	13.2	13.3
-3" / +1"	0.279	0.243	0.226	0.225	36.2	34.5	33.0	32.8
-1" / +½"	0.285	0.204	0.205	0.284	13.1	11.0	12.0	12.7
-½" / +#10	0.287	0.165	0.166	0.170	17.1	17.5	17.4	18.3
-#10	0.423	0.130	0.148	0.154	7.4	12.2	12.7	11.6
Total	**0.290**	**0.224**	**0.211**	**0.215**	100	100	100	100
Direct Analysis	**0.281**	**0.251**	**0.201**	**0.207**				
		Cu Extraction			**Cu extraction**			
		Cu Extraction by size (%) Line 1 Upper 3 m	Line 2 Medium 3 m	Line 3 Lower 3 m		**Cu weighted extraction (%)** Line 1 Upper 3 m	Line 2 Medium 3 m	Line 3 Lower 3 m
+5"		-24%	5%	7%		-2.8	0.6	0.8
-5" / +3"		5%	4%	23%		0.7	0.6	3.1
-3" / +1"		13%	19%	19%		4.4	6.3	6.4
-1" / +½"		28%	28%	0%		3.1	3.4	0.0
-½" / +#10		43%	42%	41%		7.4	7.3	7.5
-#10		69%	65%	64%		8.5	8.2	7.4
Total						**21.4**	**26.4**	**25.2**
					Total Cu weighted extraction			**24.3**

It is observed that the finest fractions, less than 1 inch, contribute the greatest to copper recovery, so increasing the fine fraction through blast optimisation should be considered an upside to increasing copper recovery.

The acceleration between Cu extraction and particle size is the basis for optimising recovery by increasing the fine particle fraction through blasting in the mine. It is observed that the minus 1-inch fraction, which represents 38% of the ore fed, has recoveries higher than the test average.

13.4.3.5 Laboratory reagent SX tests

The PLS copper-rich solutions from Heap and Dump leaching will have a minimum chloride content of 120 g/l that could affect the performance of the extractive reagents used in the SX plant. For this reason, amenability tests of these reagents were scheduled with two supply laboratories operating in Chile, to verify the recovery parameters, % of extractant required and the quality of the raffinate obtained. Table 13.39 and Table 13.40 present the results obtained by the Cytec laboratory.

Table 13.39 Cytec tests solution feed

Sample	Cu (g/L)	pH (25°C)	Mn (g/L)	FeT (g/L)	Fe+2 (g/L)	Cl- (g/L)	Nitrate (g/L)	ORP (mV)
PLS - Cu Medium grade	2.26	0.81	0.18	4.55	3.2	163	1.67	661
PLS - Cu High grade	5.88	1.22	0.38	1.70	0.6	142	2.23	696

Table 13.40 Performance Cytec tests

Item	Cu Medium grade	Cu Medium grade	Cu High grade	Cu High grade
PLS Cu (g/L)	2.26	2.26	5.88	5.88
PLS pH	1.2	1.2	1.4	1.4
Raffinate Cu (g/L)	0.225	0.220	0.557	0.472
Cu Recovery (%)	90.03%	90.27%	90.52%	91.98%
Copper Transfer (tpd)	36.62	36.72	95.81	97.35
PLS Flowrate (m3/h)	750	750	750	750
Organic Flowrate (m3/h)	750	863	750	863
Lean electrolyte flowrate (m3/h)	102	102	266	270
Extraction O/A ratio	1.0	1.15	1.0	1.15
Stripping O/A ratio	7.36	8.44	2.82	3.19
% Extractant	25.0%	25.0%	30.0%	30.0%
Loaded Organic Cu (g/L)	6.21	5.93	9.67	9.02
Loaded %	65.4%	62.5%	71.8%	67.0%
Unloaded organic Cu (g/L)	4.18	4.16	4.35	4.32
Lean electrolyte Cu (g/L)	35	35	35	35
Lean electrolyte Acid (g/L)	180	180	180	180
Electrolyte Cu delta (g/L)	15	15	15	15
Rich electrolyte Cu (g/L)	50	50	50	50
Trains	1	1	1	1
Daily copper transfer (tpd)	36.62	36.72	95.81	97.35
SX Plant Availability (%)	98%	98%	98%	98%

Item	Cu Medium grade	Cu Medium grade	Cu High grade	Cu High grade
Annual copper transfer (tpy)	13100	13135	34270	34823

Table 13.41 summarizes the results obtained by the Pochteca laboratory, which carried out tests only with the low-grade PLS solution.

Table 13.41 Pochteca test summary

Simulation #	[Cu]pls g/l	pH	Rec. Cu (%)	TPD	TPA	Cu Raffinate g/l
1	2,63	1,50	92,25	41,9	15289	0,204
2	2,63	1,30	92,11	41,8	15260	0,208
3	2,63	1,10	92,22	41,9	15276	0,205

Tests carried out in both laboratories indicate that they have adequate reagents to process PLS solutions from hypersaline leaching with a concentration of 2 to 5 g/l Cu and a pH range of 1.1 to 1.5, obtaining a recovery greater than 90% and a raffinate of 0.2 g/l of Cu.

13.4.3.6 Predictive Models

The following sections describe the development of predictive dump leach models for Productora. Cortadera has also been modelled but is not considered part of the Pre-Feasibility Study.

In October 2024, Upside Geometallurgy reviewed the Nova Mineralis variability and scaling studies for the Productora Dump leach. Upside's scope was to assess test results and assist in developing revised predictive models for copper recovery and acid consumption. This work is included later in the Section in the discussion on acid consumption and copper recovery.

13.4.3.6.1 Productora Copper Recovery Predictive Model

For the Productora deposit, ICP head assay data was reviewed for correlations with the copper recovery from the nine mini-columns (mini-columns with oxide material were not considered). The MS-6 was omitted from this regression as it was from the Alice porphyry deposit and therefore not representative of the Productora mineralisation. The two 1 m columns (Col-30 and Col-31) prepared from ROM pad material (MOP-8) were originally interpreted as oxide samples, but on review of QXRD results, these samples have been redefined as fresh to transitional and have been included in the dataset used to create the copper recovery regression shown in the next paragraph.

A negative linear relationship was drawn between copper recovery and lithium. The relationship has an R^2 value of ~0.9. QXRD completed by Nova Mineralis on the 1 m columns at Productora indicates that Li has a negative correlation with tourmaline and a positive correlation with calcite. When comparing the entire drillhole database and a more extensive QXRD dataset collected on two cross-sections at Productora (6822215mN and 6820850 mN), Li can be seen to occur at lower concentrations in the Kspar and Sericite alteration domains where both Cu grades and tourmaline abundance is highest. The magnetite amphibole, sodic calcic and background (unaltered) domains primarily occur on the western edge of the tourmaline breccia and into the wall rocks. These three domains on average have twice the Li content of the Kspar and Sericite domains. The Li can therefore be interpreted to represent the variation in gangue mineralogy between alteration domains.

Table 13.42 Productora Dump Leach Metal Recovery

Dump Leach Copper Recovery	0.51 * (-2.614 * LI_PPM + 95.97) (Minimum 10%)

13.4.3.6.2 Productora Acid Consumption Predictive Model

Analytical acid consumption (AAC) tests were completed on pulps of Productora samples that had been ground to 75 microns. In these tests, sulfuric acid was added directly to the pulps to estimate the gross acid consumption. The acid consumption from these tests is significantly higher than the acid consumption on the column leach tests because:

- In these tests, a specific pH value was not targeted (as opposed to the column tests).
- Unlike the column tests, these tests were not operated in parallel with a solvent extraction/electrowinning (SX/EW) circuit. Subsequently, there is not an acid credit coming back from the EW circuit which reduces the overall acid consumption to produce a net acid consumption.
- The particle size distributions of the pulps are much finer than the material in the 1 m column tests.

The scaled net acid consumption was calculated using the following steps:

- Firstly, estimating the gross acid consumptions on dump leach material (at 120 mm) by applying scaling factors to the AAC gross acid consumptions on the pulps (at 75 microns). This was based on benchmarked scaling factors.
- Secondly, estimating the theoretical re-generated acid credits in a heap leach circuit which report from the electrowinning stage to the leaching stage.
- Finally, net acid consumption was calculated by subtracting the acid credit value from the scaled AAC value.

While HCH considers the scaling factors reasonable for a PFS-level project (with an expected variance ± of 25%), the QP has recommended that, prior to the FS, additional project-scale testwork is completed with Productora samples.

Table 13.43 shows the average scaled net acid consumption values by the geometallurgical domain at Productora.

Table 13.43 Average Scaled Net Acid Consumptions by Geometallurgical Domain at Productora

Numeric Domain Identifier	Alteration (Lithogeochemical) Domain	Acid Consumption (kg/t H_2SO_4)
1	Albite	7
2	Background	15
3	Kaolinite	26
4	Kspar	3
5	Magnetite Amphibole	26
6	Sericite Albite	3
7	Sericite	6
8	Sodic Calcic	16
9	Not Classified	12
-99	Outside of Model Area	99

13.5 QP Comments to Sulphide Processing

The sulphide ore testwork has demonstrated the comminution and separation characteristics of the four Costa Fuego Project deposits and especially the two main deposits in the Project resource. Sufficient comminution properties have been measured to design the grinding circuit and sufficient flotation testwork has been performed to identify the appropriate reagents, develop flowsheets and estimate recoveries from the deposits. Definitive testing has demonstrated that the use of seawater as the Project process water provides flotation results equal to or slightly better than tests conducted in fresh water.

Additional testwork is required to explore the processing of ore blends from the main deposits. It is especially important to understand if the selected flowsheets will have sufficient flexibility for operators to keep the plant at optimal performance, regardless of blend.

13.6 QP Comments to Heap and Dump Leaching

The hydrometallurgy QP certifies that the testing program for the design of the heap and dump leaching processes is very robust, as it included representative samples from the Productora and Cortadera mines, included laboratory column tests for the heap leaching of 100 to 240 days of leaching, and IBC containers for the leaching dump pilot test that lasted one year. It also included mineralogical analyses and chemical characterisation by ICP. Therefore, a complete database was obtained for heap and dump design that considers leaching cycles, acid dosages (kg/t), irrigation rates (l/h/m^2), leaching ratio (m^3/t) and granulometry.

14 Mineral Resource Estimates

14.1 Introduction

The historical Mineral Resource Estimates informing this Report were last updated in February 2024. The MREs were prepared using Leapfrog 2023.1, Datamine Studio RM 2.0.66.0, and Snowden Supervisor v8.14.

Mineral Resource estimation practices are in undertaken in accordance with CIM "Estimation of Mineral Resource and Mineral Reserve Best Practice Guidelines" (29 November 2019) and CIM "Environmental, Social and Governance Guidelines for Mineral Resources and Mineral Reserve Estimation" (8 September 2023) and reported in accordance with CIM Definition Standards for Mineral Resources and Mineral Reserves (10 May 2014) that are incorporated by reference into NI 43-101.

Mineral resources that are not mineral reserves do not have demonstrated economic viability. MREs do not account for mine-ability, selectivity, mining loss and dilution. These MREs include inferred mineral resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as mineral reserves. It is reasonably expected that the majority of Inferred mineral resources could be upgraded to Indicated.

The QP responsible for the Mineral Resource Estimates is Ms. Elizabeth Haren, Director of Haren Consulting. Ms. Haren has over 25 years of experience in Mineral Resource estimation and is independent of HCH.

14.2 Location

The four deposits which comprise the Costa Fuego Project (Productora, Cortadera, Alice, and San Antonio) lie proximal to one another, at low altitude (800 m to 1,000 m), approximately 600 km north of Santiago. The La Verde exploration area is located 35 km south of Productora.

14.3 Mineral Resource Estimation Process

The MRE process has been developed by HCH with input and review from the QP responsible for the MREs. It follows a standard operating procedure which outlines each critical step in the generation of the estimates.

Working data is captured in a model process workbook, with peer reviews completed at predetermined stages (i.e., following modelling of the mineralised envelopes, statistical analysis, final estimation of grades). The peer reviewer must be satisfied with the decisions made before the commencement of the next step in the process. Any feedback and required remediation is captured in the workbook.

Final sign-off of the MRE is the responsibility of the QP and follows formal review sessions with all individuals responsible for generating the estimate. At a minimum, the following items are reviewed: geological models, mineralisation envelopes, statistical analysis, grade estimates, reporting considerations, and final reported Mineral Resource.

14.4 Database

14.4.1 Database Validation

All databases informing the MREs had the following checks completed before use:

- Drill collar locations – comparison of planned vs. actual survey pick up coordinates, checked in three dimensional (3D) vs. expected location on surveyed drill pads
- Down hole surveys - cross-referenced against planned survey orientation, and drill traces were checked visually in 3D for obvious errors
- Lithologies – reviewed based on surrounding drill holes, with drill core or RC chips logging updated where necessary
- Assays - 3D validation of assays to check for sample swaps and smearing and/or contamination.

The above validation steps ensure that the database is suitable for resource estimation.

14.4.2 Summary of Data Used in Estimate

All drilling data is stored in the HCH acQuire drill hole database.

The Productora MRE had holes which were drilled for metallurgical sampling but were not assayed. These have been excluded from the database, along with holes which were missing sampling information. The excluded hole list is in Table 14.1.

No holes were excluded from the Cortadera, Alice, or San Antonio MREs.

Table 14.1 : Drill Holes Excluded from the Productora MRE Database

HOLEID	HOLEID	HOLEID	HOLEID
MET001	MET011	MET023	PR-12
MET002	MET012	MW12	PR-13
MET003	MET014	NS7AD	PR-14
MET004	MET015	PR-1	PR-15
MET005	MET016	PR-4	PR-16
MET006	MET017	PR-6	PR-17
MET007	MET018	PR-7	PR-18
MET008	MET020	PR-8	PR-19
MET008B	MET021	PR-10	PR-24
MET009	MET022	PR-11	

A summary of the drilling data used for the MREs is listed in Table 14.2. Note that this table only includes drillholes that pass through the limits of the outermost estimation domain (i.e., may exclude some regional exploration drilling outside of the model area).

Table 14.2 : Drill Holes Included in Mineral Estimate Database – by Project			
Project	**Drilling Method**	**Holes**	**Metres**
Productora	RC	970	240,924
	DD	38	13,158
	RC with DD tail	104	44,206
	Total	**1 113**	**298,288**
Alice	RC	53	16,028
	DD	2	1,020
	RC with DD tail	5	1,745
	Total	**60**	**18,793**
Cortadera	RC	148	28,878
	DD	45	30,073
	RC with DD tail	58	48,849
	Total	**251**	**107,800**
San Antonio	RC	54	6,931
	DD	3	495
	Underground Drillhole	69	4 994
	Total	**126**	**12,420**

14.4.3 Data Manipulation

Non-numeric, zero, and negative values are used in the assay database table to indicate specific events or conditions. Edits were made to remove these values and ensure assays are suitable for use in the Mineral Resource.

For copper, gold, molybdenum, and silver:

- If the assay value was less than zero, the assay was set to 'absent'
- If the assay value was equal to zero and there was a recorded weight of sample the assay was set to half of the detection limit (0.0005 g/t Au, 0.005% Cu, 0.5 ppm Mo, 0.25 ppm Ag)
- If the assay value was equal to zero and there was no recorded weight of sample the assay was set to 'absent'.

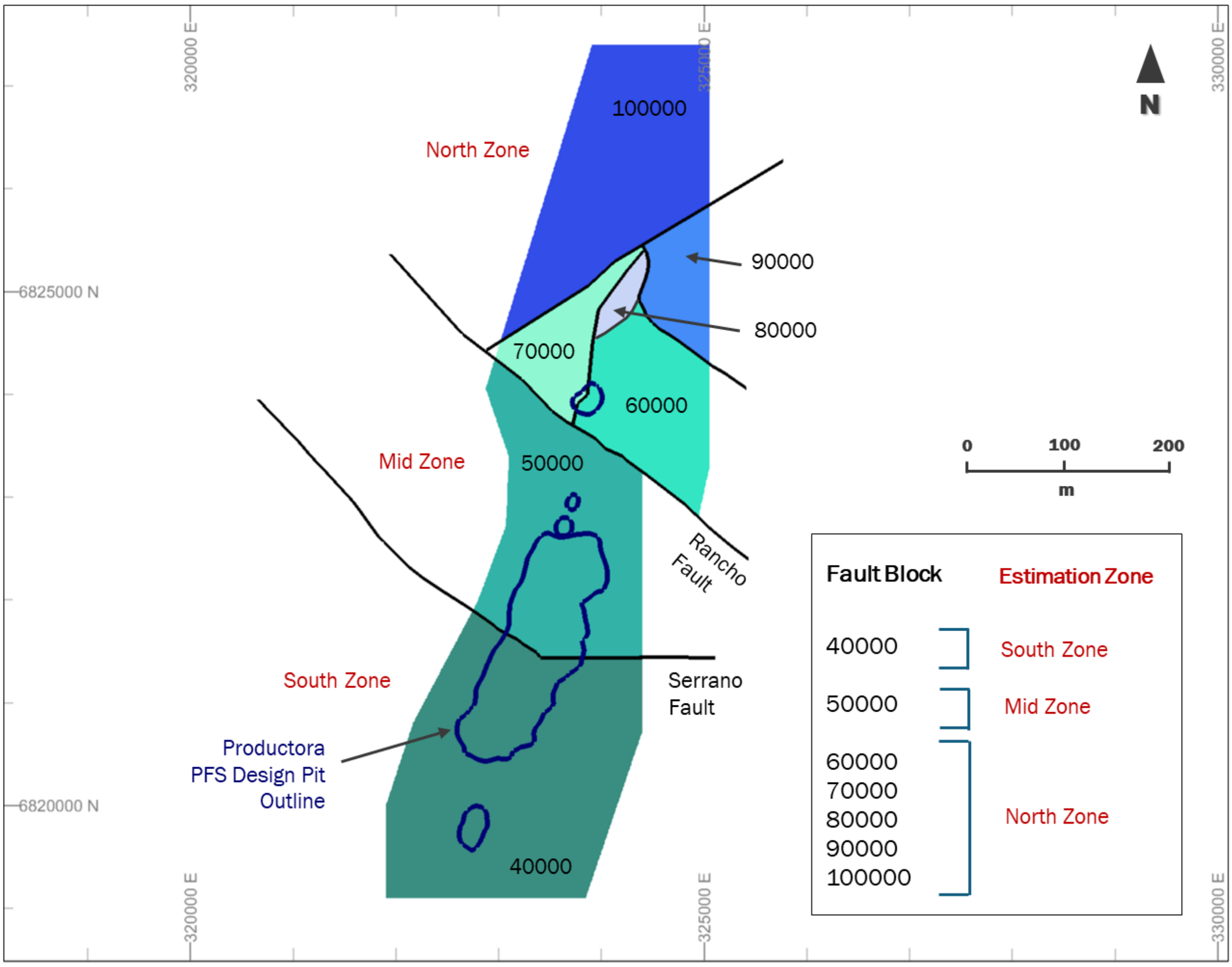
The assay tables are then checked to ensure no non-numeric, zero, and negative values are still present.

14.5 Modelling of the Mineralised Envelopes

14.5.1 Productora

The Productora model area is approximately 3,000 m east-west and 8,000 m north-south and has been split into three zones (North, Mid, and South) defined by the Rancho and Serrano faults (Figure 14.1).

Figure 14.1 : Productora Model Area with Fault Blocks



Mineralisation of copper, gold, molybdenum, and silver at Productora is developed mostly within a large intrusive hydrothermal breccia-dominated system that trends in a north-northeasterly direction.

The host breccia has been modelled from drill hole data over a strike length of 7,900 m. The breccia does not outcrop within the lease area although it has been observed extensively in drill core and in the underground workings.

Breccias tend to be narrow, north to northeast trending, tourmaline-cemented bodies. Sub-vertical feeder stocks (2 m to 5 m width) at depth, increase in thickness near-surface. These wider brecciated zones vary in orientation with central lodes tending to be sub-vertical. Flanking shallower eastern and western lodes dip moderately west and east respectively. There are also some locally steeply east-dipping lodes in the mid zone.

Deeper drilling at Productora suggests that lodes narrow at depth, with more ductile features also displayed.

While copper, gold, and molybdenum mineralisation is strongly coincident with potassic alteration, determining the protolith within and proximal to mineralisation is difficult, due to structural and hydraulic damage and extensive fluid-alteration overprinting present.

Secondary and relatively lower-grade mineralised material controls are evident as manto or manto-like horizons in the southern, far northern, and far eastern flanks of Productora. Lodes within the manto horizons are typically shallow dipping at 20° to 30° to the east or west.

14.5.1.1 2013 Predictive Modelling

A 3D predictive mineral system alteration model was completed for Productora in 2013. The predictive model comprises four principal components derived from alteration geochemical indices, lithogeochemical indices, and geological and geotechnical logging. A brief overview is provided below.

- Alteration Model: a set of alteration indices was derived from a ~160,000 drill hole sample database through application of multivariate X-Y and ternary analysis of geochemical data. The result of this was an alteration classification being applied to every drillhole sample interval, from which Leapfrog was used to create a 3D model of alteration domains
- Tourmaline Breccia Model: a 3D interpolant of the tourmaline breccia carapace was created in Leapfrog using a combination of geological and alteration logging, presence/absence of tourmaline, and fracture frequency data (using the assumption that hydraulic fracturing has increased the frequency of fractures around the tourmaline breccia bodies)
- Pyrite Enrichment Model: weight percent pyrite was calculated for each sample interval from multielement geochemistry to demonstrate the relationship between pyrite enrichment, copper grade, and the acid alteration domain
- S:Na Alteration Intensity Index: S:Na ratios provide a workable proxy for a magmatic hydrothermal alteration index, demonstrating a broad correlation between elevated S:Na (>2:1) and elevated copper grades.

14.5.1.2 Categorical Domaining

The Productora model incorporates the use of geochemical associations to help define domains for estimation, as defined in the predictive modelling of 2013 (described above). Note that the modelling has been refined on multiple occasions as more information has become available, most recently in 2023 for the February 2024 MRE.

The following observations were made regarding the predictive modelling outputs:

- Alteration wireframes were difficult to understand spatially, though k-feldspar was correlated broadly to highest copper
- Alteration wireframes have good relationships with iron and sulphur
- Alteration coding to drill holes via rock geochemistry is difficult to interpret, lacking easily identifiable and cohesive zones.

The lack of coherent and consistent mineralisation between and along sections renders a discrete mineralisation model unsuitable for Productora. Instead, a categorical kriging approach was used to model the individual zones of mineralisation within the deposit.

Drill hole data was coded with binary indicator fields ('1' being above the grade/value specified, '0' being below) as described in Table 14.3. Various ratios were also calculated and applied as shown in Table 14.4.

A total of 18 indicators and 17 element ratios were tested along with the calculated silica.

Table 14.3 : Categorical Indicator Coding of Drill Holes	
Indicator Field	**Test for Indicator to be 1, else 0**
SIND	S_PCT_D >= 0.4
COIND1	CO_PPM_D >= 50
COIND2	CO_PPM_D >= 300
MOIND	MO_PPM_D >= 50
CUIND05	CU_PCT_D >= 0.05
CUIND1	CU_PCT_D >= 0.1
CUIND2	CU_PCT_D >= 0.2
CUIND3	CU_PCT_D >= 0.3
CUIND4	CU_PCT_D >= 0.4
CUIND5	CU_PCT_D >= 0.5
AGIND03	AG_PPM_D > = 0.3
VEINPCTIND1	VEIN1PCT >= 1 OR VEIN2PCT >= 1
CAPIND	CA_PCT_D >= 0.8 or CA_PCT_D >= 0.3 AND R_P_CA < 800
CUSIND	R_CU_S >= 1.0
CUSIND2	R_CU_S >= 0.6 exclude Oxide
KALIND	R_K_AL >= 0.47
KSIND	R_K_S >= 10
ALSIND	R_AL_S >= 20
FESIND	R_FE_S >= 5

Table 14.4 : Ratio Calculation for Drill Holes	
Ratio Field	**Calculation of Ratio**
R_CU_S	CU_PCT_D / S_PCT_D
R_CU_AU	CU_PCT_D / AU_PPM_D
R_CU_MO	CU_PCT_D / MO_PPM_D
R_K_S	K_PCT_D / S_PCT_D
R_K_NA	K_PCT_D / NA_PCT_D
R_NA_AL	NA_PCT_D / AL_PCT_D
R_AL_K	AL_PCT_D / K_PCT_D
R_K_AL	K_PCT_D / AL_PCT_D
R_FE_AL	FE_PCT_D / AL_PCT_D
R_K_FE	K_PCT_D / FE_PCT_D
R_AL_S	AL_PCT_D / S_PCT_D
R_CA_NA	CA_PCT_D / NA_PCT_D
R_CA_MG	CA_PCT_D / MG_PCT_D
R_P_CA	P_PPM_D / CA_PCT_D
R_FE_S	FE_PCT_D / S_PCT_D
R_S_NA	S_PCT_D / NA_PCT_D

Table 14.4 : Ratio Calculation for Drill Holes	
Ratio Field	**Calculation of Ratio**
R_V_SC	V_PPM_D / SC_PPM_D

Additionally, a combined variable was created and used to create a combined indicator using the formula:

*COMBASS = Cu*10 + Au*10 + Mo/100. If COMBASS > 0.4 then IND_COMB = 1 else IND_COMB = 0.*

Due to the variable strike, dip and plunge over the Productora area, dynamic anisotropy was used to locally adjust the orientation of the search ellipse and variogram model. Trend wireframes were used to create a point file where each point relates to a triangle centroid and contains the true dip and true dip direction of the wireframe triangle. This point file was then used to estimate the local true dip and dip direction into the block model for each block. The estimates of true dip (TRDIP) and dip direction (TRDIPDIR) were subsequently used to locally adjust the variogram and search orientations during the categorical indicator estimation and some of the grade estimations.

The trend models used were restricted to the specific fault blocks as described Table 14.5.

Table 14.5 : Dynamic Anisotropy Trend Wireframes		
Wireframe	**Zone Field**	**Code**
DA_TREND_V5_02_STHOF27	FAULTBLOCK	40000
DA_TREND_V5_02_NTHOF27	FAULTBLOCK	50000
DA_TREND_V2_NTH_FB678_B	FAULTBLOCK	60000
DA_TREND_V2_NTH_FB678_B	FAULTBLOCK	70000
DA_TREND_V2_NTH_FB678_B	FAULTBLOCK	80000
DA_TREND_V2_NTH_FB910_B	FAULTBLOCK	90000
DA_TREND_V2_NTH_FB910_B	FAULTBLOCK	100000

The indicator and ratio data were used to generate variogram models reflecting the continuity of each of the indicators and ratios (where possible). FAULTBLOCK = 50000 (mid area) had the largest amount of data and was the area which produced the most robust variogram models.

The resulting variogram models used for estimation of the indicators and ratios for FAULTBLOCK = 50000 is presented in Table 14.6.

Table 14.6 : Categorical Variogram Models – Mid Area (FAULTBLOCK = 50000)											
VREF	**Variable**	**Rotation (ZXZ)**			**Nugget**	**Structure 1**		**Structure 2**		**Structure 3**	
					C0	**C1**	**R1**	**C1**	**R1**	**C1**	**R1**
1	CUIND05	110	115	0	0.12	0.38	60	0.23	110	0.27	700
							20		100		260
							15		130		200
2	CUIND15	110	115	0	0.12	0.17	20	0.44	25	0.27	135
							15		75		85
							15		20		40
3	MOIND	110	110	0	0.15	0.26	20	0.30	25	0.29	100

Table 14.6 : Categorical Variogram Models – Mid Area (FAULTBLOCK = 50000)											
VREF	**Variable**	**Rotation (ZXZ)**			**Nugget**	**Structure 1**		**Structure 2**		**Structure 3**	
					C0	**C1**	**R1**	**C1**	**R1**	**C1**	**R1**
							20		70		100
							10		20		60
4		100	110	0	0.1	0.46	50	0.14	120	0.30	600
	COIND1						35		135		280
							20		105		230
5		100	110	0	0.1	0.36	20	0.20	30	0.34	200
	COIND2						30		90		95
							20		60		90
6		115	160	0	0.06	0.28	50	0.23	180	0.43	1,100
	SIND						50		80		380
							20		180		180
7		110	100	0	0.15	0.30	60	0.30	70	0.25	800
	CALC_SI						25		80		250
							15		55		250
8		120	50	0	0.11	0.33	40	0.22	200	0.34	1,050
	R_P_CA						40		350		560
							40		200		300
9		120	90	0	0.11	0.31	40	0.24	200	0.34	1,050
	R_P_CA						70		300		300
							30		140		380
10		105	110	0	0.09	0.47	40	0.17	105	0.27	460
	R_K_AL						40		150		260
							20		65		200
11		110	170	0	0.03	0.41	50	0.27	170	0.29	800
	R_CU_S Ox/Tr						15		200		200
							15		100		130
12		110	110	0	0.11	0.21	50	0.34	60	0.34	800
	R_CU_S						15		80		320
							15		40		280
13		120	120	0	0.22	0.16	30	0.62	150	-	-
	AGIND03						30		150		-
							15		40		-

To perform the categorical kriging, new block models were created using a smaller parent block size of 5 mE by 5 mN by 5 mRL size.

The search strategy for estimation either used the established dynamic anisotropy to locally tune the search orientations or used the search orientations derived from the continuity analysis. The search strategy for all variables estimated within FAULTBLOCK = 50000 is contained in Table 14.7 with the combination of search and variogram parameters listed in Table 14.8.

Table 14.7 : Search Strategy for Categorical Estimation for Mid Area (FAULTBLOCK = 50000)												
SREF	**Orientation**			**Search**			**2nd Search Factor**	**No. of Composites**				
								First Search		**Second Search**		**Max Per Drill Hole**
	Rot1	**Rot2**	**Rot3**	**D1**	**D2**	**D3**		**Min**	**Max**	**Min**	**Max**	
1	Dynamic			50	50	20	4	6	12	6	12	3
6	115	160	0	50	50	20	4	6	12	6	12	3
8	120	50	0	50	50	20	4	6	12	6	12	3
9	120	90	0	50	50	20	4	6	12	6	12	3
11	110	170	0	50	50	20	4	6	12	6	12	3

Table 14.8 : Combination of Variogram and Search for Categorical Estimation Mid Area (FAULTBLOCK = 50000)				
Variable	**Model Variable**	**SREF**	**VREF**	**Description**
CUIND05	CUIND05	1	1	Using dynamic search
CUIND3	CUIND3	1	2	Using dynamic search
MOIND	MOIND	1	3	Using dynamic search
COIND1	COIND1	1	4	Using dynamic search
COIND2	COIND2	1	5	Using dynamic search
SIND	SIND	6	6	Using static search
CALC_SI	CALC_SI	1	7	Using dynamic search
CAPIND	CAPIND	8	8	Using static search
CAPIND	CAPIND2	9	9	Using static search
KALIND	KALIND	1	10	Using dynamic search
CUSIND	CUSIND	11	11	Using static search
CUSIND2	CUSIND2	1	12	Using dynamic search
AGIND03	AGIND03	12	13	Using dynamic search

The estimate was visually compared to the drill hole data in detail to fine tune the estimation parameters to reflect the spatial distribution of the conceptual mineralisation model described previously.

14.5.1.3 Interpretation of Categorical Domains

Escolme (2016) interpreted the distribution of breccia facies, based on graphic core logging, core photo library, drill hole database, detailed hand specimen, and thin section observations and WLSQ-QXRD data. Breccia cross-cutting relationships are indicated by the inclusion of clasts of earlier breccia stages. Stage 2 breccias are cross-cut by stage 3 and overprint stage 3 due to later fault reactivation. Variable breccia morphology (clast supported vs. cement supported) in stage 3 breccias are shown by fill pattern. Highest copper grade is associated with Facies 3B-1, 3B-2 and stage 2 breccias.

Using the Escolme interpretation as a guide, various combinations of the indicators and ratios were used to define geological/chemical material types. The results show that:

- Indicators for sulphur and molybdenum together could broadly outline the facies 3B-1 and facies 3B-2 – "Breccias"
- Within the breccias the K:Al ratios or cobalt indicators broadly outline the stage 2 material
- Calcium and phosphorus broadly outline a wedge of high calcium material between Productora and Alice which is generally very poorly mineralised
- Cu:S ratios outline oxidised sub horizontal features and faults where deep weathering occurs – pyrite higher in south
- Copper indicators at 0.05% Cu, 0.1% Cu, and 0.3% Cu were used to define higher- and lower-grade copper sub-domains.

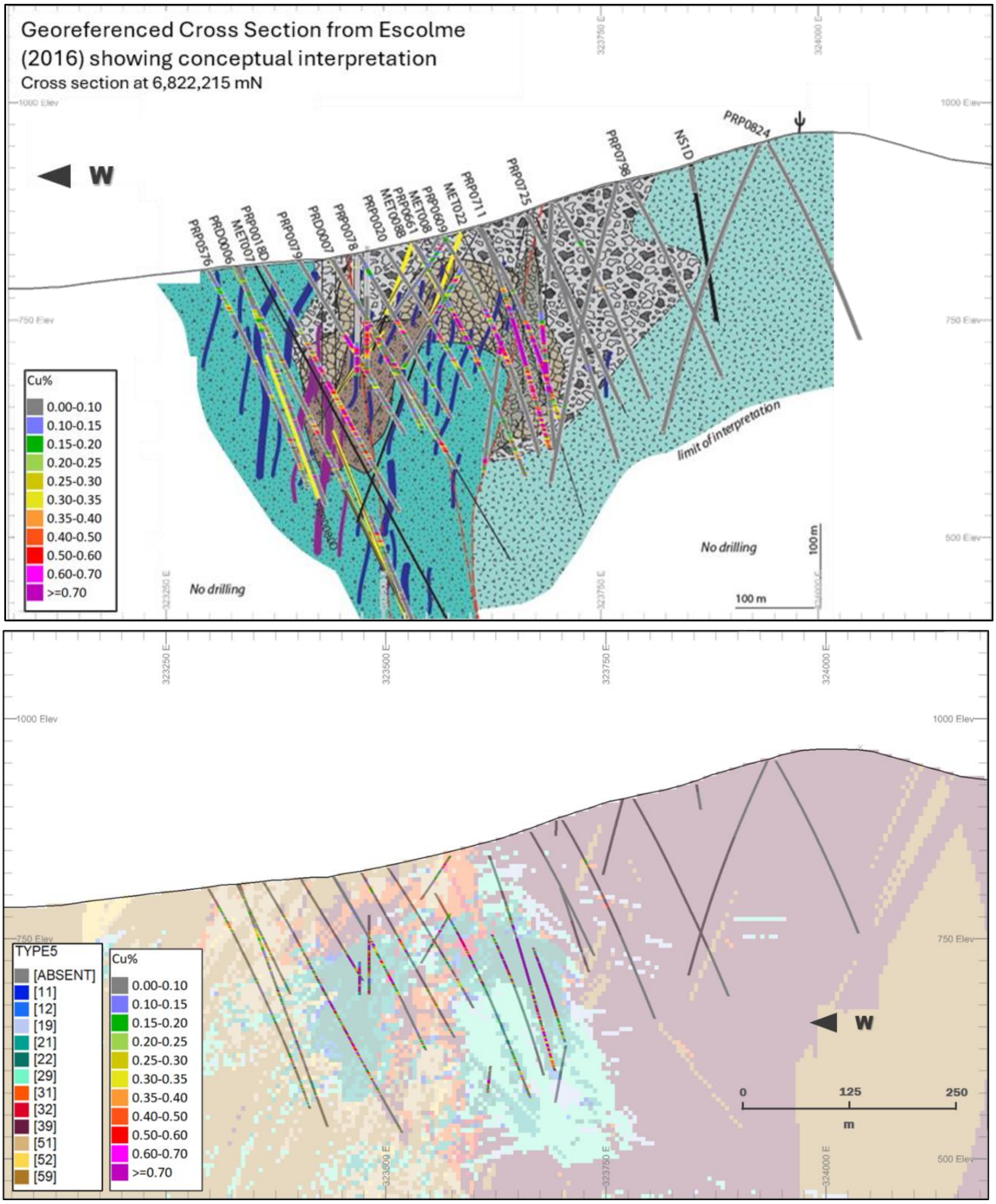
14.5.1.4 Copper Domains

Model 'Types' were coded based on a combination of indicator and ratio as shown in Table 14.9 and compared to the Escolme interpretation. An example east-west section at 6,822,215 mN is shown in Figure 14.2.

Table 14.9 : Model Coding for Copper Domaining

Description	Condition	Type
Facies 3B(1,2) LG	SIND>=0.30 and MOIND>=0.10 and CUIND05>=0.60	11
Facies 3B(1,2) HG	SIND>=0.30 and MOIND>=0.10 and CUIND3>=0.35	12
Facies 3B(1,2) VLG	SIND>=0.30 and MOIND>=0.10 and CUIND05<0.60	19
Stage 2 LG	SIND>=0.30 and MOIND>=0.10 and CUIND05>=0.60 and (KALIND>=0.50 OR COIND2>=0.1)	21
Stage 2 HG	SIND>=0.30 and MOIND>=0.10 and CUIND3>=0.35 and (KALIND>=0.50 OR COIND2>=0.1)	22
Stage 2 VLG	SIND>=0.30 and MOIND>=0.10 and CUIND05<0.60 and (KALIND>=0.50 OR COIND2>=0.1)	29
Rhyolite/Rhyodacite lapilli tuff LG	CAPIND>=0.20 and CAPIND2>=0.20 and CUIND05>=0.60	51
Rhyolite/Rhyodacite lapilli tuff HG	CAPIND>=0.20 and CAPIND2>=0.20 and CUIND3>=0.35	52
Rhyolite/Rhyodacite lapilli tuff VLG	CAPIND>=0.20 and CAPIND2>=0.20 and CUIND05<0.60	59
East oxide LG	WEATHZONE == 10 OR WEATHZONE == 20	61
East oxide HG	WEATHZONE == 10 OR WEATHZONE == 20	62
East oxide VLG	WEATHZONE == 10 OR WEATHZONE == 20	69
Background LG	Not previously coded and CUIND05>=0.60	31
Background HG	Not previously coded and CUIND3>=0.35	32
Background VLG	Not previously coded and CUIND05<0.60	39

Figure 14.2: Comparison of Conceptual Interpretation (Top), to domain coded Block Model (Bottom) on east-west Section 6,822,215 mN



In Figure 14.2, there is a reasonable correlation between the Escolme interpretation (above) and the coded model types for the February 2024 MRE (below). In particular, the geometry of the domains which host much of the copper mineralisation (Facies 3B – blue, and Stage 2 – green) match well across both interpretations.

14.5.1.5 Silver Domains

Silver forms as a halo around the copper-gold-molybdenum mineralisation, with the highest concentrations distal to the primary copper domains. The silver domaining was coded using a combination of indicators and ratios as shown in Table 14.10.

Table 14.10 : Model Coding for Silver Domaining		
Description	**Condition**	**Type**
Facies 3B(1,2)	SIND >= 0.3 AND MOIND >= 0.1 AND AGIND03 >= 0.3	1015
Rhyolite/Rhyodacite lapilli tuff	CAPIND >= 0.2 AND AGIND03 >= 0.3	1050
Flat Oxide	WEATH < 3000 AND AGIND03 >= 0.3	1060
Background	Not previously coded and AGIND03>= 0.3	1099
Facies 3B(1,2)	SIND >= 0.3 AND MOIND >= 0.1 AND AGIND03 < 0.3	9015
Rhyolite/Rhyodacite lapilli tuff	CAPIND >= 0.2 AND AGIND03 < 0.3	9050
Flat Oxide	WEATH < 3000 AND AGIND03 < 0.3	9060
Background	Not previously coded and AGIND03 <=0.3	9099

14.5.1.6 Molybdenum Domains

The model was coded with a combination of indicators and ratios for molybdenum domains as shown in Table 14.11.

Table 14.11 : Model Coding for Molybdenum Domaining		
Description	**Condition**	**MODOM1**
High Mo Facies 3B(1,2) LG	SIND>=0.30 and MOIND>=0.30 and CUIND05>=0.60	111
High Mo Facies 3B(1,2) HG	SIND>=0.30 and MOIND>=0.30 and CUIND3>=0.35	112
High Mo Facies 3B(1,2) VLG	SIND>=0.30 and MOIND>=0.30 and CUIND05<0.60	119
High Mo Stage 2 LG	SIND>=0.30 and MOIND>=0.30 and CUIND05>=0.60 and (KALIND>=0.50 OR COIND2>=0.1)	121
High Mo Stage 2 HG	SIND>=0.30 and MOIND>=0.30 and CUIND3>=0.35 and (KALIND>=0.50 OR COIND2>=0.1)	122
High Mo Stage 2 VLG	SIND>=0.30 and MOIND>=0.10 and CUIND05<0.60 and (KALIND>=0.50 OR COIND2>=0.1)	129
High Mo Rhyolite/Rhyodacite lapilli tuff LG	CAPIND>=0.20 and CAPIND2>=0.20 and CUIND05>=0.60 and MOIND>=0.3	151
High Mo Rhyolite/Rhyodacite lapilli tuff HG	CAPIND>=0.20 and CAPIND2>=0.20 and CUIND3>=0.35 and MOIND>=0.3	152
High Mo Rhyolite/Rhyodacite lapilli tuff VLG	CAPIND>=0.20 and CAPIND2>=0.20 and CUIND05<0.60 and MOIND>=0.3	159
High Mo Background LG	Not previously coded and CUIND05>=0.60 and MOIND>=0.3	131
High Mo Background HG	Not previously coded and CUIND3>=0.35 and MOIND>=0.3	132
High Mo Background VLG	Not previously coded and CUIND05<0.60 and MOIND>=0.3	139
Facies 3B(1,2) LG	SIND>=0.30 and MOIND<0.30 and CUIND05>=0.60	911
Facies 3B(1,2) HG	SIND>=0.30 and MOIND<0.30 and CUIND3>=0.35	912

Table 14.11 : Model Coding for Molybdenum Domaining		
Description	**Condition**	**MODOM1**
Facies 3B(1,2) VLG	SIND>=0.30 and MOIND<0.30 and CUIND05<0.60	919
Stage 2 LG	SIND>=0.30 and MOIND<0.30 and CUIND05>=0.60 and (KALIND>=0.50 OR COIND2>=0.1)	921
Stage 2 HG	SIND>=0.30 and MOIND<0.30 and CUIND3>=0.35 and (KALIND>=0.50 OR COIND2>=0.1)	922
Stage 2 VLG	SIND>=0.30 and MOIND<0.30 and CUIND05<0.60 and (KALIND>=0.50 OR COIND2>=0.1)	929
Rhyolite/Rhyodacite lapilli tuff LG	CAPIND>=0.20 and CAPIND2>=0.20 and CUIND05>=0.60 and MOIND<0.30	951
Rhyolite/Rhyodacite lapilli tuff HG	CAPIND>=0.20 and CAPIND2>=0.20 and CUIND3>=0.35 and MOIND<0.30	952
Rhyolite/Rhyodacite lapilli tuff VLG	CAPIND>=0.20 and CAPIND2>=0.20 and CUIND05<0.60 and MOIND<0.30	959
Background LG	Not previously coded and CUIND05>=0.60 and MOIND<0.30	931
Background HG	Not previously coded and CUIND3>=0.35 and MOIND<0.30	932
Background VLG	Not previously coded and CUIND05<0.60 and MOIND<0.30	939

14.5.1.7 Other Elements

Indicators and ratios for soluble copper, cobalt, aluminium, calcium, potassium, iron, and sulphur are not included in this Report.

14.5.1.8 Drill Hole Coding

Block model domains were back-flagged onto the drill holes to ensure consistency between the model and the data used for grade estimation.

14.5.1.9 Correlations

Correlations between all elements within the copper mineralised domains were calculated to assess potential relationships that should be preserved in grade estimation. Correlation was measured using the correlation coefficient, which is a measure of the linear relationship between variables and varies between +1 and -1, with a positive correlation coefficient indicating a positive relationship and a negative correlation coefficient indicating a negative (inverse) relationship. A correlation coefficient close to zero indicates there is no relationship between the variables. Elements which display strong correlations should have similar spatial continuity, with variogram models showing similar ranges. The correlations between elements should be preserved as much as possible in the block grade estimates.

The correlation coefficients for the four highest grade copper domains are presented in Table 14.12 for Facies 3B (1,2) low grade copper, Facies 3B (1,2) high grade copper, Stage 2 low grade copper, and Stage 2 high grade copper.

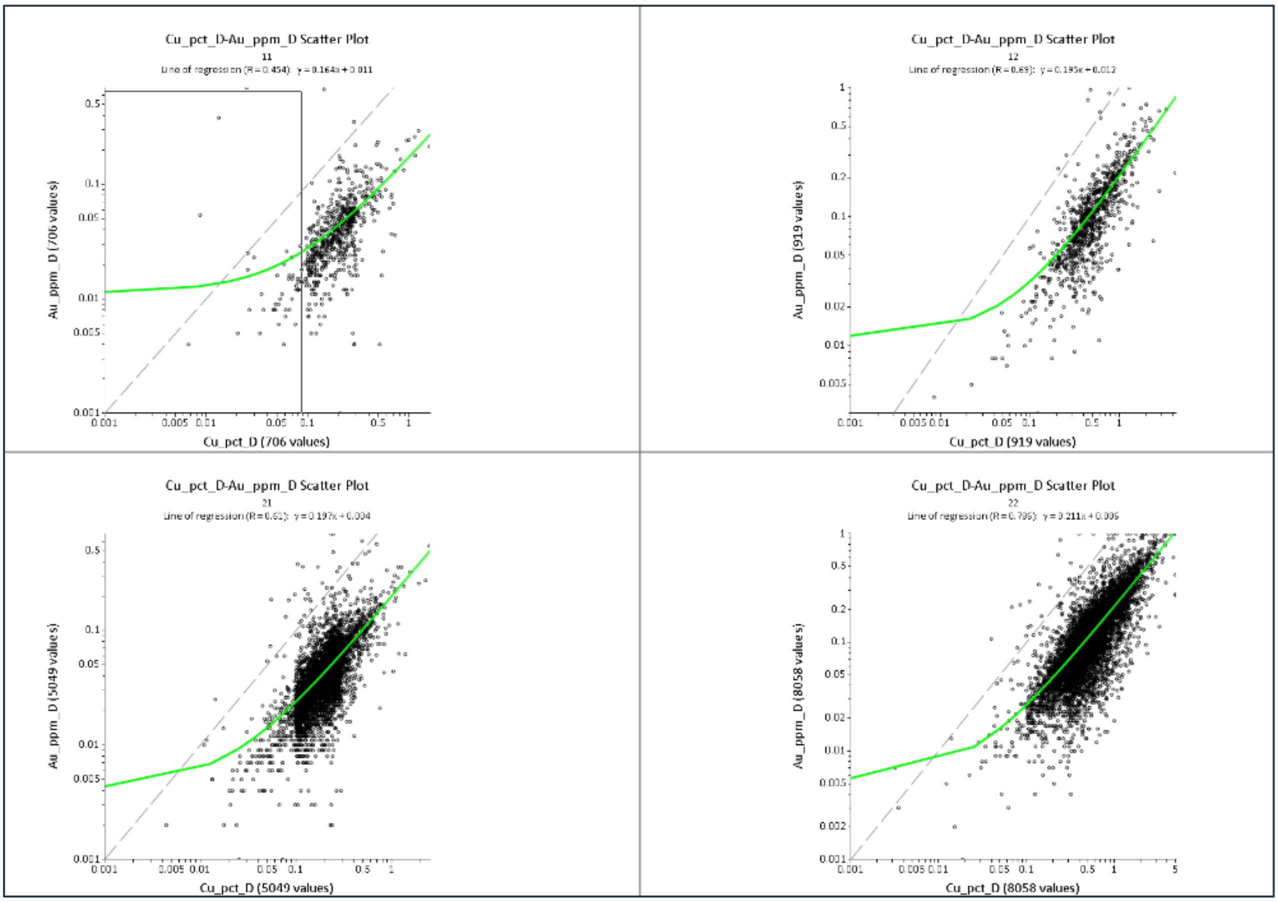
The only elements which show strong correlations are copper with gold, and cobalt with sulphur. A comparison of the copper and gold grades is presented as a scatterplot in Figure 14.3.

Table 14.12 : Correlation Matrices										
Mid Area – F3B (1,2) LG Cu – 50011										
	Au	**Cu**	**Mo**	**Ag**	**Al**	**Ca**	**Co**	**Fe**	**K**	**S**
Au	1.00	0.43	0.08	0.22	0.10	0.08	0.01	0.02	-0.03	0.10
Cu	0.43	1.00	0.16	0.26	-0.08	0.01	0.13	0.11	0.09	0.15
Mo	0.08	0.16	1.00	0.05	-0.05	0.01	0.03	-0.06	-0.01	0.06
Ag	0.22	0.26	0.05	1.00	0.04	0.19	0.01	-0.09	0.02	0.04
Al	0.10	-0.08	-0.05	0.04	1.00	0.13	0.05	-0.12	0.33	0.03
Ca	0.08	0.01	0.01	0.19	0.13	1.00	-0.20	0.26	-0.17	-0.12
Co	0.01	0.13	0.03	0.01	0.05	-0.20	1.00	0.17	0.12	0.75
Fe	0.02	0.11	-0.06	-0.09	-0.12	0.26	0.17	1.00	-0.09	0.15
K	-0.03	0.09	-0.01	0.02	0.33	-0.17	0.12	-0.09	1.00	0.07
S	0.10	0.15	0.06	0.04	0.03	-0.12	0.75	0.15	0.07	1.00
Mid Area – F3B (1,2) HG Cu – 50012										
	Au	**Cu**	**Mo**	**Ag**	**Al**	**Ca**	**Co**	**Fe**	**K**	**S**
Au	1.00	0.69	0.10	0.30	-0.10	-0.05	0.16	0.08	0.02	0.31
Cu	0.69	1.00	0.15	0.31	-0.18	-0.05	0.18	0.07	0.03	0.26
Mo	0.10	0.15	1.00	0.07	-0.07	0.09	-0.01	-0.04	-0.06	0.04
Ag	0.30	0.31	0.07	1.00	-0.15	0.11	0.03	0.06	-0.06	0.16
Al	-0.10	-0.18	-0.07	-0.15	1.00	0.05	-0.04	-0.19	0.29	-0.10
Ca	-0.05	-0.05	0.09	0.11	0.05	1.00	-0.15	0.26	-0.12	-0.03
Co	0.16	0.18	-0.01	0.03	-0.04	-0.15	1.00	0.16	0.06	0.66
Fe	0.08	0.07	-0.04	0.06	-0.19	0.26	0.16	1.00	-0.08	0.22
K	0.02	0.03	-0.06	-0.06	0.29	-0.12	0.06	-0.08	1.00	0.07
S	0.31	0.26	0.04	0.16	-0.10	-0.03	0.66	0.22	0.07	1.00
Mid Area – Stage 2 LG Cu – 50021										
	Au	**Cu**	**Mo**	**Ag**	**Al**	**Ca**	**Co**	**Fe**	**K**	**S**
Au	1.00	0.61	0.10	0.05	-0.03	-0.03	0.09	0.04	-0.05	0.05
Cu	0.61	1.00	0.16	0.10	-0.09	-0.01	0.13	0.13	0.02	0.09
Mo	0.10	0.16	1.00	0.03	-0.03	0.06	0.07	0.01	-0.03	0.05
Ag	0.05	0.10	0.03	1.00	0.00	0.04	0.00	0.00	-0.03	-0.01
Al	-0.03	-0.09	-0.03	0.00	1.00	-0.04	0.02	-0.24	0.31	0.01
Ca	-0.03	-0.01	0.06	0.04	-0.04	1.00	-0.10	0.23	-0.17	0.08
Co	0.09	0.13	0.07	0.00	0.02	-0.10	1.00	0.20	-0.15	0.69
Fe	0.04	0.13	0.01	0.00	-0.24	0.23	0.20	1.00	-0.20	0.17
K	-0.05	0.02	-0.03	-0.03	0.31	-0.17	-0.15	-0.20	1.00	-0.08
S	0.05	0.09	0.05	-0.01	0.01	0.08	0.69	0.17	-0.08	1.00
Mid Area – Stage 2 HG Cu – 50022										
	Au	**Cu**	**Mo**	**Ag**	**Al**	**Ca**	**Co**	**Fe**	**K**	**S**
Au	1.00	0.79	0.16	0.12	-0.06	-0.05	0.27	0.05	-0.06	0.36
Cu	0.79	1.00	0.19	0.17	-0.11	-0.04	0.32	0.08	-0.07	0.41
Mo	0.16	0.19	1.00	0.04	-0.06	0.04	0.07	-0.02	-0.04	0.12

Table 14.12 : Correlation Matrices

Ag	0.12	0.17	0.04	1.00	-0.04	0.11	0.04	0.06	-0.05	0.05
Al	-0.06	-0.11	-0.06	-0.04	1.00	-0.07	0.04	-0.31	0.37	-0.02
Ca	-0.05	-0.04	0.04	0.11	-0.07	1.00	-0.12	0.31	-0.20	-0.02
Co	0.27	0.32	0.07	0.04	0.04	-0.12	1.00	0.11	-0.12	0.60
Fe	0.05	0.08	-0.02	0.06	-0.31	0.31	0.11	1.00	-0.25	0.24
K	-0.06	-0.07	-0.04	-0.05	0.37	-0.20	-0.12	-0.25	1.00	-0.06
S	0.36	0.41	0.12	0.05	-0.02	-0.02	0.60	0.24	-0.06	1.00

Figure 14.3 : Scatterplots Between Copper and Gold for Highest Grade Domains



14.5.2 Alice

Mineralised domain interpretation for Alice was completed using Leapfrog 2023.1 based on assay grades for diamond and RC drillholes. Mineralised intervals were flagged to the drillhole for each domain, with tie-lines and boundary strings used as controls for volume and orientation where required.

For copper mineralisation, interpolants were created at 0.4% Cu, 0.2% Cu, and 0.025% CuEq based on natural breaks in the grade population.

Gold grades are low at Alice, although a strong correlation still exists between gold and copper. Gold grade distributions were checked within each copper domain, with no significant changes in grade between domains. For this reason, the gold estimate was only constrained by the 0.025% CuEq domain.

Molybdenum exhibits a different mineralisation control, albeit still contained within the broader copper anomaly. Field and drilling observations indicated that the molybdenum was limited to specific vein arrays rather than being pervasive in the porphyry groundmass.

For other mineralisation and metallurgical domains (except sulphur and iron), assay grades were used to construct the solids for estimation constraint. For sulphur and iron, a combined pyrite interpolant was created using logged percentages.

In all cases, grade distributions were assessed in Supervisor to ascertain the suitable cut-off grade (or grades) used.

14.5.3 Cortadera

Mineralisation at Cortadera is concentrated on three multi-phase tonalitic intrusions (Cuerpo 1, 2 and 3), as defined in the geological model.

Continuity of grade and geology is controlled by the emplacement of the mineralised intrusions into the gently southeasterly dipping host stratigraphic units. While these intrusions have a reasonably consistent pipe-like geometry, grade distribution is complex and extends into the host stratigraphy. Statistical analysis suggests that the copper grade decreases outwards from the porphyry core and that gradational boundary conditions exist between different rock units. While the distribution of rock types has guided mineralised zone interpretations, it has not been used to constrain the mineralised domains.

Mineralisation domains were constructed in Leapfrog 2023.1 independently for each estimated element using cut-off grades guided by breaks in each element's grade distribution, shown in Table 14.13 below. Mineralisation domains for gold, silver and molybdenum were created using grade interpolants on validated drillholes, composited to 10 m.

Copper mineralisation domains were created using a set of geological conditions on drill holes composited to 10 m intervals. These conditions are described below:

- Chalcopyrite (cpy) (as logged by site geologists) above a set cut-off
- Calculated mineralogy (ICP-MS) for chalcopyrite above a set cut-off
- Copper assays
- Logged quartz-rich A- and B-type vein abundance above a set cut-off.

Table 14.13 : Cut-Off Grades Used for Cortadera Mineralisation Domains

Cuerpo	Element	HG Domain Cut-off (DOM = 20)	LG Domain Cut-off (DOM = 30)	SLG Domain Cut-off (DOM = 50)
1	Cu	1.0% cpy	0.15% Cu	0.05% CuEq
	Au	0.07 g/t Au	-	
	Ag	1.0 g/t Ag	0.6 g/t Ag	
	Mo	70 ppm Mo	50 ppm Mo	

Table 14.13 : Cut-Off Grades Used for Cortadera Mineralisation Domains				
Cuerpo	**Element**	**HG Domain Cut-off (DOM = 20)**	**LG Domain Cut-off (DOM = 30)**	**SLG Domain Cut-off (DOM = 50)**
2	Cu	1.0% cpy	0.15% Cu	
	Au	0.17 g/t Au	0.07 g/t Au	
	Ag	1.0 g/t Ag	0.6 g/t Ag	
	Mo	70 ppm Mo	50 ppm Mo	
3	Cu	1.2% cpy	0.15% Cu	
	Au	(Cu Domain Used)	(Cu Domain Used)	
	Ag	1.0 ppm Ag	0.6 ppm Ag	
	Mo	200 ppm Mo	150 ppm Mo	

All mineralisation domains were built using the intrusion tool within Leapfrog Geo. Additional points and/or strings were used to guide the interpretation in areas of lower data density or complex geology.

While mineralisation domains do not always directly correlate with geological domains, each mineralisation domain is reconciled against the geological interpretation to ensure all observations (i.e. geological logging, surface mapping and knowledge of regional and local structural trends) are given proper consideration.

The presence of a calcium-rich alteration front is considered to exert a significant geological control on mineralisation and appears to correlate well with zones of higher A- and B-type quartz vein abundances and copper grades that extend outward from the mineralised porphyry intrusions. This geometrical relationship is consistent with the addition of potassium and sodium to the porphyry core (along with copper, gold, molybdenum, silver, and other metals), where calcium has been depleted. The calcium has been remobilised and driven outwards along permeable pathways that developed in zones of higher fracture- and vein-abundance and within adjacent competent hornfels and permissive stratigraphic units.

A strong correlation exists between gold and copper grades (statistical correlation R=~0.7). Gold interpolants were validated against the copper interpolants to ensure that this correlation is preserved. This is especially important as there are fewer gold samples available (historic Minera Fuego holes did not assay for gold), so the interpolation can be controlled using interpretation points and strings derived from the copper interpolant.

For molybdenum and silver, the Ann-Mason Porphyry deposit at Yerington has been used as a guide for the interpretation. Copper and gold form the “core” of the mineralised system, with accumulations of molybdenum and silver forming “lungs” around the fringes.

A 0.05% CuEq interpolant defines the outer extent of the mineralisation. The CuEq equation considers assayed copper, gold, silver, and molybdenum and provides volume constraint for the low-grade estimate for copper and gold. This is referred to as the super low-grade (“SLG”) domain.

The geometry of the SLG domain has a forced anisotropy of approximately 1.5:1, with mineralisation volumes extending along the gently south-easterly dipping front that broadly conforms to the orientation and dip-direction of the host stratigraphic units.

The statistical, spatial, and geological characteristics of the data were examined within each estimation domain.

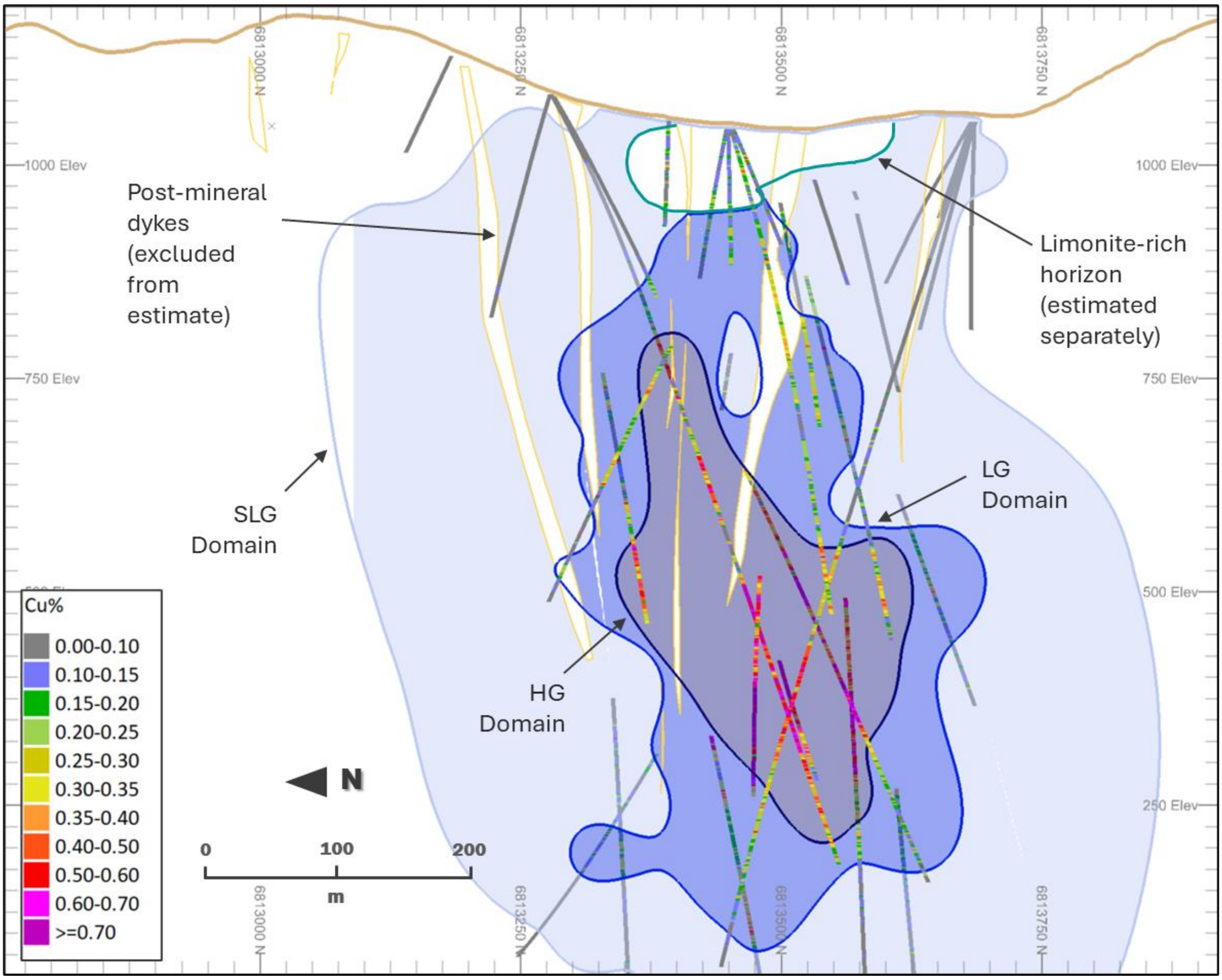
The SLG which comprises the low-grade boundary was analysed individually for each Cuerpo, but the output statistics suggested no hard boundary existed. For this reason, the SLG domain was analysed as a single domain straddling all three Cuerpos.

The effect of weathering was examined, with statistical analysis indicating a soft grade boundary between both weathering horizons (fresh to transitional, and transitional to oxide).

For the copper and gold estimates within the limonite-rich horizons, further domaining was applied through use of the categorical indicator kriging estimation methodology (CIK).

Figure 14.4 below shows the domaining applied for the estimation of Copper in Cuerpo 3.

Figure 14.4 : Oblique Cross Section of Cuerpo 3 Showing Cu% Estimation Domains Used for Mineral Resource



14.5.3.1 Categorical Domaining

The Cortadera model uses categorical indicator kriging for copper (Cuerpo 1, 2, and 3) and gold (Cuerpo 2 and 3) to help define final domains for estimation.

Drill hole data was coded with binary indicator fields ('1' being above the grade/value specified, '0' being below) as described in Table 14.14.

Table 14.14 : Categorical Indicator Coding of Drill Holes	
Indicator Field	**Test for Indicator to be 1, else 0**
Cuerpo 1	
HG CU% (DOM_CU1 = 20)	CU_PCT_D >= 0.3
LG CU% (DOM_CU1 = 50)	CU_PCT_D >= 0.2
SLG CU (DOM_CU1 = 90)	CU_PCT_D >= 0.1
Cuerpo 2	
HG CU (DOM_CU1 = 20)	CU_PCT_D >= 0.3
LG CU (DOM_CU1 = 50)	CU_PCT_D >= 0.2
SLG CU (DOM_CU1 = 90)	CU_PCT_D >= 0.1
HG AU (DOM_AU1 = 20)	AU_PPM_D >= 0.25
LG AU (DOM_AU1 = 50)	AU_PPM_D >= 0.10
SLG AU (DOM_AU1 = 90)	AU_PPM_D >= 0.08
Cuerpo 3	
HG CU% (DOM_CU1 = 20)	CU_PCT_D >= 0.5
LG CU% (DOM_CU1 = 50)	CU_PCT_D >= 0.2
SLG CU (DOM_CU1 = 90)	CU_PCT_D >= 0.1
HG AU (DOM_AU1 = 20)	Uses Cu Indicator
LG AU (DOM_AU1 = 50)	Uses Cu Indicator
SLG AU (DOM_AU1 = 90)	Uses Cu Indicator

The indicator data was used to generate variogram models reflecting the continuity of each of the indicators (where possible). The resulting variogram models used for estimation of the indicators are presented in Table 14.15.

Table 14.15 : Categorical Variogram Models											
AREA	**Domain**	**Rotation (ZXZ)**			**Nugget**	**Structure 1**		**Structure 2**		**Structure 3**	
					C0	**C1**	**R1**	**C1**	**R1**	**C1**	**R1**
Cuerpo 1	DOM_CU1 = 20	20	130	0	0.29	0.09	75	0.20	125	0.42	230
							20		75		100
							20		50		80
Cuerpo 1	DOM_CU1 = 30	-150	150	10	0.42	0.10	50	0.24	100	0.24	175
							50		100		175
							25		40		80
Cuerpo 1	DOM_CU1 = 50	-140	80	20	0.38	0.29	25	0.22	70	0.11	200
							25		70		150
							25		60		100
Cuerpo 2	DOM_CU1 = 20	-130	60	60	0.34	0.27	20	0.16	30	0.23	130
							20		30		100
							20		30		80

Table 14.15 : Categorical Variogram Models											
AREA	Domain	Rotation (ZXZ)			Nugget	Structure 1		Structure 2		Structure 3	
					C0	C1	R1	C1	R1	C1	R1
Cuerpo 2	DOM_CU1 = 30	-140	90	20	0.33	0.13	30	0.24	80	0.30	140
							20		60		140
							20		60		140
Cuerpo 2	DOM_CU1 = 50	-140	80	20	0.38	0.29	25	0.22	70	0.11	200
							25		70		150
							25		60		100
Cuerpo 2	DOM_AU1 = 20	-160	60	50	0.31	0.14	10	0.37	30	0.18	60
							10		30		50
							10		30		50
Cuerpo 2	DOM_AU1 = 30	-140	80	20	0.38	0.29	25	0.22	70	0.11	200
							25		70		150
							25		60		100
Cuerpo 2	DOM_AU1 = 50	-140	80	20	0.38	0.29	25	0.22	70	0.11	200
							25		70		150
							25		60		100
Cuerpo 3	DOM_CU1 = 20	-120	70	90	0.20	0.40	15	0.23	35	0.17	175
							15		35		125
							15		35		125
Cuerpo 3	DOM_CU1 = 30	-140	80	20	0.38	0.29	25	0.22	70	0.11	200
							25		70		150
							25		60		100
Cuerpo 3	DOM_CU1 = 50	-140	80	20	0.38	0.29	25	0.22	70	0.11	200
							25		70		150
							25		60		100

To perform the categorical kriging, new block models were created using a smaller parent block size of 5 mE by 5 mN by 5 mRL size. The estimation was split via Cuerpo as shown in the table above.

The search strategy for estimation used the search orientations derived from the continuity analysis. The search strategy for each area and variable estimated is contained in Table 14.16.

Table 14.16 : Search Strategy for Categorical Estimation												
AREA	Domain	Orientation			Search			2nd Search Factor	No. of Composites				
									First Search		Second Search		Max Per Drill Hole
		Rot1	Rot2	Rot3	D1	D2	D3		Min	Max	Min	Max	
Cuerpo 1	DOM_CU1 = 20	20	130	0	150	70	50	1.5	8	18	8	18	6
Cuerpo 1	DOM_CU1 = 30	-150	50	10	120	120	60	1.5	8	18	8	18	6

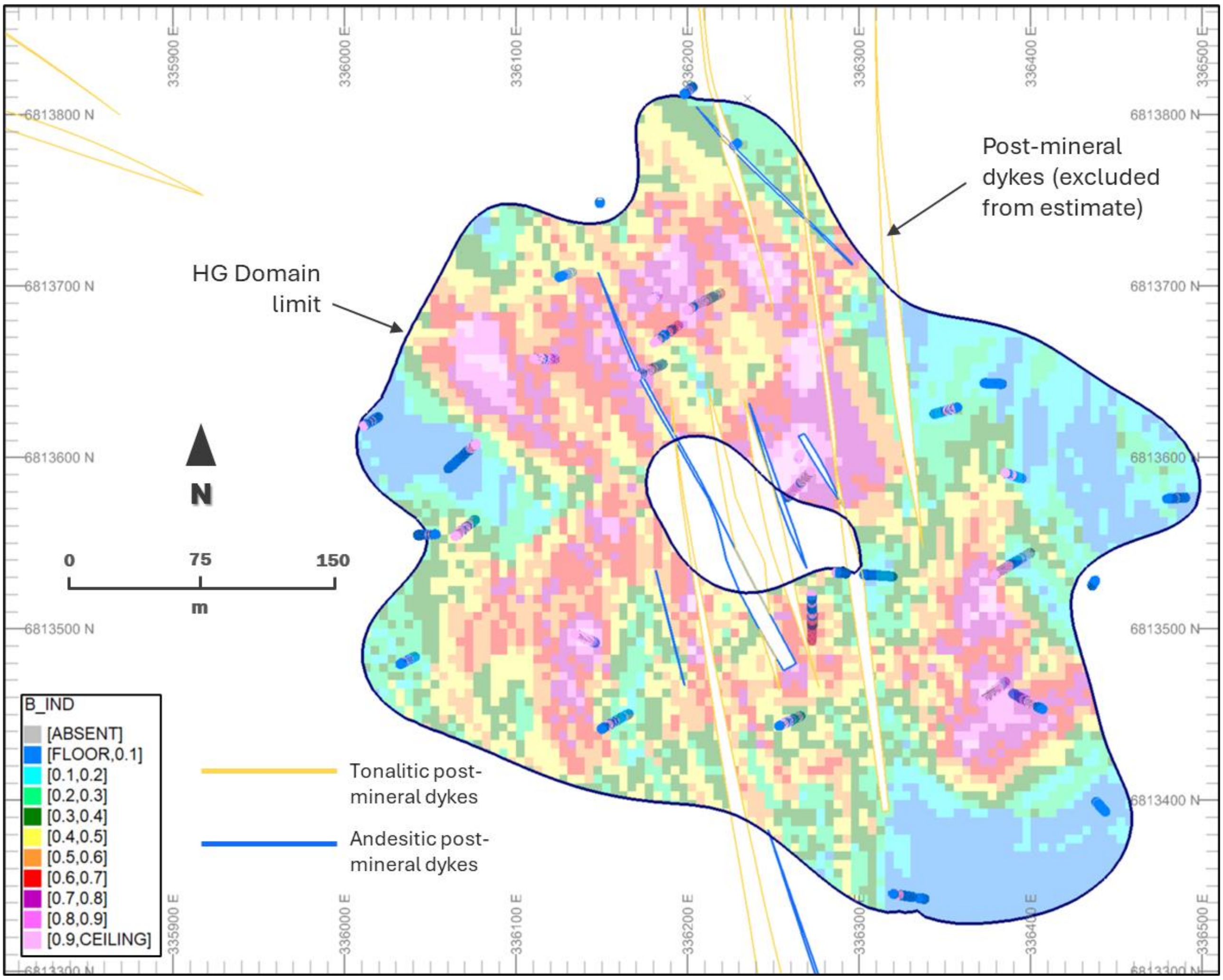
Table 14.16 : Search Strategy for Categorical Estimation													
AREA	Domain	Orientation			Search			2nd Search Factor	No. of Composites				
									First Search		Second Search		Max Per Drill Hole
		Rot1	Rot2	Rot3	D1	D2	D3		Min	Max	Min	Max	
Cuerpo 1	DOM_CU1 = 50	-140	80	20	200	100	100	1.5	10	20	10	20	6
Cuerpo 2	DOM_CU1 = 20	-130	60	60	90	70	50	1.5	10	20	10	20	6
Cuerpo 2	DOM_CU1 = 30	-140	90	20	90	50	50	1.5	8	18	8	18	6
Cuerpo 2	DOM_CU1 = 50	-140	80	20	200	100	100	1.5	10	20	10	20	6
Cuerpo 2	DOM_AU1 = 20	-160	60	50	40	35	35	1.5	10	20	10	20	6
Cuerpo 2	DOM_AU1 = 30	-140	80	20	200	100	100	1.5	10	20	10	20	6
Cuerpo 2	DOM_AU1 = 50	-140	80	20	200	100	100	1.5	10	20	10	20	6
Cuerpo 3	DOM_AU1 = 20	-120	70	90	120	80	80	1.5	10	20	10	20	6
Cuerpo 3	DOM_AU1 = 30	-140	80	20	200	100	100	1.5	10	20	10	20	6
Cuerpo 3	DOM_AU1 = 50	-140	80	20	200	100	100	1.5	10	20	10	20	6

For each indicator estimate, a third pass was run with a search factor of 10 and minimum sample count of 1 to ensure all blocks were populated with a value. Very few blocks populate on the third pass, and the limits of this methodology are considered when applying resource classification.

The estimate was visually compared to the drill hole data in detail to fine tune the estimation parameters to reflect the spatial distribution of the conceptual mineralisation model described previously.

An example of an indicator estimate for the Cuerpo 3 HG domain (DOM_CU1=20) is shown in Figure 14.5 below.

Figure 14.5 : Indicator Estimate on Binary Fields Showing the Probability of a Block Being Above 0.5% for Cuerpo 3 HG Domain. Plan View Section at 480 mRL



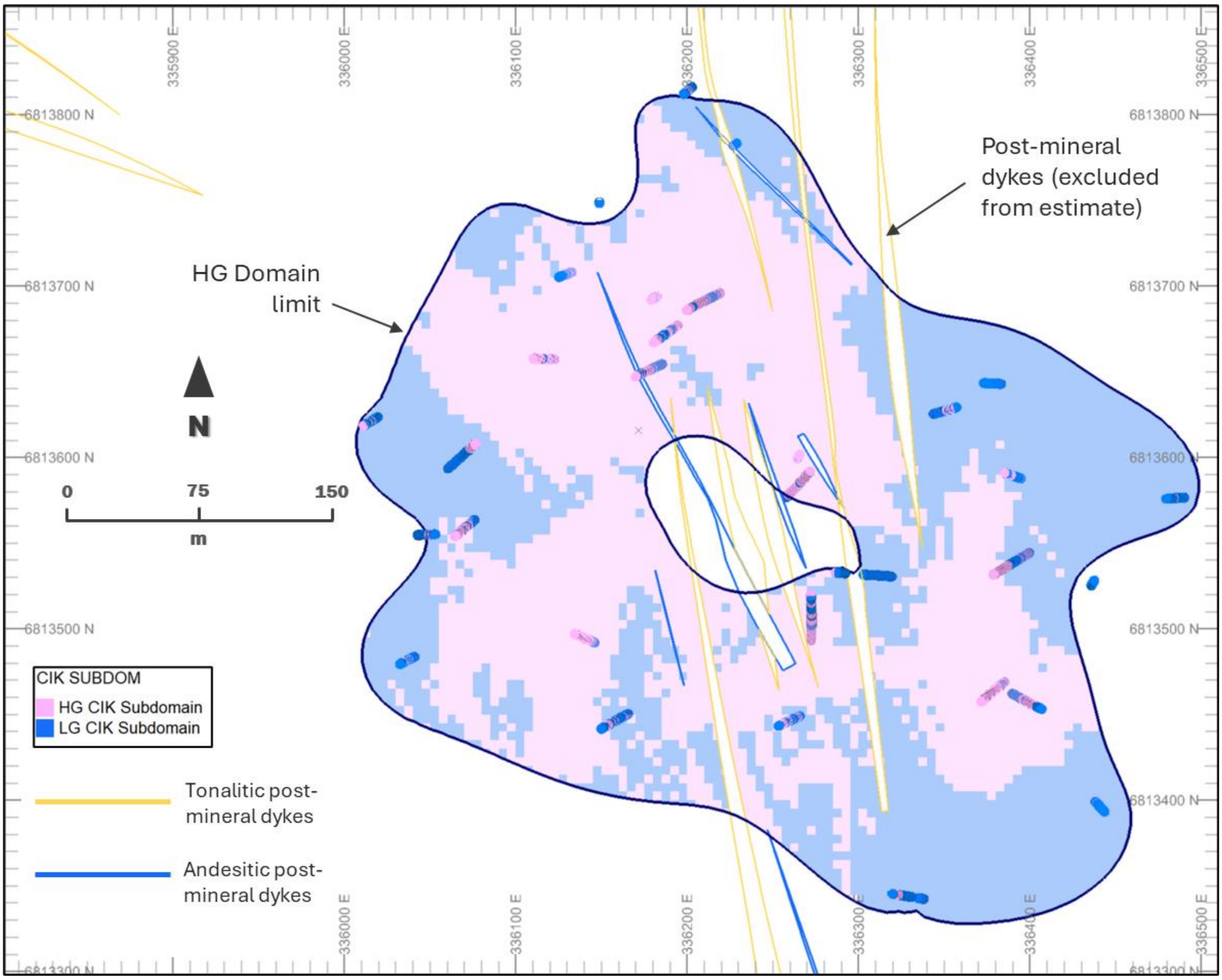
14.5.3.2 Estimation Subdomains

Estimation subdomains were coded based on indicator thresholds as shown in Table 14.17. An example of indicator estimate subdomains for the Cuerpo 3 HG domain (DOM_CU1=20) is shown Figure 14.6.

Table 14.17 : Model Coding for Copper Domaining		
AREA	**Domain**	**Condition**
Cuerpo 1	DOM_CU1 = 20	HG_CIK Domain - BIND >= 0.4, LG_CIK Domain – BIND < 0.4
Cuerpo 1	DOM_CU1 = 30	HG_CIK Domain - BIND >= 0.4, LG_CIK Domain – BIND < 0.4
Cuerpo 1	DOM_CU1 = 50	HG_CIK Domain - BIND >= 0.5, LG_CIK Domain – BIND < 0.5
Cuerpo 2	DOM_CU1 = 20	HG_CIK Domain - BIND >= 0.4, LG_CIK Domain – BIND < 0.4
Cuerpo 2	DOM_CU1 = 30	HG_CIK Domain - BIND >= 0.4, LG_CIK Domain – BIND < 0.4
Cuerpo 2	DOM_CU1 = 50	HG_CIK Domain - BIND >= 0.5, LG_CIK Domain – BIND < 0.5
Cuerpo 2	DOM_AU1 = 20	HG_CIK Domain - BIND >= 0.4, LG_CIK Domain – BIND < 0.4

Table 14.17 : Model Coding for Copper Domaining		
AREA	**Domain**	**Condition**
Cuerpo 2	DOM_AU1 = 30	HG_CIK Domain - BIND >= 0.4, LG_CIK Domain – BIND < 0.4
Cuerpo 2	DOM_AU1 = 50	HG_CIK Domain - BIND >= 0.4, LG_CIK Domain – BIND < 0.4
Cuerpo 3	DOM_AU1 = 20	HG_CIK Domain - BIND >= 0.4, LG_CIK Domain – BIND < 0.4
Cuerpo 3	DOM_AU1 = 30	HG_CIK Domain - BIND >= 0.5, LG_CIK Domain – BIND < 0.5
Cuerpo 3	DOM_AU1 = 50	HG_CIK Domain - BIND >= 0.5, LG_CIK Domain – BIND < 0.5

Figure 14.6 : Indicator Estimate Subdomains for Cuerpo 3 HG Domain . Plan View Section at 480 mRL



Significant iterative testwork has been completed to understand the impact of indicator threshold selection on the Cortadera MRE.

14.5.4 San Antonio

Mineralised copper domain interpretation for San Antonio was completed using Leapfrog 2023.1 based primarily on surface mapping and the resultant mafic dyke model, which is thought to be a critical control on mineralisation.

Mineralised intervals were flagged to the drillhole for each domain, with tie-lines and boundary strings used as controls for volume and orientation where required. Where the mineralisation is interpreted to be truncated at a structure (for instance the Agua de los Burros fault to the east), the fault mesh has been used as a constraint.

In total, nine mineralised domains were created, with strike lengths ranging from 50 m to 500 m. An additional domain was created to represent a surface stockpile which has been collared through in several drillholes.

Mineralised copper domains have been reviewed against underground workings, with a strong relationship between location of workings and interpreted domains.

In addition to the mineralised copper domains created for San Antonio, a low-grade encompassing halo has been produced, using a low cut-off grade of 0.05% Cu. This will allow for the additional of incremental metal tonnes outside of the primary mineralised domains.

14.6 Modelling of Weathering Domains

14.6.1 Productora

The Productora MRE utilises a combination of quantitative (i.e. ratio of soluble copper to total copper) and qualitative (i.e. proximity to structures and logged regolith) data used to model the oxide, transitional, and fresh weathering zones. This technique attempts to account for the impact of structural complexity on weathering at Productora and was a new addition to the 2024 Productora MRE.

The weathering model was produced using the following three-step process:

1) Create initial weathering 'domains' in Leapfrog using copper-sulphur ratio (Cu:S)
2) Run indicator estimates on 6 weathering related variable within each domain (using suitable variograms, search neighbourhoods, and anisotropy)
3) Assign weightings to each of the indicators based on confidence in the data, and then determine the likely weathering type (oxide/transition/fresh etc) within a block (at 5 m (X) x 5 m (Y) x 5 m (Z) scale).

The initial weathering domains control the subsequent orientation of the indicator estimates; a flat orientation for the traditional near-surface weathering, and a steep orientation for the fault-controlled, discrete, deeper weathering. Only two surfaces were created, corresponding to 'weathered' and 'fresh' for this step.

Domain boundaries were created in Leapfrog 2023.1, using Cu:S ratio (composited to 10 m intervals and filtered for data above 0.1% copper to remove outliers around assay lower detection limits). The Cu:S ratio was used due to its availability in almost every drillhole interval, and its quantitative nature (i.e. analysed via assay). This data was supported by logged weathering and diamond core and reverse circulation chip photographs.

This approach mimics that used to create weathering domains at Productora prior to 2024, but which did not account for the structural complexity observed by geologists at Productora from surface mapping and drillhole logging. In Figure 14.7, it should be noted that several low Cu:S intervals (<5) sit within the interpreted weathered zone. Conversely, there are intervals with high Cu:S in the interpreted fresh zone, often following the interpreted structures. This highlights the inherent inconsistencies present in the weathering at Productora and justifies the move away from a traditional horizontally interpreted weathering domain approach.

In total, six variables have been used to determine weathering classification (Table 14.18).

These data sets have been broken down into:

- Analysed variables (based on laboratory analysis) - three variables
- Implied variables (implied from laboratory analysis on similar samples) - one variable
- Subjective variables (based on geologist logging of core or chips) - two variables.

Table 14.18 : Variables Used To Determine Weathering Classification

Variable Type	Variable	Classification
Analysed Variable	Ratio Soluble Copper to Total Copper	Oxide (> 0.7)
		Transitional (0.3 – 0.7)
		Fresh (< 0.3)
	Ratio of Copper to Sulphur	Oxide (> 30)
		Transitional (5 – 30)
		Fresh (< 5)
	Copper Speciation (from Sequential Leach)	Oxide
		Transitional (including 'TransOxide' and 'TransFresh')
		Fresh (Sulphide)
Implied Variable	Copper Speciation (from Machine Learning)	Oxide
		Transitional (including 'TransOxide' and 'TransFresh')
		Fresh (Sulphide)
Subjective Variable	Logged Regolith Code	Oxide
		Transitional
		Fresh
	Logged Weathering Code	Oxide
		Transitional
		Fresh

Each variable has used a probabilistic estimate to determine the likelihood of each block being a certain weathering classification. A binary coding within each domain as defined above was estimated using ordinary kriging, resulting in an estimated value of between 0 and 1. A threshold value for each probability is then selected to determine the final classification of that block, verified by visual checks of the input data and geostatistical validation.

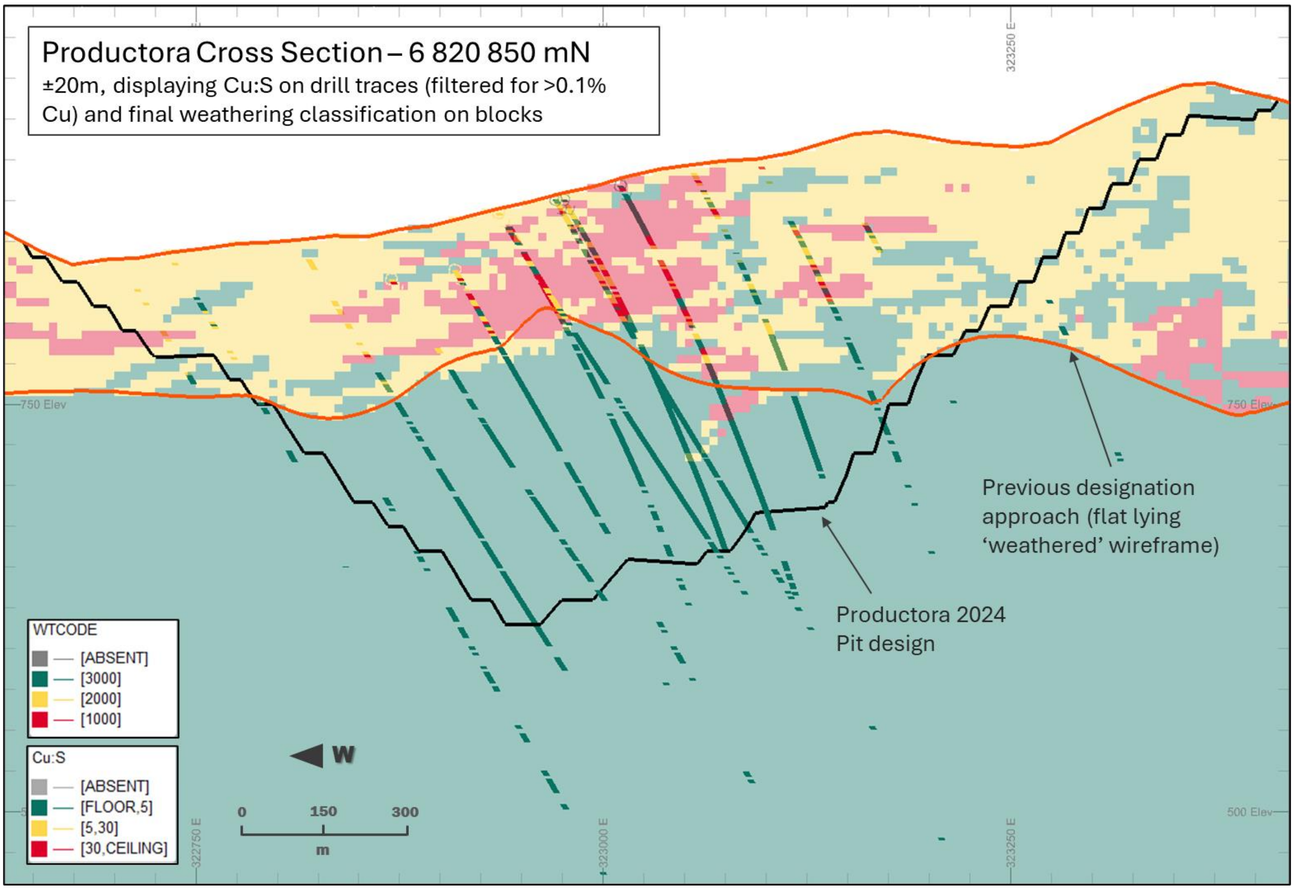
For the flat lying weathered zone, all six variables are estimated. For the vertical zone, there was no information available for ratio of soluble copper to total copper, so only five variables are estimated.

The resultant classification of weathering for each variable was validated against the input data set to ensure that the estimate was reasonable.

From each of the input estimates, a final decision on weathering classification for each block was made using a decision tree approach. Highest priorities were given to the analysed variables, followed by the implied variables, then the subjective variables.

Final weathering classification for a cross-section of Productora at 6,820,850 mN is shown in Figure 14.7 below.

Figure 14.7 : Final Assigned Weathering Classification – Cross Section at 6,820,850 mN Looking North. WTCODE = 1000 refers to oxide, 2000 refers to transition and 3000 to fresh material.



14.6.2 Alice

Weathering surfaces were constructed using a combination of qualitative measures such as visual geological logging of drilling for weathering, and quantitative analysis such as multi-element geochemistry (Cu:S) and Soluble Cu% to Total Cu% ratio.

Grade was estimated across weathering domains, due to the current assumption that most of the mineralisation within the oxide domain is in situ and comparable to primary mineralisation in tenor.

In some areas, possible supergene mineralisation and subsequent lateral movement is present, but this is not currently well understood. Variography and sample selection therefore did not differentiate between weathering domains.

14.6.3 Cortadera

The modelling of weathering domains at Cortadera is based on:

- Copper species flagging using Cu:S ratio
- Total contained sulphide (S2-) percent
- Regolith logging from diamond core and RC chips (validated by viewing photographs, where possible).

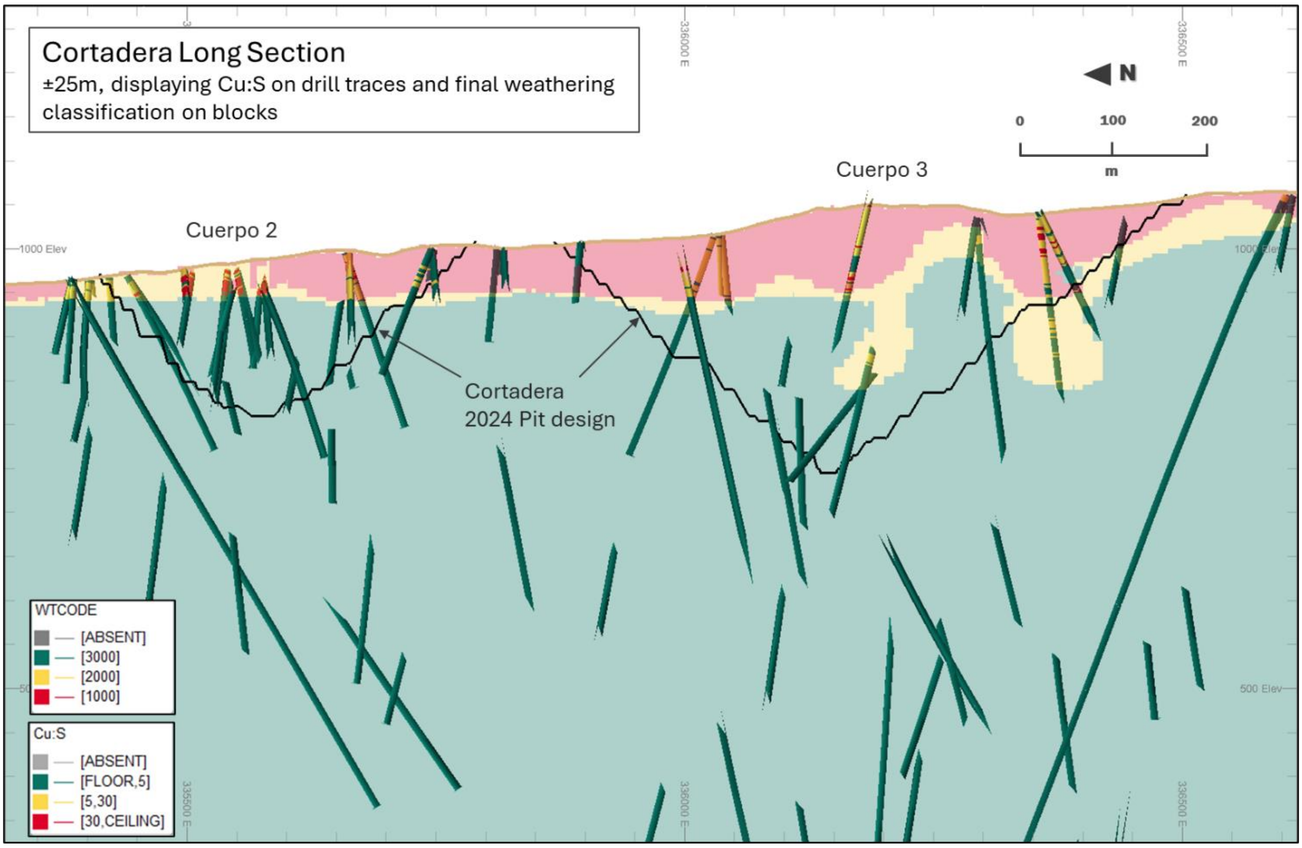
Where a quantitative method was used for interpretation of the weathering domains (i.e. Cu:S or total contained sulphide), validation was completed using the regolith logging from diamond core and RC chips.

When no quantitative information was available, regolith logging from diamond core and RC chips was used to interpret the weathering domains, which was checked by viewing core or RC chip photographs.

Validation of the surfaces and refinement of the weathering model included resolving conflicting logging in twinned drill holes and reassigning anomalous intervals to help build geologically coherent weathering model surfaces.

Figure 14.8 shows an example of the weathering model in cross-section. Note that the weathering profile is much deeper around the edges of the porphyry intrusion. This may be related to faulting, with the rock in these areas generally highly fractured. This observation was validated by viewing core and RC chip photographs and is supported by both the total contained S^{2-}% and the Cu:S ratio.

Figure 14.8 : Cross Section (Looking North-West) Showing Final Weathering Model (Fresh is green, Transitional is Yellow, Oxide is Red) Compared to Drill Holes Displaying Cu:S



14.6.4 San Antonio

The modelling of weathering domains at San Antonio is based on:

- Copper species flagging using Cu:S ratio
- Regolith logging from diamond core and RC chips (validated by viewing photographs, where possible).

Where a quantitative method was used for interpretation of the weathering domains (i.e. Cu:S) validation was completed using the regolith logging from diamond core and RC chips.

When no quantitative information was available, regolith logging from diamond core and RC chips was used to interpret the weathering domains, which was checked by viewing core or RC chip photographs.

14.7 Data Flagging

Geology, mineralisation, alteration, structure, and weathering wireframes were used to flag the drill hole data and block model for each model area. Model fields and coding in described in Table 14.19.

Table 14.19 : Drill Holes and Models Coding - by Project			
Project	**Field**	**Code**	**Description**
Productora	FAULTBLOCK	40000	South of Serrano fault
	FAULTBLOCK	50000	Between Serrano and Rancho faults
	FAULTBLOCK	60000	North of Rancho fault sub-domain
	FAULTBLOCK	70000	North of Rancho fault sub-domain
	FAULTBLOCK	80000	North of Rancho fault sub-domain
	FAULTBLOCK	90000	North of Rancho fault sub-domain
	TOU	0	Outside tourmaline breccia
	TOU	100	Inside tourmaline breccia
	CENDOM	1	Eastern orientation domain in Mid area
	CENDOM	2	Not eastern orientation domain
	SUL	0	Outside S:Na interpolant
	SUL	10	Inside S:Na interpolant
	PYR	0	Outside pyrite interpolant
	PYR	1	Inside pyrite interpolant
	WEATH	1000	Weathered
	WEATH	3000	Fresh
Alice	LTCODE	1	Intermediate Volcaniclastics Lithology
	LTCODE	3	Felsic Volcanics Lithology
	LTCODE	20	Felsic Porphyry Lithology
	WTCODE	1000	Oxide Weathering
	WTCODE	2000	Transitional Weathering
	WTCODE	3000	Fresh Weathering
	CUEQAREA	1	Inside of 0.025% CuEq Interpolant
	DOM_CU1	10	Inside of 0.4% Cu Interpolant
	DOM_CU1	20	Inside of 0.2% Cu Interpolant
	DOM_AG1	10	Inside of 0.2 ppm Ag Interpolant
	DOM_MO1	10	Inside of 50 ppm Mo Interpolant
	DOM_MO1	20	Inside of 20 ppm Mo Interpolant
	DOM_CO1	10	Inside of 60 ppm Co Interpolant
	DOM_K1	10	Inside of 1.0% K Interpolant
	DOM_CA1	10	Inside of 1.5% Ca Interpolant
	DOM_PY1	10	Inside of 1.0% Pyrite Interpolant
Cortadera	CUERPO	1	Cuerpo 1 model area
	CUERPO	2	Cuerpo 2 model area
	CUERPO	3	Cuerpo 3 model area
	LTCODE	6	Distal Skarn lithology coding
	LTCODE	5	Proximal Skarn lithology coding

Table 14.19 : Drill Holes and Models Coding - by Project			
Project	**Field**	**Code**	**Description**
	LTCODE	2	Volcanic host coding
	LTCODE	20	Intramineral porphyry coding
	LTCODE	10	Early mineralised porphyry coding
	LTCODE	32	Late mineral felsic dyke stock coding
	LTCODE	31	Late mineral felsic dyke eye coding
	LTCODE	30	Late mineral felsic dyke system coding
	LTCODE	40	Late mineral andesitic dyke system coding
	TOPO	10000	Topographic surface coding
	WTCODE	1	Oxide weathering coding
	WTCODE	2	Transitional weathering coding
	WTCODE	3	Fresh rock weathering coding
San Antonio	DOM_CU1	100	Low Grade Cu interpolant
	DOM_CU1	0	Main lode Cu interpolant
	DOM_CU1	1	Main splay Cu interpolant
	DOM_CU1	2	Splay lode Cu interpolant
	DOM_CU1	3	Splay lode Cu interpolant
	DOM_CU1	4	Splay lode Cu interpolant
	DOM_CU1	5	Splay lode Cu interpolant
	DOM_CU1	6	Splay lode Cu interpolant
	DOM_CU1	7	Splay lode Cu interpolant
	DOM_CU1	8	Splay lode Cu interpolant
	DOM_CU1	9	Splay lode Cu interpolant
	DOM_CU1	10	Splay lode Cu interpolant
	TOPO	1000	Topographic surface coding
	WTCODE	3000	Fresh Weathering
	WTCODE	2000	Transitional Weathering
	WTCODE	1000	Oxide Weathering

14.8 Block Modelling

For each model area, parent block size was selected to ensure a realistic grade estimate was achieved in each block, considering the average drill hole spacing and mineralisation orientation. Sub-celling was set at a level to provide sufficient resolution of the blocks compared to the wireframes and mineralisation characteristics. Block model dimensions are shown in Table 14.20.

Table 14.20 : Block Model Dimensions						
Project	**Dimension**	**Minimum**	**Maximum**	**Extent (m)**	**Block Size (m)**	
					Parent	**Minimum**
Productora	Easting	321,850	325,050	2,800	5.0	5.0

Table 14.20 : Block Model Dimensions						
Project	Dimension	Minimum	Maximum	Extent (m)	Block Size (m)	
					Parent	Minimum
	Northing	6,819,000	6,827,400	8,400	20.0	5.0
	Elevation	200	1 200	1,000	5.0	5.0
Alice	Easting	322,340	323,220	880	10.0	5.0
	Northing	6,822,100	6,823,000	900	10.0	5.0
	Elevation	30	1,030	1,000	10.0	5.0
Cortadera (small block)	Easting	334,100	337,100	3,000	10.0	1.0
	Northing	6,812,700	6,814,780	2,080	10.0	1.0
	Elevation	-400	1,280	1,680	10.0	1.0
San Antonio	Easting	342,000	342,700	700	10.0	1.0
	Northing	6,818,200	6,819,500	1,300	10.0	1.0
	Elevation	900	1,300	400	10.0	1.0

14.9 Mining, Tenement, and Royalty Flagging in the Model

14.9.1 Productora

Historic underground working exist off two separate portals at Productora. Surveys of the underground workings are available, and they have been depleted from the reported Mineral Resource.

Productora tenements are flagged into the 'TENEMENT' field in the final estimate using binary code (0 = inside of HCH tenement boundary and 1 = outside of tenement boundary). All tenements within the Productora model area are under HCH control.

There are three tenements within the model area which contain a royalty. These are coded into the final model using the RYLTCODE field.

- RLTYCODE = 0 : No royalty owed
- RYLTCODE = 1 : Zapa Royalty (Zapa 1 Al 6 : Subject to 1% Gross royalty on all products)
- RYLTCODE = 2 = Montosa Royalty (Montosa 1 al 4 : Subject to 3% NSR royalty on all products)
- RLTYCODE = 3 : CCHEN Royalty (Uranio 1 al 70 : Subject to 2% non-gold NSR royalty, 4% gold NSR royalty, and 5% non-metallic NSR royalty)

14.9.2 Alice

All tenements within the Alice model area are under HCH control.

There are two tenements which contain a royalty. These are coded into the final model using the RYLTCODE field.

- RYLTCODE = 0 : No Royalty Owed
- RYLTCODE = 1 : Zapa Royalty (Zapa 1 Al 6 : Subject to 1% Gross royalty on all products)
- RYLTCODE = 2 = Montosa Royalty (Montosa 1 al 4 : Subject to 3% NSR royalty on all products)

No surface workings have been completed in the Alice model area.

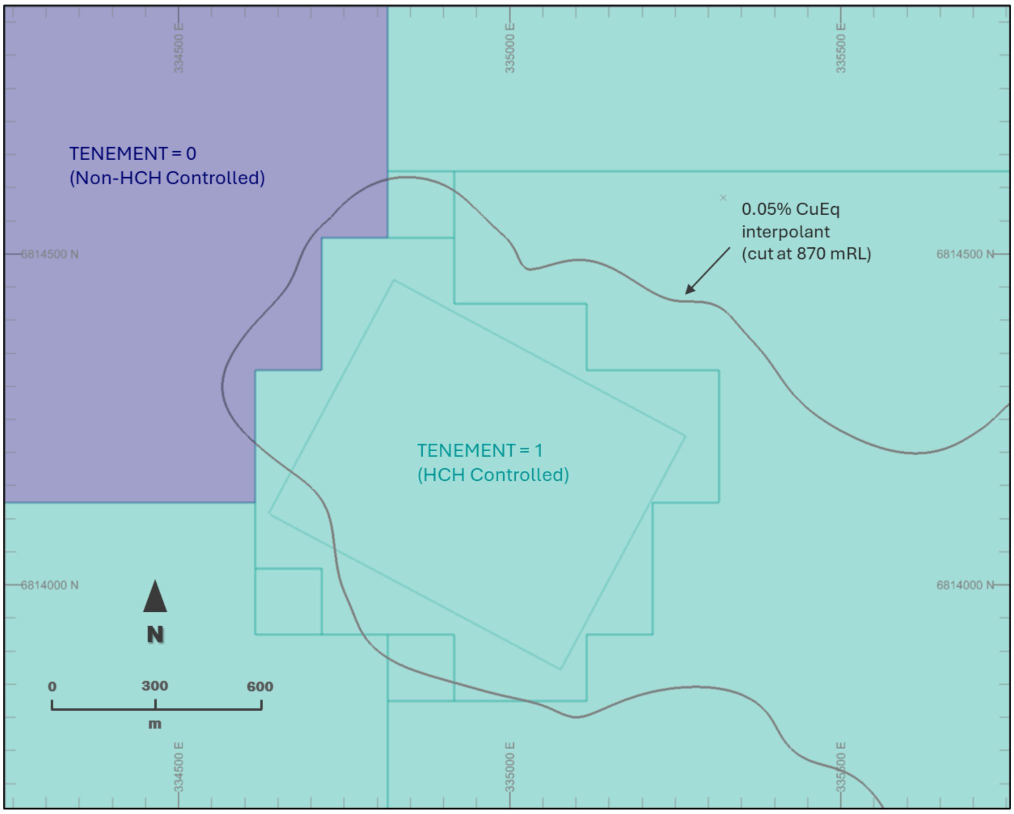
14.9.3 Cortadera

Cortadera tenements are flagged into the 'TENEMENT' field in the final estimate using binary code (0 = inside of HCH tenement boundary and 1 = outside of tenement boundary).

As shown below (Figure 14.10) a small section of Cuerpo 1 lies outside the current HCH tenement boundary. This material has been excluded from the reported Mineral Resource.

Only limited surface workings have been completed in the Cortadera model area. Depletion of this material is completed by use of a detailed topography model.

Figure 14.9 : Plan View of the Area of Cuerpo 1 Which Lies Outside the Current HCH Tenement Boundary



14.9.4 San Antonio

The San Antonio model is situated inside a single tenement, so tenements were not flagged into the Resource Estimate.

A drone survey using a mobile laser scanner was completed in August 2021 by Aerodyne Chile. In total, 23 scans were competed of the underground environment, as well as eight scans covering the outdoor terrain. The purpose of the survey was to confirm location and extent of the current San Antonio underground workings (including development headings and stoped voids).

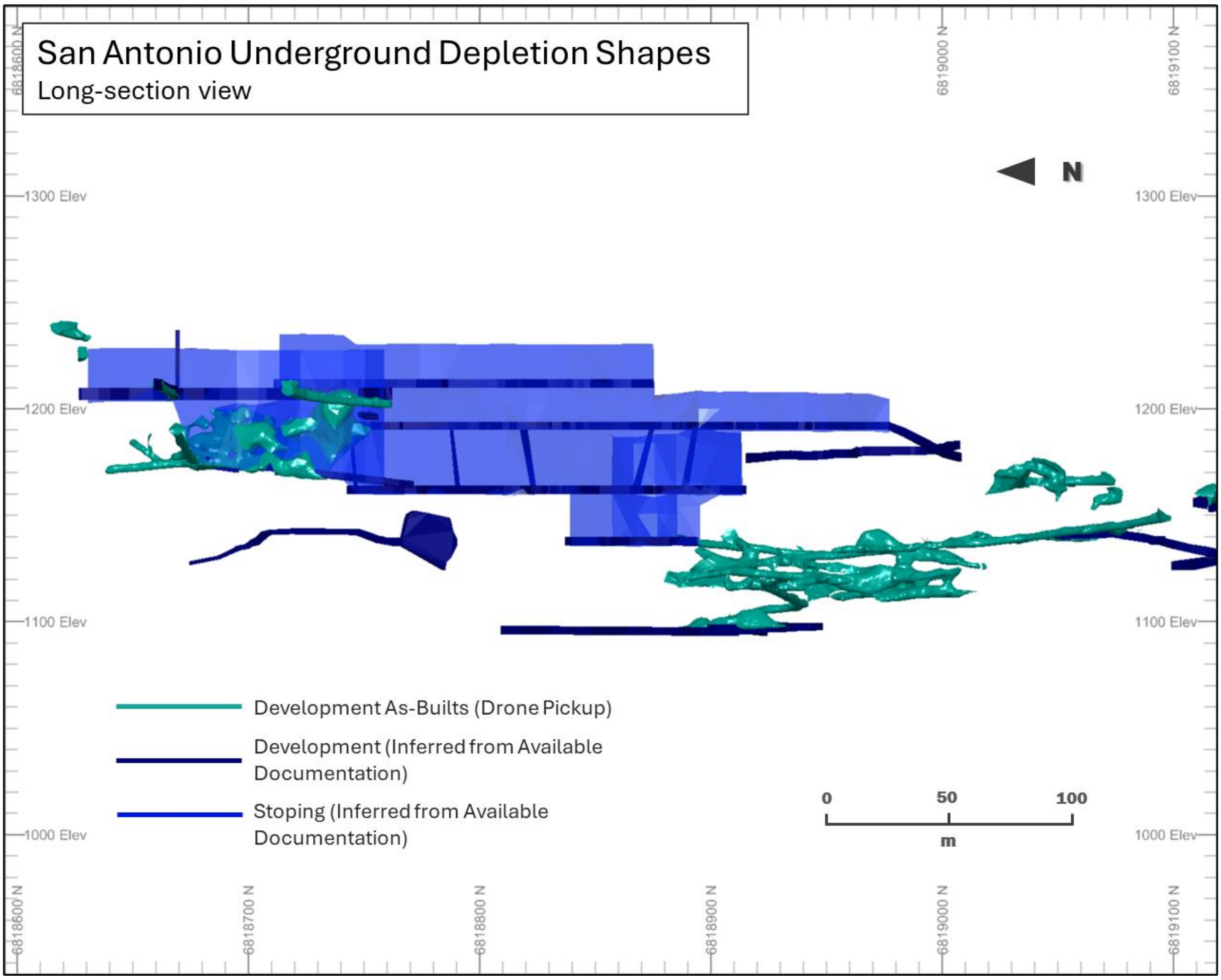
While the drone survey was successful in validating the spatial interpretation of mineralised wireframes, it was unable to access all areas of the mine due to poor ground conditions (falls of ground). As a result, assumptions have been made with regards to the depletion of the San Antonio resource.

It is also noted that underground mining is ongoing by a third-party at San Antonio as part of HCHs option agreement on the Project. The agreement allows for a maximum of 50 kt to be extracted per annum, which comprises less than 1% of the total Mineral Resource at San Antonio, so is not considered material.

For the current MRE , an interpreted depletion shape has been created by digitising sections on 10 m spacing (E-W). This shape combines the drone survey, inferred development shapes, and ongoing underground mining (Figure 14.10).

Total depletion for San Antonio is 1.5 Mt @ 1.1% CuEq (with no grade cut-off applied). This excludes open pit depletion, which cannot be calculated due to the lack of pre-mining topography at San Antonio.

Figure 14.10 : Long Section Looking East Showing Inputs into Final Depletion Shape



14.10 Compositing

For the Productora, Alice, and San Antonio projects, a majority of drill sampling was completed at 1 m intervals. At Cortadera, 80% of the drill sampling was at either 1 m or 2 m intervals. In all cases, these proportions increased within the mineralised domains where copper was greater than 0.1%.

For Productora and San Antonio, the decision to composite data to 1 m was based on the length of the sample interval, the short-range mineral and structural control and relatively narrow mineralised zones in some areas.

For Alice, compositing was completed using 2 m intervals to avoid splitting of data (where single assay intervals are broken up into multiple equal assay value, shorter intervals. 2 m is considered suitable at Alice due to the homogeneity of mineralisation.

For Productora, Alice, and San Antonio, the compositing process used the relevant domain and weathering as a boundary to ensure no composites were created across domains.

For Cortadera, 2 m was chosen as a suitable composite length due to the relative homogeneity of the mineralisation. Compositing used mineralisation, lithology, and weathering domains as boundaries to ensure no composites were created across domains.

Table 14.21 outlines the percentages of samples at 1 m intervals globally and inside of the 0.1% copper interpolant for each project area.

Table 14.21 : Sample Lengths to Support Compositing Regime

Project	Percentage of samples at 1 m interval	Percentage of samples at 1 m interval (inside of 0.1% Copper interpolant)
Productora	82%	97%
Alice	70%	91%
Cortadera	80% (1 m and 2 m combined)	95%
San Antonio	90%	95%

14.11 Statistical Analysis

Statistical analysis of copper, gold, molybdenum, silver, cobalt, calcium, potassium, aluminium, and rhenium were undertaken using Snowden Supervisor Version 8.14.3.2 software and Microsoft Excel. Analysis was completed to understand the global distribution of each element and account for any bias introduced by clustering of data or outliers.

14.11.1 Top Cutting

Where required, top-cuts were applied to the composited sample data to reduce the impact of outlier values on the mean grade and coefficient of variation (CV), and subsequent estimation of grades. Using outliers in an estimate can result in material overestimation of grade and metal. For each project area, element and domain, log histograms, log probability plots, and grade disintegration were examined. The top-cuts were chosen to reduce the potential smearing of extremely high grades.

Top-cuts for each project area are described in more detail below.

14.11.1.1 Productora

For Productora, due to the comprehensive domaining applied, few top cuts were required. This is supported by the absence of any genuine outliers within the domains, as well as the low coefficient of variation (CV).

Statistical analysis of the composites and effect of top-cutting is shown for copper and gold in Table 14.22.

Table 14.22 : Productora - Copper and Gold Top-Cut Analysis

Domain				Raw				Top-cut		%Diff Raw to Top-cut			
Code	Lith	Min	Number	Min	Max	Mean	CV	Value	Num cut	Mean	CV	Mean (%)	CV (%)
12	F3B(1,2)	HG Cu	1,058	0.002	4.28	0.52	0.80	5.00	-	0.52	0.80	0.0%	0.0%
22	Stg2	HG Cu	8,925	0.001	10.55	0.60	0.77	5.00	4	0.60	0.75	-0.2%	-2.8%
32	Bckg	HG Cu	504	0.001	4.03	0.56	0.91	5.00	-	0.56	0.91	0.0%	0.0%
52	RRLT	HG Cu	719	0.001	3.72	0.42	0.77	5.00	-	0.42	0.77	0.0%	0.0%
62	OxFlat	HG Cu	2,838	0.002	11.47	0.67	0.98	5.00	7	0.66	0.90	-0.8%	-7.4%
11	F3B(1,2)	LG Cu	1,706	0.001	1.59	0.14	0.86	5.00	-	0.14	0.86	0.0%	0.0%
21	Stg2	LG Cu	9,198	0.001	2.48	0.16	0.76	5.00	-	0.16	0.76	0.0%	0.0%
31	Bckg	LG Cu	4,572	0.003	1.83	0.11	0.81	5.00	-	0.11	0.82	0.0%	0.0%
51	RRLT	LG Cu	7,601	0.001	2.42	0.11	0.86	5.00	-	0.11	0.86	0.0%	0.0%
61	OxFlat	LG Cu	11,868	0.001	2.73	0.12	0.70	5.00	-	0.12	0.70	0.0%	0.0%
19	F3B(1,2)	VLG Cu	6,630	0.001	4.17	0.12	1.90	1.50	20	0.12	1.80	-1.4%	-5.4%
29	Stg2	VLG Cu	15,325	0.001	5.01	0.22	1.48	1.50	116	0.21	1.40	-1.7%	-5.3%
39	Bckg	VLG Cu	56,678	0.001	3.83	0.04	2.49	1.50	7	0.04	2.46	-0.1%	-1.2%
59	RRLT	VLG Cu	52,388	0.001	3.01	0.04	2.11	1.50	11	0.04	1.48	-0.2%	-2.5%
69	OxFlat	VLG Cu	64,929	0.001	4.94	0.06	2.71	1.50	134	0.06	2.43	-2.5%	-11%
12	F3B(1,2)	HG Au	919	0.003	1.12	0.12	0.94	1.50	-	0.12	0.94	0.0%	0.0%
22	Stg2	HG Au	8,058	0.001	3.11	0.14	0.96	1.50	8	0.14	0.91	-0.4%	-5.9%
32	Bckg	HG Au	425	0.001	0.87	0.11	0.96	1.50	-	0.11	0.96	0.0%	0.0%
52	RRLT	HG Au	598	0.001	2.56	0.11	1.69	1.50	2	0.11	1.42	-2.8%	-16%
62	OxFlat	HG Au	1,935	0.001	3.07	0.09	1.64	1.50	5	0.09	1.32	-2.3%	-20%
11	F3B(1,2)	LG Au	706	0.001	1.44	0.05	1.47	1.50	-	0.05	1.47	0.0%	0.0%
21	Stg2	LG Au	5,049	0.001	0.97	0.05	0.91	1.50	-	0.05	0.91	0.0%	0.0%
31	Bckg	LG Au	1,587	0.002	0.57	0.04	1.17	1.50	-	0.04	1.17	0.0%	0.0%
51	RRLT	LG Au	2,831	0.001	2.74	0.04	1.82	1.50	1	0.04	1.54	-1.0%	-16%
61	OxFlat	LG Au	4,648	0.001	14.15	0.05	4.42	1.50	8	0.05	1.99	-6.7%	-55%
19	F3B(1,2)	VLG Au	2,088	0.001	2.34	0.06	1.66	1.50	2	0.06	1.49	-0.9%	-10%
29	Stg2	VLG Au	7,434	0.001	1.45	0.08	1.07	1.50	-	0.08	1.07	0.0%	0.0%
39	Bckg	VLG Au	17,108	0.001	57.2	0.03	16.3	1.50	20	0.02	3.36	-25%	-79%
59	RRLT	VLG Au	8,417	0.001	1.57	0.03	1.94	1.50	1	0.03	1.93	-0.0%	-0.3%
69	OxFlat	VLG Au	10,319	0.001	2.29	0.06	1.69	1.50	5	0.06	1.64	-0.3%	-3.0%

14.11.1.2 Alice

For Alice, due to the homogenous mineralisation combined with the use of 2 m composites, only limited top-cutting of outlier values was required. Statistical analysis of the composites and the effect of top-cuts for key elements is provided in Table 14.23. This is supported by the absence of any genuine outliers within the domains, as well as the low coefficient of variation (CV).

Table 14.23 : Alice – Copper, Gold, Silver, and Molybdenum Top-Cut Analysis

Domain			Raw				Top-cut		%Diff Raw to Top-cut			
Domain	**Element**	**Number**	**Min**	**Max**	**Mean**	**CV**	**Value**	**Num cut**	**Mean**	**CV**	**Mean (%)**	**CV (%)**
DOM_CU1 = 10	Cu	558	0.01	1.85	0.55	0.45	-	-	0.55	0.45	0.0%	0.0%
DOM_CU1 = 20	Cu	645	0.01	0.85	0.29	0.50	-	-	0.29	0.50	0.0%	0.0%
DOM_CU1 = 50	Cu	5,118	0.01	1.08	0.04	1.34	0.40	7	0.04	1.28	-0.5%	-4.0%
DOM_CU1 = 90	Cu	3,374	0.01	0.41	0.01	1.73	0.15	2	0.01	1.49	-1.6%	-14.0%
DOM_CU1 = 50	Au	702	0.001	2.54	0.02	5.08	0.25	2	0.02	1.18	-23.9%	-76.8%
DOM_CU1 = 90	Au	121	0.003	0.08	0.01	1.71	0.04	3	0.01	1.33	-8.2%	-22.3%
DOM_AG1 = 20	Ag	995	0.01	12.7	0.86	1.60	5.00	5	0.81	1.40	-6.0%	-13.0%
DOM_AG1 = 50	Ag	5,381	0.01	3.85	0.08	2.78	-	-	0.01	3.85	0.0%	0.0%
DOM_AG1 = 90	Ag	3,323	0.01	2.80	0.04	3.40	-	-	0.01	2.80	0.0%	0.0%
DOM_MO1 = 10	Mo	635	1.12	623	84.8	0.96	400	9	83.6	0.91	-1.3%	-5.5%
DOM_MO1 = 20	Mo	1,042	0.1	448	32.6	1.05	200	8	31.8	0.88	-2.4%	-15.9%
DOM_MO1 = 50	Mo	4,648	0.1	381	6.25	1.97	100	12	6.08	1.57	-2.8%	-20.2%
DOM_MO1 = 90	Mo	3,374	0.1	285	3.00	3.65	40.0	19	2.54	1.76	-15.2%	-51.6%

14.11.1.3 Cortadera

For Cortadera, due to the homogenous mineralisation combined with the use of 2 m composites, only limited top-cutting of outlier values was required. Statistical analysis of the composites and the effect of top-cuts for key elements is provided in Table 14.24.

Table 14.24 : Cortadera - Copper, Gold, Silver, and Molybdenum Top-Cut Analysis											
Element	Mine Area	Domain	CIK Sub-domain	Raw				Top cut			
				Number	Max	Mean	CV	Value	# Cut	Mean	CV
Cu	Cuerpo 1	DOM_CU1 = 20	HG_CIK	687	2.54	0.50	0.58	1.5	9	0.49	0.54
			LG_CIK	520	0.83	0.22	0.42	-	-	0.22	0.42
		DOM_CU1 = 30	HG_CIK	440	0.98	0.22	0.42	0.5	4	0.22	0.36
			LG_CIK	1,210	0.56	0.14	0.44	0.5	2	0.14	0.44
		DOM_CU1 = 50	HG_CIK	591	0.66	0.14	0.51	-	-	0.14	0.51
			LG_CIK	4,692	1.58	0.11	0.92	0.3	158	0.11	0.81
	Cuerpo 2	DOM_CU1 = 20	HG_CIK	855	3.88	0.49	0.86	1.5	26	0.46	0.59
			LG_CIK	396	1.90	0.22	0.60	1.5	1	0.22	0.55
		DOM_CU1 = 30	HG_CIK	909	1.58	0.27	0.45	0.65	3	0.27	0.45
			LG_CIK	1,421	1.03	0.15	0.44	0.65	1	0.15	0.44
		DOM_CU1 = 50	HG_CIK	715	0.86	0.13	0.48	0.3	11	0.13	0.36
			LG_CIK	7,168	1.58	0.11	0.93	0.3	161	0.11	0.81
	Cuerpo 3	DOM_CU1 = 20	HG_CIK	2,684	1.99	0.65	0.33	1.4	16	0.65	0.32
			LG_CIK	3,754	1.14	0.34	0.37	-	-	0.34	0.37
		DOM_CU1 = 30	HG_CIK	3,473	1.11	0.27	0.37	-	-	0.27	0.37
			LG_CIK	3,723	1.41	0.16	0.43	1.4	1	0.16	0.43
		DOM_CU1 = 50	HG_CIK	84	1.14	0.31	0.52	0.5	13	0.29	0.39
			LG_CIK	18,299	1.58	0.11	0.91	0.5	185	0.11	0.87
Au	Cuerpo 1	DOM_CU1 = 30	-	549	0.64	0.10	0.68	0.3	10	0.10	0.58
		DOM_CU1 = 50	-	5,834	0.48	0.02	1.26	0.1	73	0.02	1.00
	Cuerpo 2	DOM_CU1 = 20	HG_CIK	178	1.31	0.35	0.60	1.0	1	0.32	0.52
			LG_CIK	440	2.10	0.21	1.01	0.4	7	0.18	0.83
		DOM_CU1 = 30	HG_CIK	908	0.97	0.13	0.49	0.4	3	0.13	0.42
			LG_CIK	1,642	2.10	0.14	1.20	0.4	66	0.12	0.78
		DOM_CU1 = 50	HG_CIK	128	0.56	0.10	0.67	0.25	3	0.10	0.53
			LG_CIK	6,105	0.71	0.02	1.05	0.25	2	0.02	0.97
	Cuerpo 3	DOM_CU1 = 20	HG_CIK	2,249	0.98	0.25	0.48	-	-	0.25	0.48
			LG_CIK	4,216	0.57	0.13	0.54	-	-	0.13	0.54
		DOM_CU1 = 30	HG_CIK	3,489	0.58	0.09	0.53	-	-	0.09	0.53
			LG_CIK	10,210	0.98	0.13	0.83	-	-	0.13	0.83
		DOM_CU1 = 50	HG_CIK	74	0.49	0.07	1.14	-	-	0.07	1.14
			LG_CIK	12,112	4.22	0.02	2.05	0.4	9	0.02	1.16
		DOM_AG1 = 20	-	444	8.70	1.23	0.82	-	-	1.23	0.82

Table 14.24 : Cortadera - Copper, Gold, Silver, and Molybdenum Top-Cut Analysis											
Element	**Mine Area**	**Domain**	**CIK Sub-domain**	**Raw**				**Top cut**			
				Number	**Max**	**Mean**	**CV**	**Value**	**# Cut**	**Mean**	**CV**
	Cuerpo 1	DOM_AG1 = 30	-	775	10.3	0.69	1.18	5.0	5	0.68	1.11
Ag		DOM_AG1 = 50	-	4,907	3.00	0.16	1.93	1.0	131	0.15	1.80
		DOM_AG1 = 20	-	440	9.00	1.23	0.95	-	-	1.23	0.95
	Cuerpo 2	DOM_AG1 = 50	-	1,381	14.9	0.74	0.85	2.5	17	0.66	0.75
		DOM_AG1 = 30	-	6,171	14.3	0.34	0.85	1.0	163	0.31	0.81
		DOM_AG1 = 20	-	2,932	100	1.28	1.56	3.5	30	1.22	0.51
	Cuerpo 3	DOM_AG1 = 30	-	5,630	8.00	0.73	0.68	2.0	113	0.66	0.60
		DOM_AG1 = 50	-	17,189	59.6	0.33	1.99	1.0	404	0.13	1.64
Mo		DOM_MO1 = 20	-	429	523	91.9	0.68	250	14	89.8	0.61
	Cuerpo 1	DOM_MO1 = 30	-	932	719	59.9	0.71	170	14	58.6	0.57
		DOM_MO1 = 50	-	5,022	2 100	18.5	2.03	100	73	17.5	1.14
		DOM_MO1 = 20	-	195	581	122	0.78	350	6	118	0.68
	Cuerpo 2	DOM_MO1 = 30	-	344	308	62.5	0.64	200	3	61.8	0.60
		DOM_MO1 = 50	-	8,280	1 100	15.9	1.65	100	96	15.1	1.17
		DOM_MO1 = 20	-	817	2 980	339	0.91	1 250	14	328	0.77
	Cuerpo 3	DOM_MO1 = 30	-	1,225	2 410	180	0.91	600	23	173	0.71
		DOM_MO1 = 50	-	23,984	2 670	34.5	1.72	100	1 842	29.0	1.05

In addition to conventional top-cutting, additional distance-controlled capping was used. This was deemed necessary to prevent high-grades from spreading indiscriminately across soft boundaries (described in 14.13.3), especially in areas of sparser data density. The Studio RM COKRIG function allows for a 'capdist' (distance beyond which the cap is applied) and 'capgrade' (the capping grade value).

The capping methodology 'capped' the high value and replaced with the cap value (i.e., a value of 2.5% Cu with a cap value of 1.0% Cu would be replaced with the 1.0% value beyond the cap distance).

For the Cortadera estimate, cap distances and grades were informed by boundary analysis completed between domains which had a soft boundary applied, and by the combined domains data distribution. Distance-controlled capping was only used for copper and gold estimates.

Distance-controlled capping is summarised in Table 14.25.

Table 14.25 : Cortadera Distance Controlled Capping Summary					
AREA	**Element**	**Domain**	**Cap Value**	**Cap Distance (m)**	**Condition**
Cuerpo 1	Cu	DOM_CU1 = 20	0.50	10	One-way semi-soft boundary between HG_CIK and LG_CIK domains
		DOM_CU1 = 30	0.50	30	One-way semi-soft boundary between DOM_CU1 = 20 and HG_CIK domains
		DOM_CU1 = 30	0.35	30	One-way semi-soft boundary between HG_CIK and LG_CIK domains
		DOM_CU1 = 50	0.50	50	One-way semi-soft boundary between DOM_CU1 = 30 and HG_CIK domains
		DOM_CU1 = 50	0.30	50	One-way semi-soft boundary between HG_CIK and LG_CIK domains
Cuerpo 2	Cu	DOM_CU1 = 20	0.50	10	One-way semi-soft boundary between HG_CIK and LG_CIK domains
		DOM_CU1 = 30	1.00	30	One-way semi-soft boundary between DOM_CU1 = 20 and HG_CIK domains
		DOM_CU1 = 30	0.65	30	One-way semi-soft boundary between HG_CIK and LG_CIK domains
		DOM_CU1 = 50	0.75	50	One-way semi-soft boundary between DOM_CU1 = 30 and HG_CIK domains
		DOM_CU1 = 50	0.50	50	One-way semi-soft boundary between HG_CIK and LG_CIK domains
	Au	DOM_CU1 = 20	0.50	10	One-way semi-soft boundary between HG_CIK and LG_CIK domains
		DOM_CU1 = 30	0.75	30	One-way semi-soft boundary between DOM_CU1 = 20 and HG_CIK domains
		DOM_CU1 = 30	0.40	30	One-way semi-soft boundary between HG_CIK and LG_CIK domains
		DOM_CU1 = 50	0.25	50	One-way semi-soft boundary between DOM_CU1 = 30 and HG_CIK domains
		DOM_CU1 = 50	0.10	50	One-way semi-soft boundary between HG_CIK and LG_CIK domains
Cuerpo 3	Cu	DOM_CU1 = 20	0.50	10	One-way semi-soft boundary between HG_CIK and LG_CIK domains
		DOM_CU1 = 30	1.00	30	One-way semi-soft boundary between DOM_CU1 = 20 and HG_CIK domains
		DOM_CU1 = 30	0.65	30	One-way semi-soft boundary between HG_CIK and LG_CIK domains
		DOM_CU1 = 50	0.75	50	One-way semi-soft boundary between DOM_CU1 = 30 and HG_CIK domains
		DOM_CU1 = 50	0.50	50	One-way semi-soft boundary between HG_CIK and LG_CIK domains

	Au	DOM_CU1 = 20	0.40	10	One-way semi-soft boundary between HG_CIK and LG_CIK domains
		DOM_CU1 = 30	0.30	30	One-way semi-soft boundary between DOM_CU1 = 20 and HG_CIK domains
		DOM_CU1 = 30	0.20	30	One-way semi-soft boundary between HG_CIK and LG_CIK domains
		DOM_CU1 = 50	0.20	50	One-way semi-soft boundary between DOM_CU1 = 30 and HG_CIK domains
		DOM_CU1 = 50	0.10	50	One-way semi-soft boundary between HG_CIK and LG_CIK domains

14.11.1.4 San Antonio

Top-cuts for copper and silver used for the San Antonio project summarised in Table 14.26. No other elements had top-cuts applied due to an absence of outlier values.

Table 14.26 : San Antonio – Copper, and Silver Top-Cut Analysis

		Raw				Top-Cut			
Element	**Domain**	**Number**	**Max**	**Mean**	**CV**	**Value**	**# Cut**	**Mean**	**CV**
Cu	DOM_CU1=0	295	8.55	1.60	0.68	3.5	1	1.49	0.58
	DOM_CU1=1	66	4.65	1.03	1.20	2.5	5	0.88	1.04
	DOM_CU1=2	46	2.66	0.63	1.41	2.0	7	0.58	1.08
	DOM_CU1=3	101	5.40	1.38	1.03	3.5	12	1.10	0.84
	DOM_CU1=4	71	3.75	0.96	0.93	3.0	2	0.94	0.89
	DOM_CU1=5	31	3.86	0.81	1.13	2.0	4	0.74	0.82
	DOM_CU1=6	5	1.43	0.78	0.55	-	-	0.78	0.55
	DOM_CU1=7	41	2.17	0.59	0.89	1.5	3	0.56	0.84
	DOM_CU1=8	18	3.00	0.99	0.93	2.5	1	0.96	0.90
	DOM_CU1=9	15	0.88	0.28	1.29	-	-	0.88	0.28
	DOM_CU1=10	41	0.56	0.25	0.61	-	-	0.25	0.61
Ag	DOM_CU1=0	134	8.50	2.80	0.69	6.0	9	2.74	0.66
	DOM_CU1=1	60	13.8	2.92	1.14	8.0	7	2.64	1.01
	DOM_CU1=2	18	5.80	1.93	0.86	4.0	2	1.75	0.74
	DOM_CU1=3	47	14.2	3.63	1.00	7.0	6	3.04	0.79
	DOM_CU1=4	67	82.5	3.73	2.66	6.0	5	2.51	0.77
	DOM_CU1=5	30	9.00	2.21	1.07	6.0	3	2.05	0.98
	DOM_CU1=6	5	2.30	1.17	0.56	-	-	1.17	0.56
	DOM_CU1=7	32	9.40	1.66	1.23	3.0	5	1.26	0.83
	DOM_CU1=8	16	6.90	3.20	0.70	-	-	3.20	0.70
	DOM_CU1=9	9	2.50	0.77	0.98	-	-	0.77	0.98
	DOM_CU1=10	39	1.40	0.59	0.50	-	-	0.59	0.50

14.12 Variography

Snowden Supervisor Version 8.14.3.2 software was used to generate and model the variograms for each of the elements and domains in all project areas. A normal score transformation was applied to each of the variables prior to experimental variogram calculation. The normal score variogram model's variance was back-transformed to traditional space after modelling to adjust the variance using hermite polynomials in the Supervisor software. All variograms used spherical models.

Knowledge of the underlying geological conditions was always considered when constructing variograms. Variogram orientations were overlain onto each domain during the validation step to ensure trends are coherent and agree with the broader geological model.

Care was taken to ensure reasonable correlation between any sub-domains (orientations, nugget effect and variogram ranges).

14.12.1 Productora

Correlation coefficients between different elements were used to guide the variogram modelling, with moderate to high correlations between elements indicating that similar ranges of continuity should be observed for those elements.

In some cases, domains with similar characteristics were combined for continuity analysis to provide the most robust data for analysis.

Due to the reduced composite count and more widely spaced data, the variogram models for the north area are not as robust as for the mid and south areas. In this case there has been only one variogram model created for each element, which has generally been interpreted from the higher-grade domains for the elements.

The back-transformed variogram model parameters used for resource estimation, broken down by FAULTBLOCK, are shown for copper and gold in Table 14.27.

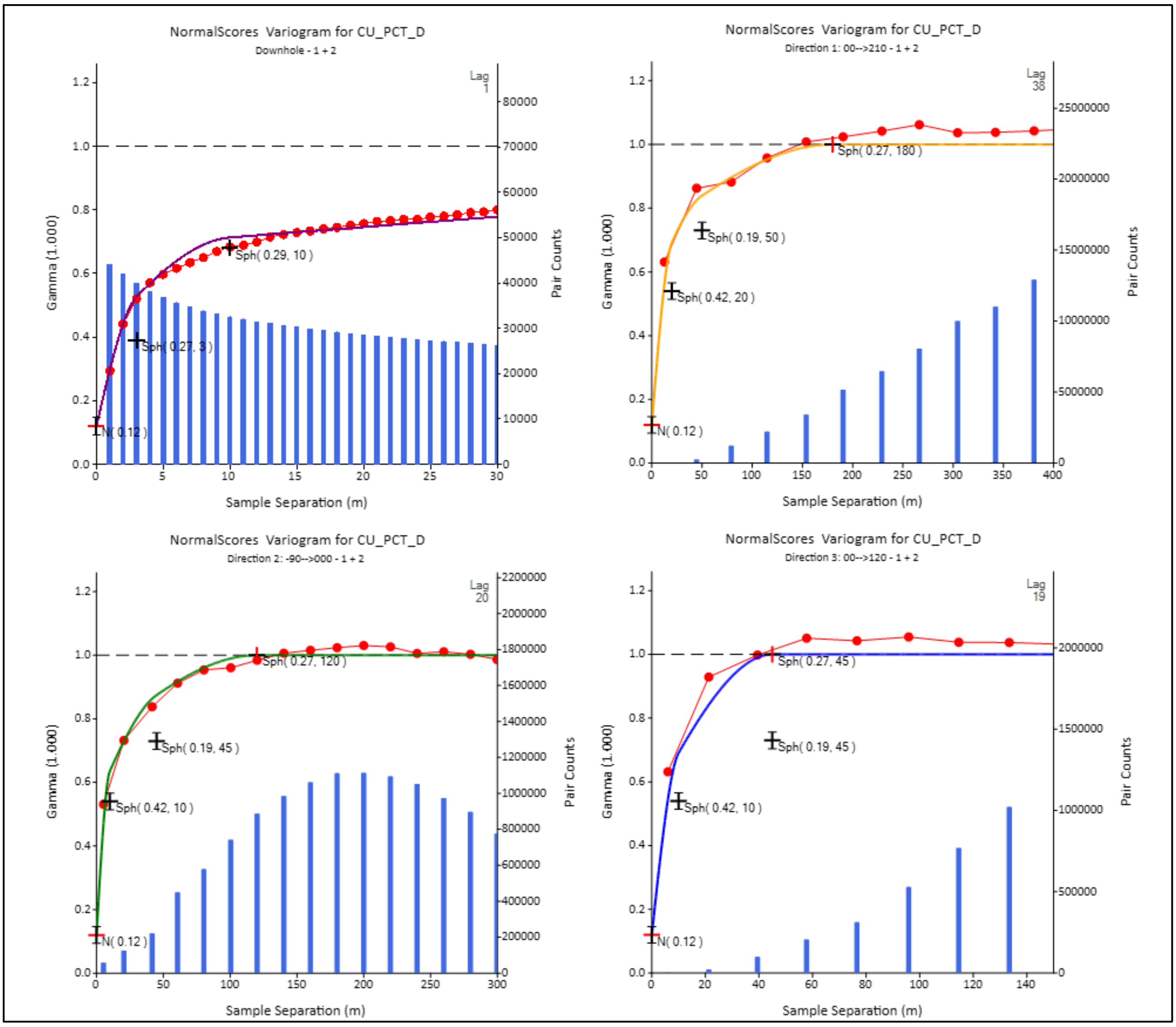
The normal score variogram model for mineralised copper is illustrated in Figure 14.11.

Table 14.27 : Productora - Grade Variogram Models – Cu/Au Estimate

VREF	Variable	Rotation			Nugget	Structure 1		Structure 2		Structure 3	
		(Datamine ZXZ)			C_0	C_1	R_1	C_1	R_1	C_1	R_1
FAULTBLOCK = 50000											
1	Cu	120	90	0	0.17	0.49	20	0.16	50	0.18	180
							10		45		120
							10		45		45
2	Au	120	90	0	0.26	0.50	20	0.12	50	0.12	180
							10		55		140
							10		45		55
18	Cu (Flat Oxide)	120	170	0	0.09	0.63	20	0.19	60	0.09	260
							20		60		260
							10		40		40

Table 14.27 : Productora - Grade Variogram Models – Cu/Au Estimate											
VREF	**Variable**	**Rotation**			**Nugget**	**Structure 1**		**Structure 2**		**Structure 3**	
		(Datamine ZXZ)			**C_0**	**C_1**	**R_1**	**C_1**	**R_1**	**C_1**	**R_1**
19	Au (Flat Oxide)	120	170	0	0.09	0.63	20	0.19	60	0.09	260
							20		60		260
							10		40		40
FAULTBLOCK = 40000											
23	Cu	120	165	-90	0.14	0.44	15	0.20	25	0.22	100
							15		25		100
							10		15		20
24	Au	120	165	-90	0.14	0.44	15	0.20	25	0.22	100
							15		25		100
							10		15		20
29	Cu (Flat Oxide)	125	135	0	0.17	0.52	25	0.14	100	0.17	300
							25		40		180
							10		15		70
30	Au (Flat Oxide)	125	135	0	0.18	0.53	25	0.14	100	0.15	200
							25		40		150
							15		25		100
FAULTBLOCK = 60000 to 100000											
60	Cu	100	25	0	0.41	0.37	35	0.13	110	0.10	120
							25		50		80
							5		12		17
61	Au	100	25	0	0.41	0.38	35	0.12	110	0.09	120
							25		50		80
							5		12		17

Figure 14.11 : Normal Score Variogram Model for Mid Area Copper High Grade and Low Grade Combined



14.12.2 Alice

The variogram model parameters used for grade estimation of key elements are shown below for copper and silver in Table 14.28 and Table 14.29.

The normal score variogram model for mineralised copper is illustrated in Figure 14.12.

The experimental variograms calculated for some domains were difficult to model therefore a constructed variogram with a nominal nugget of 0.2 and isotropic search of 200 m was used to estimate grade. These variograms are consistent with the geologists' conceptual model of continuity for these domains. These areas are predominately lower grade, and distal to the Alice mineralised porphyry.

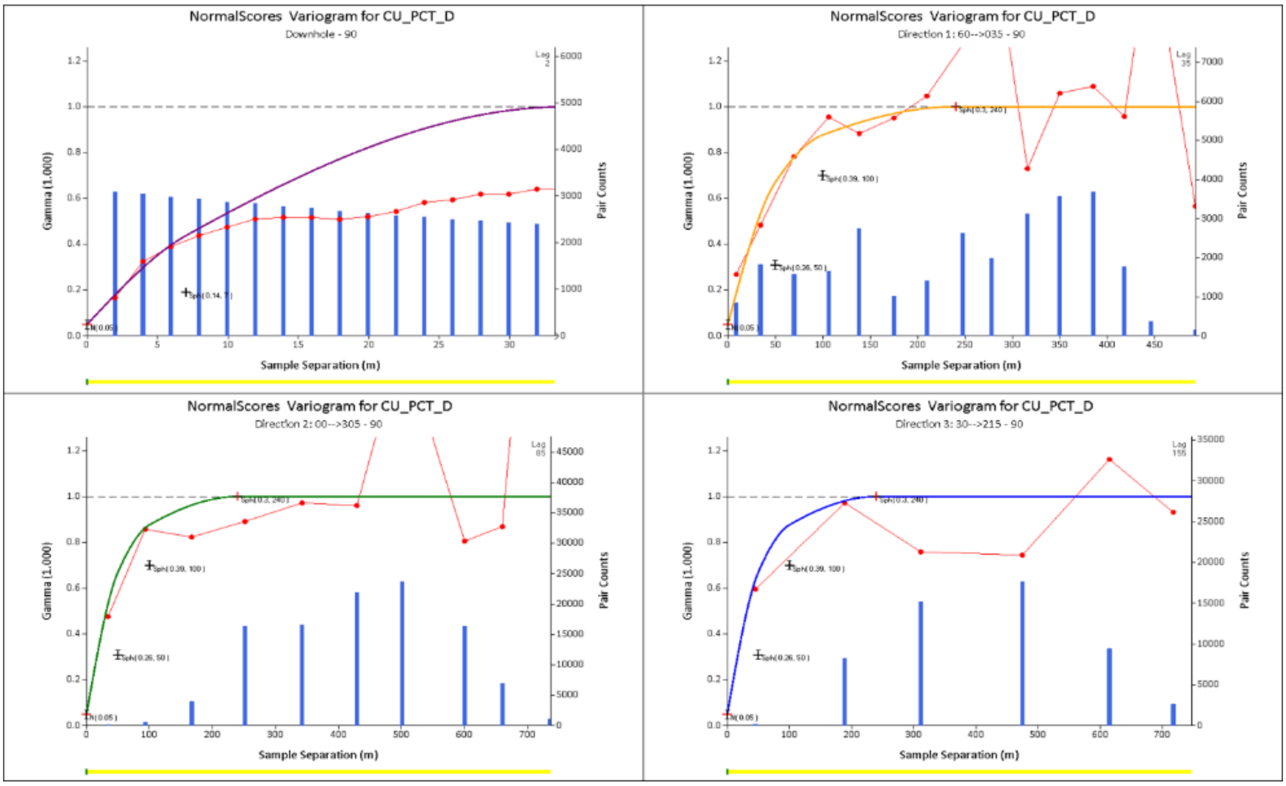
Table 14.28 : Alice - Grade Variogram Models – Cu Estimate

Domain	Rotation (Datamine ZXZ)			Nugget C_0	Structure 1 C_1	Structure 1 R_1	Structure 2 C_1	Structure 2 R_1	Structure 3 C_1	Structure 3 R_1
DOM_CU1 = 10	140	110	100	0.14	0.24	50	0.39	90	0.25	150
						50		90		150
						50		90		150
DOM_CU1 = 20	130	100	0	0.27	0.06	50	0.38	180	0.29	210
						30		90		150
						20		50		100
DOM_CU1 = 50	100	160	0	0.06	0.35	35	0.25	80	0.34	350
						35		80		220
						35		80		200
DOM_CU1 = 90	145	120	160	0.12	0.37	60	0.27	220	0.25	240
						20		100		240
						20		50		170

Table 14.29 : Alice - Grade Variogram Models – Ag Estimate

Domain	Rotation (Datamine ZXZ)			Nugget C_0	Structure 1 C_1	Structure 1 R_1	Structure 2 C_1	Structure 2 C_0	Structure 3 C_1	Structure 3 R_1
DOM_AG1 = 20	-60	60	0	0.32	0.18	30	0.28	130	0.23	240
						20		70		150
						20		70		120
DOM_AG1 = 50	140	90	170	0.08	0.38	100	0.33	170	0.21	300
						100		170		300
						80		150		250
DOM_AG1 = 90	0	0	-90	0.2	0.8	200	-	-	-	-
						200		-		-
						200		-		-

Figure 14.12 : Normal Score Variogram Model for Copper High Grade Estimate



14.12.3 Cortadera

The variogram model parameters used for grade estimation of copper and gold are shown below (Table 14.30 to Table 14.31).

Table 14.30 : Cortadera - Grade Variogram Models – Cu Estimate

AREA	Domain	CIK Sub Domain	Rotation			Nugget	Structure 1		Structure 2		Structure 3	
			(Datamine ZXZ)			C_0	C_1	R_1	C_1	R_1	C_1	R_1
Cuerpo 1	DOM_CU1 = 20	HG_CIK	0	110	70	0.19	0.25	20	0.20	55	0.36	75
								20		40		65
								15		30		40
		LG_CIK	-150	60	-10	0.25	0.19	90	0.36	170	0.20	220
								40		60		80
								40		60		80
	DOM_CU1 = 30	HG_CIK	-110	60	90	0.19	0.26	40	0.30	80	0.25	200
								40		80		200
								40		80		150
		LG_CIK	-110	60	90	0.19	0.26	40	0.30	80	0.25	200

Table 14.30 : Cortadera - Grade Variogram Models – Cu Estimate												
AREA	**Domain**	**CIK Sub Domain**	**Rotation**			**Nugget**	**Structure 1**		**Structure 2**		**Structure 3**	
			(Datamine ZXZ)			**C_0**	**C_1**	**R_1**	**C_1**	**R_1**	**C_1**	**R_1**
								40		80		200
								40		80		150
	DOM_CU1 = 50	HG_CIK	-120	70	80	0.34	0.38	25	0.13	50	0.15	220
								25		50		150
								25		50		150
		LG_CIK	-130	100	50	0.34	0.18	60	0.31	100	0.17	150
								40		80		125
								40		60		100
Cuerpo 2	DOM_CU1 = 20	HG_CIK	-150	130	90	0.30	0.34	10	0.13	30	0.24	100
								10		25		60
								10		15		40
		LG_CIK	-150	130	90	0.30	0.34	10	0.13	30	0.24	100
								10		25		60
								10		15		40
	DOM_CU1 = 30	HG_CIK	-115	90	60	0.41	0.15	40	0.30	70	0.15	150
								40		70		150
								40		70		150
		LG_CIK	-115	90	60	0.41	0.15	40	0.30	70	0.15	150
								40		70		150
								40		70		150
	DOM_CU1 = 50	HG_CIK	-120	70	80	0.34	0.38	25	0.13	50	0.13	220
								25		50		150
								25		50		150
		LG_CIK	-130	100	50	0.34	0.18	60	0.31	100	0.17	150
								40		80		125
								40		60		100
Cuerpo 3	DOM_CU1 = 20	HG_CIK	-150	70	70	0.39	0.10	30	0.12	50	0.39	110
								30		50		110
								30		50		110
		LG_CIK	-110	60	90	0.23	0.21	20	0.27	50	0.30	150
								20		50		120
								20		50		80
		HG_CIK	-120	70	80	0.34	0.38	25	0.13	50	0.15	220

Table 14.30 : Cortadera - Grade Variogram Models – Cu Estimate												
AREA	**Domain**	**CIK Sub Domain**	**Rotation**			**Nugget**	**Structure 1**		**Structure 2**		**Structure 3**	
			(Datamine ZXZ)			**C_0**	**C_1**	**R_1**	**C_1**	**R_1**	**C_1**	**R_1**
								25		50		150
								25		50		150
	DOM_CU1 = 30							60		100		150
		LG_CIK	-130	100	50	0.34	0.18	40	0.31	80	0.17	125
								40		60		100
								25		50		220
		HG_CIK	-120	70	80	0.34	0.38	25	0.13	50	0.15	150
	DOM_CU1 = 50							25		50		150
								60		100		150
		LG_CIK	-130	100	50	0.34	0.18	40	0.31	80	0.17	125
								40		60		100

Table 14.31 : Cortadera - Grade Variogram Models – Au Estimate												
AREA	**Domain**	**CIK Sub Domain**	**Rotation**			**Nugget**	**Structure 1**		**Structure 2**		**Structure 3**	
			(Datamine ZXZ)			**C_0**	**C_1**	**R_1**	**C_1**	**R_1**	**C_1**	**R_1**
								15		50		150
	DOM_CU1 = 20, 30	-	-150	80	90	0.33	0.29	15	0.20	50	0.18	150
Cuerpo 1								15		30		75
								50		150		350
	DOM_CU1 = 50	-	-150	80	0	0.18	0.33	20	0.25	70	0.24	200
								20		70		200
								30		50		225
		HG_CIK	-140	60	160	0.17	0.30	10	0.23	25	0.30	150
	DOM_CU1 = 20							10		25		100
								30		50		225
		LG_CIK	-140	60	160	0.17	0.30	10	0.23	25	0.30	150
Cuerpo 2								10		25		100
								50		100		200
		HG_CIK	-150	70	0	0.19	0.27	20	0.23	50	0.32	200
								20		50		150
	DOM_CU1 = 30							50		100		200
		LG_CIK	-150	70	0	0.19	0.27	20	0.23	50	0.32	200
								20		50		150

Table 14.31 : Cortadera - Grade Variogram Models – Au Estimate

AREA	Domain	CIK Sub Domain	Rotation			Nugget	Structure 1		Structure 2		Structure 3	
			(Datamine ZXZ)			C_0	C_1	R_1	C_1	R_1	C_1	R_1
	DOM_CU1 = 50	HG_CIK	-150	70	0	0.19	0.27	50	0.23	100	0.32	200
								20		50		200
								20		50		150
		LG_CIK	-150	70	0	0.19	0.27	50	0.23	100	0.32	200
								20		50		200
								20		50		150
Cuerpo 3	DOM_CU1 = 20	HG_CIK	-150	70	70	0.39	0.10	30	0.12	50	0.39	110
								30		50		110
								30		50		110
		LG_CIK	-110	60	90	0.23	0.21	20	0.27	50	0.17	150
								20		50		125
								20		50		100
	DOM_CU1 = 30	HG_CIK	-120	70	80	0.34	0.38	25	0.13	50	0.13	220
								25		50		150
								25		50		150
		LG_CIK	-130	100	50	0.34	0.18	60	0.31	100	0.17	150
								40		80		125
								40		60		100
	DOM_CU1 = 50	HG_CIK	-120	70	80	0.34	0.38	25	0.13	50	0.13	220
								25		50		150
								25		50		150
		LG_CIK	-130	100	50	0.34	0.18	60	0.31	100	0.17	150
								40		80		125
								40		60		100

14.12.4 San Antonio

Due to low sample counts at San Antonio, the construction of a coherent variogram was only possible for the primary San Antonio lode. All other domains use the same variogram and kriging neighbourhood for estimation as the primary lode. Given the style of mineralisation, grade population, and orientation are reasonably consistent between domains, this was considered reasonable.

Due to the strong correlation between copper and silver, copper variograms and search neighbourhoods have been used for the silver estimate.

The experimental variograms calculated for the molybdenum and gold were difficult to model therefore a constructed variogram has been used with a nominal nugget of 0.2 and a spherical search of 150 m for estimation. This variogram is consistent with the geologists' conceptual model of continuity for molybdenum and gold.

Hard boundaries have been used between all mineralisation domains. Soft boundaries have been used between weathering domains, no statistical difference in grade population exists across different weathering conditions.

Variogram parameters used for the San Antonio estimate are included in Table 14.32 below.

Table 14.32 : Grade Variogram Models – San Antonio

Element	Domain	Rotation (Datamine ZXZ)			Nugget C_0	Structure 1		Structure 2		Structure 3	
						C_1	R_1	C_1	R_1	C_1	R_1
Cu	All	120	55	20	0.15	0.21	30	0.26	50	0.38	240
							20		45		120
							15		40		80
Ag	All	120	55	20	0.15	0.21	30	0.26	50	0.38	240
							20		45		120
							15		40		80
Au	All	0	0	-90	0.20	0.80	150	-	-	-	-
							150		-		-
							150		-		-
Mo	All	0	0	-90	0.20	0.80	150	-	-	-	-
							150		-		-
							150		-		-

14.13 Estimation

For all project areas, a suite of elements (copper, gold, molybdenum, cobalt, calcium, iron, sulphur, potassium, aluminium, and rhenium) was estimated using via ordinary kriging in Datamine Studio RM.

Kriging accounts for the spatial distribution and grade continuity of the input data. Kriging is also able to account for the clustering of samples caused by variation in drilling density throughout the deposit.

Composite data was top-cut prior to estimation as discussed in 14.10.

No octant searches were used.

14.13.1 Productora

For the Productora estimate, composites were selected from within a search ellipse of radius 100 m in the principal direction along strike, 100 m in the down dip direction and 50 m across the plane of mineralisation. The search strategy for grade estimation largely used the established dynamic anisotropy to locally tune the search orientations, with the exception of cobalt and copper oxide where a static search orientation was derived from the continuity analysis.

Hard boundaries were utilised between domains cut by the Serrano fault and the Rancho fault, with soft boundaries between other fault blocks in the north area.

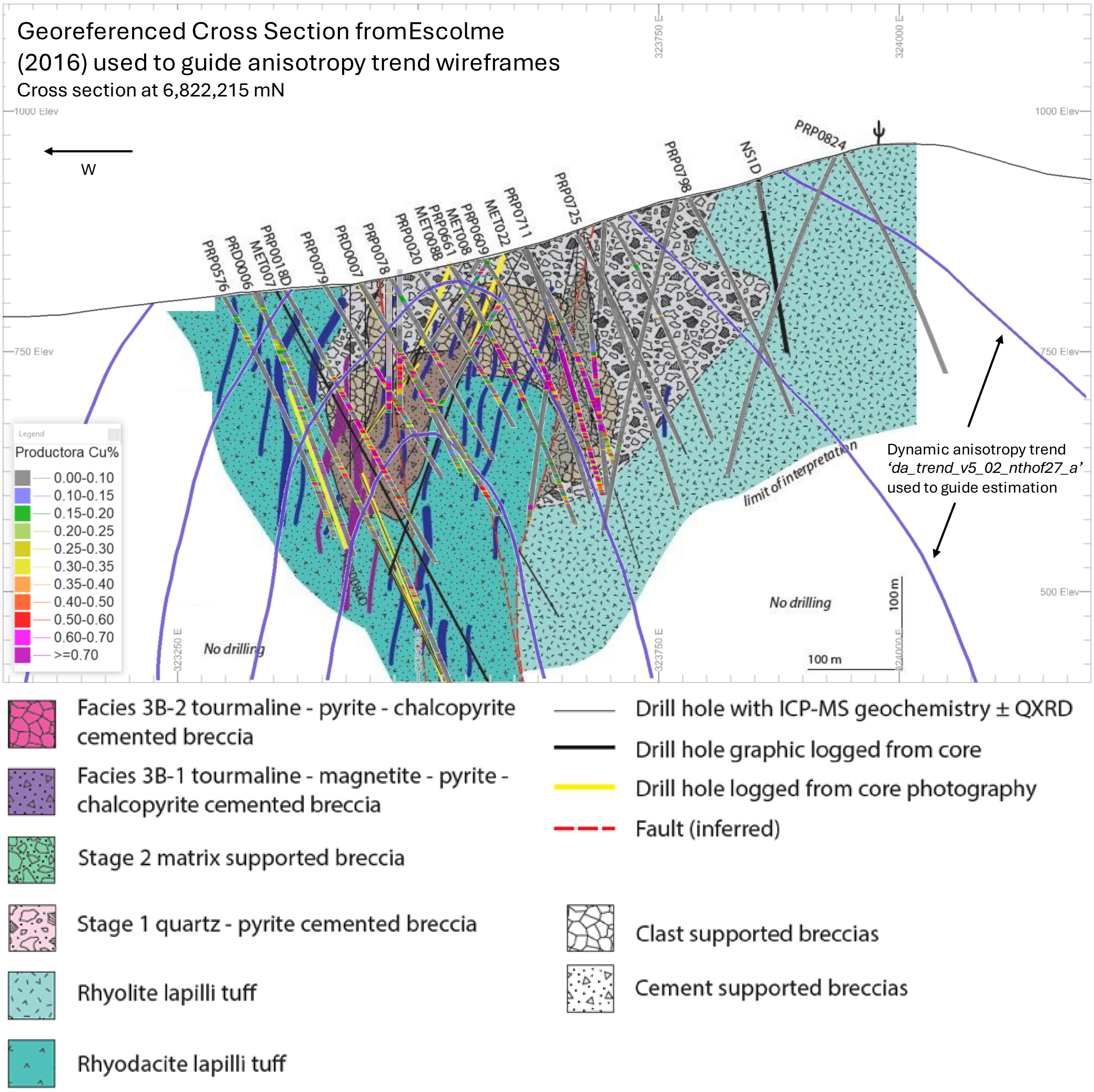
Interpretation of dynamic anisotropy surfaces was based on graphic core logging, core photo library, drill hole data base, detailed hand specimen and thin section observations, and WLSQ-QXRD data. These observations, in conjunction with iterative indicator estimation testwork, resulted in the final manually-created fault block-specific dynamic anisotropy trend wireframes. An example of dynamic anisotropy is included in Figure 14.13.

The search strategies for copper and gold are shown in Table 14.33 below.

Table 14.33 : Productora - Search Strategy for Grade Estimation – Cu/Au

Type	SREF	Orientation			Search			2nd Search Factor	Number of Composites				
		Rot1	Rot2	Rot3	D1	D2	D3		First Search		Second Search		Max per Drill Hole
									Min	Max	Min	Max	
FAULTBLOCK = 50000													
Cu/Au	1	Dynamic Anisotropy			100	100	50	3	6	12	6	12	7
Cu/Au Flat Oxide	18	120	170	0	100	100	50	3	6	12	6	12	7
FAULTBLOCK = 40000													
Cu/Au	23	Dynamic Anisotropy			100	100	50	2	6	12	6	12	5
Cu/Au Flat Oxide	29	125	135	0	100	100	50	3	6	12	6	12	5
FAULTBLOCK = 40000													
Cu/Au	23	Dynamic Anisotropy			100	100	50	2	6	12	6	12	5

Figure 14.13 : Cross Section at 6822215 mN Displaying a Georeferenced Interpreted Distribution of Breccia Facies from Escolme (2016)



14.13.2 Alice

Mineralisation was estimated using semi-soft boundaries according to the domain conditions for each element. Semi-soft boundaries allow for restricted sharing of samples across domain boundaries. For the Alice estimate, the restriction was controlled by the Datamine Studio RM 'MAXKEY' field. Multiple estimate iterations tested the impact of different boundary conditions, with the final estimate best representing the geological understanding of the mineralisation, as well as boundary analysis completed.

The boundaries between oxidation states were soft.

Table 14.34 : Alice - Search Strategy for Grade Estimation – Cu												
Domain	Orientation			Search			2nd Search Factor	Number of Composites				
	Rot1	Rot2	Rot3	D1	D2	D3		First Search		Second Search		Max per Drill Hole
								Min	Max	Min	Max	
DOM_CU1 = 10	140	110	90	100	100	100	1.5	10	20	10	20	6
DOM_CU1 = 20	130	100	0	140	100	70	1.5	10	20	10	20	6
DOM_CU1 = 50	100	160	0	230	150	150	1.5	12	24	12	24	6
DOM_CU1 = 90	145	120	160	150	150	110	1.5	12	24	12	24	6

14.13.3 Cortadera

The search strategies for copper and gold are shown in Table 14.35 and Table 14.36 below.

Mineralisation was estimated using one-way semi-soft boundaries according to the domain conditions for each element. This approach is based on the observation that the mineralised system comprises a high-grade "core" with gradational copper grade decreasing outwards to the edge of the porphyry intrusion and into wall rock.

Rigorous testwork has shown that the CIK approach with one-way soft boundaries is the optimal way to estimate the observed grade trends. This is also supported geostatistically by boundary analysis completed between the CIK sub-domains (Figure 14.14).

For copper and gold estimates, high-grades across the semi-soft boundary are controlled using the 'cap distance' in Datamine, discussed in Section 14.11.1.

The boundaries between oxidation states were soft.

Table 14.35 : Cortadera - Search Strategy for Grade Estimation – Cu														
Area	Domain	CIK Subdomain	Orientation			Search			2nd Search Factor	Number of Composites				
			Rot1	Rot 2	Rot 3	D1	D2	D3		First Search		Second Search		Max per Drill Hole
										Min	Max	Min	Max	
Cuerpo 1	DOM_CU1 = 20	HG_CIK	0	110	70	50	45	25	1.5	10	20	10	20	6
		LG_CIK	-110	60	90	100	80	55	1.5	10	20	10	20	6
	DOM_CU1 = 30	HG_CIK	-110	60	90	130	130	100	1.5	10	20	10	20	6
		LG_CIK	-110	60	90	130	130	100	1.5	10	20	10	20	6
	DOM_CU1 = 50	HG_CIK	-120	70	80	150	100	100	1.5	10	20	10	20	6
		LG_CIK	-130	100	50	100	85	70	1.5	10	20	10	20	6

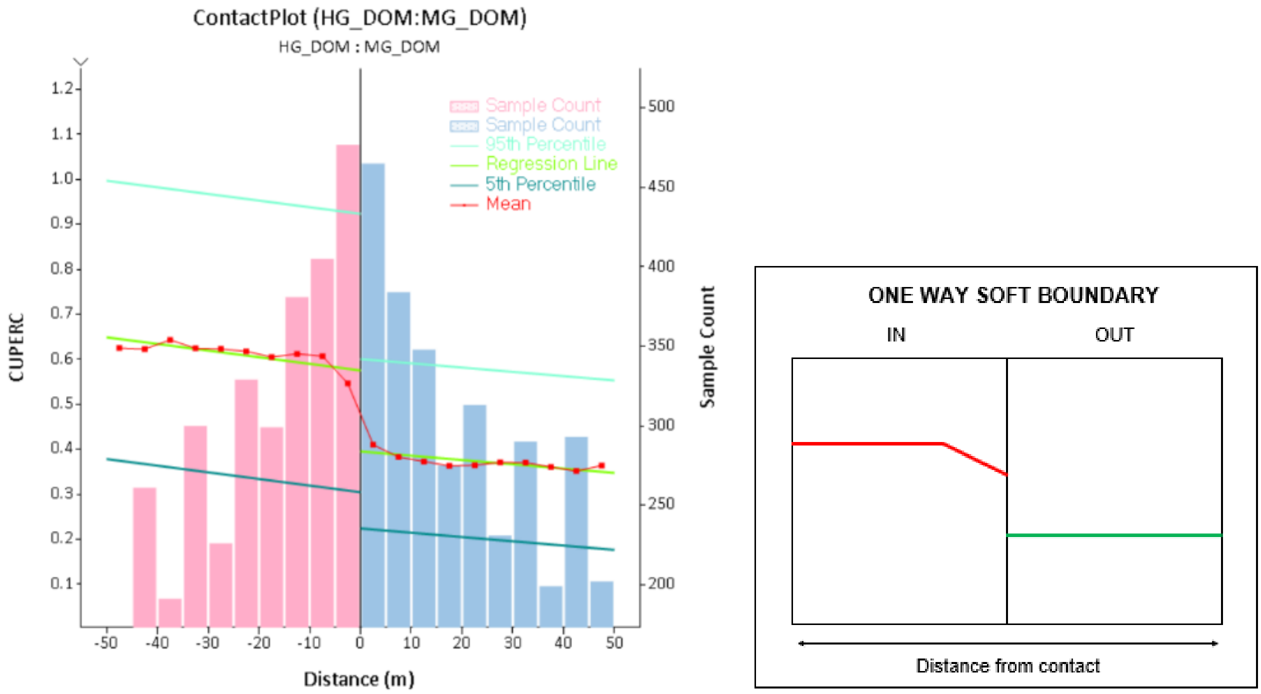
Table 14.35 : Cortadera - Search Strategy for Grade Estimation – Cu

Area	Domain	CIK Subdomain	Orientation			Search			2nd Search Factor	Number of Composites				
			Rot1	Rot 2	Rot 3	D1	D2	D3		First Search		Second Search		Max per Drill Hole
										Min	Max	Min	Max	
Cuerpo 2	DOM_CU1 = 20	HG_CIK	-150	130	90	60	40	40	1.5	10	20	10	20	6
		LG_CIK	-110	60	90	100	80	55	1.5	10	20	10	20	6
	DOM_CU1 = 30	HG_CIK	-115	90	60	100	100	100	1.5	10	20	10	20	6
		LG_CIK	-115	90	60	100	100	100	1.5	10	20	10	20	6
	DOM_CU1 = 50	HG_CIK	-120	70	80	150	100	100	1.5	10	20	10	20	6
		LG_CIK	-130	100	50	100	85	70	1.5	10	20	10	20	6
Cuerpo 3	DOM_CU1 = 20	HG_CIK	-150	70	70	75	75	75	1.5	10	20	10	20	6
		LG_CIK	-130	100	50	100	80	55	1.5	10	20	10	20	6
	DOM_CU1 = 30	HG_CIK	-120	70	80	150	100	100	1.5	10	20	10	20	6
		LG_CIK	-130	100	50	100	85	70	1.5	10	20	10	20	6
	DOM_CU1 = 50	HG_CIK	-120	70	80	150	100	100	1.5	10	20	10	20	6
		LG_CIK	-130	100	50	100	85	70	1.5	10	20	10	20	6

Table 14.36 : Search Strategy for Grade Estimation – Au

Area	Domain	CIK Subdomain	Orientation			Search			2nd Search Factor	Number of Composites				
			Rot1	Rot 2	Rot 3	D1	D2	D3		First Search		Second Search		Max per Drill Hole
										Min	Max	Min	Max	
Cuerpo 1	DOM_CU1 = 20, 30	-	-150	80	90	100	100	50	1.5	10	20	10	20	6
	DOM_CU1 = 50	-	-150	80	0	230	130	130	1.5	10	20	10	20	6
Cuerpo 2	DOM_CU1 = 20	HG_CIK	-140	60	160	150	100	60	1.5	10	20	10	20	6
		LG_CIK	-140	60	160	150	100	60	1.5	10	20	10	20	6
	DOM_CU1 = 30	HG_CIK	-150	70	0	130	130	100	1.5	10	20	10	20	6
		LG_CIK	-150	70	0	130	130	100	1.5	10	20	10	20	6
	DOM_CU1 = 50	HG_CIK	-150	70	0	130	130	100	1.5	10	20	10	20	6
		LG_CIK	-150	70	0	130	130	100	1.5	10	20	10	20	6
Cuerpo 3	DOM_CU1 = 20	HG_CIK	-150	70	70	75	75	75	1.5	10	20	10	20	6
		LG_CIK	-110	60	90	100	80	55	1.5	10	20	10	20	6
	DOM_CU1 = 30	HG_CIK	-120	70	80	150	100	100	1.5	10	20	10	20	6
		LG_CIK	-130	100	50	100	85	70	1.5	10	20	10	20	6
	DOM_CU1 = 50	HG_CIK	-120	70	80	150	100	100	1.5	10	20	10	20	6
		LG_CIK	-130	100	50	100	85	70	1.5	10	20	10	20	6

Figure 14.14 : Boundary Analysis (Completed Using Snowden Supervisor) Comparing HG and MG Cu% CIK Sub-Domains for Cuerpo 3 Justifying the Use of a One-Way Soft Boundary. Schematic Shows the Typical Trend Displayed by One-Way Soft Boundaries



One-way semi-soft boundaries are controlled using the Datamine MAXKEY approach. Correct application of the soft boundary is checked using the SAMPOUT file created during the estimation process, with outputs the samples used to estimate each block, as well as the kriging weight applied.

Table 14.37 details the soft boundaries used for each estimate, including minimum and maximum sample counts for the estimate.

Table 14.37 : Cortadera - Soft Boundary Usage for Cu% Estimates						
Area	**Domain**	**CIK Subdomain**	**MAXKEY FIELD**	**MAXKEY Value**	**Minimum Sample Count**	**Maximum Sample Count**
Cuerpo 1	DOM_CU1 =20	HG_CIK	BHID	6	10	20
		LG_CIK	LGDOMSFT	6	10	20
	DOM_CU1 =30	HG_CIK	HGDOMSFT	6	10	20
		LG_CIK	LGDOMSFT	6	10	20
	DOM_CU1 =50	HG_CIK	HGDOMSFT	6	10	20
		LG_CIK	LGDOMSFT	6	10	20
Cuerpo 2	DOM_CU1 =20	HG_CIK	BHID	6	10	20
		LG_CIK	LGDOMSFT	6	10	20
	DOM_CU1 =30	HG_CIK	HGDOMSFT	6	10	20
		LG_CIK	LGDOMSFT	6	10	20
	DOM_CU1 =50	HG_CIK	HGDOMSFT	6	10	20
		LG_CIK	LGDOMSFT	6	10	20
Cuerpo 3	DOM_CU1 =20	HG_CIK	BHID	6	10	20
		LG_CIK	LGDOMSFT	6	10	20
	DOM_CU1 =30	HG_CIK	HGDOMSFT	6	10	20
		LG_CIK	LGDOMSFT	6	10	20
	DOM_CU1 =50	HG_CIK	HGDOMSFT	6	10	20
		LG_CIK	LGDOMSFT	6	10	20

Test iterations (variably comparing estimation methodology, block size, boundary conditions, and probability thresholds (for CIK)), were compared against one another, with the estimate best reflecting the input data and the geological understanding of the mineralised system selected.

14.13.3.1 Estimation Inside of Limonites

Limonite-rich iron oxide horizons are found above each of the Cuerpos at Cortadera, into which copper, gold, silver, and molybdenum were estimated.

As the limonite-rich iron oxides are separated by significant distance, each one was estimated independently. A hard boundary was used between the iron oxide horizon and the fresh-rock underneath as they comprise separate grade populations.

The interpretation comprises a geological domain (including volume where lower grades exist, to ensure geometric continuity. As a result of this, mixed grade populations exist for some elements in the iron oxide horizon, which necessitated the use of a CIK estimate.

Variograms and search neighbourhoods were determined for each horizon and element independently, with the estimates completed using top-cut, composited samples.

14.13.4 San Antonio

A conventional estimation strategy has been used for San Antonio, with the mineralised zone interpretation producing copper grade populations suitable for ordinary kriging.

Due to the undulating nature of the structurally controlled mineralised domains, it was necessary to translate some domains and composites into two-dimensional space to ensure artefacts are not introduced during estimation.

14.14 Validation

All estimates were validated using a three-stage comparison between top-cut composites and the estimated variables. The first stage involves calculating the global statistics of the composites compared to the tonnage weighted averages of estimated variables. The second stage involves comparing statistics in slices along the mineralisation, and the third involves a detailed visual comparison by section to ensure the estimated variables honour the input composite data.

14.14.1 Productora

Global comparisons were completed using the hard boundary domain coding. The global comparison for the copper domains is presented in Table 14.38. In general, the comparisons are excellent for the primary variables. Comparisons are poor for the extremely low-grade variables and low-grade domains where small differences in grade will result in large percentage differences. Comparisons are also poorer for those variables where the grade ranges vary over the mineralisation and the data distributions are mixed.

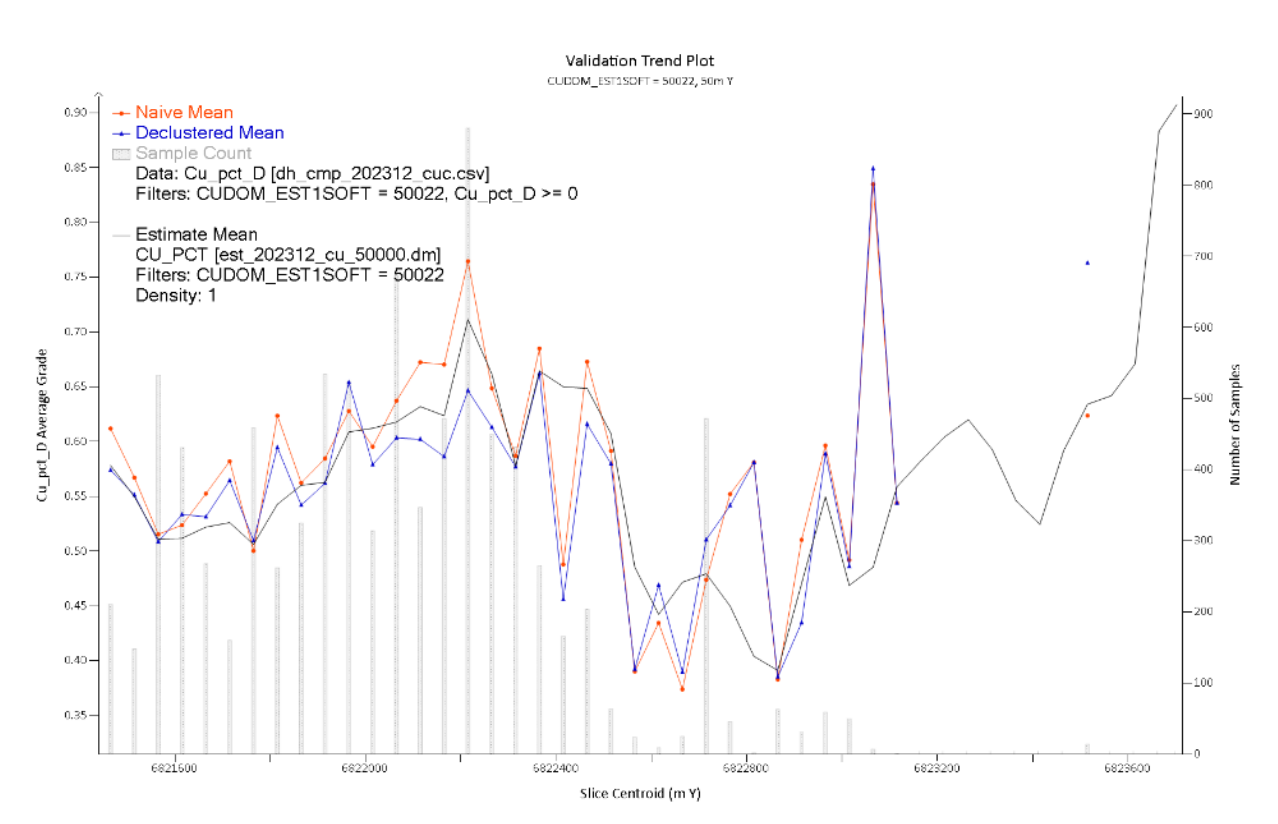
Table 14.38 : Productora - Global Comparison of Copper Composites and Estimates

CUDOM	Composites		Model	% Difference		% Est	Model	% Difference	
	Top-cut	Top-cut Declustered	1st pass	Top-cut	Top-cut Declustered	1st pass	All	Top-cut	Top-cut Declustered
50022	0.60	0.56	0.57	-5.3%	+1.7%	23%	0.57	-5.9%	+1.0%
50012	0.52	0.52	0.54	+3.2%	+3.0%	80%	0.53	+1.7%	+1.5%
50032	0.56	0.51	0.61	+5.6%	+9.8%	53%	0.58	+4.9%	+7.4%
50052	0.42	0.44	0.43	+0.1%	-2.8%	19%	0.44	+3.1%	+0.9%
50062	0.67	0.68	0.68	+2.5%	+0.8%	47%	0.68	+1.1%	+0.3%
50021	0.16	0.16	0.15	-2.0%	-1.6%	67%	0.15	-5.6%	-5.3%
50011	0.14	0.14	0.13	-3.0%	-4.0%	29%	0.14	-0.4%	-1.4%
50031	0.11	0.11	0.11	-4.2%	-6.5%	81%	0.10	-7.9%	-10.1%
50051	0.11	0.11	0.12	+3.2%	+1.4%	56%	0.12	+3.4%	+1.5%
50061	0.12	0.12	0.12	-1.9%	+1.0%	7%	0.12	-3.1%	-0.1%
50029	0.03	0.04	0.03	+5.8%	-7.3%	48%	0.03	+9.1%	-14.8%
50019	0.03	0.03	0.02	-4.0%	-8.1%	49%	0.03	+12.4%	-7.3%
50039	0.04	0.03	0.01	-75.6%	-69.1%	69%	0.01	-79.1%	-73.5%
50059	0.02	0.02	0.02	-14.4%	-23.7%	34%	0.02	-21.8%	-30.3%
50069	0.02	0.02	0.02	-37.6%	-36.7%	62%	0.01	-44.3%	-43.5%

The average block values were compared to the average input composites over northing slices. These slices represent equal intervals through the model and input data, with 80 m slices used. An example for the copper estimation domain 50022 is presented in Figure 14.15.

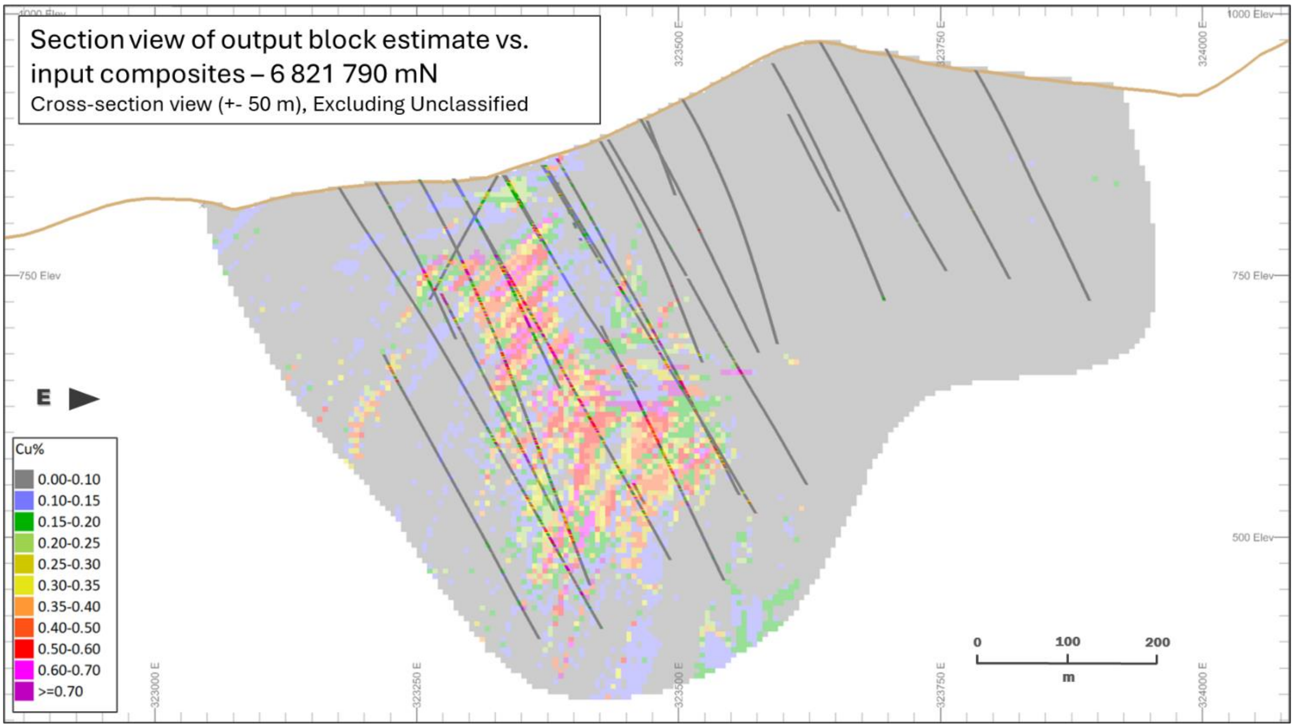
The grade trends input data in the model slices are comparable to grade trends of the block estimates. The estimated grades are, as expected, smoother than the composites though the variation in grades can be clearly seen.

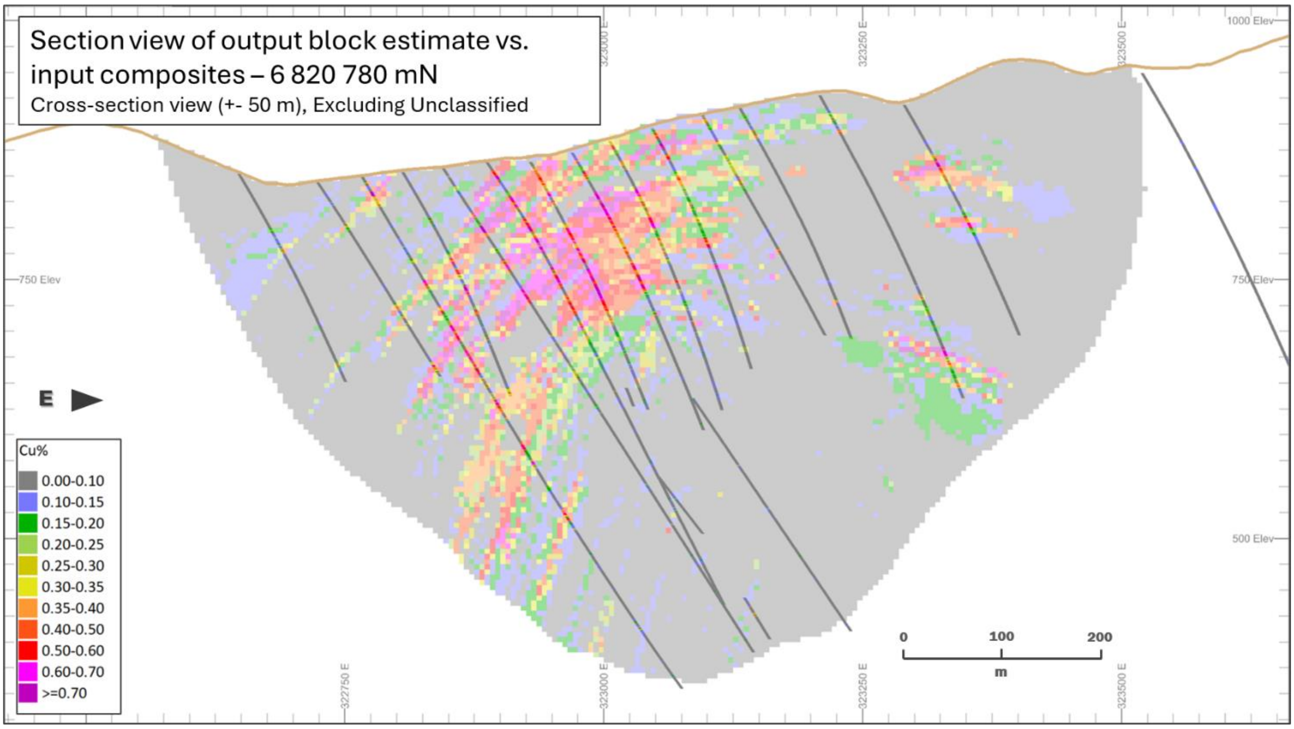
Figure 14.15: Productora - Northing Trend Validation for Copper in Domain 50022



The block model grades were visually compared with the input data. Overall, the comparisons are good, with the block grades following the trends in the input data as illustrated for copper in Figure 14.16.

Figure 14.16 : Productora - Visual Validation of Copper for two East-West Sections





14.14.2 Alice

The Alice estimation was considered acceptable in honouring the input data across all three validation techniques utilised.

Examples of a northing trend plot and cross section of the copper estimate are shown in Figure 14.17 and Figure 14.18.

Figure 14.17 : Alice - Northing Trend Validation for Alice Copper Estimate

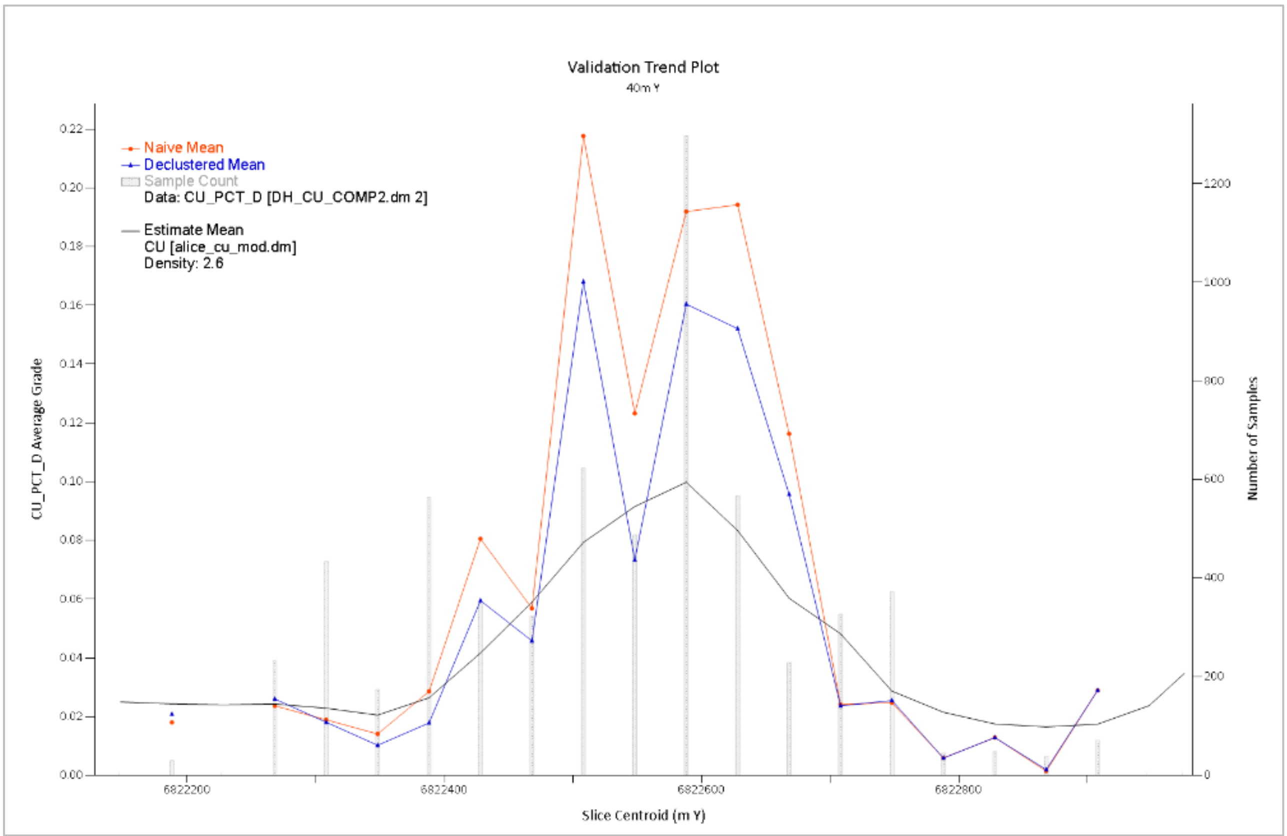
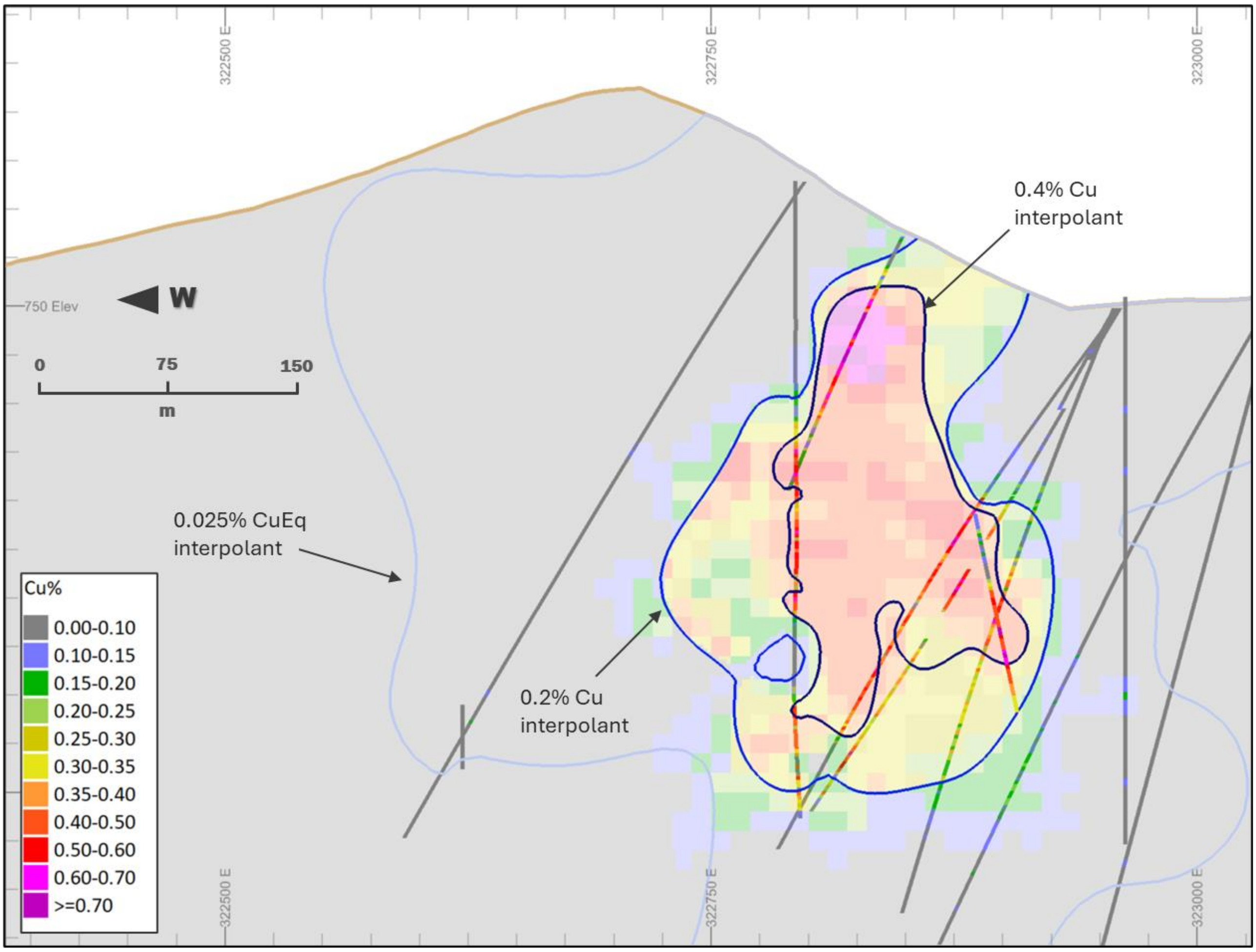


Figure 14.18 : Alice - Visual Validation of Alice Copper Estimation for East-West Section 6822215 mN



14.14.3 Cortadera

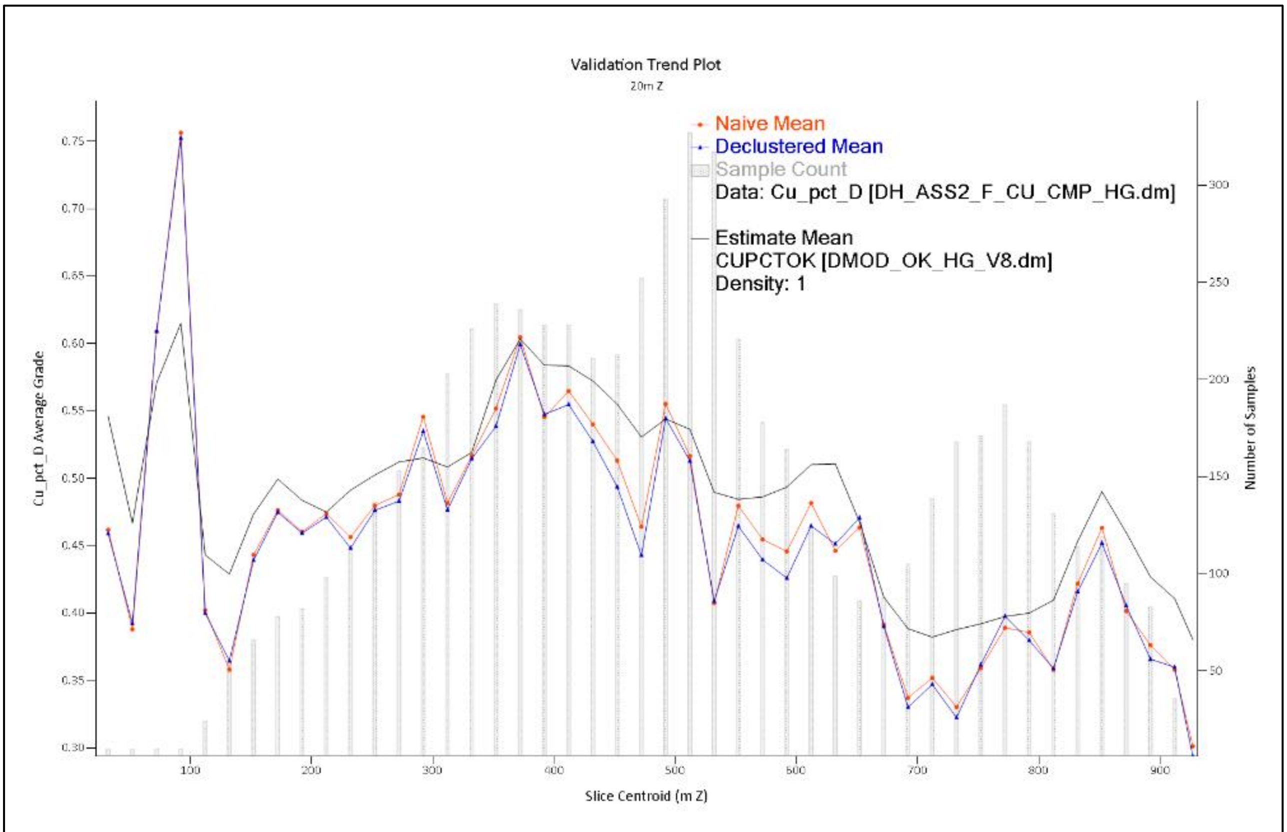
At Cortadera, CIK sub-domains were validated independently and as a combined estimate and dataset (sub-domains combined).

The comparison of average grade of the composites to the block grade estimate (excluding blocks estimated on third pass) for a selection of elements is in Table 14.39.

Table 14.39 : Cortadera Estimation Validation – Comparison of Composites to Output Block Model					
Element	**AREA**	**Domain**	**Composites – Top-cut and Declustered**	**Model (Excluding Third Pass Blocks)**	**% Difference**
Cu	Cuerpo 1	DOM_CU1 = 20	0.34	0.36	+7.3%
		DOM_CU1 = 30	0.16	0.17	+3.8%
		DOM_CU1 = 50	0.07	0.06	-8.1%
	Cuerpo 2	DOM_CU1 = 20	0.36	0.39	+6.3%
		DOM_CU1 = 30	0.16	0.18	+11.1%
		DOM_CU1 = 50	0.08	0.09	+8.1%
	Cuerpo 3	DOM_CU1 = 20	0.47	0.50	+6.4%
		DOM_CU1 = 30	0.21	0.22	+4.1%
		DOM_CU1 = 50	0.09	0.09	-5.4%

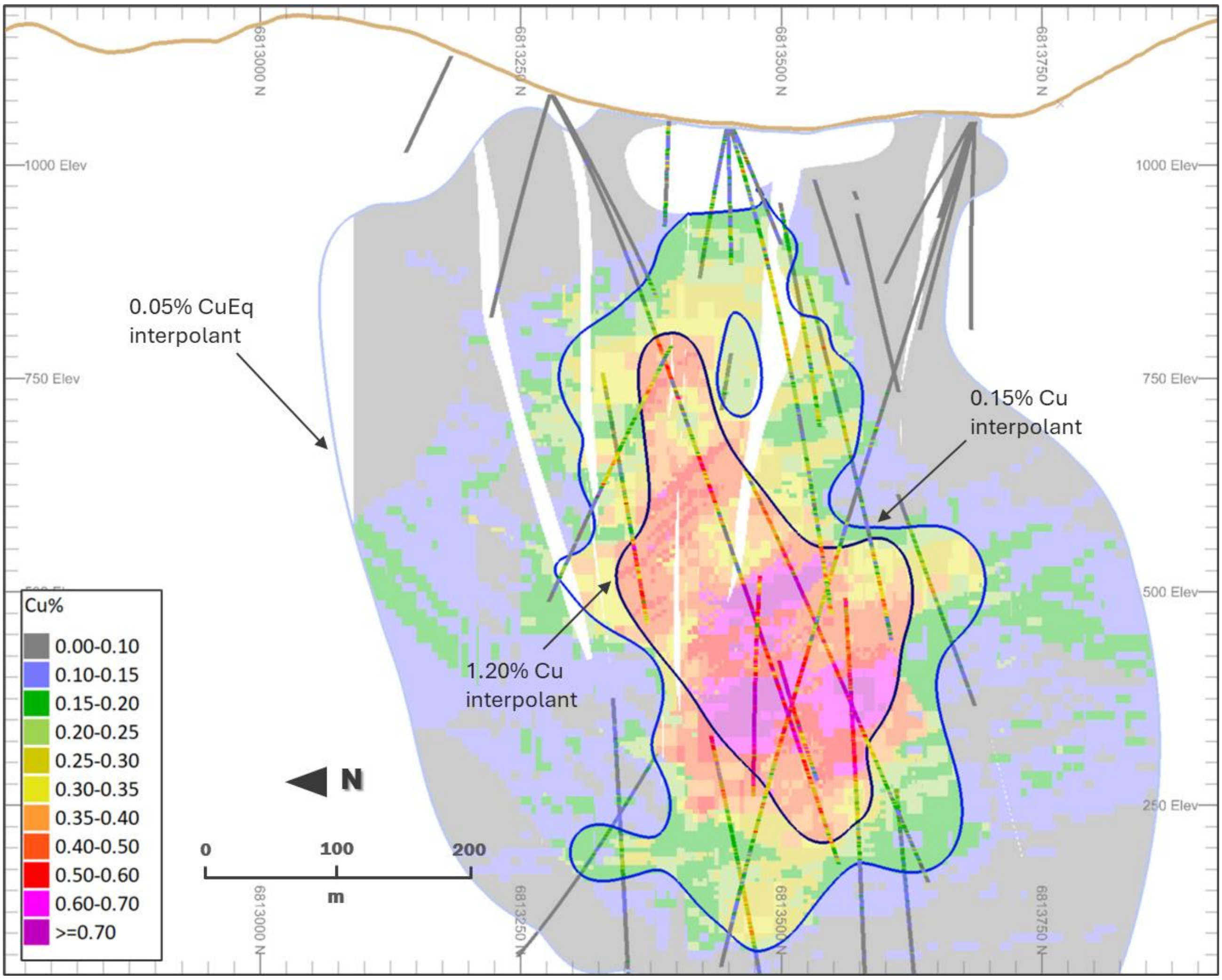
An example of trend analysis is shown in Figure 14.19 for the High-Grade Cuerpo 3 copper estimate.

Figure 14.19 : Cortadera - RL Trend Validation for Cuerpo 3 High-Grade (DOM_CU1=20) Copper Estimate



Visual validation was completed between drill hole grades and model grades (Figure 14.20). The validations show a reasonable reflection of the input composite grades in the output model, although comparisons are poor for the extremely low-grade variables and low-grade domains where small differences in grade result in large percentage differences. Comparisons are also poorer for those variables where the grade ranges vary over the mineralisation and the data distributions are mixed.

Figure 14.20 : Cortadera - Visual Validation of Drill Holes Versus Modelled Cu Grade through Cuerpo 3



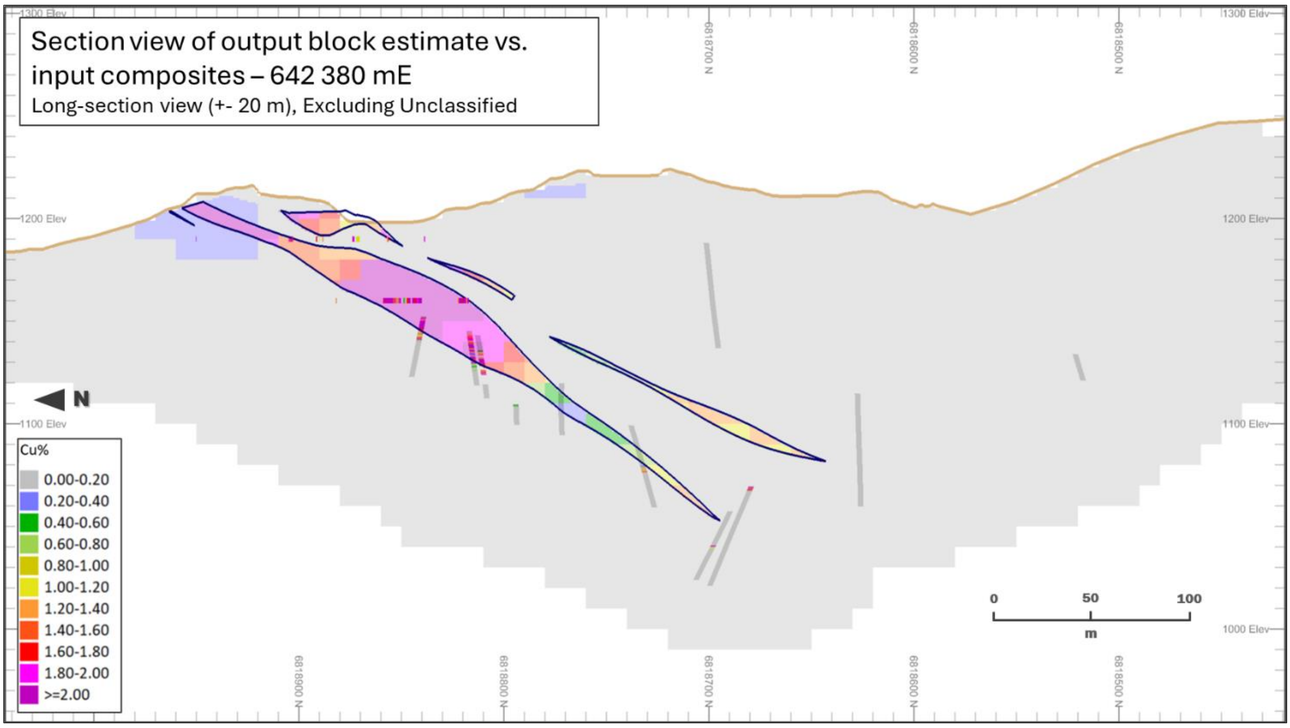
14.14.4 San Antonio

The San Antonio estimation was considered appropriate in honouring the input data across all three validation techniques utilised.

A comparison of average grade of the composites to the block grade estimate (including blocks estimated on first pass only) for the copper estimate is in Table 14.40. Visual validation was completed between drill hole grades and model grades as shown in Figure 14.21.

Table 14.40 : San Antonio - Estimation Global Validation – Cu			
Domain	**Composites – Top-Cut, Declustered**	**Model – 1st Estimation Pass**	**% Difference of 1st Pass**
DOM_CU1=0	1.49	1.48	-0.7%
DOM_CU1=1	0.88	0.91	+3.4%
DOM_CU1=2	0.58	0.52	-11.5%
DOM_CU1=3	1.10	1.18	+7.2%
DOM_CU1=4	0.94	0.93	-0.9%
DOM_CU1=5	0.74	0.79	+6.8%
DOM_CU1=6	0.78	0.79	+1.0%
DOM_CU1=7	0.56	0.62	+10.7%
DOM_CU1=8	0.96	0.91	-6.1%
DOM_CU1=9	0.88	0.96	+7.4%
DOM_CU1=10	0.25	0.25	-0.1%

Figure 14.21 : San Antonio - Visual Validation of Input Data Versus Model Copper Grade - Long Section View



14.15 Bulk Density

Sampling for dry, in-situ bulk density was completed using the water displacement methodology. Sampling included taking a 10 cm piece of whole core every 30 metres and submitting for analysis. The method is based on the Archimedes Principle as the submerged sample experiences an upward force equal to the weight of fluid it displaces. The volume is thus the difference of the weight of the sample as measured in water and as measured in air (Lomberg, 2021).

14.15.1 Productora

A total of 2,164 measurements were available for estimation of bulk density at Productora.

A top-cut of 3.00 t/m^3 was applied to remove any outlier values. Statistical analysis by weathering domain indicated very low variability in average bulk density across the different weathering types.

Bulk Density at Productora was estimated using Ordinary Kriging, using the same Dynamic Anisotropy trends as the mineralisation estimates, with hard boundaries used between weathering domains.

A large data set of pycnometer data is also available for Productora, however due to know the known presence of porous areas within the Productora deposit, this data set was not considered suitable for use.

14.15.2 Alice

In total, 137 bulk density values have informed the Alice Resource.

Of these 137 values, 129 were taken in fresh porphyry unit 'LTCODE = 20' (which contains most of the mineralisation), and only five were taken within the weathered zones.

Given the lack of measurements in the weathered zones, a 'step-down' approach has been used to determine coded density values, with 10% removed from the fresh rock density for the transitional zones, and 20% removed for the oxide zones.

It is noted that the estimate of bulk density will be more reliable in the fresh porphyry unit than the remainder of the Mineral Resource area. Further bulk density measurements will be required for advancement of the Project.

Table 14.41 : Bulk Density Value Assignment

WTCODE	LTCODE	Density Assigned (t/m^3)
1000 (oxide)	1	2.10
	3	2.10
	20	2.22
2000 (trans)	1	2.31
	3	2.31
	20	2.46
3000 (fresh)	1	2.63
	3	2.63
	20	2.72

14.15.3 Cortadera

1,304 bulk density samples were analysed and average density values for each lithology were determined for the fresh, transitional, and oxide material.

As most density measurements have been taken in fresh rock (1,264 in fresh, 20 in transitional, and 20 in oxide), for oxide and transitional zones similar lithologies have had been grouped, dictated by rock characteristics. (For instance, early-, intra- and post-mineral felsic intrusions have been grouped).

These densities were then applied to the block model based on the coded lithology and weathering in the final block model.

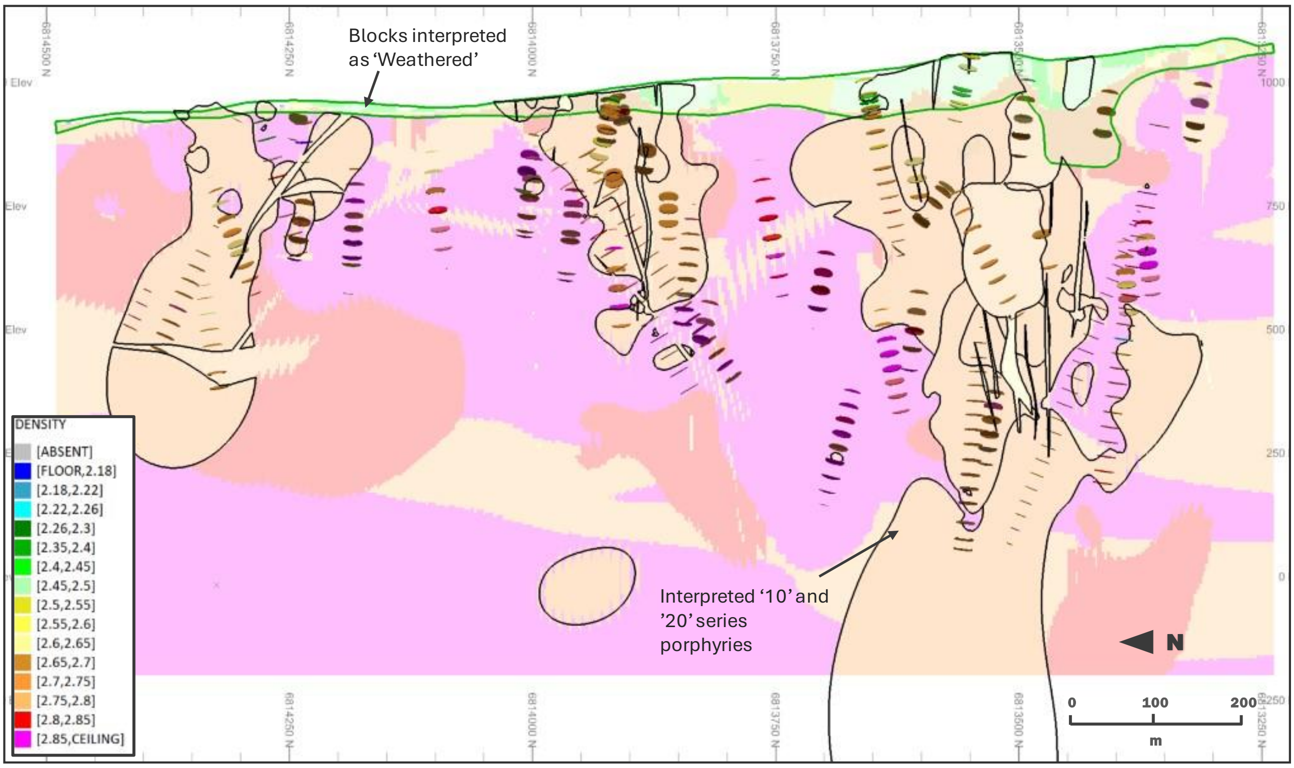
Densities as coded are tabulated in Table 14.42.

Table 14.42 : Bulk Density Value Assignment

LTCODE	WTCODE	Density Assigned (t/m³)
1 = Sediments	1000 (Oxide)	2.44
2 = Volcanics		2.47
5 = Hornfels Proximal		2.50
6 = Hornfels Distal		2.40
10 = Early Mineralised Porphyry		2.43
20 = Intramineral Porphyry		2.47
30 = Late Mineral Felsic Dyke		2.47
31 = Late Mineral Dyke Eye		2.47
32 = Late Mineral Dyke stockwork		2.47
40 = Post Mineral Andesitic Dyke		2.37
1 = Sediments	2000 (Transitional)	2.63
2 = Volcanics		2.67
5 = Hornfels Proximal		2.59
6 = Hornfels Distal		2.67
10 = Early Mineralised Porphyry		2.65
20 = Intramineral Porphyry		2.65
30 = Late Mineral Felsic Dyke		2.65
31 = Late Mineral Dyke Eye		2.65
32 = Late Mineral Dyke stockwork		2.65
40 = Post Mineral Andesitic Dyke		2.47
1 = Sediments	3000 (Fresh)	2.85
2 = Volcanics		2.76
5 = Hornfels Proximal		2.86
6 = Hornfels Distal		2.83
10 = Early Mineralised Porphyry		2.70
20 = Intramineral Porphyry		2.71
30 = Late Mineral Felsic Dyke		2.77
31 = Late Mineral Dyke Eye		2.77
32 = Late Mineral Dyke stockwork		2.77
40 = Post Mineral Andesitic Dyke		2.64

Figure 14.22 shows the densities as coded in the final block model.

Figure 14.22: Long Section Showing the Bulk Density Sampling and Assignment to the Block Model



14.15.4 San Antonio

182 bulk density samples have been collected at San Antonio.

Bulk density for the Fresh material was calculated as 2.93 t/m^3 based on an average of all measurements. Limited variability was exhibited across lithologies, so the single density value was deemed suitable.

For oxide and transitional domains, no density measurements were available. A nominal -10% and -20% have been factored into the fresh density for the transitional and oxide domains, respectively (Table 14.43).

Further bulk density measurements will be required for advancement of the Project.

Table 14.43 : Densities coded to San Antonio

WTCODE	Density Assigned (t/m^3)	Notes
1000 (oxide)	2.34	-20% from Fresh
2000 (transitional)	2.64	-10% from Fresh
3000 (fresh)	2.93	Average of analysed bulk density

14.16 Resource Classification

The four project areas which make up the combined Costa Fuego Project MRE have been classified in accordance with CIM "Estimation of Mineral Resource and Mineral Reserve Best Practice Guidelines" (29

November 2019) and CIM "Environmental, Social and Governance Guidelines for Mineral Resources and Mineral Reserve Estimation" (8 September 2023).

A range of criteria was considered in determining the Mineral Resource classification, including:

- Drill data density
- Sample/assay confidence
- Confidence in density estimate
- Geological confidence in the interpretations and, similarly, geological continuity
- Grade continuity of the mineralisation
- Estimation method and resulting estimation output variables
- Estimation performance through validation
- Prospect for eventual economic extraction.

Wireframes were constructed to flag the blocks as Indicated or Inferred.

No Measured classification was applied.

The reported grades and metal contents are presented without metallurgical factors or assumptions and should be considered total metal endowment.

14.17 Reasonable Prospects for Eventual Economic Extraction (RPEEE)

Reporting for each project area considers RPEEE for open-pit mining, with the cost basis for cut-off grade analysis utilising costs from the 2023 PEA, which was relevant at the time of the February 2024 MRE release.

Open pit shells were generated using the industry-standard Lerchs-Grossman algorithm and the parameters listed in Table 14.44. Selling costs incorporate transport and handling costs, as well as any treatment and refining costs. Payable terms assume that the copper concentrate is sold into the Asian smelter market, for which these terms are standard.

Table 14.44: Key open pit optimisation parameters applied to generate a pit shell defining Reasonable Prospects of Eventual Economic Extraction

Costa Fuego Open Pit RPEEE Parameter Table		
Value	**Units**	**Description**
6	USD/lb	Optimistic Forward-Looking Copper Price
1700	USD/oz	Gold Price
20	USD/oz	Silver Price
14	USD/lb	Molybdenum Price
1.70	USD/t	Open Pit Mining Cost (incremental cost of 0.05/10m bench) - including G&A
6.88	USD/t	Sulphide Processing Cost
1.50	USD/t	Dump Leach Processing Cost

Costa Fuego Open Pit RPEEE Parameter Table		
Value	**Units**	**Description**
6.32	USD/t	Oxide Processing Cost
45 - 49° (depending on Deposit)	Degrees	Pit Slope Angle
0.42	USD/lb	Sulphide Copper Selling Cost
0.34	USD/lb	Oxide Copper Selling Cost
5	USD/oz	Gold Selling Cost
0.62	USD/oz	Silver Selling Cost
1.72	USD/lb	Molybdenum Selling Cost
Variable by Deposit and Oxidation State	%	Sulphide Copper Recovery
	%	Sulphide Gold Recovery
	%	Sulphide Molybdenum Recovery
	%	Sulphide Silver Recovery

The controlling RPEEE surfaces for the Productora and Cortadera open pits are shown in Figure 14.23 and Figure 14.24, respectively. These figures also include sensitivity testwork completed at different copper prices, showing that the RPEEE surface is not particularly sensitive to copper price over the ranges tested.

Figure 14.23 : Productora - Cross Section View Looking North Showing Pit Shell Used to Define RPEEE

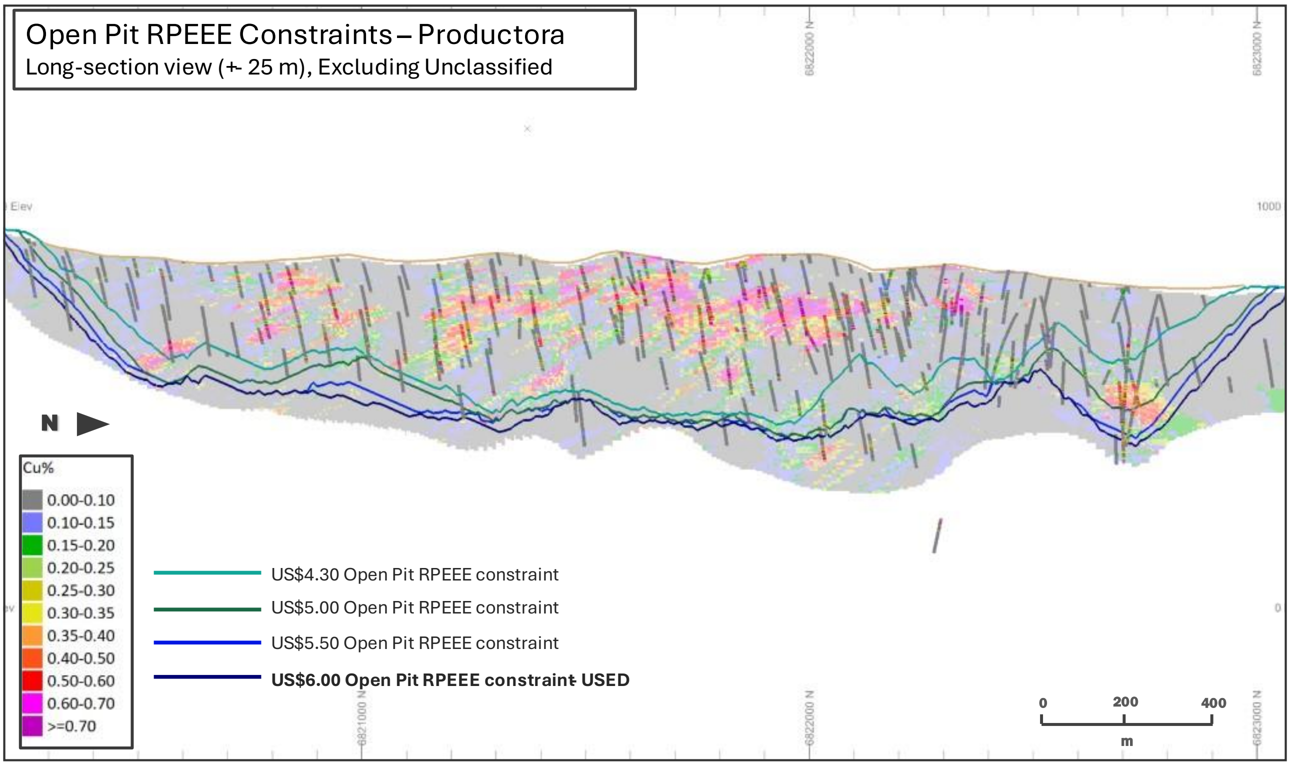
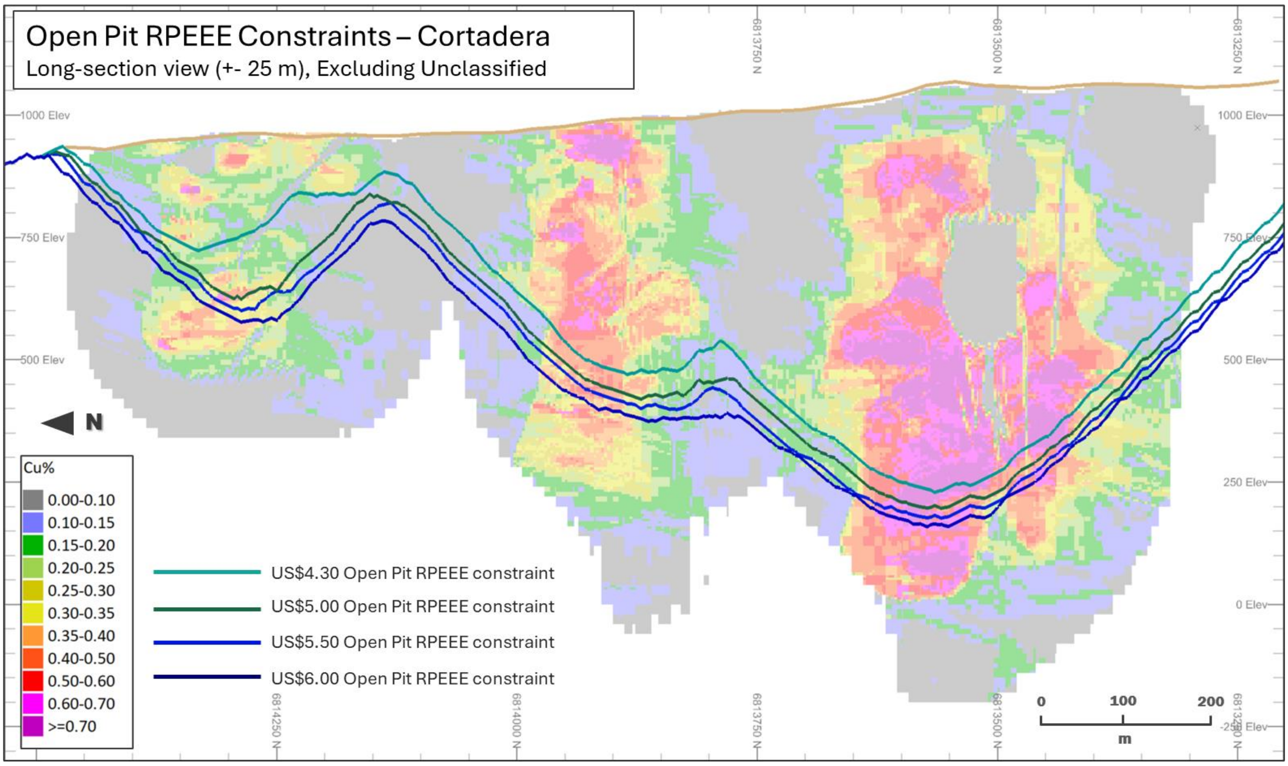


Figure 14.24 : Cortadera - Long Section View Looking Northeast Showing Pit Shell Used to Define RPEEE



Reporting for Cortadera also considers RPEEE for underground mining, with the cost basis for cut-off grade analysis utilising the costs from the 2016 study at Productora (nominally inflated and adjusted as applicable) and a benchmarking study into block-cave mining (Wood, 2020). Parameters are listed in Table 14.45.

For the underground mining scenario at Cortadera, a grade of 0.13% CuEq, was used an input cut-off grade to simulate a block cave/massive underground economic mining shape. The Resource Model was processed by mining software generally used to develop future potential economic underground shapes (Mineable Shape Optimiser/MSO) which applied a minimum-dimension mining shape. The minimum-dimension geometry applied was a cave draw shape typically 80 m wide x 80 m long x 80 m high (although sub-shapes were allowed within the MSO software, as they may be included in a draw point if occurring inside the typical footprint of the cave shape). This geometry was considered a reasonable size to initiate a cave when assuming pre-stress and a hydraulic radius of +20 m.

The resulting cave shapes produced by the MSO software serve to identify material that satisfies the RPEEE requirement by exploring a typical block cave mining strategy. The geotechnical work for a potential block cave propagation at Cortadera is currently preliminary in nature and these parameters are therefore based on conceptual geotechnical conditions.

The controlling surfaces for the underground RPEEE are shown in Figure 14.25. This figures also includes sensitivity testwork completed at different copper prices, showing that the underground RPEEE surface is not particularly sensitive to copper price at Cortadera.

Table 14.45 : Key Revenue and Cost Parameters for Estimating the Breakeven Grade for Reasonable Prospects of Eventual Economic Extraction by Block Cave Mining

Value	Units	Description
6	USD/lb	Optimistic Forward-Looking Copper Price
1700	USD/oz	Gold Price
20	USD/oz	Silver Price
14	USD/lb	Molybdenum Price
6.55	USD/t	Underground Mining Cost
6.77	USD/t	Processing Cost
0.67	USD/t	General and Administrative Cost
0.42	USD/lb	Sulphide Copper Selling Cost
5	USD/oz	Gold Selling Cost
0.62	USD/oz	Silver Selling Cost
1.72	USD/lb	Molybdenum Selling Cost
83	%	Sulphide Copper Recovery
56	%	Sulphide Gold Recovery
83	%	Sulphide Molybdenum Recovery
37	%	Sulphide Silver Recovery
40	%	Dump Leach Copper Recovery
40	m	Minimum Stope Size (X, Y, and Z)
80	m	Maximum Stope Size (X, Y, and Z)

Table 14.46 : The calculation of the breakeven caving grade to define Reasonable Prospects of Eventual Economic Extraction

CuEq %	US$/t	Description
	6.55	Mining Cost (Operating + Equipment Sustaining Capital)
	6.77	Total Processing Cost
	0.67	Fixed Costs
0.12	13.99	Required Mining & Processing Revenue Grade (USD/t)
	14.0	ROM Revenue Grade (Unplanned Dilution)
	15.6	Concentrate Revenue Grade (Flotation Recovery)
	17.4	Product Revenue Grade (Net Smelter Payability)
	0.23	6.4% Royalty on Operating Margin minus depreciation
0.13	17.6	Breakeven Development and Cave Grade

Figure 14.25 : Long-Section View Looking Showing Block Cave Mining Shapes Used to Define RPEEE

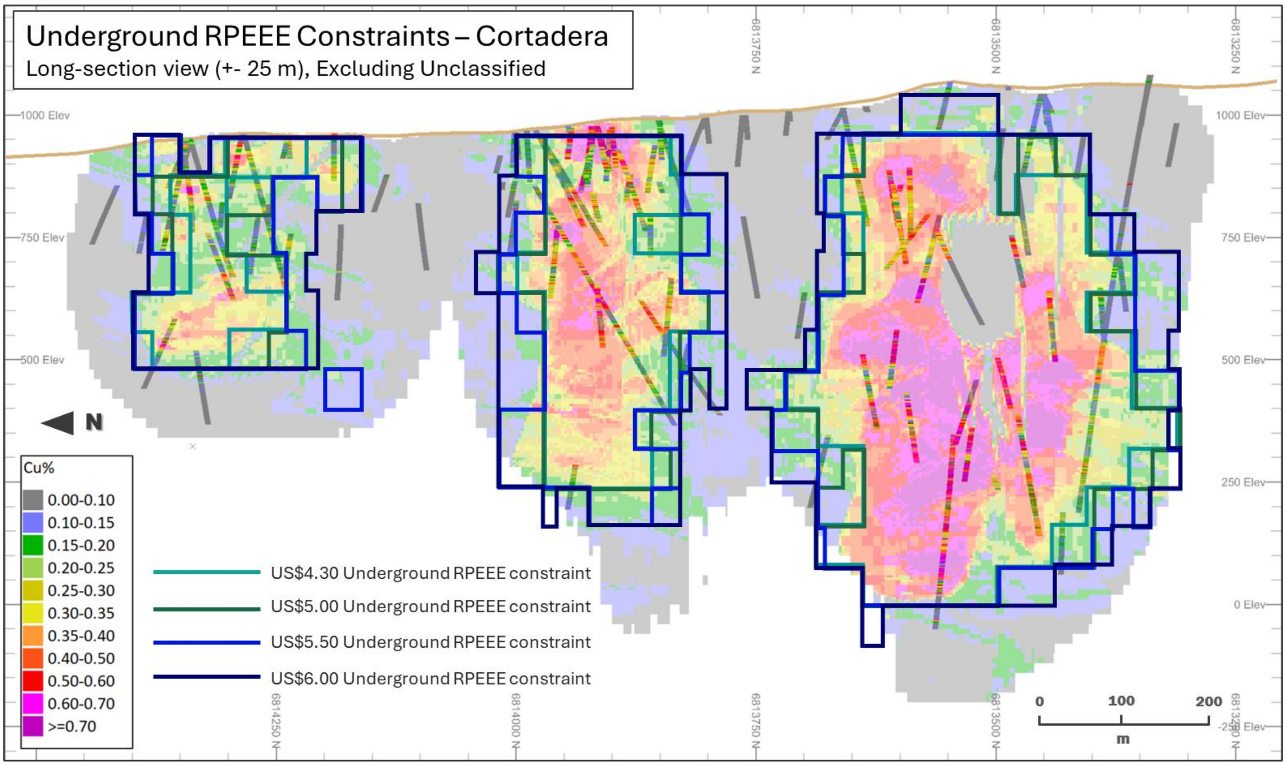
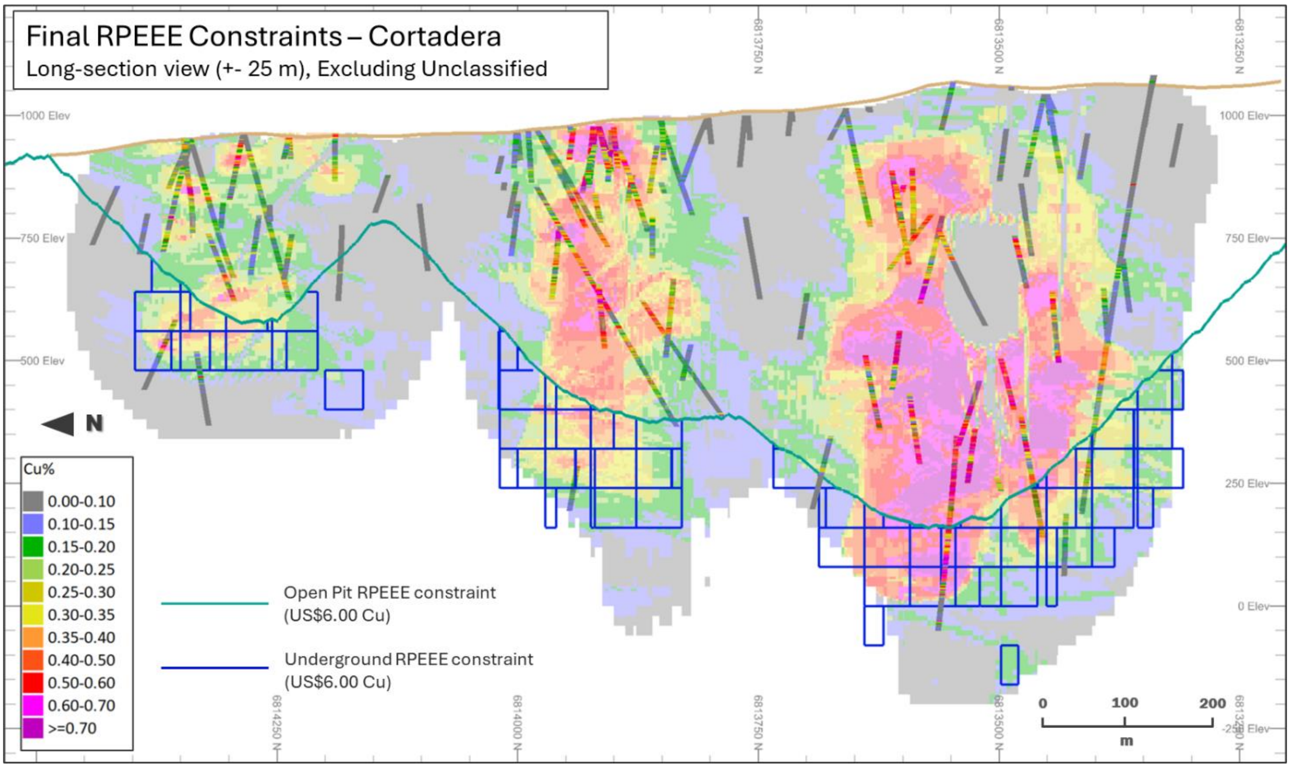


Figure 14.26 shows the final coded RPEEE classification (Open Pit or Underground) for Cortadera for the final $6.00 copper price scenario. With the significant overlap between Open Pit and Underground shapes in the near surface mineralisation, material that sits above the RPEEE pit shells is assumed as being extracted using open pit methodologies and has therefore been reported at a CuEq% cut-off grade aligned with open pit mining.

Figure 14.26 : Long Section View Showing Coding of Final Block Model With Open Pit and Block Cave Mining Reasonable Prospects of Eventual Economic Extraction – Excluding Unclassified Blocks



14.18 Calculation of Copper Equivalent for Costa Fuego

Copper equivalent (CuEq) reported for the resource was calculated using the following formula:

CuEq% = ((Cu% × Cu price 1% per tonne × Cu_recovery) + (Mo ppm × Mo price per g/t × Mo_recovery) + (Au ppm × Au price per g/t × Au_recovery) + (Ag ppm × Ag price per g/t × Ag_recovery)) / (Cu price 1% per tonne × Cu_recovery).

The metal prices applied in the calculation were:

Cu=3.00 US$/lb, Au=1,700 US$/oz, Mo=14 US$/lb, Ag=20 US$/oz.

For Cortadera, the average weighted metallurgical recoveries were:

Cu=82%, Au=55%, Mo=81%, Ag=36%.

For Productora, the average weighted metallurgical recoveries were:

Cu=84%, Au=47%, Mo=48%, Ag=18%

For Alice, the average weighted metallurgical recoveries were:

Cu=81%, Au=47%, Mo=52%, Ag=37%

For San Antonio, the average weighted metallurgical recoveries were:

Cu=85%, Au=66%, Mo=80%, Ag=63%

14.19 Costa Fuego Mineral Resource Statement

14.19.1 Mineral Resource Tables

The Cortadera, San Antonio, Productora, and Alice MREs have been combined to create the Costa Fuego Project MRE. The Costa Fuego Project MRE has an effective date of 26 February 2024.

The final reported tonnes and grade by classification are in Table 14.48 to Table 14.52 (Costa Fuego Project combined and broken down by Project area). Grade tonnage curves for both the Open Pit and Underground Resource are in Figure 14.27.

Reported grade and contained metal is total (metal recovery is not considered).

Mineral Resource estimation practices are in accordance with CIM "Estimation of Mineral Resource and Mineral Reserve Best Practice Guidelines" (29 November 2019), CIM "Environmental, Social and Governance Guidelines for Mineral Resources and Mineral Reserve Estimation" (8 September 2023) and CIM "Leading Practice Guidelines for Mineral Processing" (25 November 2022) and reported in accordance with CIM Definition Standards for Mineral Resources and Mineral Reserves (10 May 2014) that are incorporated by reference into NI 43-101. The mineral resources reported in Table 14-24 to Table 14-29, inclusively, are contained entirely within conceptual pit shells developed from the parameters discussed above. Based on these parameters, cutoff grades for Hangar Flats, West End and Yellow Pine were calculated based on a $1,250/oz gold selling price, which resulted in an open pit sulphide cutoff grade of approximately 0.45 g/t Au and an open pit oxide cutoff grade of approximately 0.40 g/t Au. Only mineral resources above these cutoffs and within the mineral resource-limiting pits are reported and, as such, mineralization falling below this cutoff grade or outside the mineral resource-limiting pit is not reported, irrespective of the grade. To demonstrate mineral resource sensitivity to gold price and cut-off grade, mineralized tonnage and grade is reported in Table 14-30 within multiple conceptual pit shells optimized at different gold selling prices.

Following release of the PEA in June 2023, a review of MRE appropriate CuEq Cut-off Grades (COG) was completed, with revisions to long-term consensus copper price assumptions and breakeven grade assessments considered.

The long-term consensus copper price assumption changed from US$ 3.30/lb Cu in 2022, to US$ 3.85/lb Cu in 2024 and to US$ 4.30 in February 2025. The change in copper price, in combination with the latest costs, as informed by the June 2023 PEA, has reduced the breakeven grade for the Costa Fuego Project.

The revised COGs reflect these changes in assumptions and have been set appropriately higher than the calculated breakeven grade.

These key assumptions in relation to COG's are summarised in Table 14.47.

Table 14.47: Summary of COG and Copper Price Changes – 2022 to 2024

Year	Copper Price	Breakeven COG	Open Pit COG	Underground COG
	US$/lb	%CuEq	%CuEq*	%CuEq
2022	3.30	0.18	0.21	0.30
2024	**3.85**	**0.15**	**0.20**	**0.27**

The information in this Report that relates to Mineral Resources for Cortadera, San Antonio, Productora, and Alice which constitute the combined Costa Fuego Project is based on information compiled by Ms. Elizabeth Haren, a Qualified Person, who is a Fellow and Chartered Professional of The Australasian Institute of Mining and Metallurgy and a Member of the Australian Institute of Geoscientists.

Table 14.48 : Costa Fuego Project Mineral Resource Summary – Reported by Classification (26 February 2024)

Costa Fuego OP Resource		**Grade**					**Contained Metal**				
Classification	Tonnes	CuEq	Cu	Au	Ag	Mo	Copper Eq	Copper	Gold	Silver	Molybdenum
(+0.20% CuEq*)	(Mt)	(%)	(%)	(g/t)	(g/t)	(ppm)	(tonnes)	(tonnes)	(ounces)	(ounces)	(tonnes)
Indicated	736	0.46	0.37	0.11	0.50	85	3,370,000	2,720,000	2,480,000	11,700,000	62,800
M+I Total	**736**	**0.46**	**0.37**	**0.11**	**0.50**	**85**	**3,370,000**	**2,720,000**	**2,480,000**	**11,700,000**	**62,800**
Inferred	170	0.30	0.25	0.06	0.36	65	520,000	420,000	340,000	1,900,000	11,000
Costa Fuego UG Resource		**Grade**					**Contained Metal**				
Classification	Tonnes	CuEq	Cu	Au	Ag	Mo	Copper Eq	Copper	Gold	Silver	Molybdenum
(+0.27% CuEq*)	(Mt)	(%)	(%)	(g/t)	(g/t)	(ppm)	(tonnes)	(tonnes)	(ounces)	(ounces)	(tonnes)
Indicated	62	0.39	0.31	0.08	0.55	85	250,000	190,000	160,000	1,100,000	5,300
M+I Total	**62**	**0.39**	**0.31**	**0.08**	**0.55**	**85**	**250,000**	**190,000**	**160,000**	**1,100,000**	**5,300**
Inferred	33	0.35	0.29	0.07	0.41	46	120,000	96,000	76,000	430,000	1,500
Costa Fuego Total Resource		**Grade**					**Contained Metal**				
Classification	Tonnes	CuEq	Cu	Au	Ag	Mo	Copper Eq	Copper	Gold	Silver	Molybdenum
	(Mt)	(%)	(%)	(g/t)	(g/t)	(ppm)	(tonnes)	(tonnes)	(ounces)	(ounces)	(tonnes)
Indicated	798	0.45	0.37	0.10	0.50	85	3,620,000	2,910,000	2,640,000	12,800,000	68,100
M+I Total	**798**	**0.45**	**0.37**	**0.10**	**0.50**	**85**	**3,620,000**	**2,910,000**	**2,640,000**	**12,800,000**	**68,100**
Inferred	203	0.31	0.25	0.06	0.36	61	640,000	516,000	416,000	2,330,000	12,500

[1] Mineral Resources are reported on a 100% Basis - combining Mineral Resource Estimates for the Cortadera, Productora, Alice and San Antonio deposits. All figures are rounded, reported to appropriate significant figures and reported in accordance with the Joint Ore Reserves Committee Code (2012) and NI 43-101. Mineral Resource estimation practices are in accordance with CIM Estimation of Mineral Resource and Mineral Reserve Best Practice Guidelines (29 November 2019),d CIM Environmental, Social and Governance Guidelines for Mineral Resources and Mineral Reserve Estimation (8 September 2023) and CIM "Leading Practice Guidelines for Mineral Processing" (25 November 2022) and reported in accordance CIM Definition Standards for Mineral Resources and Mineral Reserves (10 May 2014) that are incorporated by reference into NI 43-101.

[2] The Productora deposit is 100% owned by Chilean incorporated company Sociedad Minera El Aguila SpA (SMEA). SMEA is a joint venture (JV) company – 80% owned by Sociedad Minera El Corazón Limitada (a 100% subsidiary of Hot Chili Limited), and 20% owned by Compañía Minera del Pacífico S.A (CMP).

[3] The Cortadera deposit is controlled by a Chilean incorporated company Sociedad Minera La Frontera SpA (Frontera). Frontera is a subsidiary company – 100% owned by Sociedad Minera El Corazón Limitada, which is a 100% subsidiary of Hot Chili Limited.

[4] The San Antonio deposit is controlled through Frontera (100% owned by Sociedad Minera El Corazón Limitada, which is a 100% subsidiary of Hot Chili Limited), and Frontera has an option agreement to earn a 100% interest.

[5] The MRE in the tables above form coherent bodies of mineralisation that are considered amenable to a combination of open pit and underground extraction methods based on the following parameters: Base Case Metal Prices: Copper US$3.00/lb, Gold US$1,700/oz, Molybdenum US$14/lb, and Silver US$20/oz.
[6] All MRE were assessed for Reasonable Prospects of Eventual Economic Extraction (RPEEE) using both Open Pit and Block Cave Extraction mining methods at Cortadera and Open Pit mining methods at the Productora, Alice and San Antonio deposits.
[7] Metallurgical recovery averages for each deposit consider Indicated + Inferred material and are weighted to combine sulphide flotation and oxide leaching performance. Process recoveries: Cortadera – Weighted recoveries of 82% Cu, 55% Au, 81% Mo and 36% Ag. CuEq(%) = Cu(%) + 0.55 x Au(g/t) + 0.00046 x Mo(ppm) + 0.0043 x Ag(g/t). San Antonio - Weighted recoveries of 85% Cu, 66% Au, 80% Mo and 63% Ag. CuEq(%) = Cu(%) + 0.64 x Au(g/t) + 0.00044 x Mo(ppm) + 0.0072 x Ag(g/t). Alice - Weighted recoveries of 81% Cu, 47% Au, 52% Mo and 37% Ag. CuEq(%) = Cu(%) + 0.48 x Au(g/t) + 0.00030 x Mo(ppm) + 0.0044 x Ag(g/t). Productora – Weighted recoveries of 84% Cu, 47% Au, 48% Mo and 18% Ag. CuEq(%) = Cu(%) + 0.46 x Au(g/t) + 0.00026 x Mo(ppm) + 0.0021 x Ag(g/t). Costa Fuego – Recoveries of 83% Cu, 53% Au, 71% Mo and 26% Ag. CuEq(%) = Cu(%) + 0.53 x Au(g/t) + 0.00040 x Mo(ppm) + 0.0030 x Ag(g/t).
[8] Copper Equivalent (CuEq) grades are calculated based on the formula: CuEq% = ((Cu% × Cu price 1% per tonne × Cu_recovery) + (Mo ppm × Mo price per g/t × Mo_recovery) + (Au ppm × Au price per g/t × Au_recovery) + (Ag ppm × Ag price per g/t × Ag_recovery)) / (Cu price 1% per tonne × Cu recovery). The base case cut-off grade for Mineral Resources considered amenable to open pit extraction methods at the Cortadera, Productora, Alice and San Antonio deposits is 0.20% CuEq, while the cut-off grade for Mineral Resources considered amenable to underground extraction methods at the Cortadera deposit is 0.27% CuEq. It is the Company's opinion that all the elements included in the CuEq calculation have a reasonable potential to be recovered and sold.
[9] Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. These MRE include Inferred Mineral Resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorised as Mineral Reserves. It is reasonably expected that the majority of Inferred mineral resources could be upgraded to Measured or Indicated Mineral Resources with continued exploration.
[10] The effective date of the MRE is 26 February 2024 (the "2024 Effective Date"). Refer to ASX Announcement "Hot Chili Indicated Resource at Costa Fuego Copper-Gold Project Increases to 798 Mt" for JORC Table 1 information in this statement related to the Costa Fuego Project Mineral Resource Estimate by Competent Person Elizabeth Haren, who is also a Qualified Person (within the meaning of NI 43-101) constituting the MRE of Cortadera, Productora, Alice and San Antonio (which combine to form the Costa Fuego Project). Hot Chili Limited confirms it is not aware of any new information or data that materially affects the information included in the Resource Announcement and all material assumptions and technical parameters stated for the Mineral Resource Estimates in the Resource Announcement continue to apply and have not materially changed since the 2024 Effective Date.
[11] Hot Chili Limited is not aware of political, environmental or other risks that could materially affect the potential development of the Mineral Resources, other than those common to all such projects, including the permitting of a mining operation, access to adequate funding on reasonable terms, etc. A detailed list of Project risks is included in Section 25.12 of this Report.

Table 14.49 : Cortadera Mineral Resource Summary – Reported by Classification (26 February 2024)

Cortadera OP Resource		**Grade**					**Contained Metal**				
Classification	Tonnes	CuEq	Cu	Au	Ag	Mo	Copper Eq	Copper	Gold	Silver	Molybdenum
(+0.20% CuEq*)	(Mt)	(%)	(%)	(g/t)	(g/t)	(ppm)	(tonnes)	(tonnes)	(ounces)	(ounces)	(tonnes)
Indicated	469	0.44	0.35	0.12	0.58	59	2,070,000	1,620,000	1,800,000	8,790,000	27,500
M+I Total	**469**	**0.44**	**0.35**	**0.12**	**0.58**	**59**	**2,070,000**	**1,620,000**	**1,800,000**	**8,790,000**	**27,500**
Inferred	116	0.28	0.21	0.06	0.38	53	320,000	250,000	230,000	1,400,000	6,200
Cortadera UG Resource		**Grade**					**Contained Metal**				

Table 14.49 : Cortadera Mineral Resource Summary – Reported by Classification (26 February 2024)											
Classification	Tonnes	CuEq	Cu	Au	Ag	Mo	Copper Eq	Copper	Gold	Silver	Molybdenum
(+0.27% CuEq*)	(Mt)	(%)	(%)	(g/t)	(g/t)	(ppm)	(tonnes)	(tonnes)	(ounces)	(ounces)	(tonnes)
Indicated	62	0.39	0.31	0.08	0.55	85	250,000	190,000	160,000	1,100,000	5,300
M+I Total	**62**	**0.39**	**0.31**	**0.08**	**0.55**	**85**	**250,000**	**190,000**	**160,000**	**1,100,000**	**5,300**
Inferred	33	0.35	0.29	0.07	0.41	46	120,000	96,000	76,000	430,000	1,500
Cortadera Total Resource		**Grade**					**Contained Metal**				
Classification	Tonnes	CuEq	Cu	Au	Ag	Mo	Copper Eq	Copper	Gold	Silver	Molybdenum
	(Mt)	(%)	(%)	(g/t)	(g/t)	(ppm)	(tonnes)	(tonnes)	(ounces)	(ounces)	(tonnes)
Indicated	531	0.44	0.34	0.11	0.58	62	2,320,000	1,810,000	1,960,000	9,890,000	32,800
M+I Total	**531**	**0.44**	**0.34**	**0.11**	**0.58**	**62**	**2,320,000**	**1,810,000**	**1,960,000**	**9,890,000**	**32,800**
Inferred	149	0.30	0.23	0.06	0.38	52	440,000	346,000	306,000	1,830,000	7,700

[1] Mineral Resources are reported on a 100% Basis. All figures are rounded, reported to appropriate significant figures and reported in accordance with the Joint Ore Reserves Committee Code (2012) and NI 43-101. Mineral resource estimation practices are in accordance with CIM Estimation of Mineral Resource and Mineral Reserve Best Practice Guidelines (29 November 2019) and CIM Environmental, Social and Governance Guidelines for Mineral Resources and Mineral Reserve Estimation (8 September 2023) and reported in accordance CIM Definition Standards for Mineral Resources and Mineral Reserves (10 May 2014) that are incorporated by reference into NI 43-101.
[2] The Cortadera deposit is controlled by a Chilean incorporated company Sociedad Minera La Frontera SpA (Frontera). Frontera is a subsidiary company – 100% owned by Sociedad Minera El Corazón Limitada, which is a 100% subsidiary of Hot Chili Limited.
[3] The MRE in the tables above form coherent bodies of mineralisation that are considered amenable to open pit and underground extraction methods based on the following parameters: Base Case Metal Prices: Copper US$3.00/lb, Gold US$1,700/oz, Molybdenum US$14/lb, and Silver US$20/oz.
[4] The Cortadera MRE was assessed for Reasonable Prospects of Eventual Economic Extraction (RPEEE) using both Open Pit and Underground mining methods. Metallurgical recovery averages for Cortadera consider Indicated + Inferred material and are weighted to combine sulphide flotation and oxide leaching performance.
[5] Resource Copper Equivalent (CuEq) grades are calculated based on the formula: CuEq% = ((Cu% × Cu price 1% per tonne × Cu_recovery) + (Mo ppm × Mo price per g/t × Mo_recovery) + (Au ppm × Au price per g/t × Au_recovery) + (Ag ppm × Ag price per g/t × Ag_recovery)) / (Cu price 1% per tonne × Cu recovery). The base case cut-off grade for Mineral Resources considered amenable to open pit extraction methods at Cortadera is 0.20% CuEq while the cut-off grade for Mineral Resources considered amenable to underground extraction methods at the Cortadera deposit is 0.27% CuEq. Weighted recoveries of 82% Cu, 55% Au, 81% Mo and 36% Ag were used. CuEq(%) = Cu(%) + 0.55 x Au(g/t) + 0.00046 x Mo(ppm) + 0.0043 x Ag(g/t).
[6] Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. These Mineral Resource Estimates include Inferred Mineral Resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorised as Mineral Reserves. It is reasonably expected that the majority of Inferred mineral resources could be upgraded to Measured or Indicated Mineral Resources with continued exploration.
[7] The effective date of the MRE is 26 February 2024 (the "2024 Effective Date"). Refer to ASX Announcement "Hot Chili Indicated Resource at Costa Fuego Copper-Gold Project Increases to 798 Mt" for JORC Table 1 information in this announcement related to the Costa Fuego Project Mineral Resource Estimate by Competent Person Elizabeth Haren, who is also a Qualified Person (within the meaning of NI 43-101) constituting the MREs of Cortadera, Productora, Alice and San Antonio (which combine to form the Costa Fuego Project). Hot Chili Limited confirms it is not aware of any new information or data that materially affects the information included in the Resource Announcement and all material assumptions and

technical parameters stated for the Mineral Resource Estimates in the Resource Announcement continue to apply and have not materially changed since the 2024 Effective Date.

[8] Hot Chili Limited is not aware of political, environmental or other risks that could materially affect the potential development of the Mineral Resources, other than those common to all such projects, including the permitting of a mining operation, access to adequate funding on reasonable terms, etc. A detailed list of Project risks is included in Section 25.12 of this Report.

Table 14.50 : Productora Mineral Resource Summary – Reported by Classification (26 February 2024)

Productora Total Resource		Grade					Contained Metal				
Classification	Tonnes	CuEq	Cu	Au	Ag	Mo	Copper Eq	Copper	Gold	Silver	Molybdenum
(+0.20% CuEq*)	(Mt)	(%)	(%)	(g/t)	(g/t)	(ppm)	(tonnes)	(tonnes)	(ounces)	(ounces)	(tonnes)
Indicated	248	0.49	0.41	0.08	0.35	140	1,210,000	1,020,000	668,000	2,760,000	34,600
M+I Total	**248**	**0.49**	**0.41**	**0.08**	**0.35**	**140**	**1,210,000**	**1,020,000**	**668,000**	**2,760,000**	**34,600**
Inferred	52	0.36	0.31	0.07	0.27	92	190,000	160,000	110,000	450,000	4,800

[1] Mineral Resources are reported on a 100% Basis. All figures are rounded, reported to appropriate significant figures and reported in accordance with the Joint Ore Reserves Committee Code (2012) and NI 43-101. Mineral resource estimation practices are in accordance with CIM Estimation of Mineral Resource and Mineral Reserve Best Practice Guidelines (29 November 2019) and CIM Environmental, Social and Governance Guidelines for Mineral Resources and Mineral Reserve Estimation (8 September 2023) and reported in accordance CIM Definition Standards for Mineral Resources and Mineral Reserves (10 May 2014) that are incorporated by reference into NI 43-101.

[2] The Productora deposit is 100% owned by Chilean incorporated company Sociedad Minera El Aguila SpA (SMEA). SMEA is a joint venture (JV) company – 80% owned by Sociedad Minera El Corazón Limitada (a 100% subsidiary of Hot Chili Limited), and 20% owned by Compañía Minera del Pacífico S.A (CMP).

[3] The MRE in the tables above form coherent bodies of mineralisation that are considered amenable to open pit extraction methods based on the following parameters: Base Case Metal Prices: Copper US$3.00/lb, Gold US$1,700/oz, Molybdenum US$14/lb, and Silver US$20/oz.

[4] The Productora MRE was assessed for Reasonable Prospects of Eventual Economic Extraction (RPEEE) using an Open Pit mining method. Metallurgical recovery averages for Productora consider Indicated and Inferred material and are weighted to combine sulphide flotation and oxide leaching performance.

[5] Resource Copper Equivalent (CuEq) grades are calculated based on the formula: CuEq% = ((Cu% × Cu price 1% per tonne × Cu_recovery) + (Mo ppm × Mo price per g/t × Mo_recovery) + (Au ppm × Au price per g/t × Au_recovery) + (Ag ppm × Ag price per g/t × Ag_recovery)) / (Cu price 1% per tonne × Cu recovery). The base case cut-off grade for Mineral Resources considered amenable to open pit extraction methods at Productora is 0.20% CuEq. Weighted recoveries of 84% Cu, 47% Au, 48% Mo and 18% Ag were used. CuEq(%) = Cu(%) + 0.46 x Au(g/t) + 0.00027 x Mo(ppm) + 0.0021 x Ag(g/t).

[6] Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. These Mineral Resource Estimates include Inferred Mineral Resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorised as Mineral Reserves. It is reasonably expected that the majority of Inferred mineral resources could be upgraded to Measured or Indicated Mineral Resources with continued exploration.

[7] The effective date of the MRE is 26 February 2024 (the "2024 Effective Date"). Refer to ASX Announcement "Hot Chili Indicated Resource at Costa Fuego Copper-Gold Project Increases to 798 Mt" for JORC Table 1 information in this announcement related to the Costa Fuego Project Mineral Resource Estimate by Competent Person Elizabeth Haren, who is also a Qualified Person (within the meaning of NI 43-101) constituting the MREs of Cortadera, Productora, Alice and San Antonio (which combine to form the Costa Fuego Project). Hot Chili Limited confirms it is not aware of any new information or data that materially affects the information included in the Resource Announcement and all material assumptions and technical parameters stated for the Mineral Resource Estimates in the Resource Announcement continue to apply and have not materially changed since the 2024 Effective Date.

[8] Hot Chili Limited is not aware of political, environmental or other risks that could materially affect the potential development of the Mineral Resources, other than those common to all such projects, including the permitting of a mining operation, access to adequate funding on reasonable terms, etc. A detailed list of Project risks is included in Section 25.12 of this Report.

Table 14.51 : San Antonio Mineral Resource Summary – Reported by Classification (26 February 2024)

San Antonio Total Resource		Grade					Contained Metal				
Classification	Tonnes	CuEq	Cu	Au	Ag	Mo	Copper Eq	Copper	Gold	Silver	Molybdenum
(+0.20% CuEq*)	(Mt)	(%)	(%)	(g/t)	(g/t)	(ppm)	(tonnes)	(tonnes)	(ounces)	(ounces)	(tonnes)
Indicated	3	0.71	0.70	0.01	1.12	2	22,200	21,800	710	113,000	6
M+I Total	**3**	**0.71**	**0.70**	**0.01**	**1.12**	**2**	**22,200**	**21,800**	**710**	**113,000**	**6**
Inferred	2	0.41	0.40	0.01	0.95	2	7,800	7,500	670	57,000	4

[1] Mineral Resources are reported on a 100% Basis. All figures are rounded, reported to appropriate significant figures and reported in accordance with the Joint Ore Reserves Committee Code (2012) and NI 43-101. Mineral resource estimation practices are in accordance with CIM Estimation of Mineral Resource and Mineral Reserve Best Practice Guidelines (29 November 2019) and CIM Environmental, Social and Governance Guidelines for Mineral Resources and Mineral Reserve Estimation (8 September 2023) and reported in accordance CIM Definition Standards for Mineral Resources and Mineral Reserves (May 10, 2014) that are incorporated by reference into NI 43-101.
[2] The San Antonio deposit is controlled through Frontera (100% owned by Sociedad Minera El Corazón Limitada, which is a 100% subsidiary of Hot Chili Limited), and Frontera has an option agreement to earn a 100% interest.
[3] The MRE in the tables above form coherent bodies of mineralisation that are considered amenable to open pit extraction methods based on the following parameters: Base Case Metal Prices: Copper US$3.00/lb, Gold US$1,700/oz, Molybdenum US$14/lb, and Silver US$20/oz.
[4] The San Antonio MRE was assessed for Reasonable Prospects of Eventual Economic Extraction (RPEEE) using an Open Pit mining method. Metallurgical recovery averages for San Antonio consider Indicated + Inferred material and are weighted to combine sulphide flotation and oxide leaching performance.
[5] Resource Copper Equivalent (CuEq) grades are calculated based on the formula: CuEq% = ((Cu% × Cu price 1% per tonne × Cu_recovery) + (Mo ppm × Mo price per g/t × Mo_recovery) + (Au ppm × Au price per g/t × Au_recovery) + (Ag ppm × Ag price per g/t × Ag_recovery)) / (Cu price 1% per tonne × Cu recovery). The base case cut-off grade for Mineral Resources considered amenable to open pit extraction methods at San Antonio is 0.20% CuEq. Weighted recoveries of 85% Cu, 66% Au, 80% Mo and 63% Ag were used. CuEq(%) = Cu(%) + 0.64 x Au(g/t) + 0.00044 x Mo(ppm) + 0.0072 x Ag(g/t).
[6] Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. These Mineral Resource Estimates include Inferred Mineral Resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorised as Mineral Reserves. It is reasonably expected that the majority of Inferred mineral resources could be upgraded to Measured or Indicated Mineral Resources with continued exploration.
[7] The effective date of the estimate of MRE is 26 February 2024 (the "2024 Effective Date"). Refer to ASX Announcement "Hot Chili Indicated Resource at Costa Fuego Copper-Gold Project Increases to 798 Mt" for JORC Table 1 information in this announcement related to the Costa Fuego Project Mineral Resource Estimate by Competent Person Elizabeth Haren, who is also a Qualified Person (within the meaning of NI 43-101) constituting the MREs of Cortadera, Productora, Alice and San Antonio (which combine to form the Costa Fuego Project). Hot Chili Limited confirms it is not aware of any new information or data that materially affects the information included in the Resource Announcement and all material assumptions and technical parameters stated for the Mineral Resource Estimates in the Resource Announcement continue to apply and have not materially changed since the 2024 Effective Date.
[10] Hot Chili Limited is not aware of political, environmental or other risks that could materially affect the potential development of the Mineral Resources, other than those common to all such projects, including the permitting of a

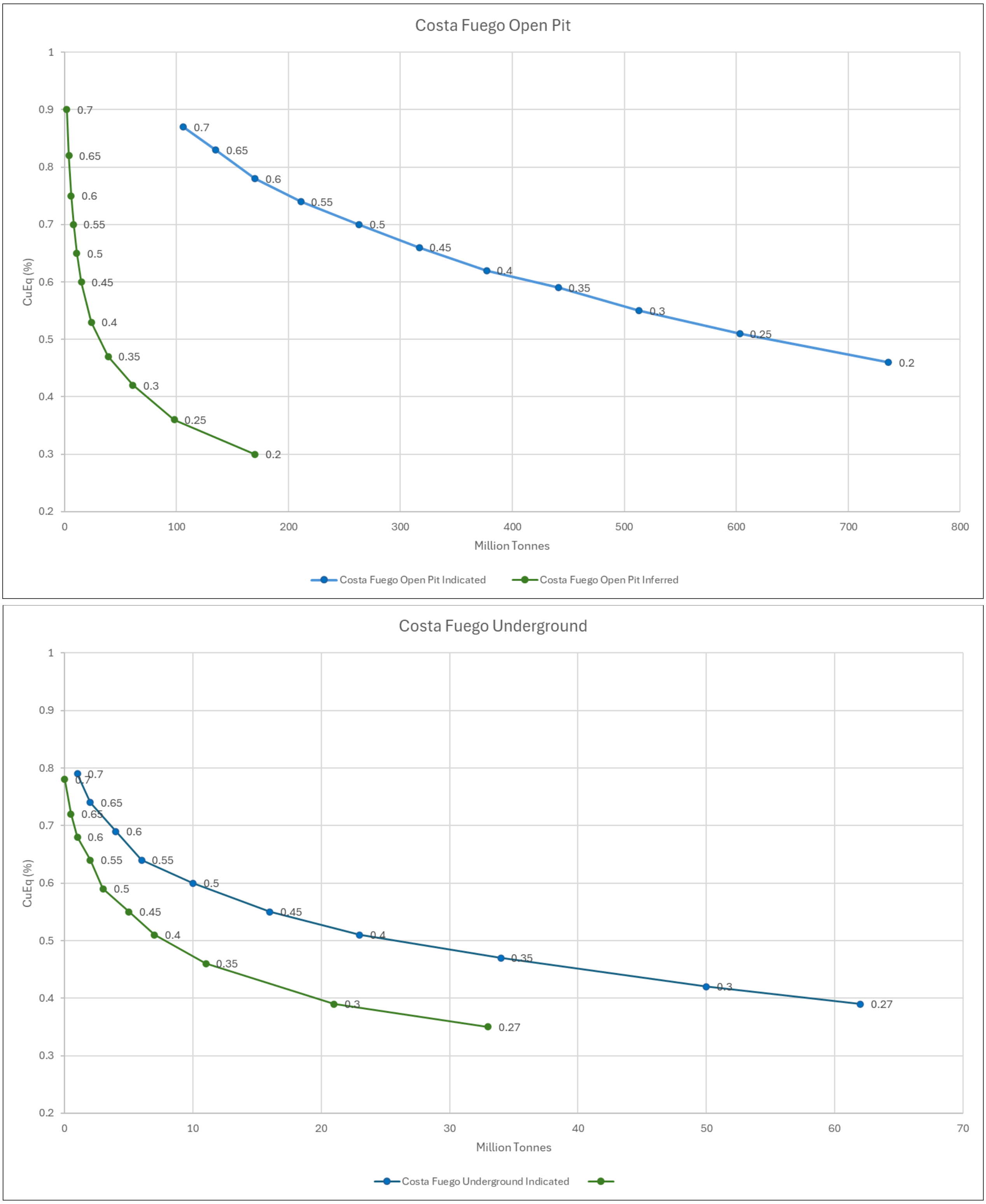
mining operation, access to adequate funding on reasonable terms, etc. A detailed list of Project risks is included in Section 25.12 of this Report.

Table 14.52 : Alice Mineral Resource Summary – Reported by Classification (26 February 2024)											
Alice Total Resource		**Grade**					**Contained Metal**				
Classification	Tonnes	CuEq	Cu	Au	Ag	Mo	Copper Eq	Copper	Gold	Silver	Molybdenum
(+0.20% CuEq*)	(Mt)	(%)	(%)	(g/t)	(g/t)	(ppm)	(tonnes)	(tonnes)	(ounces)	(ounces)	(tonnes)
Indicated	16	0.37	0.35	0.03	0.16	45	59,700	55,300	17,200	80,200	725
M+I Total	**16**	**0.37**	**0.35**	**0.03**	**0.16**	**45**	**59,700**	**55,300**	**17,200**	**80,200**	**725**
Inferred	-	-	-	-	-	-	-	-	-	-	-

[1] Mineral Resources are reported on a 100% Basis. All figures are rounded, reported to appropriate significant figures and reported in accordance with the Joint Ore Reserves Committee Code (2012) and NI 43-101. Mineral resource estimation practices are in accordance with CIM Estimation of Mineral Resource and Mineral Reserve Best Practice Guidelines (29 November 2019) and CIM Environmental, Social and Governance Guidelines for Mineral Resources and Mineral Reserve Estimation (8 September 2023) and reported in accordance CIM Definition Standards for Mineral Resources and Mineral Reserves (May 10, 2014) that are incorporated by reference into NI 43-101.
[2] The Alice deposit is 100% owned by Chilean incorporated company Sociedad Minera El Aguila SpA (SMEA). SMEA is a joint venture (JV) company – 80% owned by Sociedad Minera El Corazón Limitada (a 100% subsidiary of Hot Chili Limited), and 20% owned by Compañía Minera del Pacífico S.A (CMP).
[3] The MRE in the tables above form coherent bodies of mineralisation that are considered amenable to open pit extraction methods based on the following parameters: Base Case Metal Prices: Copper US$3.00/lb, Gold US$1,700/oz, Molybdenum US$14/lb, and Silver US$20/oz.
[4] The Alice MRE was assessed for Reasonable Prospects of Eventual Economic Extraction (RPEEE) using Open Pit mining method. Metallurgical recovery averages for Alice consider Indicated + Inferred material and are weighted to combine sulphide flotation and oxide leaching performance.
[5] Resource Copper Equivalent (CuEq) grades are calculated based on the formula: CuEq% = ((Cu% × Cu price 1% per tonne × Cu_recovery) + (Mo ppm × Mo price per g/t × Mo_recovery) + (Au ppm × Au price per g/t × Au_recovery) + (Ag ppm × Ag price per g/t × Ag_recovery)) / (Cu price 1% per tonne × Cu recovery). The base case cut-off grade for Mineral Resources considered amenable to open pit extraction methods at Alice is 0.20% CuEq.
[6] Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. These Mineral Resource Estimates include Inferred Mineral Resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorised as Mineral Reserves. It is reasonably expected that the majority of Inferred mineral resources could be upgraded to Measured or Indicated Mineral Resources with continued exploration.
[7] The effective date of the estimate of MRE is 26 February 2024 (the "2024 Effective Date"). Refer to ASX Announcement "Hot Chili Indicated Resource at Costa Fuego Copper-Gold Project Increases to 798 Mt" for JORC Table 1 information in this announcement related to the Costa Fuego Project Mineral Resource Estimate by Competent Person Elizabeth Haren, who is also a Qualified Person (within the meaning of NI 43-101) constituting the MREs of Cortadera, Productora, Alice and San Antonio (which combine to form the Costa Fuego Project). Hot Chili Limited confirms it is not aware of any new information or data that materially affects the information included in the Resource Announcement and all material assumptions and technical parameters stated for the Mineral Resource Estimates in the Resource Announcement continue to apply and have not materially changed since the 2024 Effective Date.
[8] Hot Chili Limited is not aware of political, environmental or other risks that could materially affect the potential development of the Mineral Resources, other than those common to all such projects, including the permitting of a mining operation, access to adequate funding on reasonable terms, etc. A detailed list of Project risks is included in Section 25.12 of this Report.

Table 14.53 : Costa Fuego Sensitivity to Cut-off Grade – Open Pit and Underground													
CuEq cut-off	**Costa Fuego Open Pit Indicated**						**CuEq cut-off**	**Costa Fuego Open Pit Inferred**					
	Mt	CuEq %	Cu%	Au	Ag	Mo		Mt	CuEq %	Cu%	Au	Ag	Mo
0.70	106	0.87	0.71	0.19	0.82	149	0.70	2	0.90	0.78	0.15	0.61	186
0.65	135	0.83	0.67	0.18	0.79	145	0.65	4	0.82	0.70	0.13	0.55	176
0.60	170	0.78	0.64	0.17	0.76	141	0.60	6	0.75	0.65	0.12	0.50	165
0.55	211	0.74	0.61	0.17	0.73	134	0.55	8	0.70	0.60	0.12	0.46	160
0.50	263	0.70	0.57	0.16	0.71	125	0.50	11	0.65	0.56	0.11	0.45	150
0.45	317	0.66	0.54	0.15	0.67	116	0.45	15	0.60	0.51	0.11	0.44	138
0.40	377	0.62	0.51	0.14	0.64	109	0.40	24	0.53	0.45	0.10	0.43	118
0.35	441	0.59	0.48	0.13	0.61	102	0.35	39	0.47	0.39	0.09	0.41	102
0.30	513	0.55	0.45	0.13	0.58	96	0.30	61	0.42	0.35	0.08	0.39	88
0.25	603	0.51	0.41	0.12	0.54	91	0.25	98	0.36	0.30	0.07	0.37	76
0.20	**736**	**0.46**	**0.37**	**0.11**	**0.50**	**85**	**0.20**	**170**	**0.30**	**0.25**	**0.06**	**0.36**	**65**
CuEq cut-off	**Costa Fuego Underground Indicated**						**CuEq cut-off**	**Costa Fuego Underground Inferred**					
	Mt	CuEq %	Cu%	Au	Ag	Mo		Mt	CuEq %	Cu%	Au	Ag	Mo
0.70	1	0.79	0.61	0.16	0.88	168	0.70	0	0.78	0.67	0.14	0.61	54
0.65	2	0.74	0.57	0.15	0.83	173	0.65	1	0.72	0.61	0.14	0.60	47
0.60	4	0.69	0.52	0.14	0.78	189	0.60	1	0.68	0.58	0.13	0.59	45
0.55	6	0.64	0.59	0.13	0.75	176	0.55	2	0.64	0.54	0.13	0.57	46
0.50	10	0.60	0.45	0.12	0.73	160	0.50	3	0.59	0.50	0.12	0.56	47
0.45	16	0.55	0.42	0.11	0.70	146	0.45	5	0.55	0.46	0.11	0.53	48
0.40	23	0.51	0.39	0.10	0.67	129	0.40	7	0.51	0.42	0.10	0.51	49
0.35	34	0.47	0.36	0.09	0.63	109	0.35	11	0.46	0.38	0.09	0.49	51
0.30	50	0.42	0.33	0.08	0.58	93	0.30	21	0.39	0.32	0.08	0.43	48
0.27	**62**	**0.39**	**0.31**	**0.08**	**0.55**	**85**	**0.27**	**33**	**0.35**	**0.29**	**0.07**	**0.41**	**46**

Figure 14.27 : Costa Fuego Grade-Tonnage Curves – (Open Pit (Top) and Underground (Bottom))



14.20 Relevant Factors Affecting Resource Estimates

To the best of the QP's knowledge, there are no environmental, permitting, legal, title, tax, socioeconomic, market, political or other relevant factors other than what is presented in this Report that would affect the MRE, other than those typical to the mining industry in Chile.

A detailed list of Project risks is included in Section 25.12 of this Report.

The MRE could be materially affected by future changes in the following factors:

- Changes in metal prices versus those assumed in the estimation process
- Estimated density, particularly in the oxide and transitional weathering domains, following further data collection
- Metallurgical recoveries as a result of additional test work programs or updates in process technologies
- Estimated grades and metal as a result of extensional and infill drilling programs
- Geotechnical modelling as a result of updated information from future drilling campaigns.

14.21 Comments on Section 14

Mineral resources presented herein are reported in accordance the 2014 CIM Definition Standards and were estimated in accordance with the CIM Best Practices Guidelines, as required by NI 43-101. The mineral resources reported in Table 14.48 to Table 14.52, inclusively, are constrained and reported using economic and technical criteria such that the Mineral Resource has reasonable prospects for eventual economic extraction.

The MRE is well-constrained by three-dimensional wireframes representing geologically realistic volumes of mineralisation. Grade estimation has been performed using an interpolation plan designed to minimise bias and over-smoothing and produce representative tonnes and grade of the four deposits within the Costa Fuego Project.

It is the opinion of the Qualified Person responsible for the MREs that the Mineral Resource Estimates for the Productora, Cortadera, Alice and San Antonio deposits were prepared using industry standards and best practices by qualified professionals and may be relied upon for public reporting and for estimating Mineral Reserves contained in this Report.

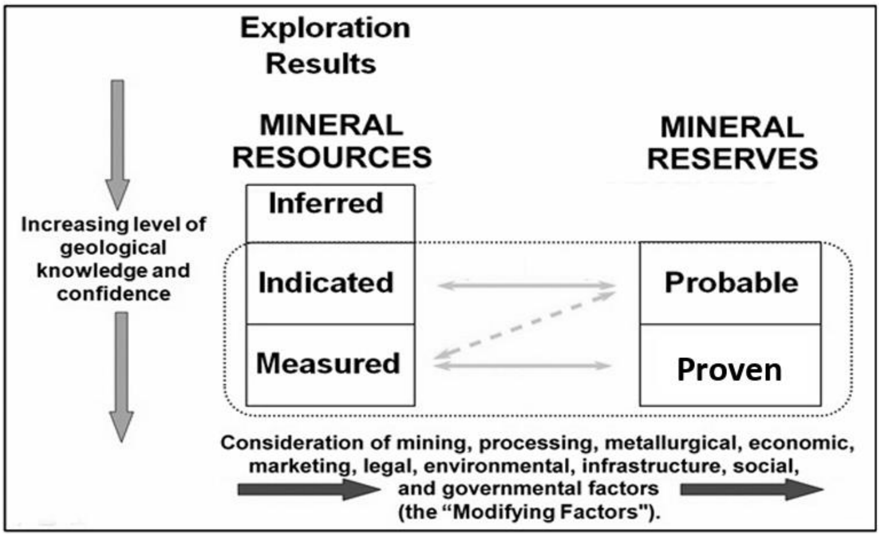
15 Mineral Reserve Estimates

The CIM Definition Standards requires the completion of a PFS as the minimum prerequisite for the conversion of Mineral Resources to Mineral Reserves.

A PFS is a comprehensive study of a range of options for the technical and economic viability of a mineral project that has advanced to a stage where a preferred mining method, in the case of underground mining, or the pit configuration, in the case of an open pit, is established and an effective method of mineral processing is determined. It includes a financial analysis based on reasonable assumptions on mining, processing, metallurgical, economic, marketing, legal, environmental, social and governmental considerations and the evaluation of any other relevant factors which are sufficient for a Qualified Person, acting reasonably, to determine if all or part of the Mineral Resource may be classified as a Mineral Reserve.

Mineral Reserves were estimated and reported using adequate modification of Mineral Resources within economic and practical mine designs (Open pit and Underground mine designs).

Figure 15.1 Relationship between Mineral Resources and Mineral Reserves (CIM) – Figure reference (CIM Definition Standards for Mineral Resources and Mineral Reserves).



A mineral reserve estimate was generated for the open pit and underground mining methods. As such the following sections explain the open pit and underground mineral reserve estimate separately. A combined mineral reserve statement is provided in section 15.4.

Pit optimisation and design inputs and methodologies are described in Section 16.

15.1 Qualified Persons Responsible for the Mineral Reserves Estimates

Mr Anton von Wielligh (FAUSIMM) of ABGM Consulting Pty Ltd is the Qualified Person responsible for the Mineral Reserve Estimates.

15.2 Mineral Reserve Estimation Process

15.2.1 Open Pit Mining

The Costa Fuego Project comprises of three distinct mining areas:

- Productora (Productora pit and Alice pit)
- Cortadera Open Pits (Cuerpo 1, Cuerpo 2 and Cuerpo 3 pits)
- San Antonio pit.

An Industry accepted open pit planning process (for converting Mineral Resources to Mineral Reserves) has been followed which is underpinned by pit optimisation (economic open pit shell development) staged pit designs, staged pit scheduling and economic evaluation. The conversion process is described in the following points, with further detail provided in subsequent sections.

- Open Pit Optimisations were developed for the open pits (each pit/area used the supplied block model comprising of regularised model blocks)
- The block models with regularised model blocks assumed the mine modifying factors of oreloss and planned mining dilution. No further mine modifying factors were applied
- Engineered mining stages were designed for Productora and Cortadera. San Antonio and Alice pits comprise of one pit design/Stage only
- The stage pit designs used the given geotechnical slope design criteria and was again assessed by the geotechnical engineers post the stage design process to ensure the stage/pit designs still complied with the minimum safe design/stability requirements as required by the Competent Person Geotechnical/Rock Engineering
- Surface roads, impoundment designs and then revised/refinements to the stage open pit designs were completed
- The pit stages were imported into MineMax Scheduler and block model blocks were categorised into mineral bins for mine scheduling
- Measured and Indicated model blocks were considered as economic contributing material (ore) during mine scheduling, however considering specific block destinations with those destination costs and recovery criteria applied
- No specific open pit mining cut-off was applied beforehand, but if a mineral block (Measured or Indicated Resource classified block model block) had to be mined within the staged pit design and the block can then be rehandled to the Run of Mine stockpile, crushed, milled and it yields a positive value (after all rehandling, processing, General and Administration costs and subsequent selling costs are applied), the block would be deemed economically viable under a typical marginal cut-off consideration and the block will be

considered for the Mineral Reserves estimation (however, specific to that block's process destination and subsequent destination costs and recovery criteria)

- As these Mineral Resources were studied for more than 10 years, this method of mining is well understood and practiced not only around the world but also in Chile and due to the reasonable confidence, the mine planning detail provided, Measured Mineral Resources was considered for the Proved Mineral Reserves and Indicated Mineral Resources were considered for the Probable Mineral Reserve estimations.

15.2.2 Open Pit Economic Cut-Off

An economic cut-off was not applied on a mineral grade basis nor considering an equivalent mineral grade basis simply as the different Mineral Resource areas have varying relationships of the key economic minerals considered for the Costa Fuego Project and therefore on a Net Smelter Return (NSR) basis or calculation. Copper (Cu), Gold (Au), Molybdenum (Mo) and Silver (Ag) were considered as economic contributing minerals to be recovered and sold as a concentrate (for the transition and fresh mineral blocks/parcels) and Copper Cathode (Cu) to be produced from the Oxide Heap Leach and Dump Leach material/ore.

To better explain the cut-off calculations considered for the Costa Fuego Project, there are two definitions that will be referred to in this section:

Marginal NSR Cut-Off: Applies to a mineralised block or parcel contain a combination of recoverable Copper, Gold, Molybdenum and Silver or at least recoverable Copper (Cu, Au, Mo and Ag or at least recoverable Cu) and where the NSR of this block ore parcel exceed the total processing cost, cost of rehabilitation, royalty and process General and Administrative cost and covers the total selling cost.

Breakeven NSR Cut-Off: Applies to a mineralised block or parcel contain a combination of recoverable Copper, Gold, Molybdenum and Silver or at least recoverable Copper (Cu, Au, Mo and Ag or at least recoverable Cu) and where the NSR of this block ore parcel exceed the total Mining Cost (Operating All-in mining cost inclusive of mining G&A), the total processing cost, cost of rehabilitation, royalty and process General and Administrative cost and covers the total selling cost.

Due to these dynamic conditions and different minerals, a Net Smelter Return (NSR) block calculation was also used during the open pit optimisation process. The following formula explains the NSR calculation for the different process destinations considered:

- Oxide (Heap Leach) Destination: NSR = Price Cu x Recovery curve (Heap Leach) x Payability – Royalty – Cathode Selling Cost
- Sulphide Low Grade (Dump Leach) Destination: NSR = Price Cu x Recovery curve (Dump Leach) x Payability – Royalty – Cathode Selling Cost
- Sulphide Grade (Concentrate) Destination: NSR = (Price Cu x Recovery curve (Concentrator) x Payability + Price Au x Au Recovery curve (Concentrator) x Payability + Price Mo x Mo Recovery (Concentrator) x Payability + Price Ag x Ag Recovery (Concentrator) x Payability) – Royalty – TCRC costs to get the concentrate sold.

Concentrator cut-off is the preferred Sulphide destination, therefore even if the value of a Sulphide block of mineral is more economically viable on a revenue – cost basis at the Dump Leach process destination but it also meets the minimum economic cut-off to be processed in the concentrator circuit, the block will be sent to the concentrator circuit or deferred concentrator stockpile. There might be opportunities to further optimise value with these marginal blocks/parcels reporting to the Dump Leach pads, but this is subject to more definitive dump Leach recovery criteria and testing the time value of the product. The total copper production is also important, and the concentrator do yield higher recoveries (theoretically) compared to the expectations of the Dump Leach copper recovery, therefore if the values are similar at either destination, it would be prudent to improve overall copper recovery/production. Further analyses are planned for these processing options.

The Oxide mineral blocks where NSR > (Processing Cost for Heap Leach + G&A + Rehandling) meets the Marginal Cut-off criteria and will be processed and considered as ore (only for Measured and Indicated category Mineral Resource blocks). Only blocks that must be mined irrespective of its value as a consequence of its position within a defined economic open pit design can be considered as ore even if these blocks are above the marginal cut-off calculation but below the break Even cut off calculation.

The following tables depict the NSR Cut-Off criteria and economic parameters considered:

Table 15.1 NSR Cut-Off Criteria and Economic Parameters

Marginal NSR Cut-Off	Rock Weathering/ Condition	Productora	Alice	Cortadera	San Antonio
		NSR/t	NSR/t	NSR/t	NSR/t
Heap Leach	Oxide	8.88	6.87	9.15	10.91
Dump Leach	Oxide/Transitional/Fresh	3.80	3.80	-	-
Concentrator	Transitional/Fresh	7.88	6.79	8.36	11.65
Breakeven NSR Cut-Off	**Rock Weathering/ Condition**	**Productora**	**Alice**	**Cortadera**	**San Antonio**
		NSR/t	**NSR/t**	**NSR/t**	**NSR/t**
Heap Leach	Oxide	11.20	9.19	11.47	13.24
Dump Leach	Oxide/Transitional/Fresh	6.12	6.12	-	-
Concentrator	Transitional/Fresh	10.21	9.11	10.68	13.97

Table 15.2 NSR Economic Parameters

Element	Price Base	Payability
Cu	USD 4.30/lb	96%
Au	USD 2,280/oz	92%
Mo	USD 20/lb	99%
Ag	USD 28/oz	90%

Table 15.3 : Metallurgical Recovery Fields and Block Model Calculations by Mine Area

Block Model Field	Calculation
Productora	
CU HEAP LEACH OXIDE	CUHLOX = (58.209 * CU_PCT) + (-7.5857 * NA_PCT) + 51.5024 *(Maximum 95%)*
CU DUMP LEACH OXIDE	CUDLOX = 0.51 * (-2.614 * LI_PPM + 95.97) *(Minimum 10%)*

Table 15.3 : Metallurgical Recovery Fields and Block Model Calculations by Mine Area	
Block Model Field	**Calculation**
CU DUMP LEACH TRANS	CUDLTR = 0.51 * (-2.614 * LI_PPM + 95.97) *(Minimum 10%)*
CU DUMP LEACH FRESH	CUDLFR = 0.51 * (-2.614 * LI_PPM + 95.97) *(Minimum 10%)*

CU CONCENTRATOR TRANS	IF(CU_PCT>0.05) CUCONTR = (19.609 *CU_PCT) + 63.443
	IF(CU_PCT<0.05) CUCONTR = 0
	(Maximum 90%)

AU CONCENTRATOR TRANS	IF(AU_PPM>0.02) AUCONTR = (145.4 * AU_PPM) + 38.549
	IF(AU_PPM<0.02) AUCONTR = 0
	(Maximum 80%)

MO CONCENTRATOR TRANS	MOCONTR = 56
AG CONCENTRATOR TRANS	AGCONTR = 40

CU CONCENTRATOR FRESH	IF(CU_PCT>0.05) CUCONTR = (9.072 * CU_PCT + 83.66)
	IF(CU_PCT<0.05) CUCONTR = 0
	(Maximum 95%)

AU CONCENTRATOR FRESH	IF(AU_PPM>0.02) AUCONTR = (145.4 * AU_PPM) + 38.549
	IF(AU_PPM<0.02) AUCONTR = 0
	(Maximum 80%)

MO CONCENTRATOR FRESH	MOCONTR = 0.9 * (5.676 * LOGN(MO_PPM) + 51.191) *(Maximum 95%)*
AG CONCENTRATOR FRESH	AGCONFR = 40
Cortadera	
CU HEAP LEACH OXIDE	Cuerpo 1: CUHLOX = 36.3 Cuerpo 2&3: CUHLOX = (-2.2209 * CO_PPM) + 93.58 *(Maximum 95%, Minimum 10%)*
CU DUMP LEACH OXIDE	Cuerpo 1: CUHLOX = 36.3 Cuerpo 2&3: CUDLOX = 0.65 * (-2.2209 * CO_PPM) + 93.58 *(Minimum 10%)*
CU DUMP LEACH TRANS	Cuerpo 1: CUHLOX = 36.3 Cuerpo 2&3: CUDLTR = 0.65 * (-2.2209 * CO_PPM) + 93.58 *(Minimum 10%)*
CU DUMP LEACH FRESH	Cuerpo 1: CUHLOX = 36.3 Cuerpo 2&3: CUDLFR = 0.65 * (-2.2209 * CO_PPM) + 93.58 *(Minimum 10%)*
CU CONCENTRATOR TRANS	CUCONTR = 17.016*LOGN(CU_PCT)+86.378 *(Maximum 80%, Minimum 8%)*
AU CONCENTRATOR TRANS	AUCONTR = 104.74*AU_PPM+29.42 *(Maximum 90%)*
MO CONCENTRATOR TRANS	MOCONTR = 21.8
AG CONCENTRATOR TRANS	AGCONTR = 27
CU CONCENTRATOR FRESH	CUCONFR = 17.016*LOGN(CU_PCT)+96.378 *(Maximum = 90%, Minimum = 18%)*

Table 15.3 : Metallurgical Recovery Fields and Block Model Calculations by Mine Area	
Block Model Field	**Calculation**
AU CONCENTRATOR FRESH	AUCONFR = 104.74*AU_PPM+29.42 *(Maximum 90%)*
MO CONCENTRATOR FRESH	MOCONFR = 0.9 * (12.563 * LOGN(MO_PPM) + 21.88) *(Maximum 92%)*
AG CONCENTRATOR FRESH	AGCONFR = 27
Block Cave	
CU CONCENTRATOR FRESH	8.615*LOGN(CU_PCT)+96.122 *(Maximum = 95%)*
AU CONCENTRATOR FRESH	AUCONFR = 30.368*AU_PPM+51.637 *(Maximum 90%)*
MO CONCENTRATOR FRESH	MOCONFR = 0.9 * (12.563 * LOGN(MO_PPM) + 21.88) *(Maximum 92%)*
AG CONCENTRATOR FRESH	AGCONFR = 38
Alice	
CU HEAP LEACH OXIDE	46 (Fixed for all oxide blocks)
San Antonio	
CU HEAP LEACH OXIDE	50 (Fixed for all oxide blocks)

The recovery formula considers the actual block or parcel grades and applied upper limits to the calculated metal recoveries.

The following table depicts the typical conversion factors used:

Table 15.4 Typical Conversion Factors	
Conversion Factors Considered	**Factor**
ppm to oz/t	31.10345
Lb to Tonnes	2204

15.2.3 Underground Mining

An Industry accepted underground mine planning process (for converting Mineral Resources to Mineral Reserves) has been followed for the identified underground potential area (Cuerpo 3 orebody at Cortadera). This process considered the best possible exploitation strategy for the area was underground block caving and therefore applied block cave mining shape optimisation (developing the optimal block cave footprint and shape using industry accepted software "footprint finder") followed by a more detailed block cave engineering and planning process using industry accepted software (known as "PCBC" or Geovia's Long Term Planner). The block cave mining shape was supported by a detailed mine access and mine development design completed using Datamine Studio UG software and scheduled to model the necessary details as required for a PFS process. The following points summarize the underground process to estimate Mineral Reserves from the Mineral Resources presented in 3D block model formats:

- Import the block model of Cortadera into Geovia's Footprint Finder software

- Input the optimisation and cost criteria considered for a block cave underground operation at Cortadera's Cuerpo 3
- Develop several optimisation footprints and block cave draw height sensitivities to ultimately recommend the best possible practical block cave footprint, the optimal undercut elevation and the optimal block cave heights
- Import the block model into Geovia's Long Term Planner ("PCBC") software
- Import the geotechnical surface that impacts the block cave mining shapes
- Import the pit design develop for Cuerpo 3 that will be influenced by and influence the underground block cave operation
- Run several PCBC schedules and simulate the best/most practical final block cave shape and design entity to be used to then design access and key mine development to access and extract
- Finalise the rock handling requirements and designs and stitch the block cave, access development and ventilation development designs together for a complete underground mine scheduling process.

The following report section details the specific process of extraction/methodology in more detail.

The underground block cave shape and economic cut-off was developed within the optimisation process within the Geovia software (Footprint Finder and then PCBC).

15.2.4 Underground Economic Cut-Off

The nominated underground mining method (best suited for Cortadera's Cuerpo 3 deposit) is underground block cave mining. As the block cave shape was developed in a 3D optimisation software suite (Footprint Finder and PCBC) the economic cut-off used are also dynamic but should meet the full Break-Even mining cut-off to be considered as ore.

Block caving is a non-selective bulk mining method and relies on caving of the insitu rock mass as the primary method of breaking the rock. To meet the requirements of a successful cave it is sometimes required to add sub-economic mining locations as part of the optimized footprint to meet required mining geometries, and/or force draw columns to mine to a minimum height of draw to ensure a successful operating cave. These sub-economic areas are included as marginal, or planned dilution as part of the reserve estimation process, and forms part of the concentrator feed material due to the non-selective mining nature of a block cave.

There are also two Cut-Off calculations for the underground Block Cave operation. The Marginal Cut-off calculation involves at least the Ropecon cost to get this material to the Productora plant and then the full concentrator processing and G&A cost. This will only apply if lower grade mineralized blocks (below the Break-Even cut-off) and dilution blocks are drawn and loaded as part of the typical underground Block Cave production activities. None of the underground block cave material was considered for Dump Leach processing. The Break-Even Cut-Off calculation involves the full underground mining Operating Costs (including underground crushing and conveyance to surface costs), the Ropecon cost to Productora, and then all associated processing and G&A costs at the Productora plant.

The following tables depict the NSR Cut-Off calculations for the Block Cave mine plan (The Cut-Off value was not entered yet the NSR value per block and then all the relevant costs were added into the Block Cave Mine Planning software).

Table 15.5 Block Cave NSR Cut-Off	
Cortadera Concentrator (Underground Block Cave - Ore)	USD/t
Underground Mining Cost (Cortadera Sulphides) - Excluding UG Crushing & Conveying Cost	6.85
Cortadera UG Crushing Cost	0.68
Cortadera UG Conveyor Cost	0.42
RopeCon Cost (Transport from Cortadera to Productora)	0.31
Productora Sulphide Concentrator Processing Cost (- UG Cortadera Primary Crusher)	6.37
Productora Sulphide Concentrator Deferred Rehandling Cost	0.97
NSR Cut-Off (Marginal - mineralised blocks and dilution blocks that is mined as part of maintaining the proper Block Cave draw/production)	6.68
NSR Cut-Off (Break-Even)	14.63
NSR Cut-Off (Break-Even) if Ore is deferred on Stockpile	15.60

15.3 Mineral Reserve Estimation

All Mineral Reserves are shown on a 100% ownership basis.

Table 15.6 Summary of Mineral Reserves by Destination as of 27 March 2025

		Grade				Contained Metal			
	Tonnes (Mt)	Cu (%)	Au (g/t)	Ag (g/t)	Mo (ppm)	Cu (kt)	Au (koz)	Ag (koz)	Mo (kt)
Open Pit									
Concentrator									
Proved	-	-	-	-	-	-	-	-	-
Probable	293	0.36	0.08	0.37	113	1,043	728	3,517	33
Total	293	0.36	0.08	0.37	113	1,043	728	3,517	33
Oxide Leach									
Proved	-	-	-	-	-	-	-	-	-
Probable	41	0.35	0.07	0.43	35	142	96	563	1
Total	41	0.35	0.07	0.43	35	142	96	563	1
Sulphide Leach									
Proved	-	-	-	-	-	-	-	-	-
Probable	22	0.13	0.03	0.23	41	29	20	168	1
Total	22	0.13	0.03	0.23	41	29	20	168	1
Combined									
Proved	-	-	-	-	-	-	-	-	-
Probable	356	0.34	0.07	0.37	98	1,213	844	4,248	35
Total	356	0.34	0.07	0.37	98	1,213	844	4,248	35
Underground									
Concentrator									
Proved	-	-	-	-	-	-	-	-	-
Probable	146	0.44	0.16	0.79	93	645	734	3,704	14
Total	146	0.44	0.16	0.79	93	645	734	3,704	14
Combined (Open Pit and Underground)									
Proved	-	-	-	-	-	-	-	-	-
Probable	502	0.37	0.10	0.49	97	1,858	1,578	7,951	49
Total	502	0.37	0.10	0.49	97	1,858	1,578	7,951	49

Table 15.7 Summary of Mineral Reserves by Deposit as of 27 March 2025									
		Grade				Contained Metal			
	Tonnes	Cu	Au	Ag	Mo	Cu	Au	Ag	Mo
	(Mt)	(%)	(g/t)	(g/t)	(ppm)	(kt)	(koz)	(koz)	(kt)
Open Pit									
Productora									
Proved	-	-	-	-	-	-	-	-	-
Probable	260	0.35	0.07	0.34	125	917	593	2,801	33
Total	260	0.35	0.07	0.34	125	917	593	2,801	33
Alice									
Proved	-	-	-	-	-	-	-	-	-
Probable	14	0.30	0.03	0.18	37	42	15	82	1
Total	14	0.30	0.03	0.18	37	42	15	82	1
Cortadera									
Proved	-	-	-	-	-	-	-	-	-
Probable	79	0.29	0.09	0.48	27	224	235	1,208	2
Total	79	0.29	0.09	0.48	27	224	235	1,208	2
San Antonio									
Proved	-	-	-	-	-	-	-	-	-
Probable	4	0.82	0.01	1.34	3	30	1	158	0
Total	4	0.82	0.01	1.34	3	30	1	158	0
Underground Block Cave									
Cortadera									
Proved	-	-	-	-	-	-	-	-	-
Probable	146	0.44	0.16	0.79	93	645	734	3,704	14
Total	146	0.44	0.16	0.79	93	645	734	3,704	14
Combined (Open Pit and Underground)									
Proved	-	-	-	-	-	-	-	-	-
Probable	502	0.37	0.10	0.49	97	1,858	1,578	7,951	49
Total	502	0.37	0.10	0.49	97	1,858	1,578	7,951	49

[1]Mineral Reserves are reported on a 100% Basis - combining Mineral Reserve estimates for the Cortadera, Productora, Alice and San Antonio deposits, and have an effective date of 27 March 2025.

[2]An Ore Reserve (declared in accordance with JORC Code 2012) was previously reported at Productora, a component of the Costa Fuego Project, on 2nd March 2016 on the ASX. The Company was not subject to the requirements of NI 43-101 at that time.

[3]Mineral Reserve estimation practices are in accordance with CIM Estimation of Mineral Resource and Mineral Reserve Best Practice Guidelines (29 November 2019) and reported in accordance CIM Definition Standards for Mineral Resources and Mineral Reserves (10 May 2014) that are incorporated by reference into NI 43-101.Mineral Reserve estimates are in

accordance with the JORC Code. References to "Mineral Reserves" mean "Ore Reserves" as defined in the JORC Code and references to "Proven Mineral Reserves" mean "Proved Ore Reserves" as defined in the JORC Code.

[4]The Mineral Reserve reported above was not additive to the Mineral Resource. The Mineral Reserve is based on the 26 February 2024 Mineral Resource.

[5]Tonnages and grades are rounded to two significant figures. All figures are rounded, reported to appropriate significant figures and reported in accordance with the Joint Ore Reserves Committee Code (2012) and NI 43-101. As each number is rounded individually, the table may show apparent inconsistencies between the sum of rounded components and the corresponding rounded total.

[6]Mineral Reserves are reported using long-term metal prices of US$4.30/lb Cu, US$2,280/oz Au, US$27/oz Ag, US$20/lb Mo.

[7]The Mineral Reserve tonnages and grades are estimated and reported as delivered to plant (the point where material is delivered to the processing facility) and is therefore inclusive of ore loss and dilution.

[8]The Productora deposit is 100% owned by Chilean incorporated company Sociedad Minera El Aguila SpA (SMEA). SMEA is a joint venture (JV) company – 80% owned by Sociedad Minera El Corazón SpA (a 100% subsidiary of Hot Chili Limited), and 20% owned by Compañía Minera del Pacífico S.A (CMP).

[9]The Cortadera deposit is controlled by a Chilean incorporated company Sociedad Minera La Frontera SpA (Frontera). Frontera is a subsidiary company – 100% owned by Sociedad Minera El Corazón SpA, which is a 100% subsidiary of Hot Chili Limited.

[10]The San Antonio deposit is controlled through Frontera (100% owned by Sociedad Minera El Corazón SpA, which is a 100% subsidiary of Hot Chili Limited) and Frontera is party to an option agreement pursuant to which it can earn a 100% interest in the property.

[11]The Mineral Reserve Estimate as of 27 March 2025 for the Costa Fuego Project was prepared by Anton von Wielligh, Fellow with the AUSIMM (FAUSIMM). Mr. von Wielligh fulfils the requirements to be a "Qualified Person" within the meaning of NI 43-101 and is the Competent Person under JORC for the Mineral Reserve.

[12]Hot Chili Limited is not aware of political, environmental, or other risks that could materially affect the potential development of the Mineral Reserves other than those that will be disclosed in a technical report for the PFS. A detailed list of Costa Fuego Project risks is also included in Section 25.12 of this Report.

16 Mining Method

The following section outlines the parameters and procedures used for the designs of the Cortadera, Productora, Alice and San Antonio mines as conventional open pits and a block cave mine, estimates of the Mineral Reserves within the open pit and block cave mine plan, and establishes a practical mining schedule for this Report. The mine plan is based on the Probable Mineral Reserves presented in Section 15 of this Report.

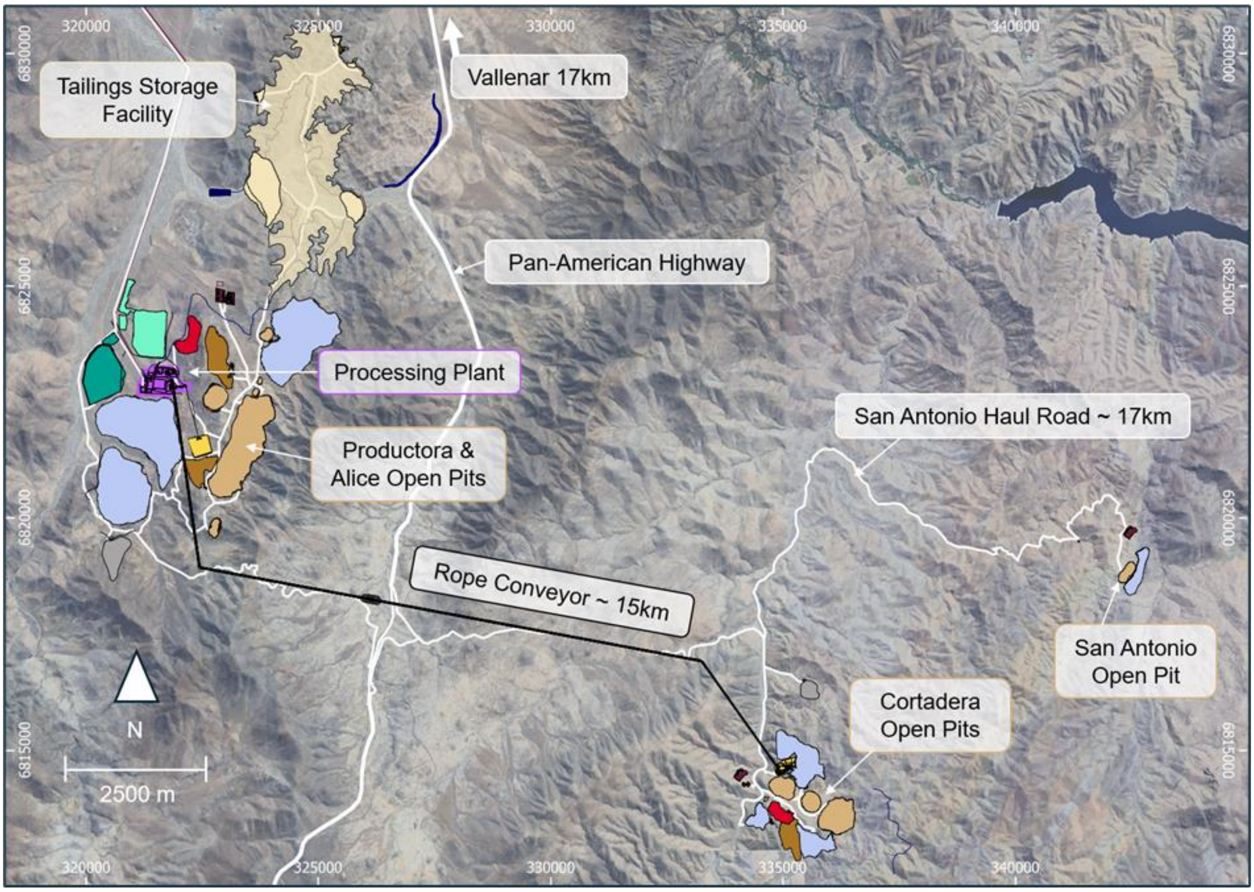
16.1 Introduction

Open pit mining methods have been selected as the key exploitation technique for the Costa Fuego Project deposits. These methods are supported by near surface mineralisation which allows for low waste to processing feed strip ratios and associated lower mining costs. There is also underground block cave exploitation potential at Cortadera's Cuerpo 3 deposit.

The Costa Fuego Project consists of the following mining areas (Figure 16.1):

- Productora, including the Productora main pit and several smaller satellite pits
- Alice
- Cortadera (broken into three separate mining areas): Cuerpo 1, Cuerpo 2, and Cuerpo 3
- San Antonio.

Figure 16.1 : Costa Fuego Project Mining Areas (HCH, 2025)



16.2 Mine Operations and Equipment

16.2.1 Mining Strategy

Mines are scheduled to work seven days per week, 365 days per year. Each day will consist of two 12-hour shifts with four mining crews required to cover the operation. The mines are envisaged to be operated by a mining contractor.

Mining operations on the mine site will be managed by the Mining Manager. Operational personnel will be employed by the nominated mining contractor with HCH personnel supervising the contractor and managing production, safety and technical activities.

Mining contractor personnel will be responsible for:

- Drilling and blasting
- Loading and hauling
- Ancillary equipment
- Process feed-rehandling

- Equipment maintenance
- Road construction and road maintenance
- Waste dump construction
- Short-term planning
- Procurement of mining supplies
- General administration of mining activities.

HCH's mining personnel will be responsible for:

- Defining standards and guidance for mining activities
- Managing mining production and safety
- Managing Mineral Resource Estimates
- Managing process feed control activities
- Managing geotechnical activities
- Running long- and medium-term mine planning.
- Managing mining costs through administration of mining contractor agreement.

16.2.2 Mine Personnel

HCH will allocate dedicated staff for managing critical mining activities. Staff will cover areas such as mine management, production supervising, safety supervising, mine planning, mine geology support, geological modelling and Mineral Resource estimation, mine and waste dump surveying, process feed sampling, geotechnical support, production tracking, and mine contract supervision.

Contractor staff will include a site manager, supervision personnel for production, safety and maintenance, operators and maintenance personnel, short term mine planning team, and personnel for associated supporting services like accounting, human resources, purchasing, catering, personnel transport, etc.

16.2.3 Mine Equipment Selection

Mine equipment was selected to perform the following duties:

- Construct roads to the initial mining areas and to the crusher, WRDS and processing feed SPs, construct additional roads as needed to support mining activity
- Complete pre-production development required to expose process feed for initial production
- Mining and transport of processing feed to the primary crushers
- Mining and transport of waste from the pit or underground to the WRD areas
- Maintenance of all the mine work areas, in-pit haul roads and external haul roads; and maintenance of the WRD areas

- Rehandling process feed and marginal process feed (load, transport and auxiliary equipment) from the SPs to feed the primary crusher.

While HCH is not advocating for specific mining equipment models, it is expected that the contractor will have suitable production equipment: typically, a ~26 m^3 bucket excavator (for use in waste zones) and a 16 m^3 hydraulic excavator (for handling processing feed). Both would be expected to load 190 t trucks, with the smaller excavators also loading 90 t haul trucks in the deeper areas of the open pits where smaller access ramps would be introduced.

Underground mine equipment is assumed to be a rubber tyred, diesel-powered fleet. Underground development equipment would typically include major front-line equipment such as face drills (Twin Boom Jumbo), bolting drills, cable bolters, loaders (load haul dump units, or LHD) (~17 t to 21 t capacity), trucks (~60 t capacity), ground support equipment (shotcrete sprayers, agitator trucks), charge-up rigs, secondary breaking rigs, integrated tool carriers, boxhole rigs (for drawbell development), and other auxiliary support equipment. The underground production fleet would typically include loaders (material movement from drawpoint to crusher), secondary breakage equipment (drills and charge-up), integrated tool carriers and water cannons.

The best suited loader for the underground operations is either the 17 t LHD or 21 t LHD, with the 21 t LTD likely the preferred option. The reasonably long LHD hauling distances due to the large block cave/draw point footprint requires larger LHDs to reduce operational and production/productivity risks.
The main ore handling system with the underground mine is LHD tipping into two underground jaw-gyratory crushers (each of a name plate capacity of ~13mtpa) and feeder conveyor belts discharging via a discharge chute onto the main conveyor belt.

16.2.4 Drilling and Blasting

As per the known vertical variability of the process feed, the open pit bench height for drilling and blasting was considered as 15 m but loading in two 7.5 m slices. Alternatively, for waste, as per the high strip ratio of the pit, there will be large areas of known waste material, so 15 m benches were selected for waste drilling, blasting, and loading. The loading activities were considered in these different bench heights for process feed and waste, to obtain the required selectivity and high productivity.

16.2.5 Loading and Hauling

The Main pit at Productora and Cuerpo 2 and 3 open pits at Cortadera assume that the mining operation could use 34 m^3 hydraulic excavators for waste and 22 m^3 hydraulic excavators for processing feed, loading 220 t trucks. This type of equipment can achieve the required productivity for an annual total material movement of approximately 92 Mt and will provide sufficient mining selectivity as required for control of processing feed grade. This main fleet will be complemented with a 15 m^3 front-end loader to add flexibility to the loading operation and for the rehandling of process feed to the mill.

The Alice pit, Satellite pits at Productora, Cuerpo 1 pit at Cortadera, and the San Antonio pit assume the use of 15 m^3 front-end loaders or a combination of hydraulic excavators with similar bucket capacities and 90 t trucks. Underground development load and haul will be via large scale rubber tyred diesel equipment including Load Haul Dump (LHD) units and trucks. Underground production will incorporate LHD units to move material from cave drawpoints to the crusher. This will then feed onto a conveyor system for removal from the underground mine.

16.2.6 Open Pit Grade Control

Blast holes will be sampled and tested for (at a minimum) copper, gold, molybdenum, and silver to define processing feed grades.

HCH personnel will conduct the sample collection and deliver grade control samples to an external laboratory in Vallenar. The grade control process and cost calculations will be based on 100% of sulphide samples being analysed for total copper and 10% of samples being analysed for gold and molybdenum. For the oxide samples, 100% will be analysed for soluble copper. The unit rates for sample preparation and laboratory analysis were obtained from a company running several laboratories in Chile and providing similar services for exploration activities in the Vallenar area.

16.2.7 Underground Grade Control

Underground block cave grade control is primarily done through infill diamond drill prior to commencement of caving. A dedicated cave preconditioning / hydro fracturing level is planned above the cave from where the entire cave footprint can be grade control drilled to a suitable resolution.

Additionally mapping and sampling of the undercut levels and production level provides short term information about the initial cave ramp-up material. During the later parts of the cave's life drawpoint sampling is typically used to calibrate the material flow settings within PCBC to more accurately determine drawpoint shut-off. It is important to keep record of material that can be identified in the drawpoint, through repeatable methods such as visual inspection (such as rock type) or chemically assaying (mineral grade, grade ratios, etc).

It is critical that accurate tonnage mined from each drawpoint is kept from the start of mining and maintained over the life of the cave. Additionally cave back propagation and tracking of the cave muck pile is used to confirm the movement of broken material within the cave from its original insitu location to where it is extracted from the drawpoints.

The grade control and material flow data is used to build up trends for each drawpoint and is used to calibrate the PCBC flow parameters both locally for individual drawpoints and globally for the cave.

16.2.8 Auxiliary Equipment

Auxiliary equipment will include track dozers, wheel dozers, motor graders and water trucks.
The mine fleet will also include the necessary equipment to rehandle the processing feed from the stockpiles to the primary crushers.

The low-grade sulphide leach and heap leach processing feed rehandling could be carried out using a 15 m^3 front-end loader and the same 90 t trucks used in the Alice and Productora satellite pits.
Equipment utilisation will be analysed in more detail during the next study phase.

16.3 Open Pit Mining

16.3.1 Methodology

An open pit optimisation exercise was carried out to define suitable open pit limits for each deposit, with the aim of maximising the economic value of the Project.

The open pit optimisation process defined technical and economic parameters for assigning a value to each block contained within the respective Mineral Resource block models. This information was combined with geotechnical information and assessed by application of the Lerchs-Grossmann algorithm within the Datamine Studio NPVS software program (NPVS). The software generates ultimate and optimal final pit shell surfaces that help identify the economic limit of mining, when considering reasonable mining and economic parameters.

As outlined in Figure 16.16.2, the process methodology behind NPVS commences with a geological block model, which is then converted to an economic model once various inputs (including economic, mining, geotechnical, processing and metallurgical properties) are applied. Pit shells are then generated with the aim of maximising net present value (NPV) and helping to create an optimised mining sequence and schedule.

Figure 16.16.2 : NPVS Process Flow

16.3.2 Dilution

The Mineral Resource models provided adopt selective mining units (SMUs) as part of the Resource modelling process.

Open pit block models were regularised to a parent block size of 5 m X, 10 m Y and 5 m Z (5-10-5/X-Y-Z). This is deemed to be an adequate modification because:

- The block volume is 250 bcm or approximately 600 to 650 t
- The block size is therefore close to three full large haul trucks and allows for bulk loading equipment
- This block size also allows for the blast muck movement which dilutes process feed along the contact.

The regularisation process from the sub-celled Mineral Resource block model added approximately 5% to 8% dilution (reduction in average grade). All mining losses and planned dilution is assumed to be captured in the regularisation blocks. No additional losses or dilution were applied outside of the regularised models.
All open pits use a minimum of 15 m high benches. For the larger open pits (Cortadera and Productora Main) double benching (30 m) is considered for planning purposes in the upper waste rock benches. These pits will utilise larger diesel-driven bulk loading equipment, before reverting to the smaller mining equipment for the lower ore benches.

16.3.3 Hydrogeological Field Campaign, Pore Pressure and Pit Dewatering Analysis

16.3.3.1 Hydrogeological Field Campaign

Hydrological and hydrogeological studies of the Costa Fuego Project have been undertaken by Artois Consulting Ltd.

Hydrogeological field campaigns carried out between 2012 and 2015 consisted of the following activities:

- Geophysical resistivity profiling by Geodatos (2012)
- Measurements of groundwater levels and flows during the exploration drilling by Blue Spec Mining and Artois. Anecdotal information about dewatering the existing underground mine was also obtained (2012-2013)
- Drilling and permeability testing of groundwater monitoring wells across the mining concession by Blue Spec Mining and Artois (2013)
- Permeability testing of geotechnical holes inside the proposed pit perimeter by Blue Spec Mining and Artois (2014)
- Geological and geotechnical mapping by Ingeroc (2014-2015).

16.3.3.2 Productora Groundwater and Pore Pressure Analysis

Based on the geological setting, limited groundwater flow in the Project area is expected along the weathered units near surface, and as fracture flow within the igneous rock formations, inducing a northwest regional flow direction. At greater depth, flow is confined to the regional fault zones and the discontinuities associated with the intensely fractured mineralised zones.
A conceptual model was generated as a representation of the groundwater flow regime within the Productora area. This model is described as follows:

- Groundwater levels occur at a depth between 75 m and 100 m in the upper part of the catchments (e.g. pit areas) and 20 m to 45 m in the bottom of the valleys.
- Groundwater flow in the catchment is generated by the infiltration of sporadic rainfall. At a 1% to 2% recharge rate, it generates a total flow volume of approximately 3 L/s to 6 L/s across the catchment.
- The groundwater flows in a northwest direction along the weathered igneous terrain towards the valley infill deposits. Due to the subsurface topography of the weathered bedrock units, the saturated zone does not form a continuous aquifer flow unit. Instead, the groundwater table is likely to form only a thin “veneer” along the soil or weathered bedrock contact and the underlying basement.
- Therefore, regardless of the intermediate permeability values of the soils and the weathered bedrock ($1x10^{-7}$ to $5x10^{-6}$ m/s), the transmissivity values across the site remain low. The values typically range between 1 and 25 m^2/day.
- At depth, groundwater flow is associated with the structures that bound the mineralised zone and regional discontinuities. Although the dykes themselves are considered low permeability, the fracturing in the intruded host rock and breccia allows for the circulation of groundwater. This water may flow under pressure and, when intercepted, rises upwards into the bore hole and the mine workings. Typically, transmissivity values are in the order of 1 - 20 m^2/day, although peaks of up to 56 m^2/day could be associated with the mineralised zone.

The pore pressure and pit wall drainage requirements were analysed using the 2D, finite element software SEEP/W (Geostudios, 2007) to simulate the saturated and unsaturated flow conditions around the proposed open pits. The study was completed assuming a final pit depth in the order of 500 metres above mean sea level (mamsl). Both the Alice and Productora pits will advance between 200 m and 250 m below the original groundwater level, therefore, the saturation level of the pit walls was incorporated into the geotechnical stability analysis.
The numerical analysis of the drainage behaviour provides the following input to the geotechnical analysis:

- The open pit wall will remain dry, or drain naturally, along the first 100 m below the ground surface. The pore pressures will remain very low or may be negative due to evaporative "pumping".
- At a depth of 100 m to 200 m below the ground surface, a drainage system will be required to lower the groundwater level and reduce the pore pressure in the pit slopes. The optimum distance to push-back the groundwater behind the pit wall is in the order of 50 m to 100 m. This can be achieved by in-pit dewatering wells and sub-horizontal drains. Under ideal conditions, the Productora pit drainage system would intercept between 2 L/s and 15 L/s of groundwater inflow. At Alice, the dewatering system will likely intercept between 1 L/s and 2 L/s.
- At depths beyond 200 m below the original ground surface, the hydraulic conductivity reduces which, in turn, limits the drainage potential of the pit slopes. Particularly in the central zone of the open pit between coordinates 6,821,500 mN and 6,822,000 mN, saturated conditions along the pit wall are likely. To avoid the accumulation of pore pressures, additional sub-horizontal drains will be required in this part of the pit. They need to be drilled to optimise the interception of the north-south and northwest-southeast faults.

16.3.3.3 Productora Pit Dewatering

The arid climate of Atacama Region, with high evaporation rates and small catchments, reduces the potential for any surface water and groundwater flow across the Project area. The monitoring results indicate a deep groundwater level (20 m to 100 m below surface) within low transmissivity sediments and igneous rock formations.

Regardless of the low permeability flow regime at the Project, pit dewatering wells and sub-horizontal drains will be required to prevent flooding and maintain stable slope angles below a depth of 100 m. The pumping system will likely intercept between 1 and 2 L/s at the Alice pit and between 2 and 15 L/s at the Productora pit. Near the Productora and Alice pit areas, the groundwater level (piezometric level) is 700 to 750 mamsl (mRL).

Based on a final pit floor, the pits will be excavated to approximately 200 to 350 m below the piezometric level. Seepage models indicate that groundwater drainage is required to maintain stable pit slopes and prevent the flooding of the excavation. Such a drainage system would consist of:

- Deep in-pit dewatering wells:
  - Three to five pit dewatering wells drilled to a depth of 250 m, each connected to a high-density polyethylene (HDPE) pipe circuit to lift the water to a central collection tank or sump located outside the pit.
- Sub-horizontal pit wall drainage holes:
  - A spread of sub-horizontal drainage holes to depths of 200 m
  - Collection and gravity flow of the water to the sump in the bottom of the pit.

16.3.3.4 Cortadera Pit Dewatering

The analysis of surface water and groundwater conditions at the Cortadera site was based on a combination of regional data of the Rio Huasco Valley and site-specific information collected over the course of a two-year field program (2021-2022).

The conclusions are summarised as follows:

- The arid climate (40 mm/year rainfall), the high evaporation rates (1 700 mm/year) and the small catchments reduce the potential for any surface water and groundwater flow across the Project area. The recharge rate is estimated to vary between 0.5 and 2% of rainfall. Geology exploration wells have so far confirmed a relatively deep groundwater level (20 to 350 m below surface).
- Limited groundwater flow occurs along the contact between the weathered bedrock and the transition zone underneath. This saturated unit has a transmissivity of 1 to 10 m^2/day. The flow converges towards the centre of Aguada Cortadera and then migrates west-ward along a steep hydraulic gradient of 10%. Contact metamorphism has generated a low permeability aureole around each of the three ore bodies which locally deflect and obstruct the flow pattern. Within the underlying fresh bedrock, flow is constrained to the regional fault zones and the structural corridors surrounding the ore bodies.
- Regardless of this low permeability flow regime, groundwater is likely to intersect the open pits below a depth of approximately 75 m below surface (or 900 mamsl). It will maintain partially saturated slopes with pore pressures ranging from 0.25 to 0.74 MPa. The slope stability can be optimised by installing sub-horizontal drains.
- Underground, undrained pressures may increase to 5.25 MPa. Groundwater seepage rates are not likely to exceed 94 m^3/day (1 L/s) in the pit and 60 m^3/day (0.7 L/s) in the underground mine.
- Due to the arid conditions and the small contributing catchments, the storm water volumes associated with a 1 in 50-year event remain small (8,000 – 22,000 m^3). A pit dewatering system consisting of a conventional pit sump and a barge mounted pump (77-311 kW) adequately manages this volume of water.
- To reduce the generation of contact water on site, an open diversion channel (0.2 m depth x 0.2 m width) will be designed to intercept natural run-off upstream and discharge it into Quebrada Las Cañas. The generation of contact water is thereby limited to direct rainfall across the waste rock materials. This will be collected in an open, lined channel constructed along the toe of each WRD (0.2 m depth x 0.2 m width). The contact water will flow into the Cuerpo 1 Pit (for WRD-South) or a lined collection pond (for WRD-North). It will be recycled for dust suppression and evaporation, forming a closed-circuit mine water management system.

Based on the data available at the time of assessment, the work confirms that conventional water management strategies and passive mine dewatering techniques can be implemented to safely operate the Cortadera open pit mine.

Additional hydrogeological and hydro chemical testing and modelling will be required to improve the designs during the next stage of work.

Surface groundwater work carried out for the open pits will also be considered for the underground. No major water bearing structures are known at this stage, but data is limited and should be further investigated in the next study stage. Hydrology test work and water balance models should be conducted and consider the combined impact of the open pit and potential underground inflows.

16.3.4 Geotechnical Analysis – Open Pit

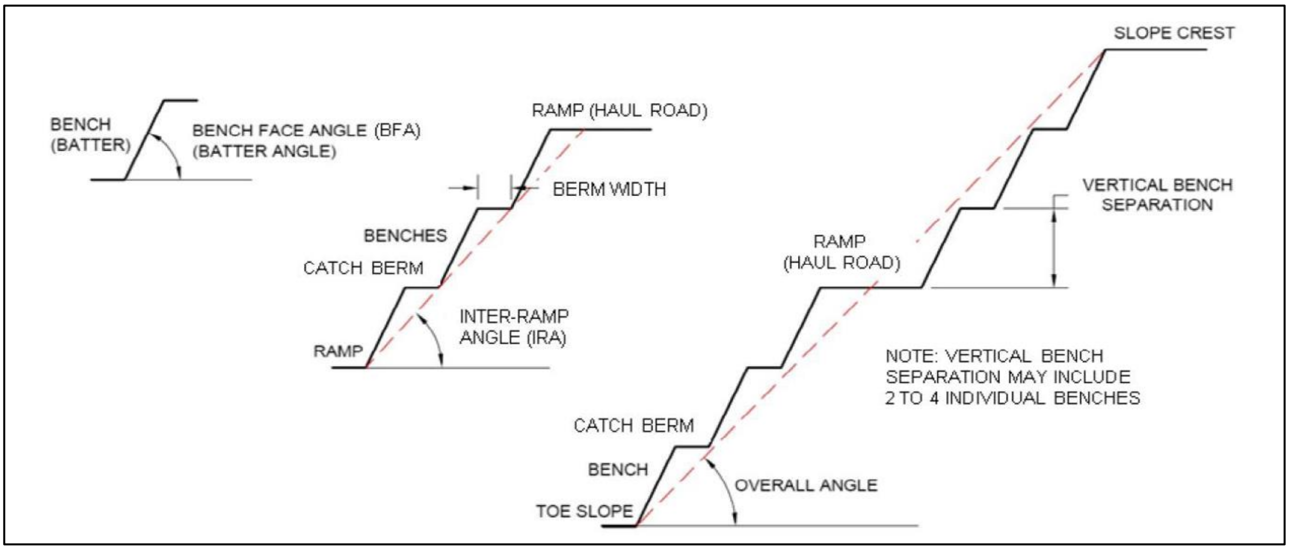
The open-pit geotechnical design was conducted by GMT (2024), based on previous geotechnical investigations reported in the Ingeroc studies for Productora and Cortadera. These studies included field investigations carried out from 2014 to 2015 for Productora and from 2021 to 2022 for Cortadera. The investigations involved drilling cored geotechnical holes (47 for Productora/Alice, 103 for Cortadera and 3 for San Antonio) with core orientation, collecting rock samples for laboratory testing, and performing surface geotechnical mapping.

Surface mapping was previously conducted by Ingeroc for both Productora and Cortadera, with additional geotechnical surface mapping undertaken in 2015 for the Alice open pit. Surface mapping for the San Antonio open pit was conducted by GMT in July 2024.

The geotechnical analysis for the open pit addresses stability at multiple scales. At the bench scale, designs ensure the containment of inter-bench failures and mitigate rockfall risks. These configurations form the basis for assessing inter-ramp and overall pit wall stability.

Pit design parameters are informed by updated geotechnical and structural models, with the application of slope design criteria alongside haul road width considerations determining the overall pit wall angle. The typical application of slope design criteria to batter, inter-ramp, and overall slope angles is illustrated in Figure 15.1.

Figure 16.3 Typical application of pit wall slope design criteria.



Finally, geotechnical design zones are established based on structural and geotechnical domains, incorporating the dip and strike direction of the pit walls. These design zones summarize the bench-scale parameters, inter-ramp heights, and angles for each geotechnical and structural domain, while accounting for the dip and strike orientation of the pit walls.

16.3.4.1 Geotechnical Slope Criteria

GMT has applied conservative assumptions to compensate for areas where information is insufficient or limited. These conservative measures, particularly in cases where data gaps exist, help to reduce uncertainty and ensure a more robust and reliable analysis. This judgement is supported by the previous experience dealing with similar situations, the numbers applied can be improved in the next study phase.

A conservative set of values was adopted for both the strength of geological structures and the horizontal seismic coefficient. Conservative values for cohesion and friction angle were used due to the unusually high results obtained from previous direct shear tests (INGEROC, 2016), which would have led to unrealistic outcomes across all slope scales of analysis. Similarly, for the horizontal seismic coefficient, a high value of k_h = 0.2, based on previous reports (INGEROC, 2020), was applied. Although this value is unusually high, it was retained to maintain consistency across project studies, as no new information has been available.

The geotechnical analysis covered assessments at the bench, inter-ramp, and overall slope levels. Stability at the bench-berm scale was evaluated using the Keyblocks method in S-Block software (developed by GS Esterhuizen), while inter-ramp and overall slope stability were analysed using the Limit Equilibrium Method (LEM) in Slide2 software by Rocscience.

The analysis results indicate the following:

- The bench-berm configuration shows that the recommended bench widths could contain potential local instabilities according to the adopted acceptability criteria. The details for each pit are the following.
  - For the Productora open pit, pit walls with a dip direction of 282° in the Southern Volcanics structural domain, a 70° bench face angle is required, rather than the 75° specified for the other structural domains under that pit orientation. However, this analysis can be improved in the next study phase, therefore, the recommendation is to not change the pit design parameters at this stage.
  - For the Alice open pit, the design parameters selected for all pit wall orientations and structural domains meet the acceptability criteria.
  - For Cortadera, in Cuerpo 1 and Cuerpo 2 open pits, pit walls with dip directions of 268º in the skarn structural domain under oxide conditions do not meet acceptability criteria. The vertical extension of this domain is between 20 m and 59 m. However, this analysis can be improved in the next study phase, therefore, the recommendation is to not change the pit design parameters at this stage.
  - For the Cuerpo 3 open pit of Cortadera, pit walls with a dip direction of 147° in the skarn structural domain under oxide conditions do not meet the acceptability criteria. The vertical extension of this domain is approximately 11 m. Like the situation at the Cuerpo 1 and Cuerpo 2 open pits, this analysis can be improved in the next study phase, therefore, the recommendation is to not change the pit design parameters at this stage
  - For San Antonio, the design parameters selected for all pit wall orientations meet the acceptability criteria.

In terms of the inter-ramp and overall scale analysis:

- For the Productora pit, all sections meet the acceptability criteria under static and pseudo-static conditions, except for Sections S06 and S18; the latter only fails to meet criteria under pseudo-static conditions. However, uncertainties in structural domain boundaries and the conservative shear strength properties of the joint sets controlling stability may underestimate slope instability.

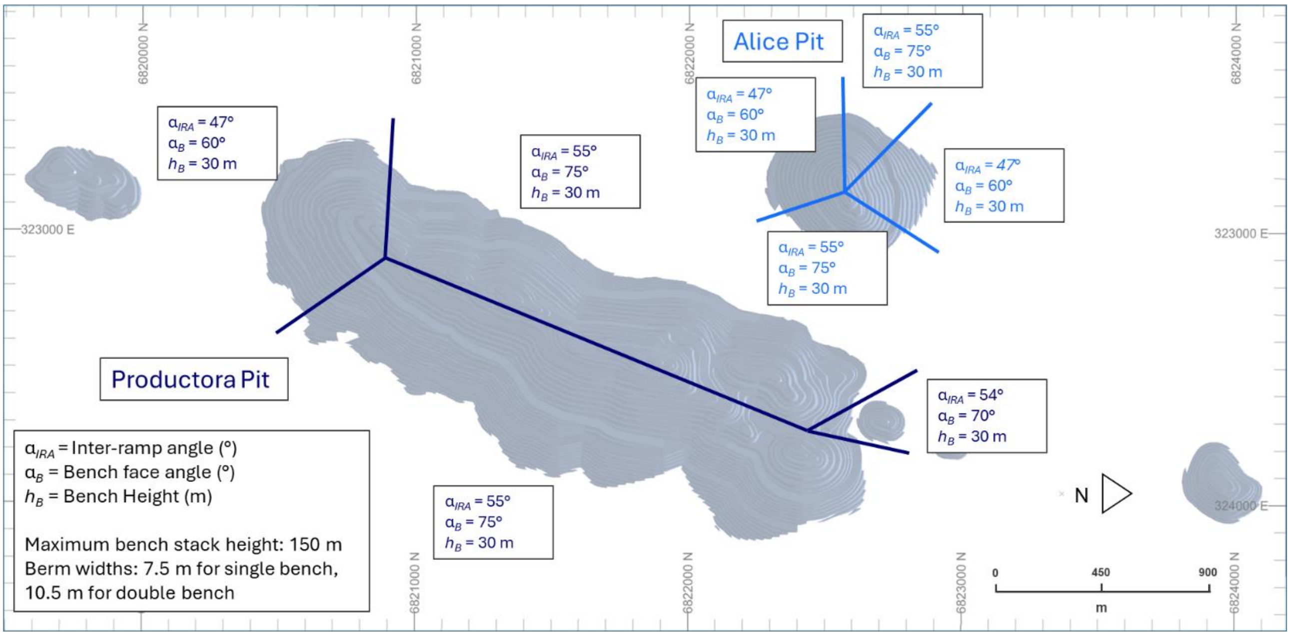
- For the Alice pit, all sections meet the acceptance criteria under both static and pseudo-static conditions.
- For the Cortadera pits, all sections meet the acceptance criteria under static conditions, except for the upper inter-ramp slopes in Sections S01 and S02 of Cuerpo 2 under pseudo-static conditions. The failure surfaces are controlled not by rock mass strength but by a fault structure labelled "076_IS_COR24_V1_0_EXTENDED", defined in the updated structural model.
- For the San Antonio pit, all sections meet the acceptance criteria under both static and pseudo-static conditions.

16.3.4.1.1 Productora

GMT defined four geotechnical design zones for both the Productora and Alice open pits (see Figure 16.4), with recommended inter-ramp angles ranging from 47° to 55°. These angles were adjusted to account for pit ramps, leading to design inter-ramp angles from 36° to 55°.

An area to consider for further studies is the Productora eastern pit wall. Uncertainties in the structural domain boundaries, and the conservative shear strength properties of the joint sets controlling stability may underestimate slope instability.

Figure 16.4 : Location and Orientation of the Geotechnical Sections of Productora and Alice Pits

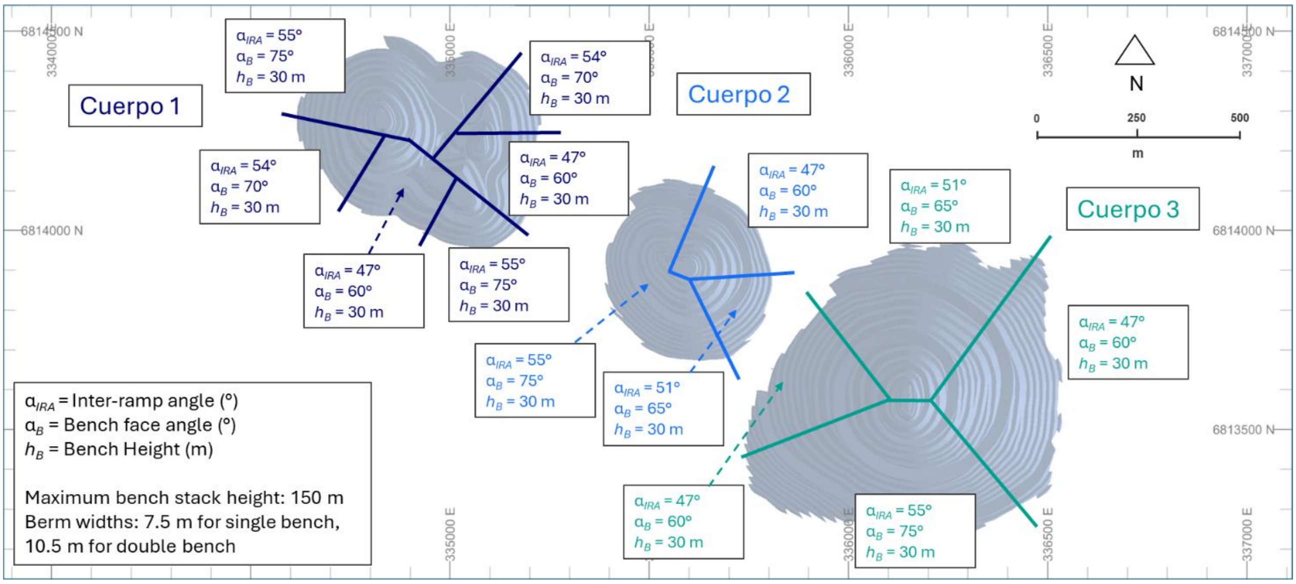


16.3.4.1.2 Cortadera

GMT defined geotechnical design zones for each of the Cortadera open pits (Cuerpo 1, Cuerpo 2 and Cuerpo 3) (Figure 16.5), with recommended inter-ramp angles ranging from 47° to 55°. These angles were adjusted to account for pit ramps, leading to design inter-ramp angles from 43° to 57°.

For the three Cortadera open pits, an area to consider for future studies is the eastern pit walls, where stability is influenced by major fault structures within the skarn structural domain under oxide weathering conditions. The location, extent, and orientation of these fault structures should be evaluated further.

Figure 16.5 : Location and Orientation of the Geotechnical Sections of Cortadera Pits

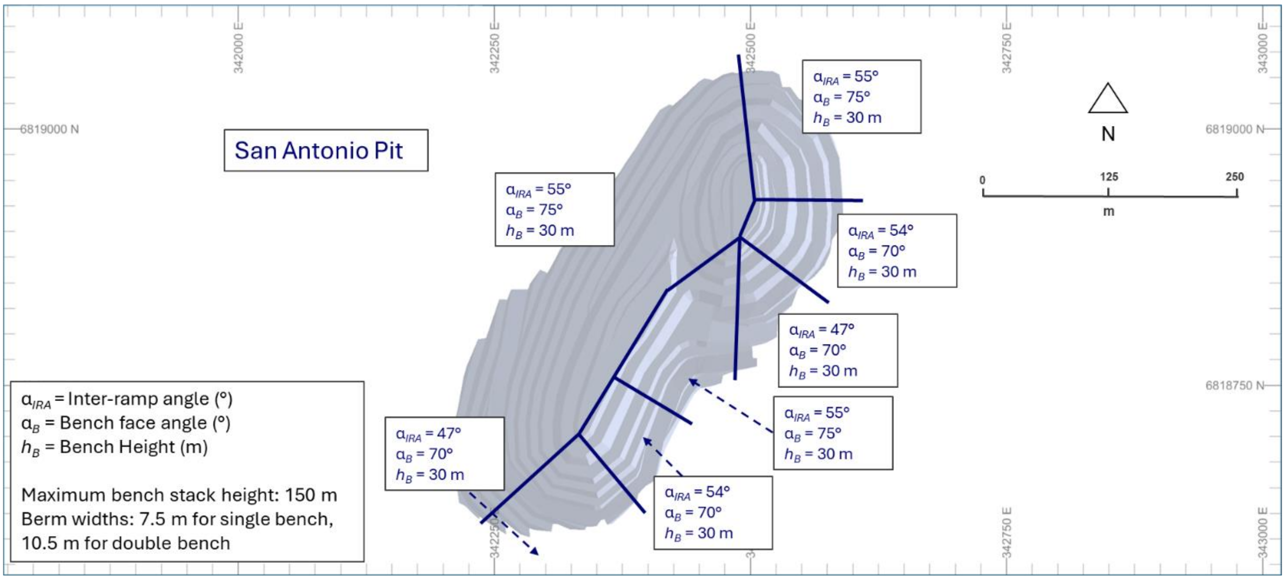


16.3.4.1.3 San Antonio

GMT defined seven geotechnical design zones for the San Antonio open pit (Figure 16.6), with recommended inter-ramp angles ranging from 47° to 55°. These angles were adjusted to account for pit ramps, leading to design inter-ramp angles from 49º to 57º.

It is recommended for later studies into the project to develop a new structural model that delineates the boundaries of structural domains and describes their respective joint sets and characteristics, including spacing, persistence, and shear strength properties. This structural model must also evaluate the existence, orientations, and extent of primary faults to enhance the quality of inter-ramp and overall stability analyses.

Figure 16.6 : Location and Orientation of the Geotechnical Sections of San Antonio Pit



16.3.5 Open Pit Optimisation Parameters

The following section summarises the economic parameters used for the open pit and underground block cave optimisation studies.

The final open pit shells assumed the 100% revenue factor (RF 100%) economic areas as developed within NPVS. The assumed metal prices used for the open pit optimisation study were lower/more conservative compared to the Project financial model metal price forecasts. This was a reasonable decision particularly as there were unknowns with update open pit mining contractor costs as the optimisation study progressed prior to being able to provide open pit contractors with suitable Request for Pricing data and mining information.

The open pit optimisation study used previous Productora open pit study mining costs escalated to reflect estimated benchmarked open pit optimisation costs for 2024. The challenge however with using escalated costs (in the absence of updated contractor costs) is these benchmarked optimisation costs makes many educated assumptions in regard to total rock moved, equipment potentially used and haulage distances that may be realised. Knowing these unforeseen cost modifications, the study team selected a lower metal price forecast to offset the risk of producing poor open pit economic perimeters.

Secondly, the study team were still contemplating considering owner mining for some of the open pits or switching from contractors to owner mining. With this in mind, the study team made it clear that should only contractor mining be considered, it is conceivable that the assumed open pit optimisation costs might be slightly optimistic, however, using lower metal price forecasts for the open pit optimisation study would again resolve any potential issues with very large and poorly created open pit economic perimeters.

The study team furthermore developed several open pit optimisation sensitivities to test the potential impact of higher metal prices or higher mining costs and also a combination of both on the pit shells that could be selected for the PFS. The decision was made to accept the open pit optimisation outputs and selected pit shells simply as the potential impact of having sub-optimal open pit perimeters were very low and seeing there could be alternate mining options (owner or contractor) that could be considered. Post the PFS check optimisations at the higher metal prices used in the Financial Model and the final open pit mining costs proved that optimal open pit perimeters/pit shells were selected for the study.

Open pit mining operating costs were based upon a mining contractor scenario. The mining cost estimate for the pit optimisation process is based on the analysis of a mining contractor quote received specifically for Productora. The cost model considers differential costs for uphill and downhill hauling. This resulted in an initial mining cost estimate of approximately US$2.03/t for Productora and Cortadera, and US$2.23/t for Alice and San Antonio, which corresponds to the average mining cost of the referred mining contractor quote plus a preliminary assessment of HCH's mining personnel. This mine operating cost was subsequently updated when the final mining production schedule became available. The study team furthermore applied incremental mining costs by depth which realised higher overall unit mining costs for each of the pit optimisations (bench incremental costs applied per bench and per tonne of material).

The following optimisation table (Table 16.1) summarises the parameters used in NPVS:

Table 16.1 Costa Fuego PFS Optimisation Parameters

Cost Parameters	Unit	Productora	Alice	Cortadera	San Antonio
Mining Costs					
Mining Unit Cost	$/t	2.03	2.23	2.03	2.23
Mining Incremental Cost-below Reference Bench	$/t/Bench	0.04	0.04	0.04	0.04
Mining Incremental Cost-Above Reference Bench	$/t/Bench	0.01	0.01	0.01	0.01
Reference Bench for Base Mining Cost	RL	800	780	980	1210
Processing Costs					
Oxide Heap Leach Unit Cost	$/t	6.74	4.84	5.24	5.64
Rehandle % assumption	%	15%	0%	15%	0%
Rehandle cost included	$/t	0.08	0.08	0.08	0
Additional Rope-con cost	$/t	0	0	0.31	0.31
Additional Road Trucking/Transport of ore	$/t	0	0	0	3.72
Total Process Cost	$/t	6.82	4.92	5.63	9.67
Dump Leach Unit Cost	$/t	2.64	4.07	4.2	n/a
Rehandle % assumption	%	0%	0%	15%	n/a
Rehandle cost included	$/t	0	0	0.08	n/a
Increased Drill & Blast Cost - Dump Leach ore	$/t	0.3	0.3	0.3	n/a
Additional Rope-con cost	$/t	0	0	0	n/a
Additional Road Trucking/Transport of ore	$/t	0	0	0	n/a
Total Process Cost	$/t	2.94	4.37	4.58	n/a
Concentrate ore Unit Cost	$/t	6.68	6.56	6.4	5.89
Rehandle % assumption	%	15%	15%	15%	100%

Table 16.1 Costa Fuego PFS Optimisation Parameters					
Cost Parameters	**Unit**	**Productora**	**Alice**	**Cortadera**	**San Antonio**
Rehandle cost included	$/t	0.08	0.08	0.08	0.58
Additional Rope-con cost	$/t	0	0	0.31	0.31
Additional Road Trucking/Transport of ore	$/t	0	0	0	3.41
Total Process Cost	$/t	6.76	6.64	6.79	10.19
Revenue Calculations					
Copper metal price	$/lb	3.5	3.5	3.5	3.5
Gold metal price	$/oz	1700	1700	1700	1700
Silver metal price	$/oz	20	20	20	20
Molybdenum metal price	$/lb	14	14	14	14
Cu payabilities	%	96.5%	96.5%	96.5%	96.5%
Au payabilities	%	90.0%	90.0%	90.0%	90.0%
Ag payabilities	%	90.0%	90.0%	90.0%	90.0%
Mo payabilities	%	98.0%	98.0%	98.0%	98.0%
Cu Recovery	%	Variable – Provided in Block model	Variable – Provided in Block model	Variable – Provided in Block model	Variable – Provided in Block model
Au recovery	%	Variable – Provided in Block model	Variable – Provided in Block model	Variable – Provided in Block model	Variable – Provided in Block model
Ag recovery	%	Variable – Provided in Block model	Variable – Provided in Block model	Variable – Provided in Block model	Variable – Provided in Block model
Mo recovery	%	Variable – Provided in Block model	Variable – Provided in Block model	Variable – Provided in Block model	Variable – Provided in Block model
Discount Rate	%	8%	8%	8%	8%
Discount per Bench applied	Benches/Year	5	5	5	5
Process Plant Throughput					
Plant throughput	Mtpa	22.3	23.2	24.2	19.4
Moisture Content in Cu Concentrate	%	8%	8%	8%	8%
Moisture Content in Mo Concentrate	%	5%	5%	5%	5%
Cu grade in Dry Concentrate	%	25%	25%	25%	25%
Mo grade in Dry Concentrate	%	50%	50%	50%	50%
Modifying Factors					
Model regularisation – Planned Dilution (mX,mY,mZ)	Regularised	5-10-5	5-10-5	5-10-5	5-10-5
Model regularisation – Planned Ore loss (mX,mY,mZ)	Regularised	5-10-5	5-10-5	5-10-5	5-10-5

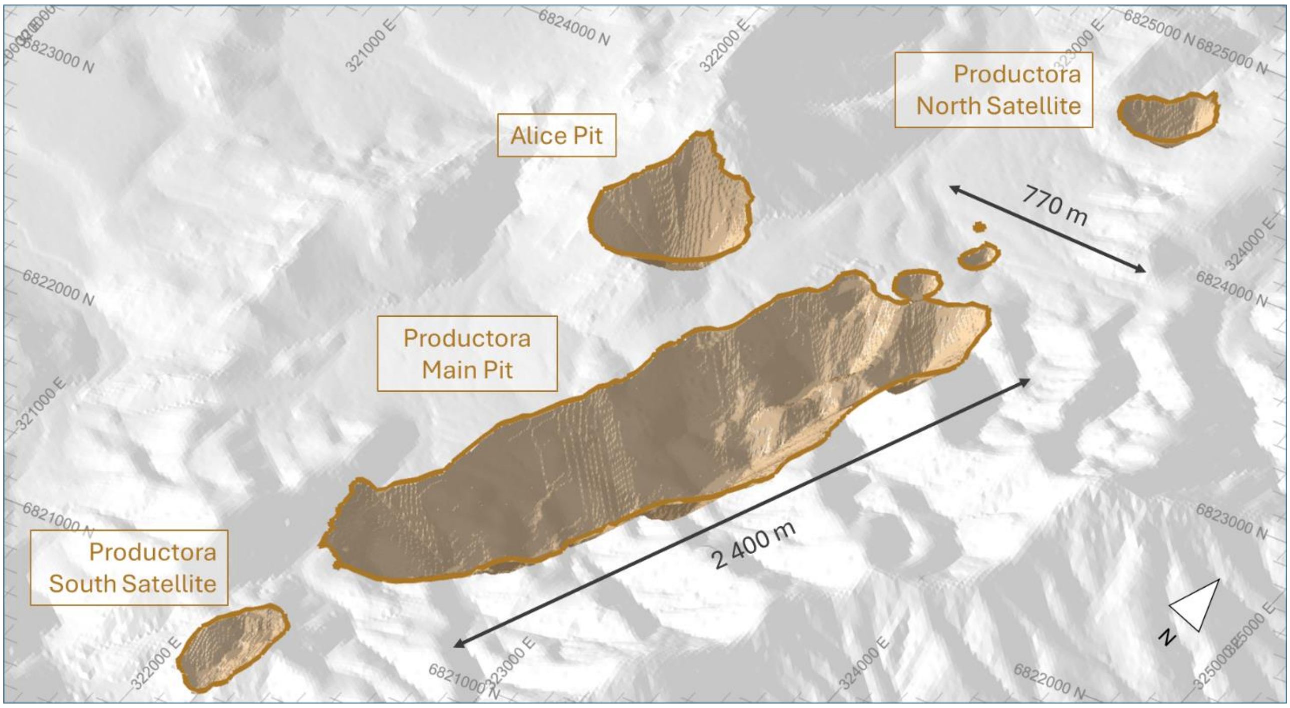
Table 16.1 Costa Fuego PFS Optimisation Parameters					
Cost Parameters	**Unit**	**Productora**	**Alice**	**Cortadera**	**San Antonio**
Optimisation inputs - Unplanned Mine Dilution	%	5%	5%	5%	5%
Optimisation inputs - Mining Recovery of ore	%	95%	95%	95%	95%
Revenue Factor pit selected	RF	100%	100%	100%	100%

Various royalty costs were also included. Two of the most notable are the CCHEN royalty at Productora and the Purisima Royalty to Cortadera. Additional unplanned mine modifying factors were considered during the open pit optimisation studies.

16.3.6 Results

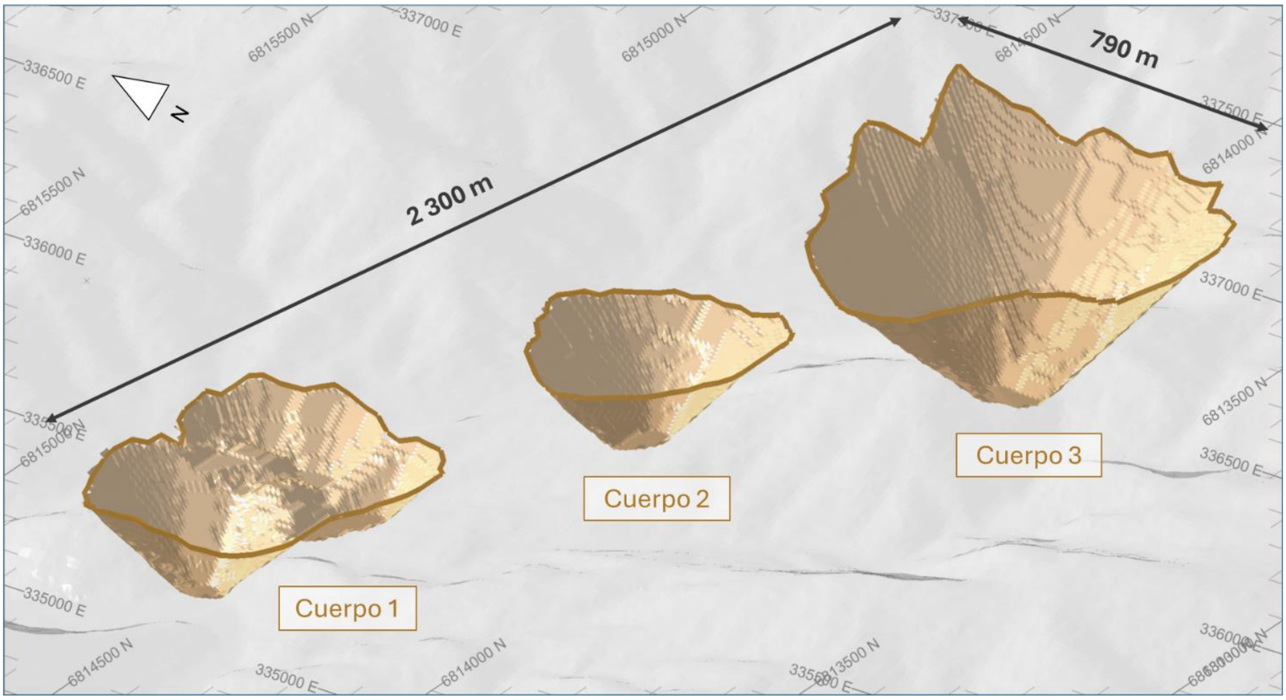
Optimisations for the Productora area produced several pits, consisting of Productora main pit (and satellite pits) and Alice pit, as shown in Figure 16.7. The Productora main pit is approximately 2.4 km in length and 770 m wide, while the Alice pit is approximately 550 m in length and 500 m wide.

Figure 16.7 : Productora and Alice Open Pits - View Looking North-West



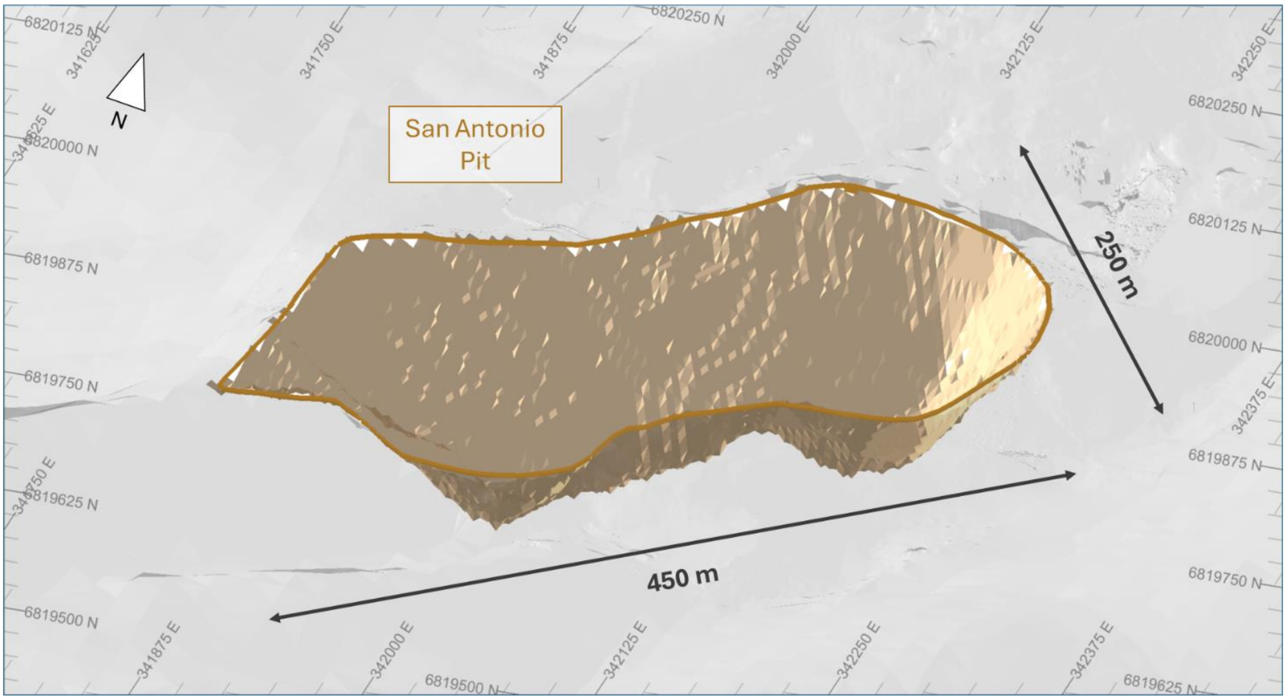
Optimisation results at Cortadera (located roughly 15 km east-southeast from Productora) produced three pits, namely Cuerpo 1, 2 and 3 (Figure 16.8). Mineralisation extends below Cuerpo 3 and is amenable to mass mining methods. The project utilised a block cave mining approach to assess the viability of this resource, which is discussed in more detail in Section 1.16.

Figure 16.8 : Cortadera Open Pits – View Looking North-West



The San Antonio pit is located roughly 8 km northeast of the Cortadera mining area and is shown in Figure 16.9.

Figure 16.9 : San Antonio Open Pit – View Looking North



Production feed is the material which provides a positive economic contribution and is contained within the optimised shell boundaries. The production feed is estimated from the regularised block models and includes diluted material coming from Mineral Resource blocks classified as Indicated. At Costa Fuego, production feed includes Oxide leach, Low Grade (LG) sulphide leach and Sulphide concentrator material.

The PFS developed both pit shells and open pit designs. The following optimisation summary report tables depict the quantity and quality of ore and waste reported within the pit shells developed.

Table 16.2 Optimisation Quantity Report Summary (All Open Pits)

Material - All Open Pits	Units	Total
Oxide Leach	Mt	28.25
Cu Grade	%	0.44
LG Dump Leach	Mt	54.38
Cu Grade	%	0.12
Sulphide Concentrator	Mt	314.43
Cu Grade	%	0.07
Au Grade	g/t	110.77
Mo Grade	Ppm	0.37
Ag Grade	g/t	545.12
Waste Rock Tonnes	Mt	28.25

Table 16.3 Optimisation Quantity Report Summary (Productora Open Pits)

Material - Productora	Units	Total
Oxide Leach	Mt	17.99
Cu Grade	%	0.52
LG Sulphide Leach	Mt	52.42
Cu Grade	%	0.12
Sulphide Concentrator	Mt	233.44
Cu Grade	%	0.36
Au Grade	g/t	0.07
Mo Grade	ppm	139.29
Ag Grade	g/t	0.33
Waste Rock Tonnes	Mt	418.48

Table 16.4 Optimisation Quantity Report Summary (Alice Open Pit)

Material - Alice	Units	Total
Oxide Leach	Mt	2.98
Cu Grade	%	0.37
LG Dump Leach	Mt	0.35
Cu Grade	%	0.18

Table 16.4 Optimisation Quantity Report Summary (Alice Open Pit)		
Material - Alice	**Units**	**Total**
Sulphide Concentrator	Mt	9.27
Cu Grade	%	0.32
Au Grade	g/t	0.03
Mo Grade	Ppm	45.13
Ag Grade	g/t	0.19
Waste Rock Tonnes	Mt	28.27

Table 16.5 Optimisation Quantity Report Summary (Cortadera Open Pits)		
Material - Cortadera	**Units**	**Total**
Oxide Leach	Mt	7.09
Cu Grade	%	0.25
LG Dump Leach	Mt	1.61
Cu Grade	%	0.18
Sulphide Concentrator	Mt	68.27
Cu Grade	%	0.30
Au Grade	g/t	0.10
Mo Grade	Ppm	27.68
Ag Grade	g/t	0.49
Waste Rock Tonnes	Mt	90.61

Table 16.6 Optimisation Quantity Report Summary (San Antonio Open Pit)		
Material - San Antonio	**Units**	**Total**
Oxide Leach	Mt	0.19
Cu Grade	%	1.12
LG Dump Leach	Mt	0.00
Cu Grade	%	0.00
Sulphide Concentrator	Mt	3.45
Cu Grade	%	0.79
Au Grade	g/t	0.01
Mo Grade	Ppm	1.86
Ag Grade	g/t	1.19
Waste Rock Tonnes	Mt	7.76

Figure 16.10 : Productora Resource Model Blocks Filtered for blocks >0.1% Cu and Indicated Resource Classification

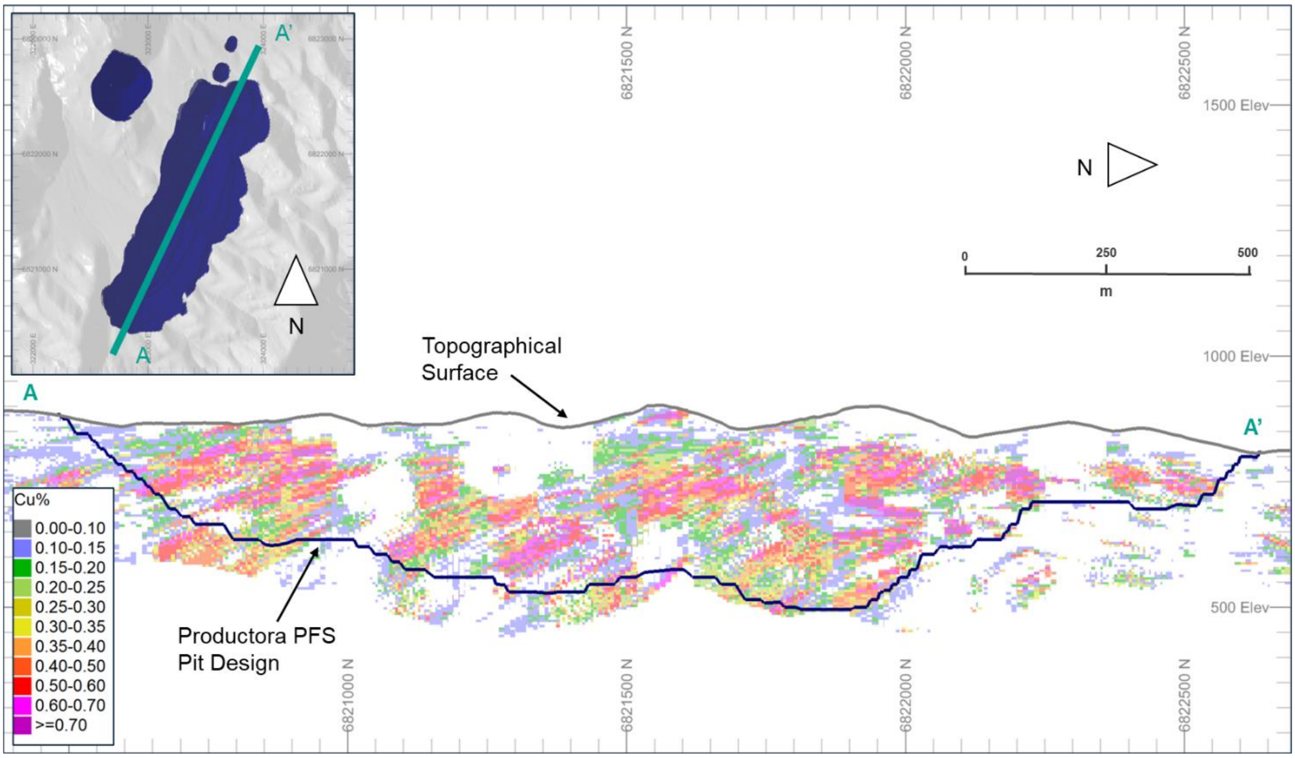


Figure 16.11 : Cortadera Resource Model Blocks Filtered for blocks >0.1% Cu and Indicated Resource Classification

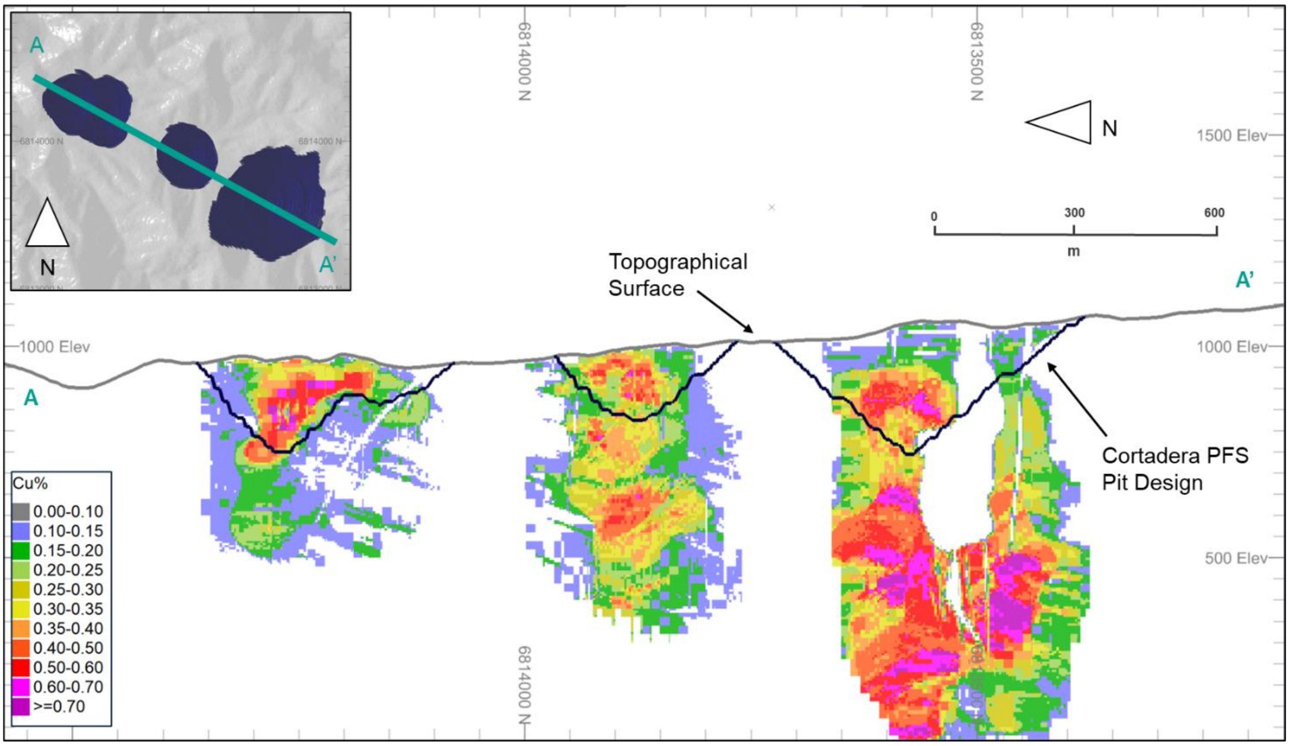
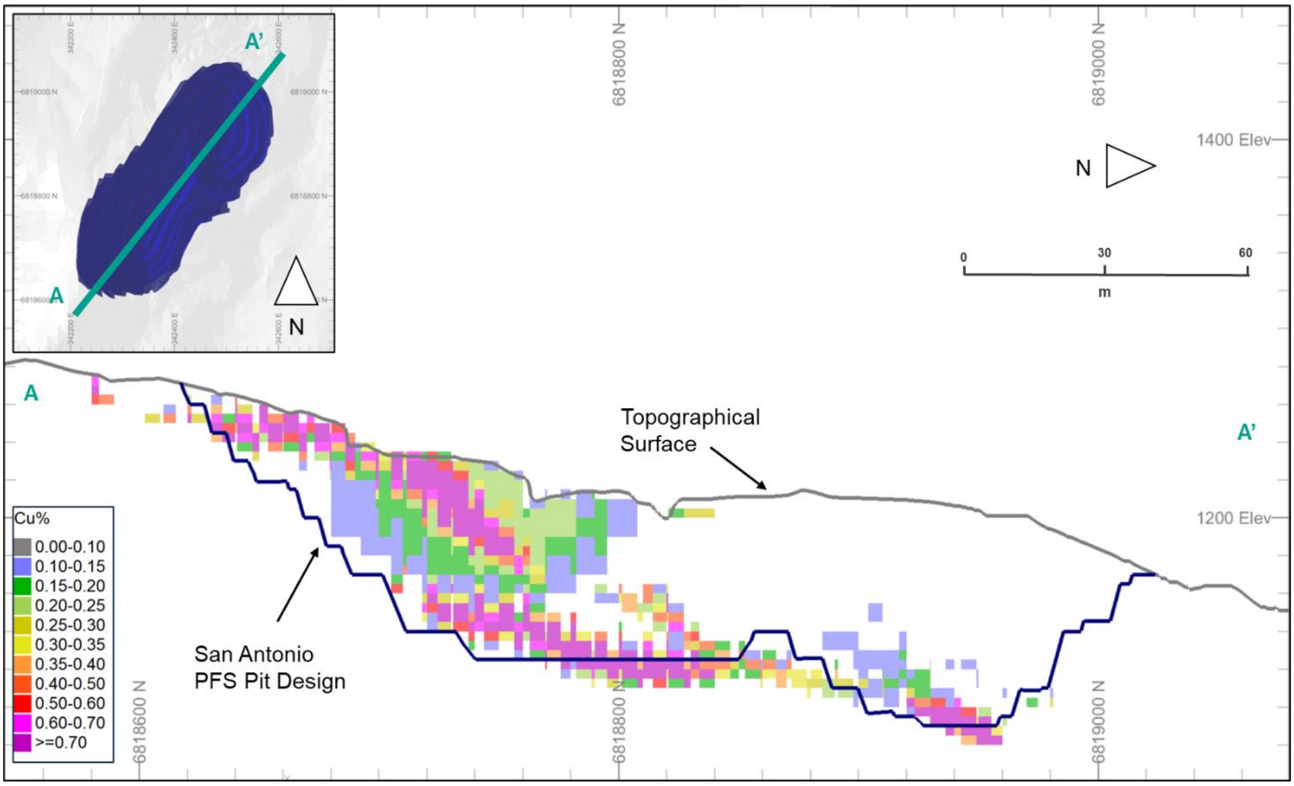


Figure 16.12 : San Antonio Resource Model Blocks Filtered for blocks >0.1% Cu and Indicated Resource Classification



16.3.7 Relevant Factors Affecting Production Feed

As of the date of this Report, there were no known environmental, permitting, legal, title, taxation, socio-economic, marketing, political or other relevant factors which could affect the mining inventory.

16.3.8 Open Pit Mine Design

The Costa Fuego Project's open pits are designed as a conventional truck-shovel operation consisting of four open pit mining areas/deposits namely Productora, Alice, Cortadera and San Antonio. There is also a proposed underground block cave exploitation potential at the Cortadera Cuerpo 3 deposit. The larger mining equipment proposed are principally considered for the bigger open pits (Productora and Cortadera) where bulk waste loading benefits timing and mining cost as two key aspects of the open pit operations for Costa Fuego Project.

The mining study considered 220 t capacity rigid body mining trucks matched with a 60 t capacity bucket size hydraulic excavator or shovel (mainly for the bulk waste and some bulk ore mining benches). As the respective pits gets deeper, there will be a transition from the larger bulk earth moving equipment to smaller trucks and excavators for improved selectivity. The smaller equipment for the deeper mining benches at both Productora and Cortadera is typically 90 t trucks with matching 12 m^3 bucket size hydraulic excavators.

Truck and Shovel/Excavator operations (diesel equipment) are also planned open pit operations at Alice and San Antonio. The proposed mining fleet at Alice is a combination of 90 t trucks matched with 12 m^3 bucket size hydraulic excavators for the waste benches and upper mining benches whilst 45 t trucks with 7 m^3 bucket size excavators are planned for the bottom ore benches. Limitations in relation to speed factors for ore mining coupled with limiting ramp widths, dictate where the smaller equipment fleets become necessary (i.e. reduction in ramp widths designed for this study and for each open pit).

16.3.8.1 Productora Mine Design

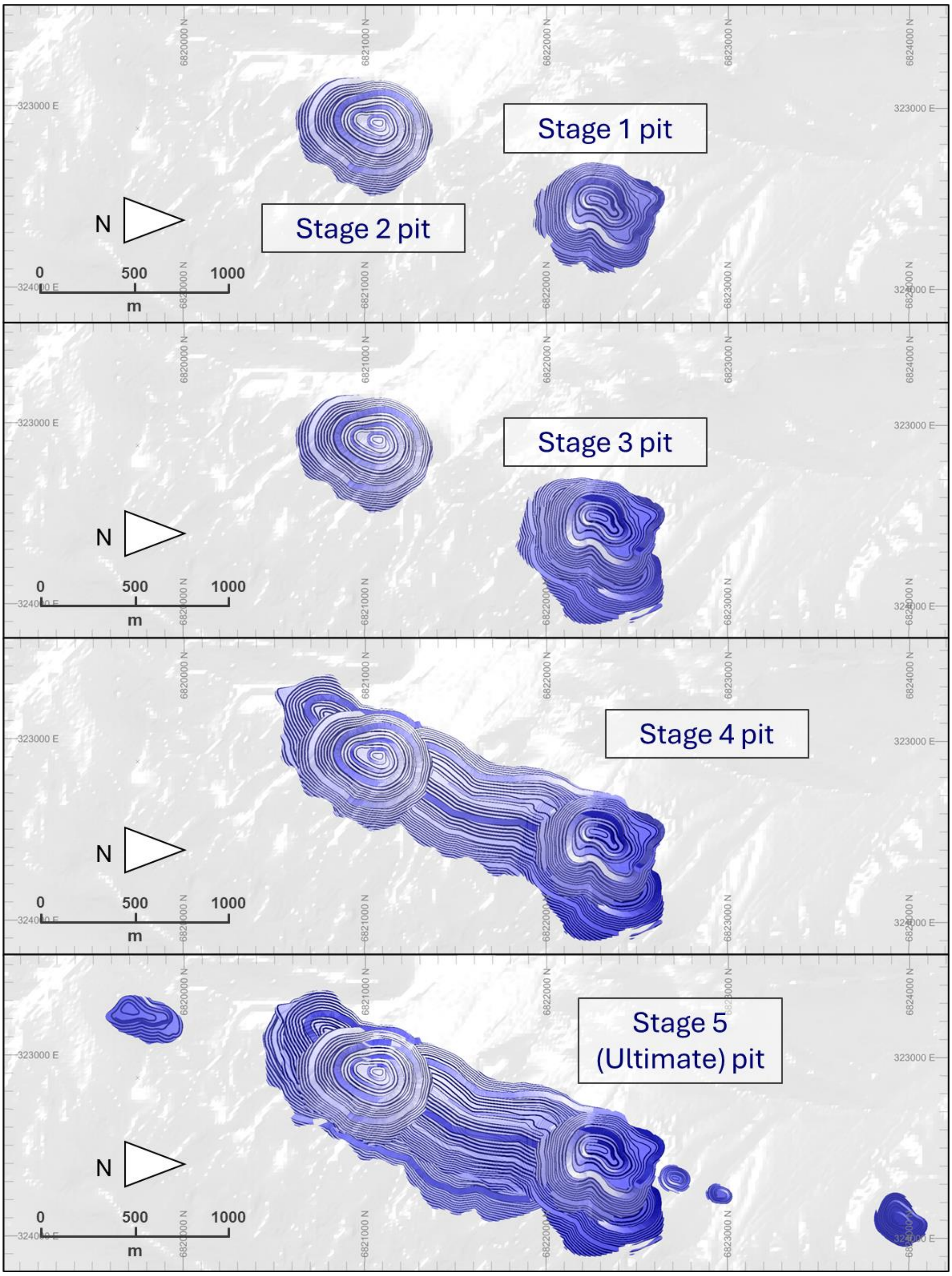
Productora pit design includes five nested stages (Stages 1 to 5) to balance stripping requirements while satisfying the processing requirements. The Stage designs are shown in Figure 16.13.

The pit is designed at 30 m ramp widths for two-way traffic with the last bench at single-lane traffic of 15 m to minimise waste stripping. The design parameters for Productora are shown in Table 16.7.

Table 16.7 Productora Mine Design Parameters

Design Parameter	Criteria
Ramp Width – Top waste benches – Dual way (Larger Trucks)	31 m
Ramp Width – Mid-depth – Semi Dual way large trucks (Dual way 90 t trucks)	23 m
Ramp Width – Bottom Benches – Single Way (90 t Trucks)	15 m
Ramp Width – Bottom Benches – Single Way reduced/bottom cuts	10 m
Road Gradient (Standard to steep bottom ramps)	10% - 12%
Double Bench Height	30 m
Single Bench Height	15 m
Berm Width – Single Bench	7.5 m
Berm Width – Double Bench	10.5 m
Bench Face Angle	Variable by sector

Figure 16.13 : Productora Pit Stages and Ultimate Pit Design



16.3.8.2 Alice Mine Design

Alice pit design has one stage with 23 m ramp widths for two-way traffic with the last bench at single-lane traffic of 7 m. The design parameters for Alice are shown in Table 16.8.

Table 16.8: Alice Mine Design Parameters	
Design Parameter	**Criteria**
Ramp Width	23 m
Road Grade	10%
Bench Height	30 m
Berm Width – Single Bench	7.5 m
Berm Width – Double Bench	10.5 m
Bench Face Angle	Variable by sector

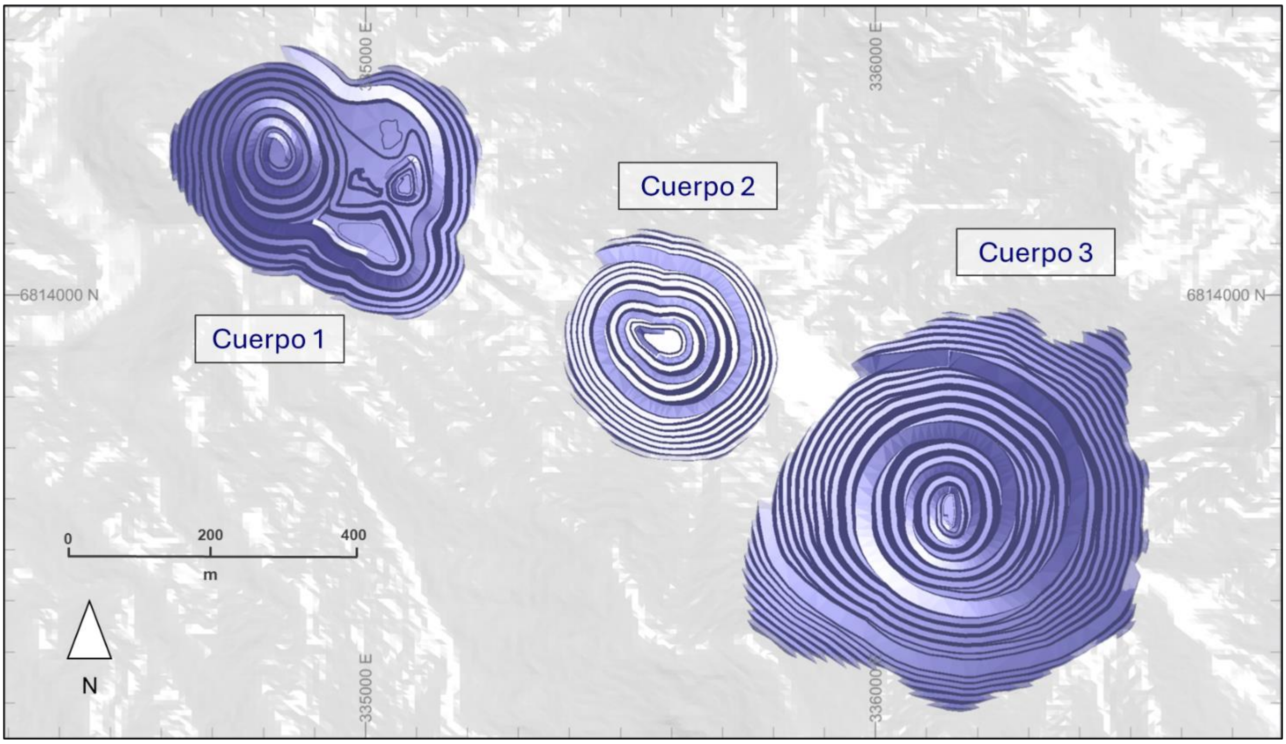
16.3.8.3 Cortadera Open Pit Mine Design

Cortadera was designed as three independent pits: Cuerpo 1, Cuerpo 2, and Cuerpo 3. The design parameters for Cortadera are shown in Table 16.9 below.

Table 16.9: Cortadera Open Pit Mine Design Parameters			
Design Parameter	**Cuerpo 1**	**Cuerpo 3**	**Cuerpo 3**
Ramp Width	23 m	30 m	30 m
Road Grade	10%	10%	10%
Bench Height	30 m	30 m	30 m
Berm Width – Single Bench	7.5 m	7.5 m	7.5 m
Berm Width – Double Bench	10.5 m	10.5 m	10.5
Bench Face Angle	Variable by sector	Variable by sector	Variable by sector

The Cortadera pit designs are shown in Figure 16.14.

Figure 16.14 : Cortadera Open Pit Designs



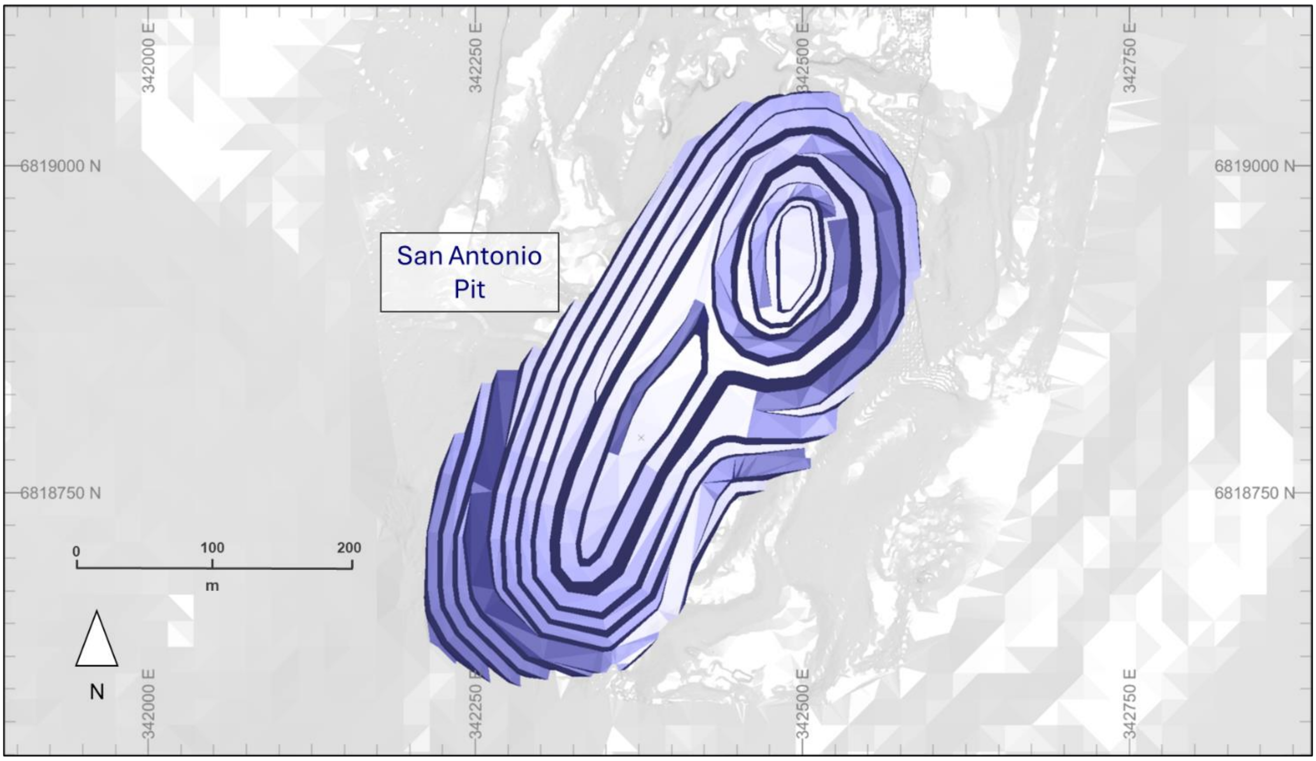
16.3.8.4 San Antonio Mine Design

San Antonio ultimate pit design has a ramp of 16 m and reduces to one-lane traffic in the last bench to 8 m wide. The design parameters for San Antonio are shown in Table 16.10 below.

Table 16.10: San Antonio Open Pit Mine Design Parameters	
Design Parameter	**Criteria**
Ramp Width	16 m
Road Grade	10%
Bench Height	30 m
Berm Width – Single Bench	7.5 m
Berm Width – Double Bench	10.5 m
Bench Face Angle	Variable by sector

The San Antonio pit design is shown in Figure 16.15.

Figure 16.15 : San Antonio Open Pit Design



16.4 Underground Mining

16.4.1 Introduction

The Cuerpo 3 deposit at Cortadera has underground exploitation potential, which forms the basis of this section. The PEA identified conventional block caving as the only underground mining method to be further investigated and no other underground mining methods are studied in this report.

16.4.2 Geotechnical Analysis – Underground

This section summarises the findings from the Geotechnical analysis completed by GMT for the PFS.

16.4.2.1 Rock quality

The geological drilling program at Cortadera has been extensive, comprising multiple campaigns with specific objectives, including exploration, resource definition, and geotechnical characterization. The data collected encompasses geotechnical parameters such as Rock Quality Designation (RQD), fracture frequency per metre (FF/m), joint orientation, and vein logging. Recent efforts, particularly the CORMET 2022 campaign, have focused on geotechnical drilling to support block caving studies, emphasizing detailed rock mass characterisation.

Drilling at Cortadera intersected Level 220 of the underground mine through 31 drill holes, yielding valuable data on rock mass characteristics. These include RQD, fracture frequency, joint alteration, joint roughness, Geological Strength Index (GSI), Q' values and Rock Mass Rating (RMR). Laboratory tests on intact rock, such as unconfined compressive strength (UCS), Brazilian tensile strength (BTS), triaxial testing, and point load tests (PLT), have been conducted as part of several campaigns.

For geotechnical analysis, the primary geological units were categorized into geotechnical units based on their weathering states: oxide, transitional, and fresh. Each geotechnical unit exhibited distinct geotechnical properties.

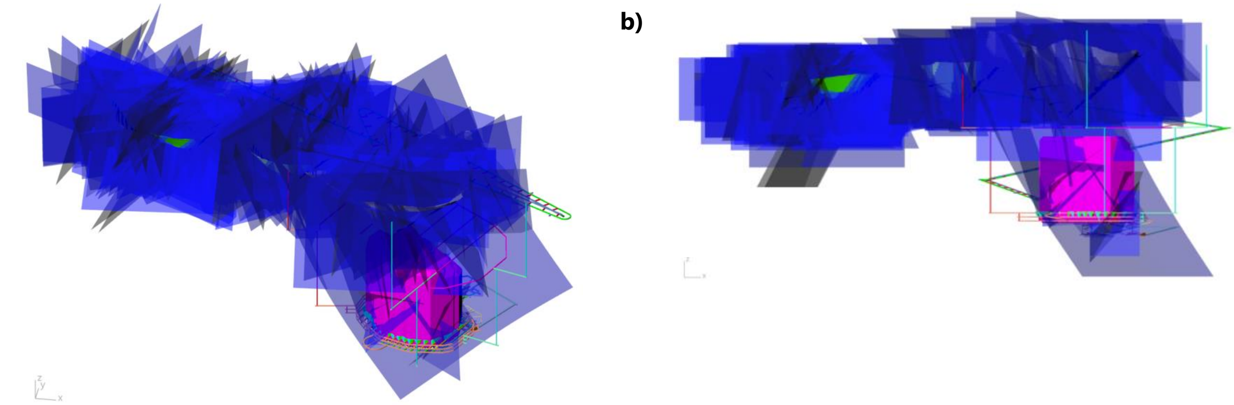
Uniaxial Compressive Strength (UCS) values highlight the unique characteristics of each geotechnical unit, reflecting the data source. For oxide rocks, UCS values derived from hardness logging range between 10 – 50 MPa. Transitional rocks, primarily evaluated through hardness logging, show UCS values mostly between 50 – 100 MPa. However, less competent geologies within the transitional unit, such as skarns, early mineralisation porphyries, and post-mineralisation dykes display lower UCS values of 25 – 50 MPa. In contrast, UCS values for fresh units, determined through laboratory testing, typically range from 100 – 137 MPa, except for the post-mineralisation dykes, which exhibit a notably lower UCS of 47 MPa.

16.4.2.2 Structural model

A Structural model for Geological structures that are relevant for Geotechnical Analysis have been developed for the Cortadera mine (Carrizo, 2024). Figure 16.16 presents isometric and front views of the 136 extended geological structures of this structural model. This structural model encompasses all three pits and the proposed underground mine.

Figure 16.16. Cortadera Structural model. a) Isometric view. b) Front view.

a) **b)**

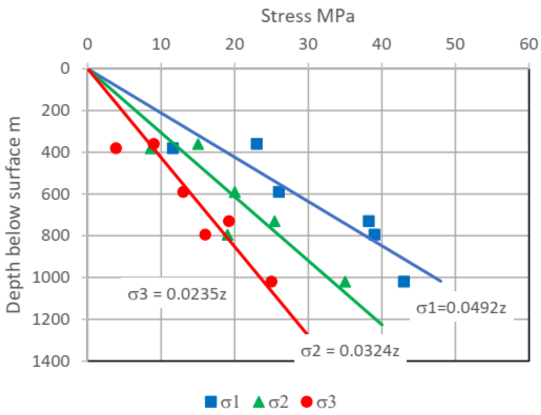


16.4.2.3 Insitu Stress

The in-situ stress was determined by the Australian Centre of Geomechanics (ACG) using deformation rate analysis (DRA) on five core samples (Dight, 2022). The stress measurements reveal a consistent trend of increasing stress with depth, as shown in Figure 16.17. The analysis also provides a mean stress orientation for numerical modelling, indicating that the major principal stress aligns SE/NW, while the minor principal stress is oriented sub-vertically.

Figure 16.17 a) Stress model defined by Dight (2022). b) Mean trend and plunge of stress model

a)



b)

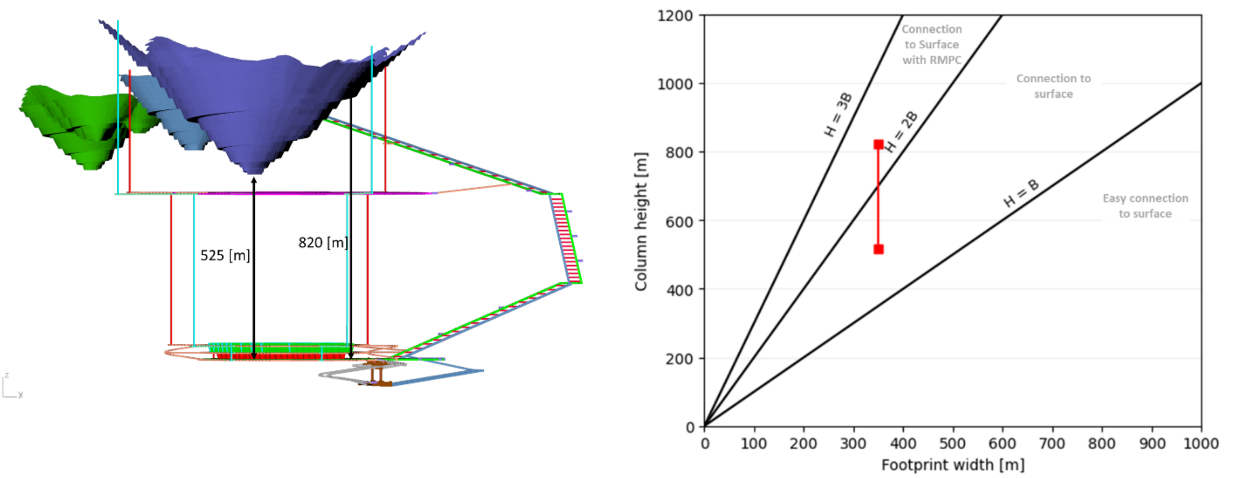
Mean Results

Stress	Trend	Plunge
σ1	143.1	9.3
σ2	50.2	17.4
σ3	260.1	70.1

16.4.2.4 Caveability assessment

The relationship between footprint dimensions and cave column heights, typically used as an initial indicator to assess if a cave can reach the desired height, was evaluated by GMT through benchmarking and empirical methods (GMT, 2024). This relationship is shown in Figure 16.18 where footprint width (B) and the cave column height (H) are used for a qualitative assessment of caveability (including the need for rock mass preconditioning). For the proposed Cortadera underground mine, the column height is estimated to range from approximately 525 m to 820 m, with a footprint width set at 350 m. The H/B ratio varies between 1.5 and 2.5. Analysis indicates that mines with similar ratios have effectively employed rock mass preconditioning (RMPC) to facilitate cave propagation, especially in scenarios involving taller cave columns.

Figure 16.18. Results of benchmarking of the relationship between the footprint width and the column height and the feasibility of connection to surface. The range of H/B values for Cortadera are indicated by a red line



Typically, initial assessments in block caving projects employ Laubscher's nomogram to estimate the minimum dimensions needed to initiate and propagate the cave. Laubscher's (1990) caving chart is widely regarded as the empirical standard method for determining the necessary Hydraulic Radius to ensure caving. This empirical approach adjusts the Intact Rock Mass Rating (IRMR) and uses these modifications to compute the Mining Rock Mass Rating (MRMR) through Equation (16.1).

$$\boldsymbol{MRMR = IRMR \times A_M \times A_O \times A_S \times A_T} \tag{16.1}$$

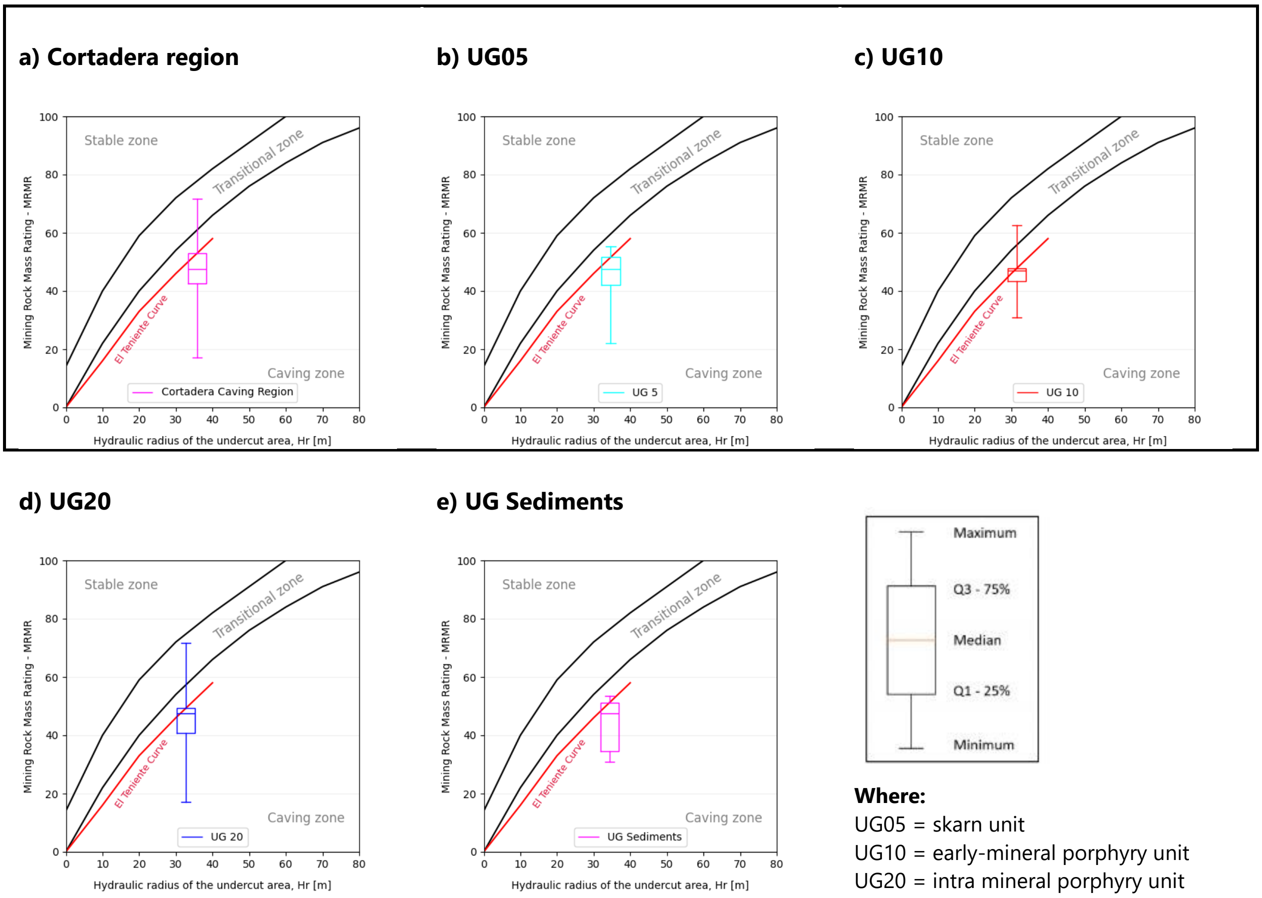
Where:

- $IRMR$: In-situ Rock Mass Rating (Laubscher & Jakubec, 2001).
- A_M: Weathering adjustment factor (between 0.3 and 1).
- A_O: Joint orientation adjustment factor (between 0.63 and 1).
- A_S: Mining induced stress factor (between 0.6 and 1.2).
- A_T: Blasting adjustment factor (between 0.8 and 1).
- A_W: Water adjustment factor (factor between 0.7 and 1.1).

For the case of the caveability of geotechnical units of Cuerpo 3: A_M=1 (no weathering), A_O=0.8 (three joint sets define the blocks, three faces inclined from the vertical, joint condition 16-30), A_S=1.0 (medium stress condition), A_T=1 (no blasting), A_W=1 (dry conditions).

Figure 16.19 presents Laubscher's empirical caving graph, displaying the average MRMR values and standard deviation for the geotechnical units defined at Cortadera mine. The plots indicate that a hydraulic radius (HR) between 30 and 40 [m], with an average of 37 [m], is sufficient to achieve continuous caving. It is important to note that the 75th percentile of the data was considered, assuming stronger rock mass conditions rather than the median value. In this Graph, the effect of the hydraulic fracturing (HF) is not considered.

Figure 16.19 Laubscher's caveability diagram (Laubscher, 1990) with the range of MRMR for the geotechnical units of Cuerpo 3 of the Cortadera Deposit. a) Cortadera Region, b) UG05 , c) UG10 (, d) UG20, e) UG Sediments.



Where:
UG05 = skarn unit
UG10 = early-mineral porphyry unit
UG20 = intra mineral porphyry unit

As experience with stronger rock masses in block caving grows, empirical methods increasingly incorporate the Mathews method. Adapted by Mawdesley (2002) for block caving from its original use in open stope design, this method uses a stability database of over 480 case histories and statistical techniques to delineate caving boundaries and isoprobability contours. It enhances excavation design optimization by adjusting the Q' value and calculating the Stability Number with Equation (16-2):

$$\boldsymbol{N = Q' \times A \times B \times C} \qquad (16\text{–}2)$$

Where:

N: Stability Number.

Q': Modified Rock Tunneling Quality Index system (Barton, 1974).

A: Rock stress factor (between 0.1 and 1).

B: Joint orientation adjustment factor (between 0.2 and 1).

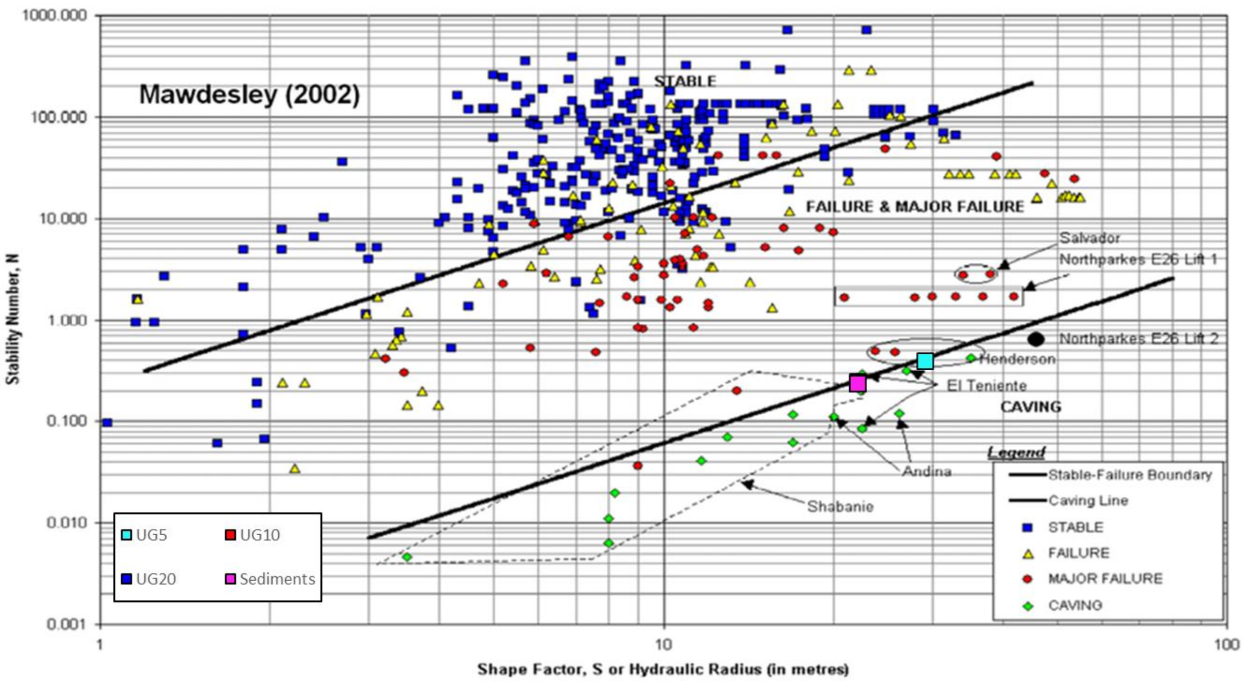
C: Design surface orientation factor (between 2 an 8).

For the case of the caveability of geotechnical units of Cortadera caving mine: $A = 0.1$ (conservative analysis using σ_c=100 [MPa] and σ_1=50 [MPa]), $B = 0.2$ (minimum B factor using the updated geological structural model for the roof), $C = 2.0$ (dip of roof equal to 0°).

Figure 16.20 presents an Extended Mathews Stability Chart, displaying the average N values for the geotechnical units defined at Cortadera. The plot indicates that a hydraulic radius (HR) between 22 m and 29 m is sufficient to achieve continuous caving.

Figure 16.20 Extended Mathews Stability Chart analysis (Mawdesley, 2002). a) Extended Mathews Stability Chart showing the average Q' values for Cortadera's geotechnical units: UG5, UG10, UG20, and Sediments, b) Summary of the Hydraulic Radius for each predominant geotechnical unit analyzed

a)



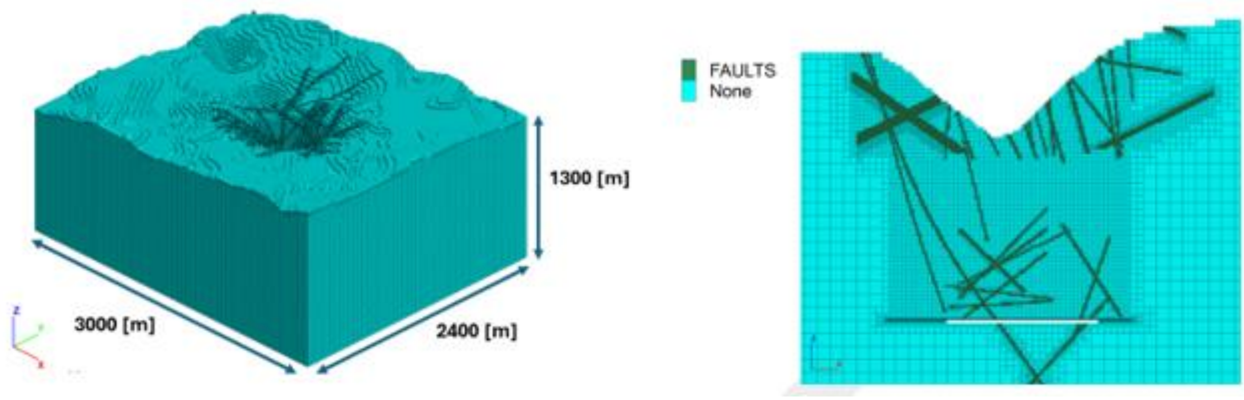
b)

UG	HR [m]	Q' (Fresh)	A	B	C	N
UG05 (Skarn)	29	11.4	0.1	0.2	2	0.46
UG10 (Early-mineral porphyry)	22	7.1	0.1	0.2	2	0.28
UG20 (intramineral porphyry)	22	6.8	0.1	0.2	2	0.27
Sediments	22	6.8	0.1	0.2	2	0.27

16.4.2.5 Cave propagation analysis

A Mine-scale FLAC3D model was developed by GMT (GMT, 2024) to simulate the caving process and associated subsidence. This model covers the entire regional extent, incorporating the final open-pit geometries of Cuerpo 1, 2, and 3, along with the structural model as shown in Figure 16.21.

Figure 16.21 a) Isometric view of numerical model dimensions. b) Section view of element size distribution.



Cave initiation and propagation were assessed both with and without the incorporation of HF. Figure 16.22 (a) and (b) illustrate cave growth under both scenarios, demonstrating that hydraulic fracturing significantly enhances cave initiation and propagation.

Regarding cave initiation, the results show that the required hydraulic radius (HR) for cave initiation decreases by 47% when HF is applied. Furthermore, the time required for the cave to break through to the open-pit surface is reduced by three-quarters.

For cave propagation, the results indicate that in the last quarter of the first operational year, the cave ratio increased from 0.5 to 3.5 with the incorporation of HF, while maintaining the same HR. By the last quarter of the second operational year, the cave ratio further increased from 3.7 to 5.4 under the influence of HF.

Figure 16.22 a) Effect of HF on simulated cave ratio by quarterly period. b) Effect of HF on simulated cave height by quarterly period.

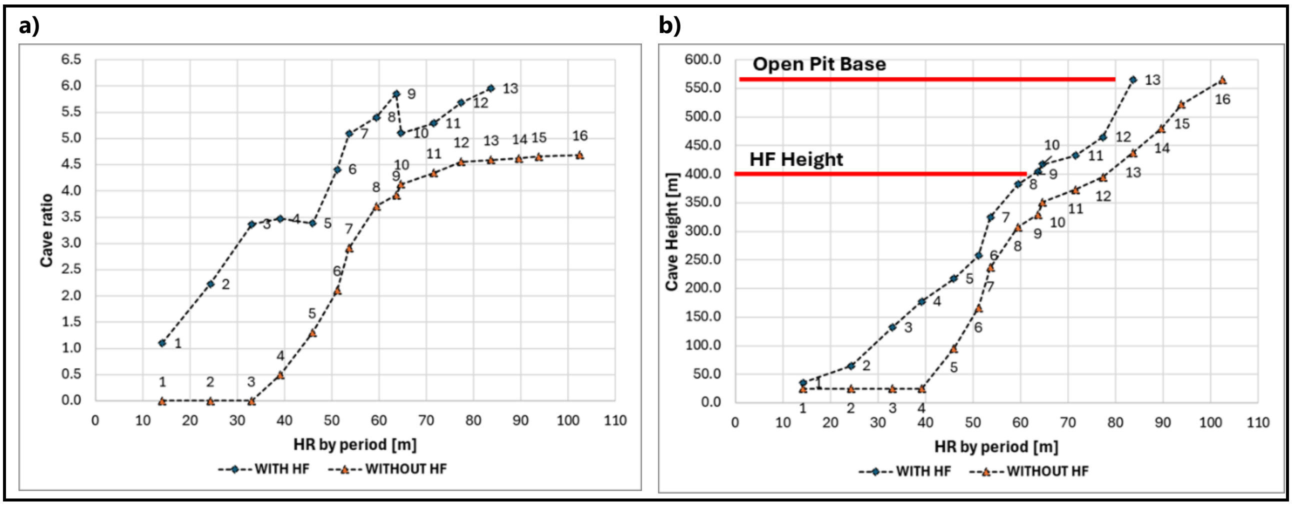
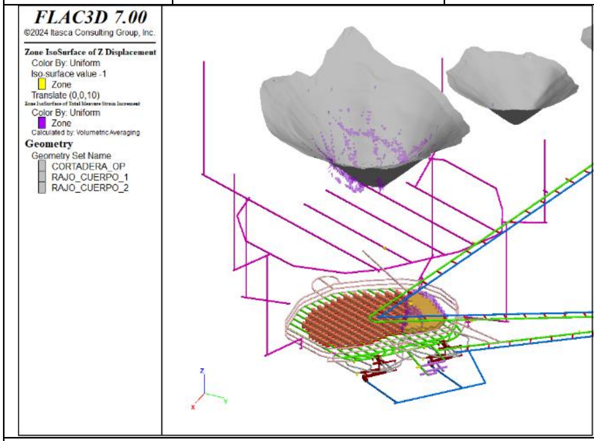
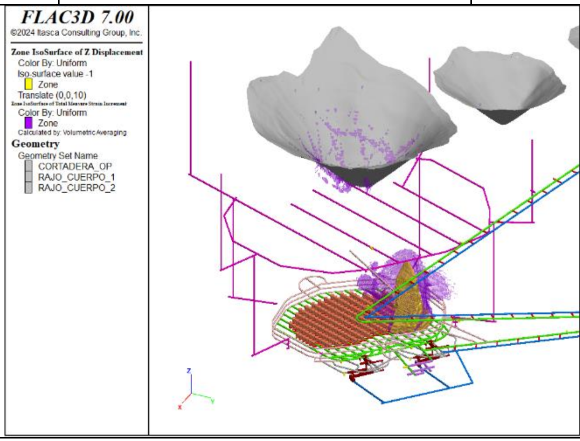
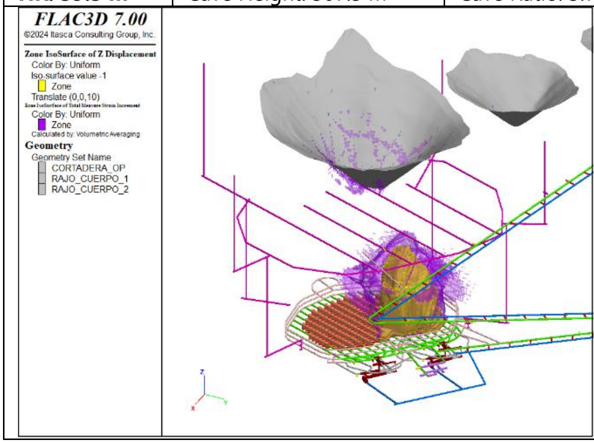
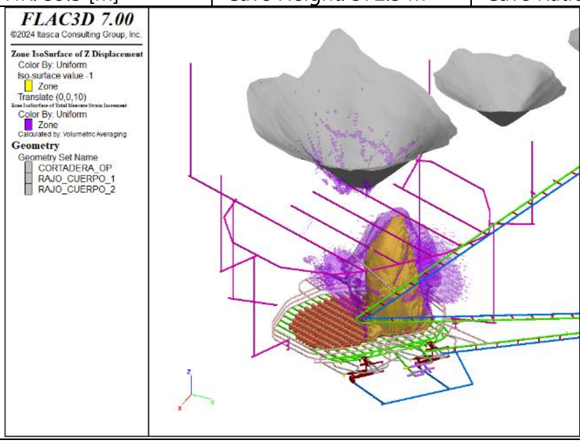


Figure 16.23. Isometric view of cave growth in model with HF and without HF

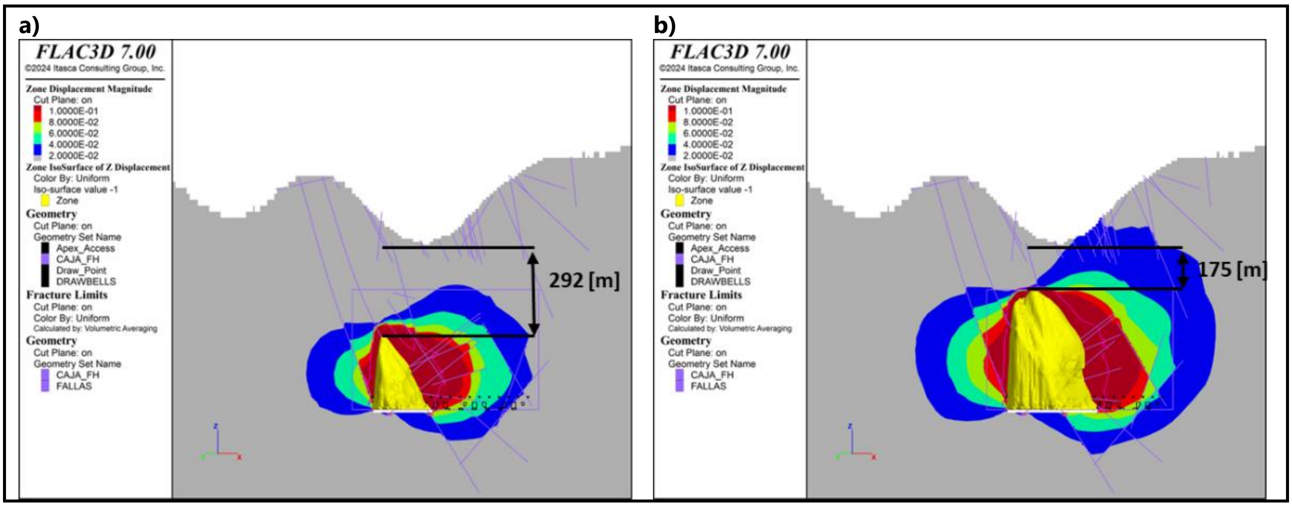
<table>
<tr><td colspan="3">a) Year 01, Quarter 04: Without HF</td><td colspan="3">b) Year 01, Quarter 04: With HF</td></tr>
<tr><td>HR: 39.2 m</td><td>Cave Height: 25 m</td><td>Cave Ratio: 0.5</td><td>HR: 39.2 m</td><td>Cave Height: 167.5 m</td><td>Cave Ratio: 3.5</td></tr>
<tr><td colspan="3">
</td><td colspan="3">
</td></tr>
<tr><td colspan="3">c) Year 02, Quarter 04: Without HF</td><td colspan="3">d) Year 02, Quarter 04: With HF</td></tr>
<tr><td>HR: 59.5 m</td><td>Cave Height: 307.5 m</td><td>Cave Ratio: 3.7</td><td>HR: 59.5 [m]</td><td>Cave Height: 372.5 m</td><td>Cave Ratio: 5.4</td></tr>
<tr><td colspan="3">
</td><td colspan="3">
</td></tr>
</table>

16.4.3 Open pit interaction

Open-pit interaction is evaluated by analysing the development of the cave shape over time and its interaction with the open-pit bottom surface. Figure 16.24 illustrates the vertical displacement at Year 02, Quarter 02, and Quarter 04, respectively, showing a reduction in crown pillar width from 292 m to 175 m, which leads to interaction between the cave back and the open pit.

The findings suggest that a crown pillar width of at least 300 m is necessary to maintain the integrity of the rock mass beneath the pit floor. The model also highlights the significant influence of geological structures on caving propagation.

Figure 16.24 Crown pillar stability analysis. Section view passing through the center of the footprint. a) Total displacements and crown pillar width at Year 02, Quarter 02, b) Total displacements and crown pillar width at Year 02, Quarter 04.



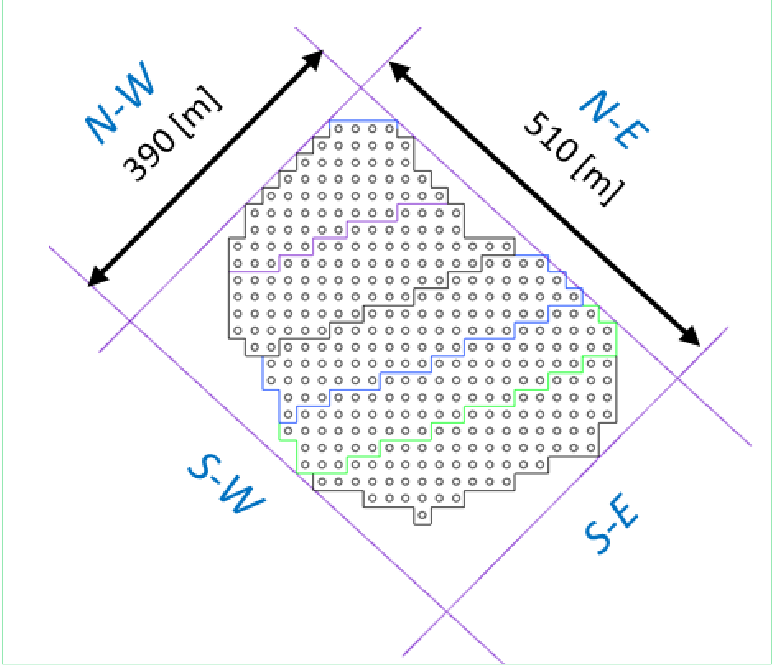
16.4.4 Subsidence assessment

16.4.4.1 Cave Angle

The cave angle is obtained by empirical methods (Laubscher, 2000) and is evaluated for each projected cave wall orientation (NW, NE, SW and SE), as described in Figure 16.25.
The estimated cave angles are:

- N-W wall: 78-79º.
- N-E wall: 80º-81º.
- S-W wall: 80º-81º.
- S-E wall: 79º-80º.

Figure 16.25 Footprint wall spans per orientation.



16.4.4.2 Numerical approach

Assessing caving-induced subsidence is critical for evaluating its effects on both underground infrastructure and surface mine facilities, as well as understanding potential environmental impacts from landscape changes. Numerical modelling defines subsidence zones to establish appropriate stand-off distances for all underground and surface critical infrastructure.

Numerical model outputs used in the subsidence analysis are associated with the following zones:

- Mobilised Zone: The model identifies this region by simulating vertical displacements exceeding 2 m, aligning with the boundaries of the subsidence crater
- Fractured Zone: Model outputs show areas where total strain exceeds 0.2%, correlating with regions expected to exhibit visible fracturing due to orebody extraction
- Continuous Subsidence Limit: The model delineates this limit based on combined horizontal strain and angular distortion values exceeding a specified threshold, representing areas with minimal but ongoing subsidence deformations (Harrison, 2011).

Figure 16.26 (a) shows that the continuous subsidence limit zone remains confined within the open pit boundary, while Figure 16.26 (b) shows that the ventilation shafts are unaffected by caving-induced strains. Additionally, Figure 16.27 shows that the interaction between the subsidence zone and underground infrastructure indicates that critical infrastructure, including declines, crusher chambers, ventilation shaft and access drives to the preconditioning level are not forecasted to be affected by subsidence.

Figure 16.26 a) Extent of subsidence zone estimated by numerical model. b) Distance between subsidence limit and ventilation chimneys.

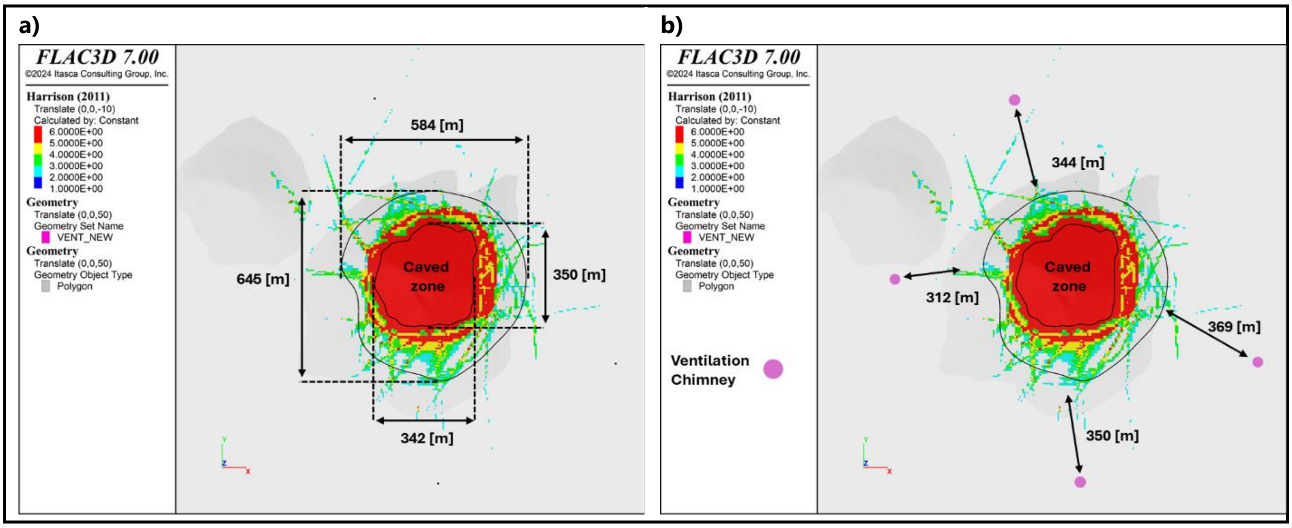
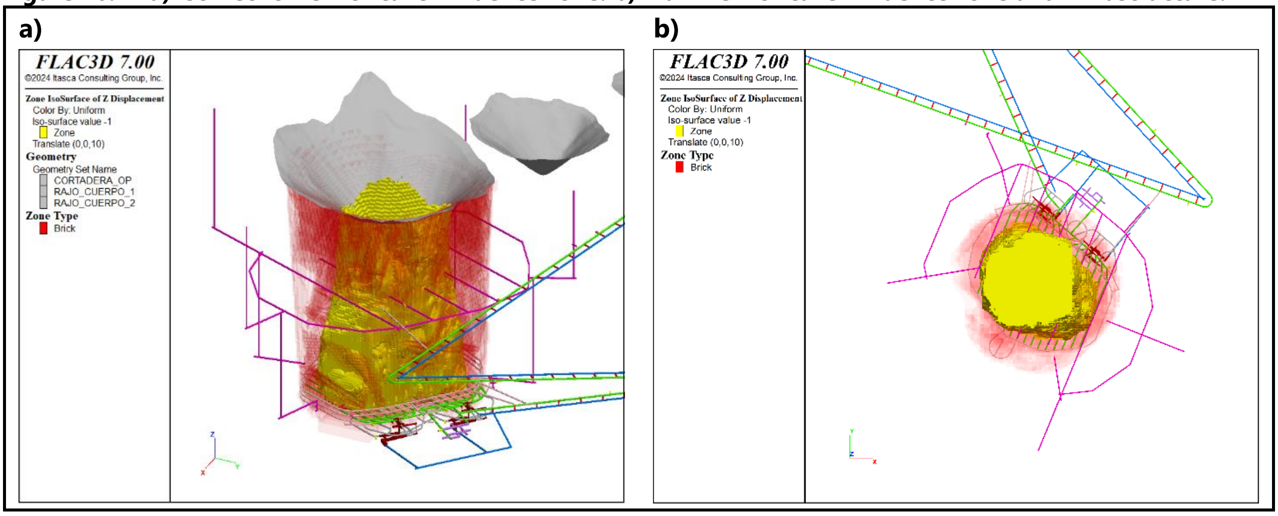


Figure 16.27 a) Isometric view of cave influence zone. b) Plan view of cave influence zone and infrastructure.



16.4.5 Stand-off distance to crusher chambers

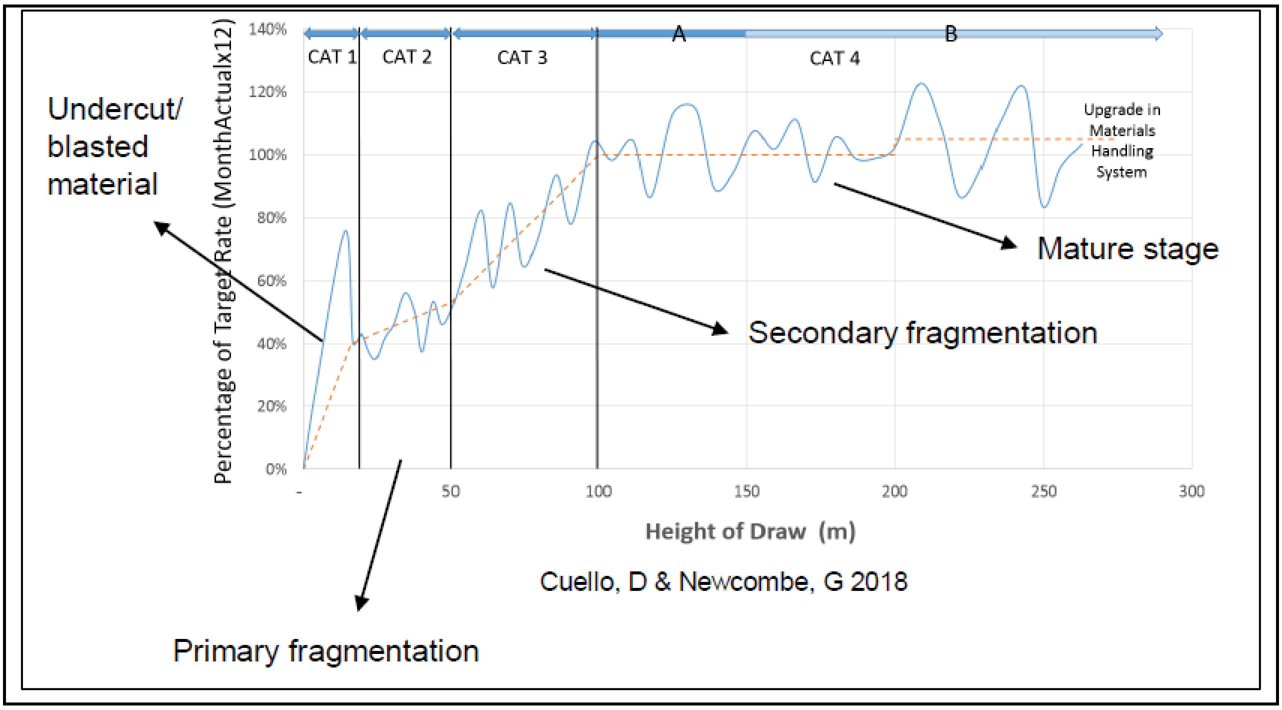
The stand-off distance between the crusher chamber and the undercut face is critical to ensure that the caving-induced stresses do not compromise the long-term integrity of the crusher chamber. In the current mine design, the crusher chambers are positioned 90 m away from the undercut front. The induced stress path for both major and minor principal stresses, from the initiation of caving to its breakthrough, has been monitored and compared against the failure envelope of the rock mass surrounding the crusher chambers. This analysis indicates that a 90 m stand-off distance is sufficient, as stress levels at the crusher chambers remain stable throughout the caving process.

16.4.6 Fragmentation

In the case of post-undercutting, early stages of fragmentation can be managed through drill and blast techniques, such as a high undercut. As shown in Figure 16.28, productivity initially increases as blasted material flows from the undercut level (Category 1), followed by a decrease as primary fragmentation develops at the base of the caved ore column (Category 2).

As mining progresses, secondary fragmentation becomes more prevalent (Category 3), reducing large blockages and increasing fines in the mature stage (Category 4). For secondary fragmentation, the choice between post- or advance-undercutting does not play a significant role. This analysis is specific to conventional caving methods and does not consider advanced undercut.

Figure 16.28 Production performance during fragmentation stages (Cuello & Newcombe, 2018).



To estimate fragmentation, the Block Cave Fragmentation (BCF) software is utilized, representing industry standards. This program integrates both analytical and empirical methodologies that describe the key processes and factors influencing expected fragmentation outcomes.

Primary fragmentation at the Cortadera mine is anticipated to yield uniform block sizes across most geotechnical domains, with P80 sizes ranging from 0.88 to 3.39 meters. Secondary fragmentation analysis indicates that block sizes decrease as the height of draw increases, facilitating manageable sizes for material handling. Despite the mine's high rock mass strength, its impact on the ramp-up process is mitigated by the absence of ore passes, as the tipping points are designed at the same elevation as the extraction level, along with the adoption of high post-undercut caving techniques and the crusher's high capacity.

16.4.7 Ground support

The ground support assessment for caving infrastructure inside the footprint, including undercut and extraction level excavations, and for the crusher chambers located outside the footprint was conducted using benchmarking of caving operations. For declines, GMT employed the empirical Q-System, which has been adapted for mining applications by Potvin et al. (2015). The benchmarking for the undercut and extraction levels involved collecting data from 17 mines across 43 productive sectors, encompassing rock quality, in situ stress, drift size and spacing, undercut methodologies, and ground support types. Additionally, the benchmarking for the crusher chambers drew on data from seven different caving mines.

16.4.8 Underground Mining Limits Optimisation

16.4.8.1 Economic Parameters

The following section summarises the economic parameters used for the underground block cave optimisation studies.

To capture the combined contribution of each metal in this polymetallic deposit a net smelter return (NSR $/t) was calculated for each block considering individual metal grades, metal prices, metal recoveries and costs. Only the metal and revenue contribution of Indicated classification blocks in the Mineral Resource model were considered. Metal and revenue contribution from lower confidence classes were set to zero, treating it as internal waste.

Processing parameters assumed the block cave will be processed through the Productora sulphide plant which will be established for the open pit operations and includes costs to transport the material from the block cave to the process plant.

Detailed first-principals cost estimate completed based on final design, development and production schedules, equipment and labour schedules.

16.4.8.2 Optimisation Methodology

The Cuerpo 3 underground block cave potential was studied in two stages using Geovia PCBC™ caving software. In Stage 1 Geovia PCBC™ caving software's Footprint Finder module (PCBC-FF) was used as a global optimisation tool to confirm the PEA results and to determine the cave's optimal extraction level elevation range, approximate geometry and sensitivity to a range of typical caving design and schedule parameters. The findings and conclusions from this strategic optimisation output stage are then used Stage 2 using Geovia PCBC™ caving software's detailed design and production scheduler (PCBC) to produce the final mine design and schedule.

Both PCBC-FF and PCBC considers the footprint establishment cost and operating cost confined to the footprint boundary when optimising the footprint and comparing results. It does not consider capital cost outside the footprint, such as access from surface or fixed infrastructure such as underground crushers, conveyors, workshops and primary ventilation. It should therefore be considered a relative discounted value to compare results instead of an absolute discounted value. An absolute discounted value will be produced when combining the overall underground design and schedule with that of the open pits in a combined site, which is outside the scope of PCBC.

The generic steps taken to define the cave footprint and schedule were:

1. Populate NSR $/t in block model
2. Deplete open pit shell from block model to avoid double counting
3. In PCBC-FF, using block model as the basis of design for Stage 1 Optimisation:
   a. Regularise block model to represent approximate draw column geometries
   b. Run PCBC-FF to test sensitivity to a range of typical cave variables, and select optimal range for further refinement in PCBC
4. In PCBC, using drawpoints as the basis of design for Stage 2 detailed design and schedule:
   a. Only regularise the model vertical dimension increments as PCBC constructs draw cones to represent the lateral dimensions based on each drawpoints reference point. Regularising in the mZ dimension allows the use of PCBC's default mixing algorithm settings which has been calibrated for standard increments in multiple previous studies
   b. Use default value for parameters where PCBC-FF showed low sensitivity, and test ranges where higher variability were found
   c. Also test typical ranges for PCBC specific parameters, i.e. parameters that cannot be tested using the more strategic PCBC-FF method
   d. Confirm final footprint layout and produce production schedules
5. In Datamine (or similar) design access and supporting infrastructure located outside the cave footprint
6. In Datamine (or similar) schedule tunnel development to enable the PCBC production schedule, and to tie into global schedule constraints, such as interaction with the process plant and open pit.
7. Produce detailed cost models for underground capital and operating costs and combined with overall site schedule (external to PCBC).

16.4.8.3 Stage 1: Footprint Finder optimisation

Footprint Finder process overview

During Stage 1 cave optimisation in PCBC-FF, the Cuerpo 3 RF1 open pit material was considered to be mined by the open pit prior to caving and was removed from the block model. The block model was then regularised to approximate eventual drawpoint dimensions using 20 m cell sizes.

PCBC-FF considers a potential block cave layout (footprint) within a user-defined range of vertical elevations (footprint elevations). Typically, a potential block cave footprint will be evaluated on each vertical level increment of the Resource model, i.e. every 20 m elevation for the regularised block model. For each footprint elevation, pre-vertical mixing is applied to each column in the using Laubscher's mixing algorithm. This process assumes unconstrained vertical caving and provides acceptable results for initial optimisation and testing sensitivity ranges, especially for caves where good cave propagation is expected either under natural caving or through assistance using cave preconditioning such as hydrofracturing.

Each pre-mixed column is then assessed to determine the best economical height of draw (BHOD), within a set of user defined constraints, selected to model operational cave constraining factors. Both the contained economical value, and discounted economical value of each column is assessed. It should be noted that the economic value is a relative value and only includes footprint establishment costs and operating costs and excludes costs outside the cave footprint, such as access from surface to the footprint elevation, surface infrastructure or underground crushers, which is assumed to remain relatively constant between different footprints elevations and will be refined for the final footprint selected.

The contained economical value is determined for each potential column height above the footprint elevation by accumulating the contained value (tonnes × NSR $/t) and then subtracting the fixed cost to establish the extraction level and undercut level (footprint cost), and total operating cost (tonnes × unit operating cost). PCBC-FF will determine the height at which the maximum column value is obtained (BHOD) to account for scenarios where it is required to mine through sub-economical material lower in the column to reach more economical material higher up in the column. The column value is also discounted based on the time required to extract the material, based on the undercutting sequence and vertical height in the column, to account for the time value of money at the selected discount factor. During this process, a basic cave schedule is created, based on the undercutting rate specified, maximum individual column vertical mining rate, overall footprint maximum tonnes ramp-up rate, and sequence of extraction. Isolated columns (clusters with three or less isolated columns) are automatically removed as these cannot be practically extracted in a real block cave footprint as they are too small. The footprint value (dollar, tonnes and grades) is recorded for each footprint on each elevation.

This process is repeated for each footprint elevation between two user defined elevations, enabling the user to review the results and assess the footprint outputs and sensitivity to the input assumptions provided. To ensure practical footprint geometries and caving geometries, the user can force mining of columns to a minimum height of draw within a user defined footprint layout.

The default PCBC-FF parameters were selected based on the PEA results. For each parameter tested, all other parameters were kept constant to understand the impact to the results when changing only that specific parameter. Based on the results, the default settings were then reviewed to create a final base case.

Footprint Finder parameters tested

The model was analysed in PCBC-FF software to define the following key block cave parameters:

- Best/most economical footprint outline or geometry
- Extraction level elevation with the highest economical value and tonnes recovered
- The most economical column heights (height of draw) to be mined
- An indicative schedule of draw
- Confirmation of default values for inputs to PCBC detailed cave design and schedule, and ranges for values to be tested further in PCBC.

PCBC-FF provides a simplistic schedule and financial evaluation of the results, and these should be treated as relative values to compare between different scenarios rather than absolute values. During Stage 2 of this study detailed mine design and scheduling and first principles cost modelling and financial evaluation was used to further refine these results and to determine the expected cave value.

Table 16.11 shows the parameters tested in PCBC-FF, and the ranges tested for each parameter. The parameters were sourced from internal project reference and personal experience. The minimum and maximum values are generally a 30% sensitivity case to the selected base case, with low and high values at 15%. The minimum value generally corresponds to settings that will result in the lowest footprint tonnes mined and / or the lowest NPV achieved (typically because of higher costs, lower grades, lower tonnes, or longer time to mine, or a combination thereof) and vice versa. In total, 14 typical cave parameters were tested, and for each five values were selected (minimum to maximum).

Mining sequence or azimuth in which the undercut front progresses through the footprint does not have a typical minimum / maximum criterion, and therefore mining sequence was tested at 22.5° azimuth increments. PCBC-FF provides a theoretical maximum and minimum NPV mining sequence by sequencing blocks in order of highest value (maximum) or lowest value (minimum), providing a theoretical best and worst sequence – this is a theoretical sequence as block caving cannot selectively mine individual blocks and have to extract the ore as a continues front, but similar to Whittle optimisations minimum and maximum sequences for open pit mining, it provides a theoretical maximum and minimum to compare other sequences to. This produced a total of 83 individual PCBC-FF output schedules.

Table 16.11 Footprint Finder parameters and ranges tested (Base Case in bold)

Parameter	Description - primary area impacted	Units	Min	Low	Base	High	Max
Sequence *	Grade sequence - grade over time, cashflow and NPV	deg	Worst	22.5° azi increments **157.5° azi base case**			Best
Cut off value	Economic cut-off - tonnes mined, maximum rate, costs	$/t opex	20.28	17.94	**15.6**	13.26	10.92
Footprint cost	Establishment cost - start-up cost and footprint size	$/t capex	3,500	3,000	**2,500**	2,250	2,000
Max Height Mined	Maximum mineable column height - tonnes recovered, maximum rate	m Max	200	300	**400**	500	600
Min Height Mined	Minimum mineable column height - planned dilution mined	m Min	240	200	**160**	120	80
Max Draw Rate	Maximum mining rate - metal produced, cashflow and NPV	Mt/yr	12.5	15	**17.5**	20	22.5
Ramp-up duration	Duration to maximum rate - cashflow and NPV	yrs	8	7	**6**	5	4
Undercut rate	Maximum undercutting rate - cashflow and NPV	m^2/yr	20,000	25,200	**30,000**	35,200	40,000
Height of Interaction	Dilution - cashflow, tonnes, NPV	m	220	200	**180**	160	140
First Dilution Entry	Dilution - cash flow, tonnes, NPV	%	78	69	**60**	51	42
Discount Rate	Discount rate - NPV	%/yr	15	12.5	**10**	7.5	5

Table 16.11 Footprint Finder parameters and ranges tested (Base Case in bold)							
Parameter	**Description - primary area impacted**	**Units**	**Min**	**Low**	**Base**	**High**	**Max**
Minimum Tonnes	Minimum tonnes per period - dilution, NPV	t/yr	36,500	27,375	**18,250**	9,125	0
Cone Height	Height to full cone overlap - tonnes, cash flow, NPV	m	125	100	**75**	50	25

* Note: Best and Worst are theoretical sequences, extracting columns in order of highest or lowest column value to test what a theoretical, although not practical, mining sequence would deliver and how the chosen sequence compare to these theoretical maximum and minimum boundaries.

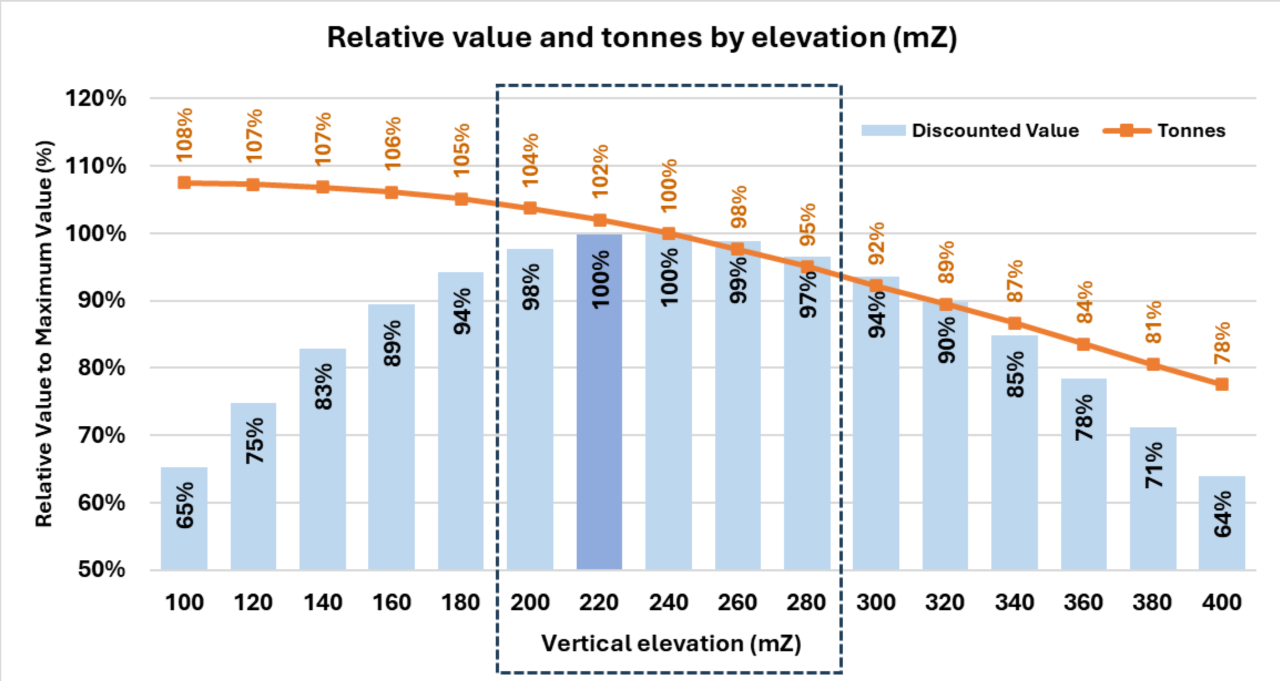
Footprint Finder Results - Value per vertical footprint elevation increment

Figure 16.29 shows the results for one of the 83 PCBC-FF schedules which were selected as the Base Case settings as highlighted in bold in Table 16.11 above. It shows footprint tonnes and relative discounted value for each 20 m vertical increment (100 mRL to 400 mRL) expressed as a relative percentage to the maximum elevation tonnes and discounted value respectively for a maximum 400 m height of draw.

The footprint results are not sensitive to the exact final elevation selected, showing less than 3% relative value difference over an 80 m vertical range (200 mRL to 280 mRL highlighted), and about 6% difference over 120 m vertical range (180 mRL to 300 mRL). The maximum relative value is returned for a footprint located on 220 mRL to 240 mRL elevation. The deeper, or lower elevation of 220 mRL extracts 2% more tonnes to achieve the same relative discounted value and was selected as the preferred Base Case extraction level floor elevation as it allows slightly more value for the same footprint should draw be increased to 500 m and allows some vertical increase for when detailed drainage and dip inclination is designed into the final footprint tunnels.

The elevation at which the maximum footprint value is achieved for each parameter, and the corresponding tonnes and discounted value at that elevation, is recorded for all 83 scenarios tested and is compared in Figure 16.35 and Figure 16.36.

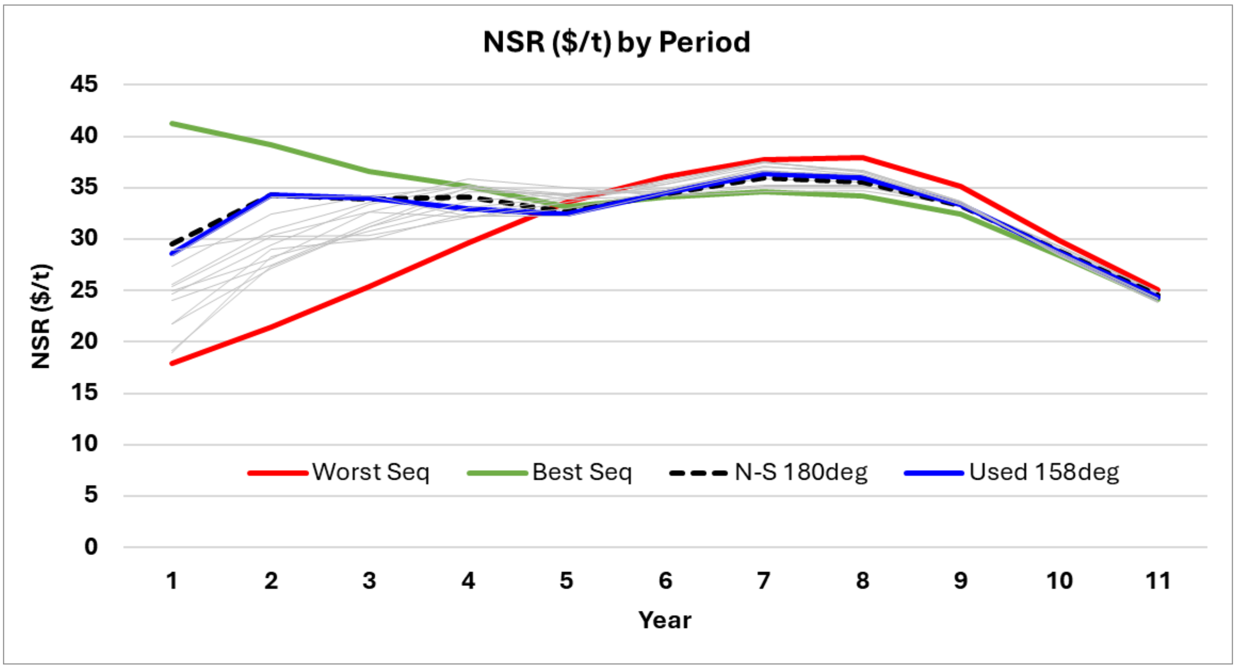
Figure 16.29 : PCBC-FF results example – relative tonnes and discounted value versus footprint elevation (mRL)



Footprint Finder Results - Undercut orientation and face length

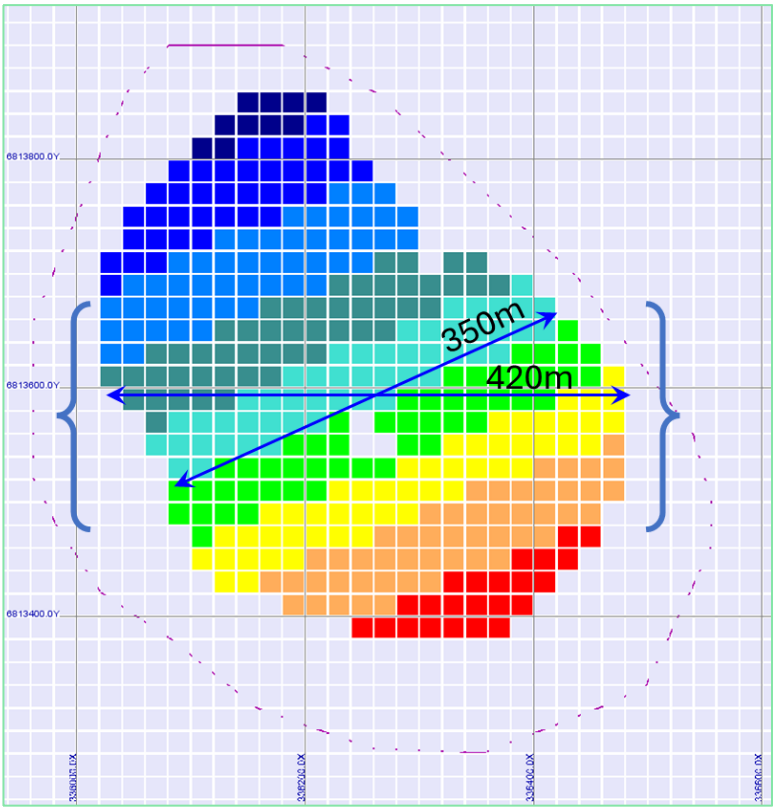
Undercut face advance direction, or azimuth was tested in 22.5° increments, and also the best and worst sequence (PCBC-FF highest column and lowest column values ranges). The Geotech recommended sequence is north to south, or 180 degrees (North = 0 degrees). The selected base case sequence is 157.5 degrees (rounded to 158 degrees for labels), as shown by the blue line. This minor change is driven by the maximum undercut face length. It has very little impact on overall discounted value as it only has a material change in grade in first year when tonnes extracted are very low from establishing the cave, and from the second year onwards it is very similar to best sequence grade, as shown in Figure 16.30.

Figure 16.30 : Annual NSR vs Undercut sequence



The Geotechnical recommended sequence starts in the north and progress south, at an azimuth of 180 degrees, results in an undercut face length of up to 420 m for the zone highlighted in { } brackets in Figure 16.31. The decision was made to rotate the direction 22.5 degrees east of south, at an azimuth of 157.5 degrees (referenced as 158 degrees for simplicity). This reduces the maximum undercut face length to about 350 m, which has been shown as a manageable face length in various caves and remains constant for most of the undercut duration. It also allows the drawpoints to be orientated near perpendicular with the advancing undercut abutment front, avoiding any drives being orientated parallel with the abutment stress.

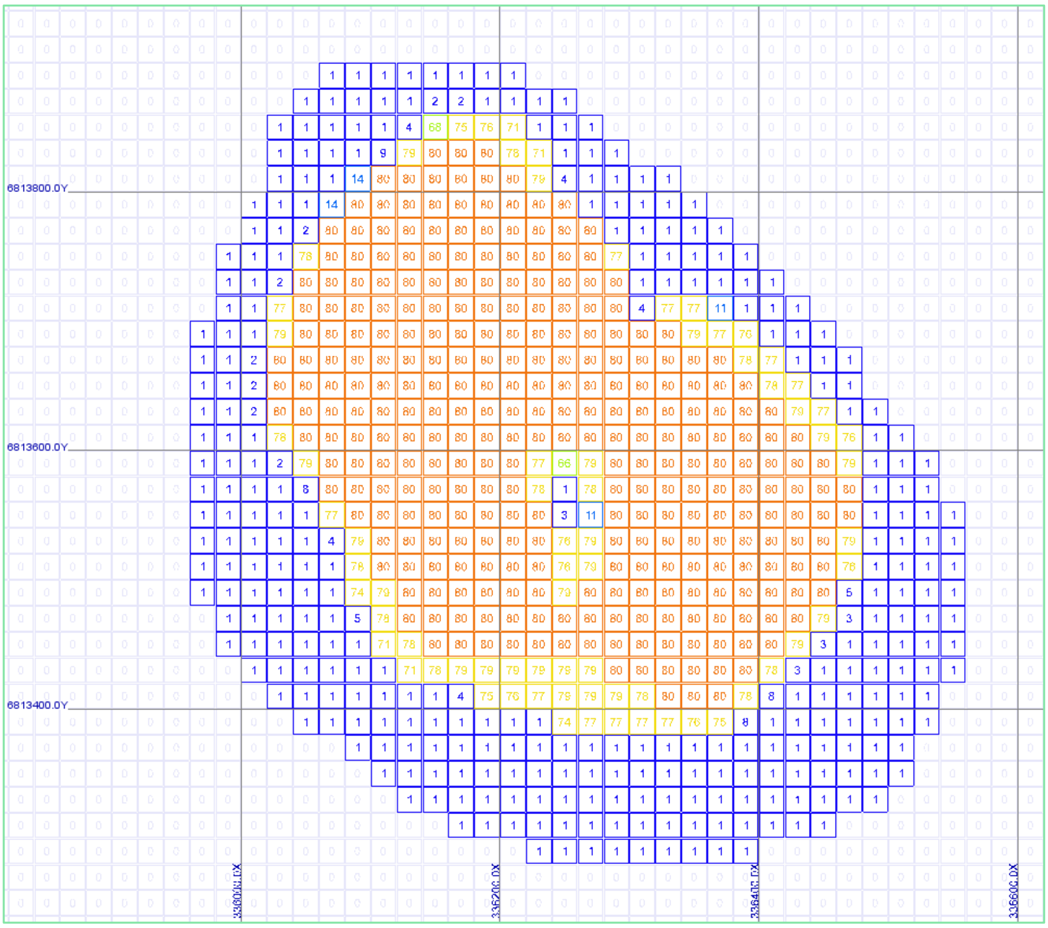
Figure 16.31 : Undercut sequence



16.4.8.4 Footprint Finder Results - Footprint boundary sensitivity

Figure 16.32 shows, for each block in the block model, the number of times that block was selected as part of the PCBC-FF selected footprint. A value of one would only have been considered in the maximum-case scenario, and a value of 83 would be included in all scenarios tested. There is a very sharp boundary of one to two blocks (20 m to 40 m) between blocks being selected for all scenarios to blocks only selected in the maximum case. This shows a very stable outer boundary of the cave footprint and provides a high level of confidence that permanent infrastructure designed outside the final footprint is not likely to be impacted by changes to the footprint resulting from updated design parameters.

Figure 16.32 : Footprint boundary sensitivity – number of times included in selected footprint



In addition to the above, it was also determined at what NSR value a block is first included in the selected footprint. For this, a maximum height of draw of 400 m was used, as per the base case assumption. Revenue factors (RF) between 50% and 150% (in 5% increments) were applied to the NSR field to represent potential changes in metal prices. This approach mimics the revenue factor approach used in open pit optimisation. For each revenue factor a PCBC-FF optimisation was then completed using the remaining base case parameters. This process was also completed using a 600 m maximum height of draw.

In Figure 16.33, the blue boundary outline shows the blocks included for the selected base case parameters, and each block shows the minimum revenue factor at which a block is first included in the footprint with 100 being the base case NSR assumption.

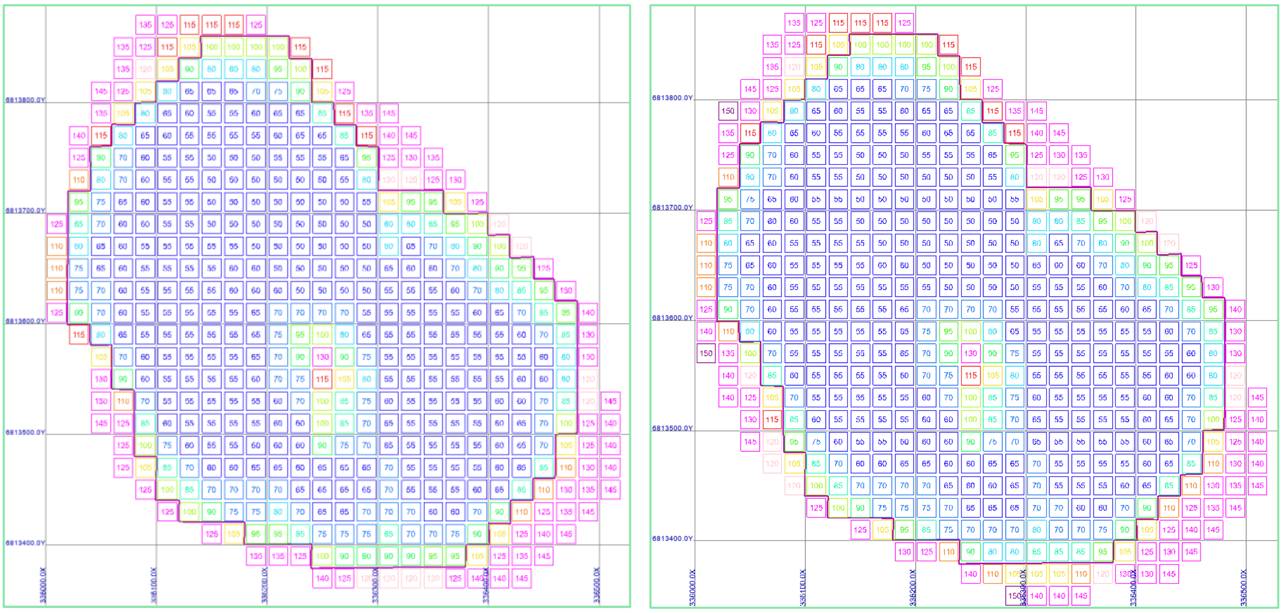
The blue core of the footprint remains selected even at a revenue factor of 70% (30% reduction in metal prices). There is a small low-grade centre of about three blocks that require a revenue factor of 105 to 130 respectively to be economical. Note this zone was forced to mine at least 200 m height of draw in PCBC to ensure full caving of the surrounding high-grade zone.

The hotter colours (red, magenta) outside boundary shows that the revenue factor can increase by 40% (RF=140) before the footprint boundary changes materially, confirming a very stable footprint layout using the

400 m base case assumptions. Most of the potential extension at very high metal prices is along the north-west to south-east axis, and these areas will be avoided in the placement of major long-term infrastructure, allowing the footprint to be expanded laterally on those direction should metal prices increase that high. The crushers are planned to the north-east where the footprint remain very stable regardless of revenue factor selected.

The image on the right shows the same result using a maximum height of draw of 600 m, and the footprint remains almost identical to the 400 m outline even at higher revenue factors, confirming the 400 m footprint will recover most of the potential upside that a 600 m height of draw will provide at higher metal price, without having to change the footprint outline, making the footprint very robust for potential future metal prices increases.

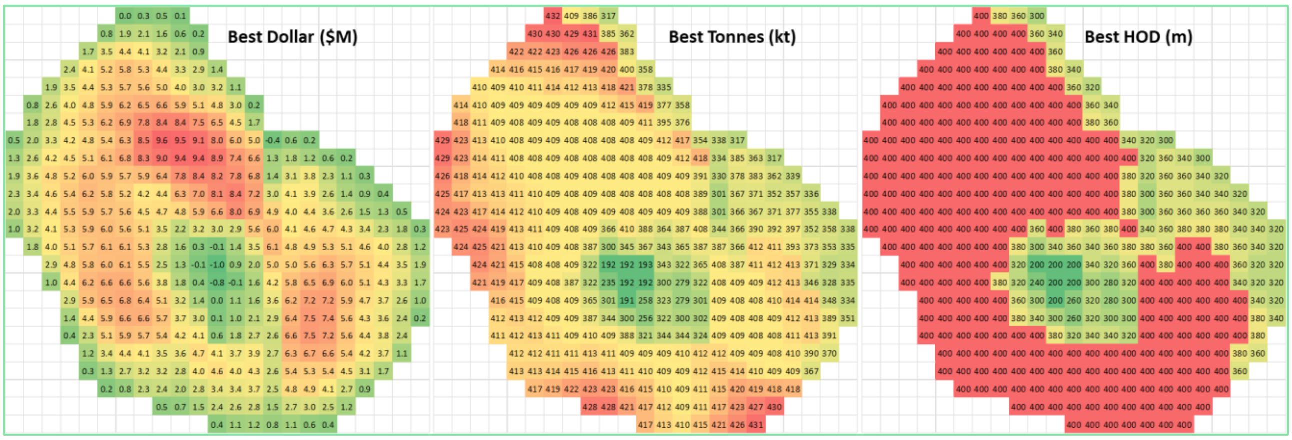
Figure 16.33 : Footprint boundary sensitivity – required revenue factor to use column (400m left, 600m right)



16.4.8.5 Footprint Finder Results - Individual column results

Figure 16.34 shows an example of the best dollar, best tonnes and best HOD reported for each column in PCBC-FF. The best dollar is the net revenue value for each column when mined to the best height of draw, or the equivalent height that provides the maximum net revenue for that column. The best tonnes are the tonnes mined for each column when mined to the best HOD. Included in the PCBC-FF optimisation is a maximum height constraint of 400 m (most of the footprint), with areas of low value forced to mine to a minimum height of 200 m.

Figure 16.34 : Column results shown for 400 m Base Case



16.4.8.5.1 Footprint Finder Results Summary

For each scenario, the footprint elevation on which the maximum discounted value was achieved was recorded. As shown in Figure 16.29, the maximum discounted value and maximum tonnes are generally found on the same vertical elevation or are within one vertical increment (20 mRL) of each other. For that reason, only the maximum value elevation was recorded.

Figure 16.35 shows the relative discounted value of each scenario to the maximum of all 83 scenarios tested, providing comparison of each parameter to the overall highest value achieved. Figure 16.36 shows the relative value of each scenario compared to the maximum value within each group of parameters tested, providing a better resolution on the sensitivity to changes to that specific parameter.

For each scenario considered, the tonnes and discounted value for the footprint elevation that produced the maximum discounted value are presented in Figure 16.35 and Figure 16.36. As shown in Figure 16.29 the elevation on which the maximum discounted value and maximum tonnes are achieved respectively are generally the same or within one block model increment (20 m) from each other with differences in tonnes being less than 2% for almost identical discounted values. For this reason, only the maximum value scenarios are shown.

In the first figure, the values for each scenario are compared to the overall maximum scenario out of all scenarios tested to determine a holistic sensitivity between parameters. In the second figure each value is compared to the highest value within each parameter tested, to compare sensitivities between minimum and maximum values within each parameter.

The Base Case used a maximum height of draw of 400 m and is highlighted in each figure for reference

Figure 16.35 : PCBC-FF result summary – parameter shown relative to maximum value for all 83 tests

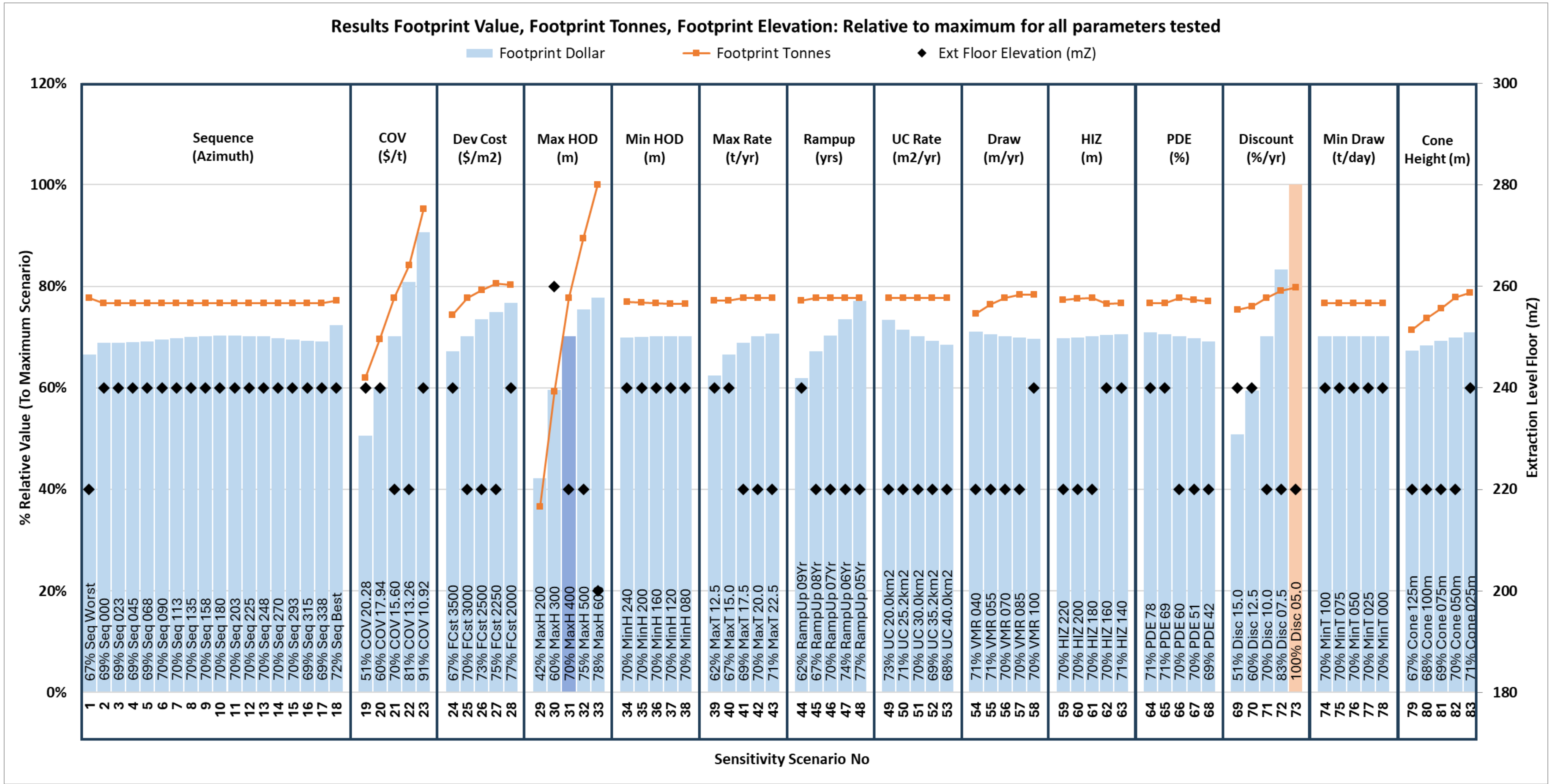
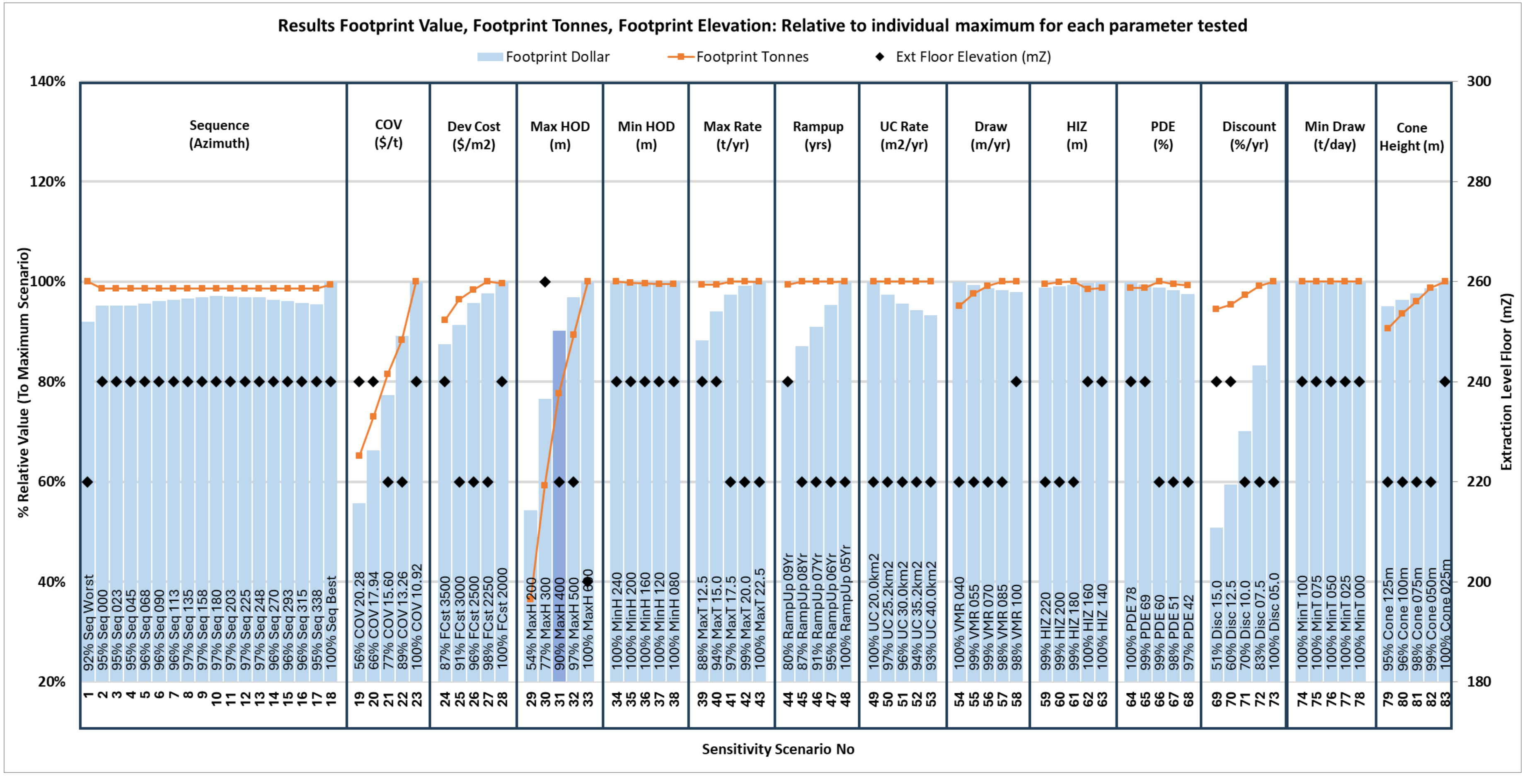


Figure 16.36 : PCBC-FF result summary – parameter shown relative to maximum value for each parameter tested



16.4.8.5.2 Footprint Finder Sensitivity Discussion

Top five sensitivities from the Footprint Finder results were, in order of maximum downside risk:

- Maximum height of draw: The downside risk at lower draw height is due to leaving potential processing feed unmined. The results are more sensitive for tonnes as the bulk of the higher-grade core remains included for all scenarios, retaining most of the value. The upside is less sensitive as most of the higher-grade material is already extracted at 400 m, leaving less upside to be recovered economically for higher columns drawn. The Base Case of 400 m is proposed for detailed work in PCBC, with 500 m as an alternative.
- Cut-off value (Operating cost): The cut-off value is almost equally sensitive to the upside and downside. Changes impact value more than tonnes as it is deducted from all tonnes for value (operating cost) but only removes tonnes where it falls below the cut-off operating cost. Tonnes for material above the maximum cut-off is thus not affected. This is modelled in more detail in PCBC.
- Discount rate: The annual discount rate impacts long-life projects more due to the longer time frame being discounted over. Long time frames are typical for most block caves; therefore, caution should be exercised when using NPV as the only factor. For this footprint there is very little impact on the actual tonnes mined, it is mostly the value of future material mined discounted to today's terms that are impacted. This is modelled in much more detail in PCBC and through a detailed financial valuation and PCBC-FF results should be treated as indicative, or relative only.
- Ramp-up duration: Similarly to the annual discount rate, the ramp-up duration impacts long life projects more due to the longer time frame being discounted over and a slower ramp-up further extending the mine life. This is modelled in much more detail in PCBC and PCBC-FF results should be treated as indicative, or relative only.
- Maximum draw rate: Similarly to the annual discount rate, the maximum annual mining rate impacts long life projects more due to the longer time frame being discounted over and a slower mining rate further extending the mine life. This is modelled in much more detail in PCBC and PCBC-FF results should be treated as indicative, or relative only. The impact on overall cashflow and processing is also not modelled directly in PCBC and requires detailed modelling in a financial valuation model.

16.4.8.6 Stage 2: PCBC optimisation

During Stage 2, detailed cave design and scheduling was performed using industry standard Geovia PCBC software. PCBC uses a detailed block cave footprint design layout with individual drawpoints, and a draw column constructed for each drawpoint, referred to as a slicefile. The material is depleted from each drawpoint in a production schedule according to a defined schedule parameters and constraints that emulates the anticipated real life cave operation and material flow.

To create the PCBC slicefile the resource model was only regularised in the vertical dimension (Z) to 20 m cell sizes. This was done to allow PCBC to utilise the raw block model X and Y dimensions when creating draw cones and using the industry standard 20 m vertical height increments for which PCBC's material flow and mixing algorithms were calibrated. PCBC allows much more detailed analysis to be performed than PCBC-FF on all cave related aspects, such as footprint layout design, vertical draw column geometry, material movement mechanisms, production sequence, schedule rates and shut-off determination, and is considered block cave industry standard for PFS, FS and operational production schedules.

Except for scenarios testing the sensitivity to that specific parameter, the following parameters were used for the base case value when testing other parameters (Table 16.12). The footprint establishment cost were increased from \$2,500/m^2 in PCBC-FF to \$3,000/m^2 based on internal discussions with South American and America caving operators regarding current experience and costs references.

Table 16.12 PCBC parameters used for the base case value when testing other parameters

Parameter	Value
Maximum Production Rate	20 Mtpa
Cut-off Value	\$15.60/t
Footprint establishment cost	\$3,000/m^2 (increased from PCBC-FF's \$2,500m^2)
Extraction Level Elevation	220 mRL
Maximum height of draw	400 m
Minimum height of draw	200 m
Sequence	158°
Undercut Rate	Averaging 3,000 m^2 per month for the maximum 350 m front length[1]
Cave Angles	Vertical[2]

[1] This is reduced when the undercut front length reduced, to model a constant front advance rate rather than a constant area in each period, since using a constant area can often overestimate the true undercut rate on short fronts

[2] Based on ground conditions and extensive high undercut blast and hydraulic fracturing preconditioning of the cave envelope

The base case for each comparison is highlighted in bold in the tables and shown in a wider line series in the charts in the following detailed sections.

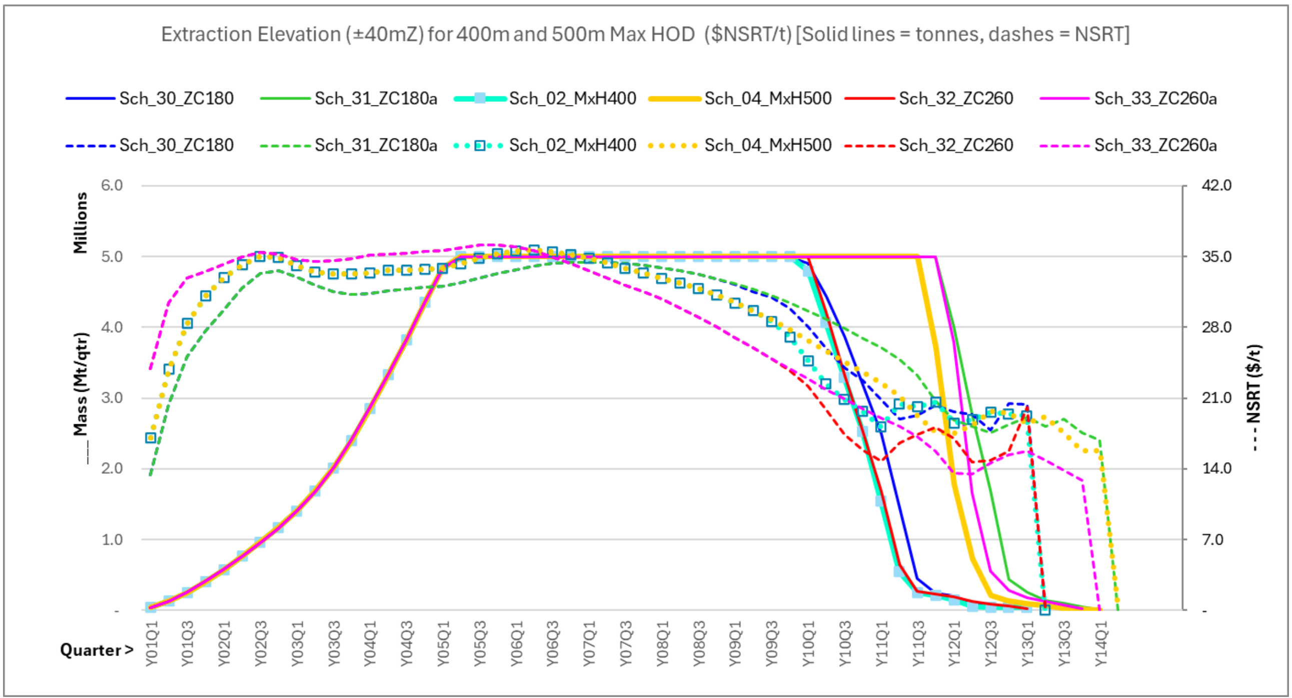
16.4.8.6.1 Extraction level elevation

PCBC-FF indicated a very low sensitivity to the preferred extraction elevation of 220 mRL. To verify this a detailed cave design was created 40 m above (260 mRL) and 40 m below (180 mRL) the preferred elevation respectively. Table 16.13 shows the results for each scenario.
Figure 16.37 shows the production tonnes and NSR \$/t schedules for both the 180 mRL and 260 mRL footprints relative to the base case 220 mRL footprint ("MxH400" and "MxH500" for 400 m and 500 m maximum height of draw base schedules respectively). Schedules "180a" and "260a" use a 500 m maximum height of draw instead of the base case 400 m height of draw.

Table 16.13: Sensitivity - Extraction level elevation (±40 m to base 220 mRL)

Scenario	Description	Tonnes (Mt)	Au (g/t)	Cu (%)	NSR (\$/t)	Relative Value (\$M)	Change %	Tonnes Var %	Value Var %
Sch_30_ZC180	400 m HOD, 40 mRL lower	147.5	0.15	0.44	31.56	980.6	-40 m	3%	-1%
Sch_02_MxH400	400 m HOD, base case	143.6	0.16	0.44	31.74	993.3	-	-	-
Sch_32_ZC260	400 m HOD, 40 mRL higher	144.3	0.16	0.43	30.66	944.4	+40 m	1%	-5%
Sch_31_ZC180a	500 m HOD, 40 mRL lower	175.5	0.15	0.43	30.52	1079.3	-40 m	5%	2%
Sch_04_MxH500	500 m HOD, base case	167.8	0.15	0.43	30.50	1063.2	-	-	-
Sch_33_ZC260a	500 m HOD, 40 mRL higher	172.7	0.15	0.41	28.68	977.7	+40 m	3%	-8%

Figure 16.37 : Sensitivity – Extraction level elevation (±40 m to base 220 mRL)



The following conclusions were drawn:

- The base case 220 mRL provides the highest relative value for a 400 m maximum height of draw and is used for future scenarios. The 180 mRL footprint provides marginally more discounted value (2%) at a 500 m maximum height of draw. The differences are relatively small and could marginally change with different assumptions in future iterations. It should be highlighted that the PCBC schedule is relative to the start of the first drawbell and therefore does not consider a start of operations reference point for determining discounted value – a lower elevation will take marginally longer with more discount and a shallower elevation faster to bring into production with slightly less discounting, which will slightly reduce the value for deeper levels and vice versa.
- Tonnes: the deeper 180 mRL footprint provides marginally more tonnes (3% total) by providing more tonnes relative to a deeper extraction level for some of the columns that did not reach the maximum height of draw restriction in the base case. The shallower 260 mRL footprint is almost identical.
- Value: The deeper 180 mRL footprint provides lower initial grades and higher grades later in the schedule as it must mine through more lower grade material at the start of the draw columns. The higher 260 mRL is the opposite, mining higher grade initially and lower grade later, but with a lower overall grade resulting in 5% less value for the 400 m and 8% less for the 500 m scenarios. This supports using the 220 mRL for the base case under the current assumptions.

16.4.8.6.2 Extraction level boundary

Similar to in PCBC-FF, PCBC can determine a best economical height of draw (BHOD), calculating the net value of each drawpoint in the footprint. This is used to determine the best footprint boundary and which drawpoints should be included or excluded from the final layout. When PCBC determines the BHOD, it applies an average vertical draw rate to the vertical column, and considers this for each drawpoint in isolation, providing a good starting point for defining the final footprint boundary.

A similar approach was used to generate the value plot per drawpoint below in Microsoft Excel, but instead the actual production schedule output from the PCBC production run was used – this allows the impact of all of the Template Mixing parameters, including cave back constraints and material rilling and toppling (PCBC material flow terminology) to be incorporated in the decision. This was considered a more accurate approach to verify the final footprint boundary, especially when connecting to the open pit above and incorporating the internal sub-economic zone shown in blue/green in Figure 16.38.

This figure shows the column value, in $M per drawpoint for the selected footprint. Boundary drawpoints are typically selected over $1,000,000 net value (revenue of incremental scheduled column tonnes and NSR per quarter less the footprint development cost to establish the drawpoint at $3,000/m^2). A few minor / marginal exceptions were included to provide a smoother footprint boundary. The central core shows seven drawpoints that break even or make a slight loss – these drawpoints were deliberate included to ensure the central core is drawn with the rest of the cave to ensure that the value contained in the surrounding drawpoints can be effectively recovered, and to avoid stress building in the centre of the footprint.

Figure 16.38 : Sensitivity – Footprint column value ($M)

				1.6	1.5	1.6		1.4																					
			2.1	2.8	2.5	2.7	2.2	2.2	1.8	2.1																			
		2.4	3.3	4.2	4.2	4.3	3.3	3.1	2.6	2.8	2.4	2.5																	
	1.8	3.1	4.5	5.3	5.5	5.5	4.6	4.5	3.8	4.2	3.5	3.8	2.5	1.5															
	2.7	3.6	5.0	5.6	6.1	6.3	6.3	6.6	6.1	6.7	5.4	6.0	4.2	3.9	1.4	0.9	1.5	1.8	1.9	1.9	1.3	1.3							
1.7	3.3	3.8	5.0	5.7	6.5	7.2	7.7	8.3	8.1	8.7	7.7	7.9	6.2	5.8	3.1	2.0	1.7	2.6	2.7	2.6	1.8	1.7	1.7	1.9	0.9	1.4			
1.9	3.2	3.9	4.9	5.7	6.8	7.4	8.4	9.1	9.2	9.5	8.8	8.8	7.6	7.2	5.1	3.8	2.1	3.1	3.3	3.3	2.7	2.7	2.1	2.5	2.0	2.1	1.9		
2.1	3.2	3.8	4.8	5.4	6.6	7.1	8.4	8.9	9.1	9.2	8.9	8.8	7.9	7.7	6.5	5.7	3.5	3.4	3.5	3.8	3.5	3.7	3.2	3.5	3.0	3.1	2.3	1.7	
2.2	3.0	3.7	4.6	5.0	6.3	7.0	7.9	8.0	8.2	7.9	8.1	7.9	7.7	7.7	6.9	6.3	4.6	4.2	3.7	4.0	4.1	4.4	4.1	4.2	3.7	3.6	2.7	2.4	
1.9	2.9	3.6	4.4	5.0	6.1	6.5	6.9	6.6	6.7	6.2	6.6	6.7	6.7	6.1	5.8	5.3	5.2	4.8	4.0	4.3	4.5	5.0	4.7	4.8	4.0	3.7	2.9	2.3	1.5
1.6	2.9	3.8	4.7	5.2	6.2	6.4	6.3	6.2	5.6	5.5	5.2	4.7	4.4	3.0	3.2	2.7	4.0	4.3	4.4	4.6	4.8	5.6	5.2	5.3	4.3	3.8	2.9	2.3	1.5
1.3	3.1	4.0	5.0	5.5	6.4	6.5	6.2	6.1	5.4	5.0	3.7	2.2	1.6	0.9	1.1	1.1	2.3	3.0	4.1	4.7	5.3	6.0	5.8	5.7	4.5	3.9	3.0	2.6	1.8
1.1	3.0	3.8	4.8	5.6	6.3	6.4	6.3	6.2	5.6	4.9	2.6	1.6	0.5	0.0	-0.3	0.0	1.2	1.9	3.3	4.2	5.5	6.4	6.2	6.0	4.8	4.4	3.4	3.1	2.4
0.4	2.7	3.5	4.8	5.4	6.3	6.5	6.4	6.4	6.0	5.3	2.8	1.8	0.6	0.2	-0.6	-0.4	0.6	1.5	2.4	3.5	5.2	6.4	6.5	6.5	5.1	4.7	3.6	3.1	2.6
	2.7	3.0	4.3	4.9	5.9	6.3	6.6	6.7	6.2	5.8	3.4	2.7	1.1	0.8	0.0	-0.1	0.4	1.1	1.8	2.5	4.6	6.0	6.6	6.5	5.5	4.9	3.5	2.9	2.0
	1.2	2.1	3.3	3.9	5.3	5.9	6.5	6.7	6.6	6.4	4.4	4.1	2.4	2.1	0.8	0.7	0.4	1.0	1.7	2.1	3.9	5.4	6.3	6.1	5.4	4.7	3.3	2.2	
			1.8	2.8	4.4	5.2	6.4	6.8	6.9	7.0	5.6	5.2	3.6	3.2	2.0	1.9	0.9	1.4	2.1	2.7	3.3	4.8	5.8	5.6	4.9	4.1	2.6	1.8	
					3.2	4.7	5.8	6.3	6.8	7.0	6.5	6.3	4.5	4.2	3.1	3.2	2.4	3.0	3.6	4.5	4.0	4.4	5.3	5.3	4.3	3.5	2.5		
							4.7	4.8	6.1	6.3	6.6	6.3	5.3	4.8	4.3	4.5	4.5	4.8	4.7	4.8	4.5	4.3	4.6	4.2	3.8	3.1	1.8		
							2.6	3.5	4.7	5.0	5.8	5.5	4.9	4.5	4.3	4.4	4.7	4.9	5.1	4.7	4.5	3.7	3.7	3.5	3.1				
									3.6	4.0	4.4	4.2	4.2	3.9	3.9	3.7	4.0	3.9	4.3	3.9	4.0	3.3	3.2						
											3.0	2.8	3.2	3.1	3.4	3.3	3.2	3.0	3.4	2.9	2.5								

N

$M
< 0
0-0.5
0.5-1.0
1.0-2.0
2.0-3.0
3.0-4.0
4.0-5.0
≥5.0

Note: Each block represents one drawpoint in the regular 32x18m El Teniente layout used

16.4.8.6.3 Minimum and Maximum height of draw

The minimum height of draw (HOD) is only specified to ensure sufficient tonnes are mined from sub-economic zones to ensure efficient caving. A minimum height of 200 m was selected to ensure the small low-grade post-mineral core is mined to at least 50% of the height of the surrounding high-grade columns, allowing for the entire footprint to be mined in a true block cave mining style.

In deposits where the maximum height of mineralisation exceeds the maximum HOD, the maximum HOD has a material impact on the tonnes that can be extracted as material is left behind unmined above the maximum HOD.

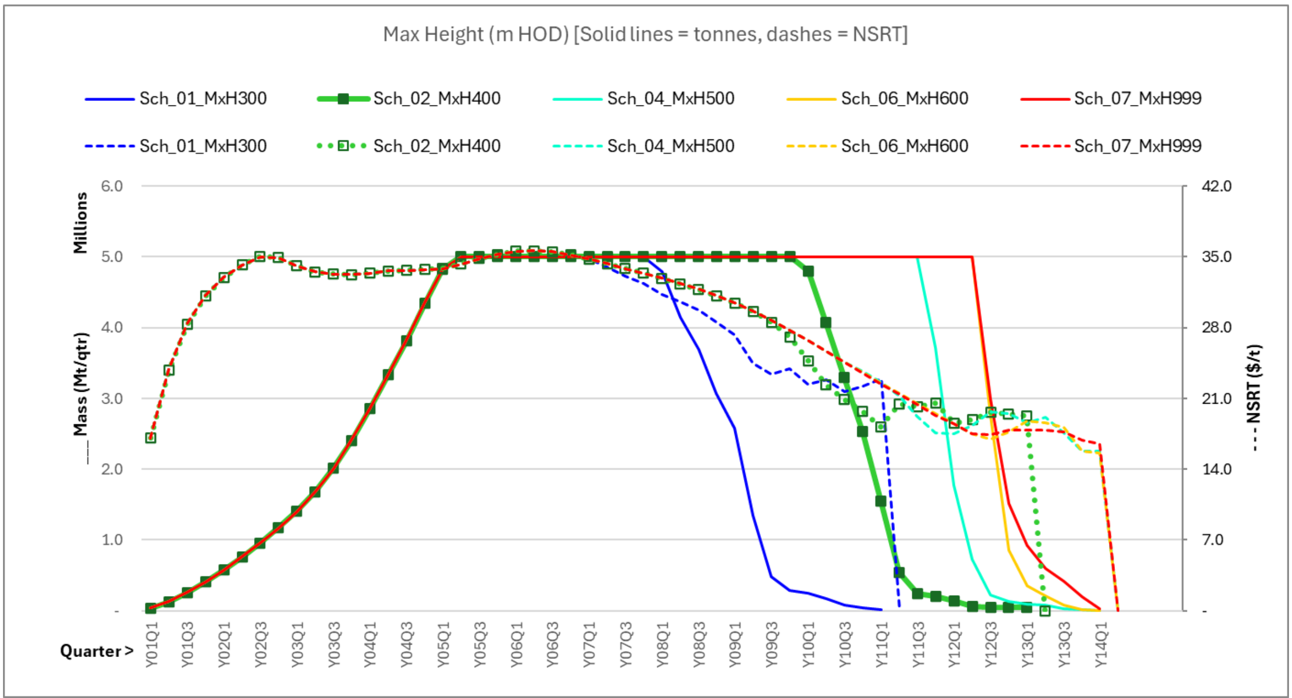
As seen in Table 16.14 and Figure 16.39, reducing the maximum height to 300 m has a material impact on the total tonnes (-26%) and value (-19%) compared to the base case 400 m maximum HOD. This is due to the 100 m column of material left unmined over most of the footprint. Increasing the maximum HOD to 500 m impacts less of the total footprint and only adds 17% tonnes for an extra 7% discounted value. As the maximum HOD is increased to 600 m, the tonnes increase a further 9% but only adds 1% value when compared to the 400 m base case. An unlimited maximum HOD restriction does not have a material change over the 600 m maximum HOD scenario, indicating that most of the draw columns are fully depleted at 600 m maximum height. This is important for two reasons:

- The open pit was removed from this scenario – should the open pit not be mined, a two-lift cave scenario should be investigated as two shorter lifts will improve the extraction of the deposit over a single 400 m lift
- Increasing the maximum HOD above 400 m adds disproportionately more tonnes than value, indicated that the extra material only marginally contributes to value. The recommendation is to retain the 400 m maximum HOD as the base case. The footprint and the initial production ramp-up are identical between the 400 m and 500 m maximum HOD scenarios and therefore if future metal prices and actual operational costs are favourable compared to current assumptions, the cave can continue to mine to 500 m maximum HOD using the same footprint.

Table 16.14: Sensitivity - Maximum height of draw

Scenario	Description	Tonnes (Mt)	Au (g/t)	Cu (%)	NSR ($/t)	Relative Value ($M)	Change %	Tonnes Var %	Value Var %
Sch_01_MxH300	300 m HOD, 220 mRL	106.9	0.16	0.46	33.21	802.1	-100 m	-26%	-19%
Sch_02_MxH400	400 m HOD, 220 mRL	143.6	0.16	0.44	31.74	993.3	-	-	-
Sch_04_MxH500	500 m HOD, 220 mRL	167.8	0.15	0.43	30.50	1063.2	+100 m	17%	7%
Sch_06_MxH600	600 m HOD, 220 mRL	180.3	0.15	0.42	29.70	1077.6	+200 m	26%	8%
Sch_07_MxH999	Unlimited HOD, 220 mRL	182.7	0.15	0.42	29.54	1078.6	+999 m	27%	9%

Figure 16.39 : Sensitivity – Maximum height of draw



16.4.8.6.4 Undercut rate

The undercut rate correlates to the number of new drawbells being installed and impacts the number of new drawpoints added to the production schedule in each period. The same 400 m maximum HOD base case was used for these scenarios. Small, immaterial changes are observed in total tonnes and average grades due to the non-linear mixing of material and the impact of change in undercutting rate on the installation of drawbells (and therefore on production ramp-up in the schedule), resulting in slightly different material mixing. This is considered immaterial, and the main conclusions are:

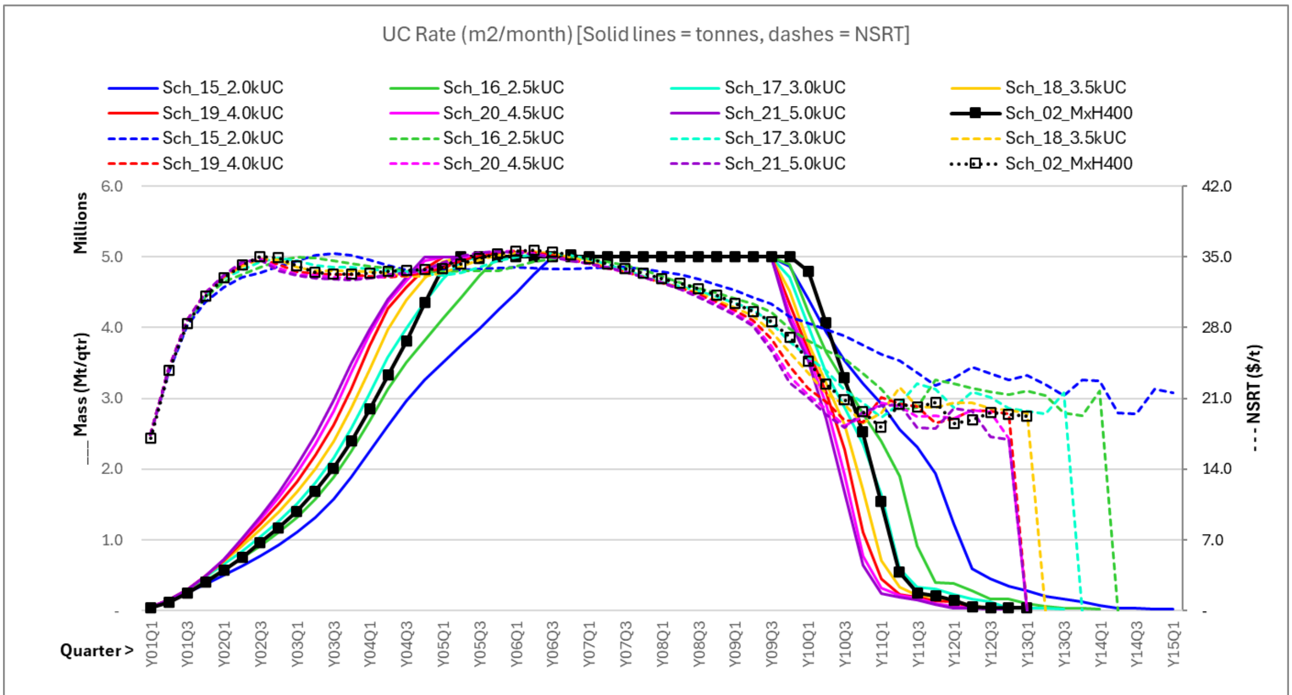
- Ramp-up profile – slower undercutting restricts the number of drawpoints added in each step, therefore restricting the production ramp-up. The impact is more pronounced at slower rates as undercutting and drawbell installation becomes the major constraint. The upside is less at higher rates as other production constraints (such as tunnel constraints) become the dominant constraint overriding the faster undercutting benefits. The use of faster undercutting rates also has other risks associated, with seismicity and increased congestion which is not modelled here but will further reduce the potential upside benefit
- Changes to undercutting impacts the drawbells installed in each period. This results in minor variations in grades mined per period as tonnes are mined from different areas in each schedule
- Increasing the undercutting rate over 3,500 m^2/month has very little benefit under the current base assumptions. This provides flexibility having more drawpoints available for production, but at increased upfront cost and little production upside as other production constraints prevents the cave from utilising the higher drawpoint capacity (for example maximum tunnel capacity).

The 400 m maximum HOD base case is similar to the 3,000 m2/month undercutting scenario, with minor changes during the first and last few periods of undercutting due to the geometry of the footprint. This equates to about 5.2 drawbells per month on a 32 x 18 m tunnel spacing. Sensitivities for each scenario are shown in Table 16.15 and Figure 16.40.

Table 16.15: Sensitivity - Undercut rate

Scenario	Description – base case of 32 x 18 m per drawbell	Tonnes (Mt)	Au (g/t)	Cu (%)	NSR ($/t)	Relative Value ($M)	Change %	Tonnes Var %	Value Var %
Sch_15_2.0kUC	2,000 m^2/month UC (3.5 bells)	143.7	0.16	0.44	31.68	954.4	-33%	0%	-4%
Sch_16_2.5kUC	2,500 m^2/month UC (4.3 bells)	143.4	0.16	0.44	31.73	984.4	-17%	0%	-1%
Sch_02_MxH400	~3,000 m^2/month base case	143.6	0.16	0.44	31.74	993.3	-	-	-
Sch_17_3.0kUC	3,000 m^2/month UC (5.2 bells)	143.3	0.16	0.44	31.75	1003.2	0%	0%	1%
Sch_18_3.5kUC	3,500 m^2/month UC (6.1 bells)	143.1	0.16	0.44	31.77	1014.8	17%	0%	2%
Sch_19_4.0kUC	4,000 m^2/month UC (6.9 bells)	143.4	0.16	0.44	31.76	1023.8	33%	0%	3%
Sch_20_4.5kUC	4,500 m^2/month UC (7.8 bells)	143.4	0.16	0.44	31.77	1029.2	50%	0%	4%
Sch_21_5.0kUC	5,000 m^2/month UC (8.7 bells)	143.3	0.16	0.44	31.78	1032.5	67%	0%	4%

Figure 16.40 : Sensitivity – Undercut rate



16.4.8.6.5 Maximum ramp-up rate

Production ramp-up in PCBC is determined primarily by the following parameters:

- Capacity:
  - Number of active drawpoints in each period – more drawpoints equate to more capacity (and vice versa)
  - Drawpoint production rate curve (PRC), or maximum tonnes per drawpoint per period versus the total cumulative tonnes mined from the drawpoint. The PRC is a combination of factors and represents the expected production rate of a drawpoint over time, considering maximum caving rate, expected delays during early production periods due to congestion from development and construction activities and oversize from initial cave primary fragmentation, and how these items change over the drawpoints production life
- Constraints such as maximum tonnes per tunnel or zone or in total in any period, which is typically related to the capacity of a loader or crusher or total materials handling system or ventilation system in each area in any period.

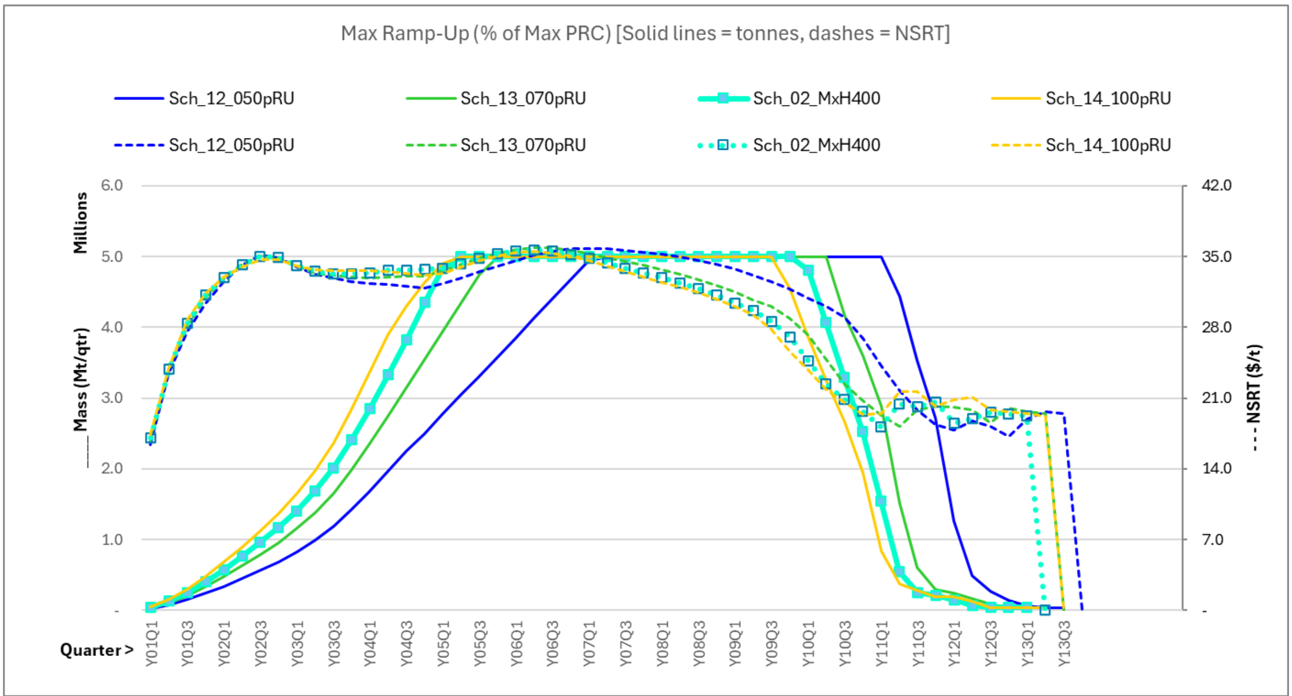
The maximum cave ramp-up rate is the minimum value of the capacity available from the drawpoints (number of drawpoints, PRC) and the constraints applicable to the overall system. A slower ramp-up defers production and therefore has a negative impact on discounted net present value, and vice versa.

Table 16.16 and Figure 16.41 show the impact of reducing the maximum ramp-up capacity in any period as a percentage of the maximum ramp-up capacity. The base case assumed 85% of the maximum ramp-up rate, which is based on a well-managed multi-staged system only achieving about 85% of the maximum theoretical capacity of the sum of the individual parts of the system. As with other ramp-up related sensitivities discussed in preceding paragraphs, the total tonnes are not materially impacted but the discounted value is impacted due to changes in the production profile over time.

Table 16.16: Sensitivity - Maximum ramp-up rate

Scenario	Description	Tonnes (Mt)	Au (g/t)	Cu (%)	NSR ($/t)	Relative Value ($M)	Change %	Tonnes Var %	Value Var %
Sch_12_050pRU	50% of maximum	143.0	0.16	0.45	31.81	880.1	-41%	0%	-11%
Sch_13_070pRU	70% of maximum	143.2	0.16	0.45	31.78	957.7	-18%	0%	-4%
Sch_02_MxH400	~85% of maximum (Base Case)	143.6	0.16	0.44	31.74	993.3	-	-	-
Sch_14_100pRU	100% of maximum	143.4	0.16	0.44	31.75	1013.3	18%	0%	2%

Figure 16.41 : Sensitivity – Maximum ramp-up rate



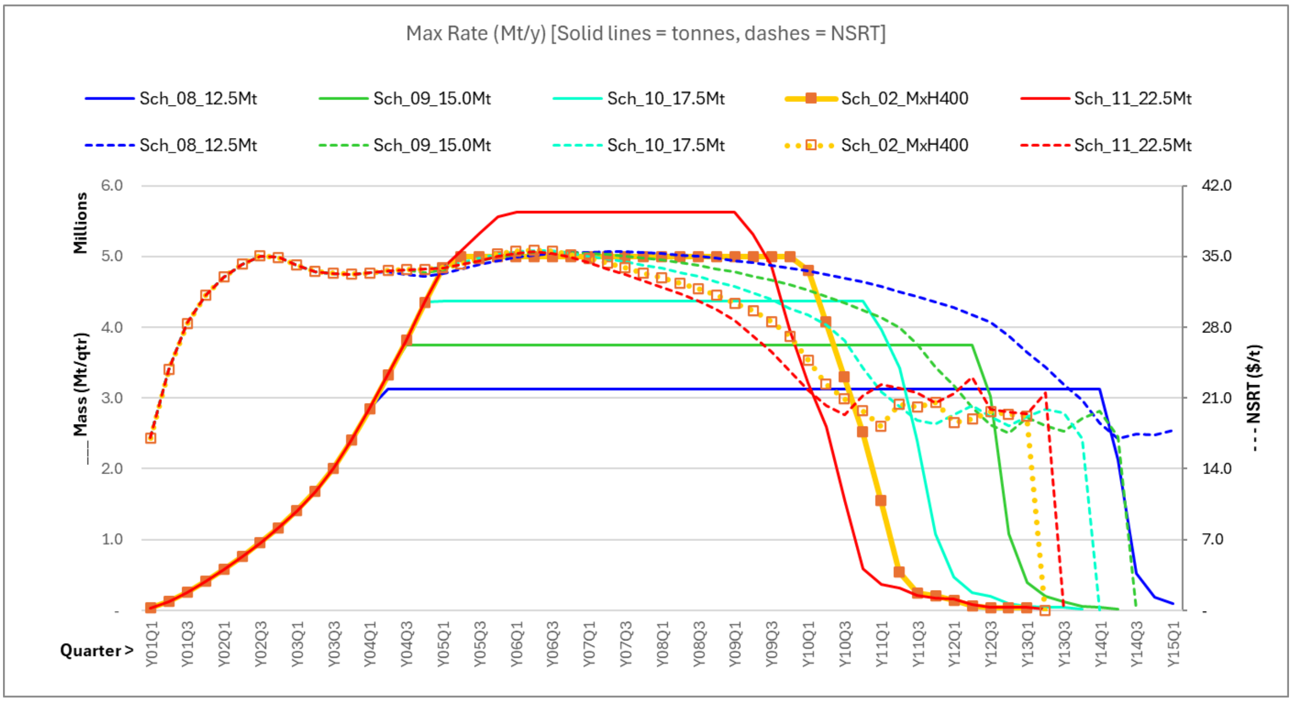
16.4.8.6.6 Maximum annual production rate

The cave footprint is of sufficient area (150,000 m^2 or almost 500 drawpoints at 32 x 18 m tunnel spacing) and draw columns sufficiently high (400 m to 500 m) to provide high-capacity production rates over multiple years. Production rates more than 20 Mtpa are restricted by the overland materials handling system and process plant and therefore only one scenario higher than 20 Mtpa was tested. Like previous sensitivities impacting throughput, reducing the total production rate constrains the throughput and therefore extends the mine life, extracting similar tonnes over a longer period and reducing discounted value. Sensitivities are shown in Table 16.17 and Figure 16.42 below.

Table 16.17: Sensitivity - Maximum annual production rate

Scenario	Description	Tonnes (Mt)	Au (g/t)	Cu (%)	NSR ($/t)	Relative Value ($M)	Change %	Tonnes Var %	Value Var %
Sch_08_12.5Mt	12.5Mtpa	142.7	0.16	0.45	31.80	860.1	-38%	-1%	-13%
Sch_09_15.0Mt	15.0Mtpa	142.9	0.16	0.45	31.79	921.8	-25%	0%	-7%
Sch_10_17.5Mt	17.5Mtpa	143.2	0.16	0.44	31.76	964.5	-13%	0%	-3%
Sch_02_MxH400	20.0Mtpa (Base case)	143.6	0.16	0.44	31.74	993.3	-	-	-
Sch_11_22.5Mt	22.5Mtpa (Upside case)	143.7	0.16	0.44	31.72	1009.1	13%	0%	2%

Figure 16.42 : Sensitivity – Maximum annual production rate



16.4.8.6.7 Cut-off operating cost

As previously discussed, the footprint boundary is not sensitive to changes in input assumptions, including the cut-off operating cost assumption used. Changes in cut-off operating costs impacts the total tonnes that can be mined from the block cave by determining when each column should be shut down at the end of its life. Due to the long life of mine and relatively low economic margin of material at the end of the mine life, the impact is more pronounced on total tonnes mined but much less on discounted value. The highest value material is extracted earlier in the schedule in most of the footprint due to a maximum HOD limit of 400 m, regardless of the cut-off cost assumptions made, thus having less impact on discounted value.

The cut-off cost sensitivity was run for both 400 m and 500 m maximum HOD to determine if the additional material for a higher maximum HOD at a lower cut-off may have a material impact on the results. The impact is more pronounced for the 500 m maximum HOD scenario as the additional 100 m of draw results in more mixing / dilution and lower grade material and is therefore more sensitive to a change in cut-off compared to the 400 m maximum HOD base case.

Table 16.18: Sensitivity - 400 m Maximum HOD Cut-off operating cost									
Scenario	**Description**	**Tonnes (Mt)**	**Au (g/t)**	**Cu (%)**	**NSR ($/t)**	**Relative Value ($M)**	**Change %**	**Tonnes Var %**	**Value Var %**
Sch_22_COV10.92	$10.92/t cut-off value	147.4	0.16	0.44	31.26	988.8	-30%	3%	0%
Sch_24_COV13.26	$13.26/t cut-off value	145.9	0.16	0.44	31.46	991.0	-15%	2%	0%
Sch_02_MxH400	$15.60/t cut-off value (base case)	143.6	0.16	0.44	31.74	993.3	-	-	-
Sch_26_COV17.94	$17.94/t cut-off value	140.5	0.16	0.45	32.07	992.7	15%	-16%	-7%
Sch_28_COV20.28	$20.28/t cut-off value	136.7	0.16	0.45	32.49	990.9	30%	-19%	-7%

Table 16.19: Sensitivity - 500m Maximum HOD Cut-off operating cost									
Scenario	**Description**	**Tonnes (Mt)**	**Au (g/t)**	**Cu (%)**	**NSR ($/t)**	**Relative Value ($M)**	**Change %**	**Tonnes Var %**	**Value Var %**
Sch_23_COV10.92b	$10.92/t cut-off value	178.5	0.15	0.42	29.44	1050.2	-30%	6%	-1%
Sch_25_COV13.26b	$13.26/t cut-off value	173.7	0.15	0.42	29.93	1058.1	-15%	4%	0%
Sch_04_MxH500	$15.60/t cut-off value (base case)	167.8	0.15	0.43	30.50	1063.2	-	-	-
Sch_27_COV17.94b	$17.94/t cut-off value	161.1	0.16	0.44	31.09	1062.9	15%	-4%	0%
Sch_29_COV20.28b	$20.28/t cut-off value	153.7	0.16	0.45	31.72	1056.8	30%	-8%	-1%

Figure 16.43 : Sensitivity – 400m Max HOD Cut-off operating cost

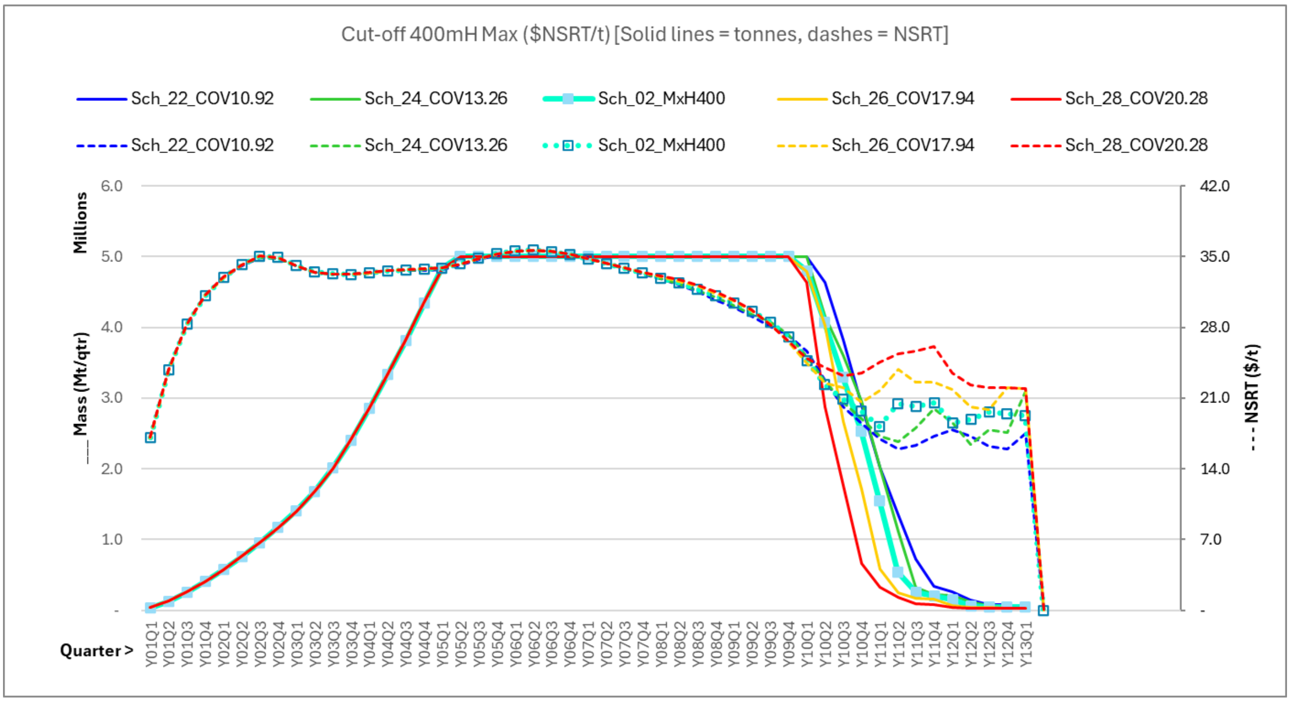
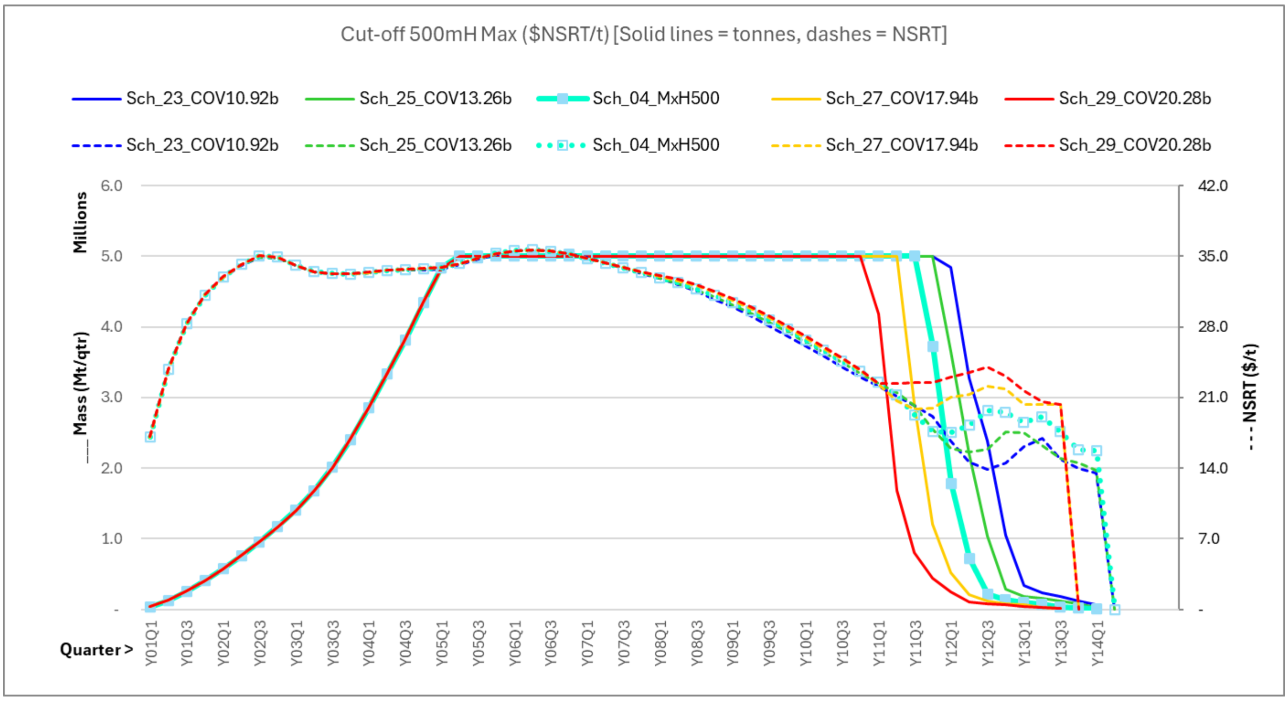


Figure 16.44 : Sensitivity – 500m Max HOD cut-off cost (operating cost cut-off)



16.4.8.6.8 Cave angle and potential ore loss

PCBC-FF and traditional PCBC setups assumes free vertical caving unless the cave propagation is constrained in the PCBC schedule using a cave back height above each drawpoint in each scheduling step, releasing only material below the height to the PCBC schedule in that step. Three approaches were used to model the impact of potential non-vertical caving:

1. Manual, fixed cave angles of 85°, 80°, 75° - lower angles were not tested as this would result in the cave stalling and not breaking into the open pit, which is very unlikely given the depth and geometry of the cave. A material loss in mined tonnes and contained value is observed for shallower angles (80° and 75°), especially for the taller 500 m maximum HOD scenarios, as more material is lost around the annulus of the cave at height. To reduce the potential impact of non-vertical cave propagation, the undercut was extended at least one drawpoint width (~20 m) past the last drawpoint, a high undercut and blast precondition was included from the extraction level, and hydro fracturing was included for the entire cave volume. This was included in the subsequent work using numerical modelling to determine the cave back propagation

2. Numerical modelling was used to model cave propagation surfaces based on the PCBC-FF schedules. These surfaces are based on the initial Footprint Finder block model layout and annual schedule increments, and does not exactly match the PCBC drawpoint layout and quarterly schedules, but was used as an interim constraint to run PCBC Template Mixing quarterly schedules using cave back constraints, which was then modelled again at a quarterly interval

3. Numerical modelling using the PCBC quarterly schedules for the first three years until cave break through into the open pit, followed by the annual surfaces from the numerical modelling step completed above. This provided refined cave back constraints on a quarterly resolution in the PCBC Template Mixing runs to model the impact of cave propagation during cave establishment and ramp-up.

The taller 500 m maximum HOD is impacted more by non-vertical caving, as can be expected, as it will remove more of the taller columns and cause more rilling of dilution higher up in the cave, further diluting the material mined.

Making allowance for a high and wide undercutting strategy with blast precondition, and hydrofracturing the remaining cave volume produce almost identical results to the free, vertical caving scenarios for both maximum HOD scenarios (Figure 16.47). The potential discounted value loss for a 75° cave back compared to a vertical cave is in the order of $137M (12% tonnes, 14% value) and $191M (21% tonnes, 18% value) for the 400 m and 500 m maximum HOD scenarios respectively (Table 16.20, Figure 16.45 and Figure 16.46), reducing mine life by almost a year. This justifies the additional expense purely on ore recovery but also has the added advantage of reduced cave stall and air blast risks, and the potential for faster production ramp-up.

Table 16.20: Sensitivity - Cave back propagation

Scenario	Description	Tonnes (Mt)	Au (g/t)	Cu (%)	NSR ($/t)	Relative Value ($M)	Tonnes Var %	Value Var %
Sch_02_MxH400	400 m (Base case) 90°	143.6	0.16	0.44	31.74	993.3	-	-
Sch_34_400_CB85	Fixed angle 85°	143.4	0.16	0.44	31.72	994.2	0%	0%
Sch_35_400_CB80	Fixed angle 80°	138.2	0.16	0.44	31.55	955.2	-4%	-4%
Sch_36_400_CB75	Fixed angle 75°	125.7	0.16	0.44	31.38	856.1	-12%	-14%
Sch_42_400FF_CB	FF annual cave back	143.8	0.16	0.44	31.59	988.0	0%	-1%
Sch_52_400mCaveBack	PCBC quarterly Year 1-3	143.8	0.16	0.44	31.59	988.0	0%	-1%
Sch_04_MxH500	500 m (Base case) 90°	167.8	0.15	0.43	30.50	1063.2	-	-
Sch_38_500_CB85	Fixed angle 85°	166.9	0.15	0.43	30.48	1060.6	-1%	0%
Sch_39_500_CB80	Fixed angle 80°	154.5	0.16	0.43	30.56	998.1	-8%	-6%
Sch_40_500_CB75	Fixed angle 75°	133.1	0.16	0.43	30.77	871.9	-21%	-18%
Sch_43_500FF_CB	FF annual cave back	167.1	0.15	0.43	30.35	1051.5	0%	-1%
Sch_53_500mCaveBack	PCBC quarterly Year 1-3	167.1	0.15	0.43	30.35	1051.1	0%	-1%

Figure 16.45 : Sensitivity – 400 m Cave angle constraint (theoretical limit inwards, non-vertical)

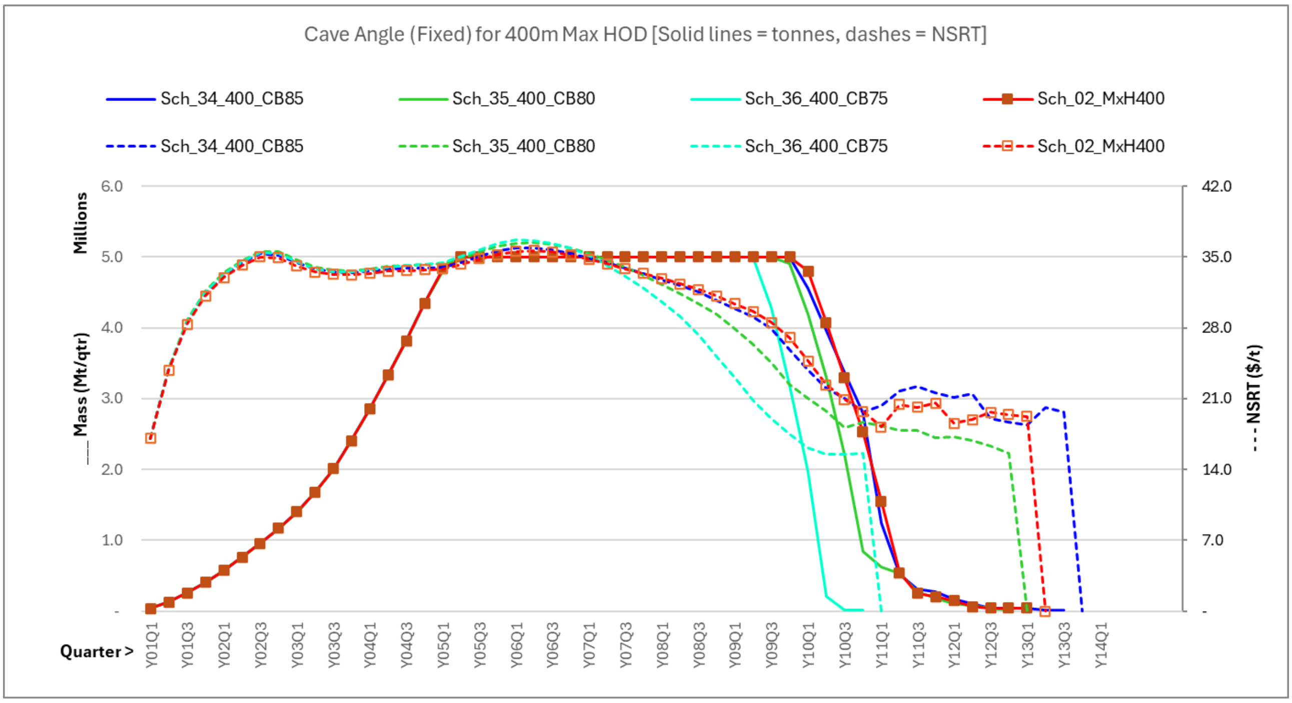


Figure 16.46 : Sensitivity – 500 m Cave angle constraint (theoretical limit inwards, non-vertical)

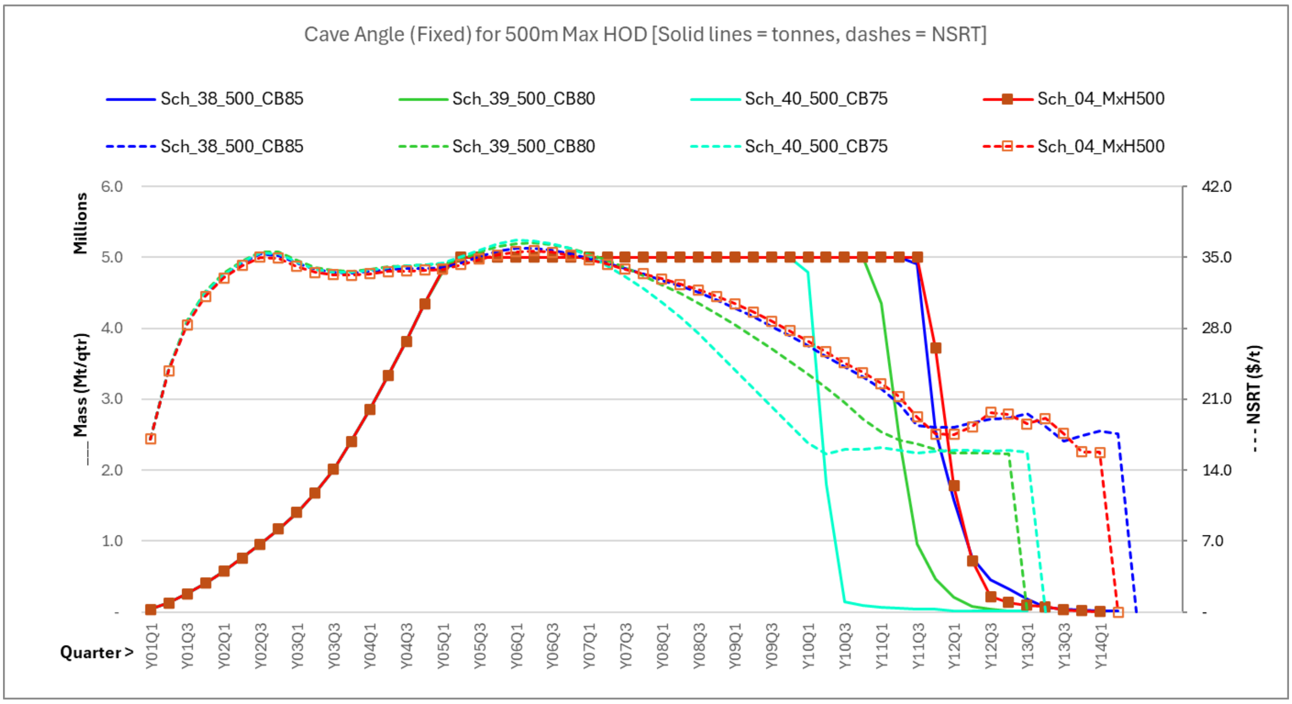
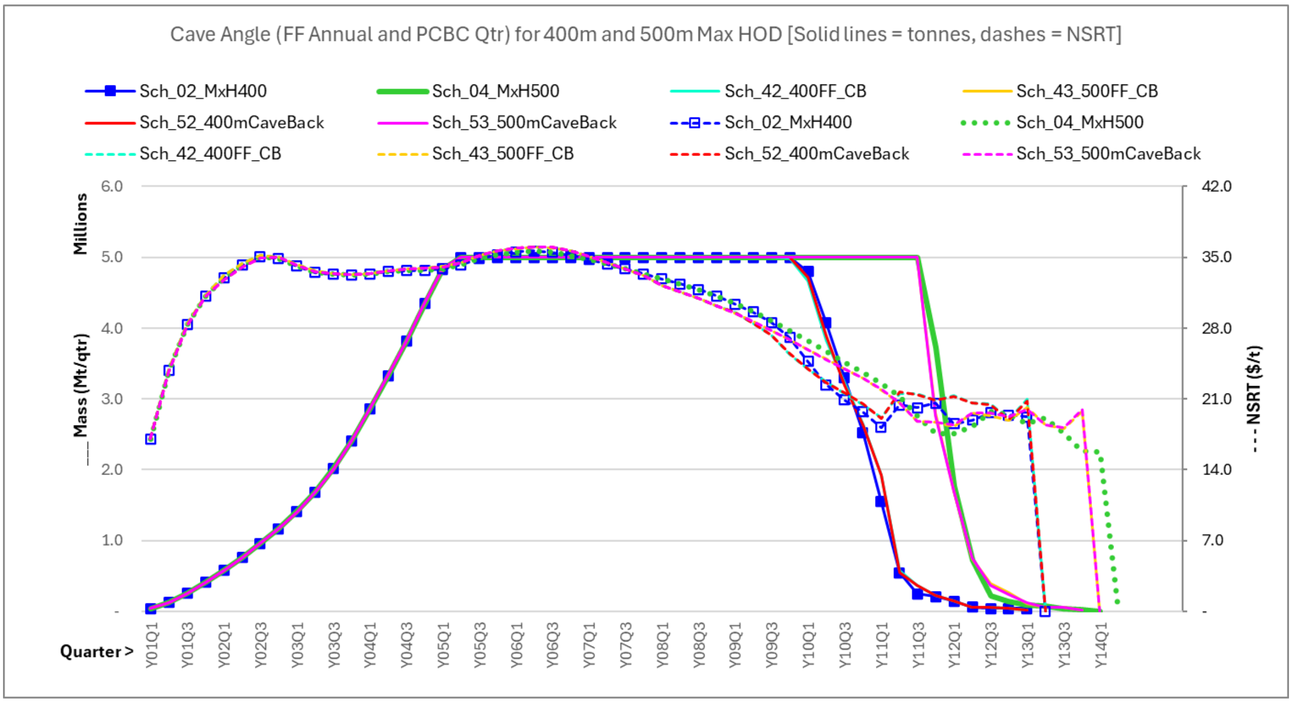


Figure 16.47 : Sensitivity – 400 m and 500 m Cave angle constraints (Numerical model results)



16.4.8.6.9 PCBC Optimisation Results Summary

All of the sensitivities typically encountered in a block cave, and which can be modelled in PCBC were tested and the results are shown in the two figures below for a maximum height of draw of 400 m and 500 m respectively. The cave is most sensitive to maximum height of draw below 400 m due to the loss of economic material. The chosen footprint elevation allows higher draw up to 500 m to be applied during operations without having to change the actual footprint layout and materials handling system. This is beneficial in long life projects where increases to the metal price in later years can allow the economic extraction of more material.

16.4.8.6.10 PCBC Final Production Runs

The initial production schedules and Block Cave sensitivities assumed a 20 Mtpa throughput based on initial processing and materials handling capacity assumptions discussed. Following development and production loading simulation work, additional mining schedules were developed for a 19.4 Mtpa throughput which also suited the larger 21 t loaders better compared to utilising 17 t LHD’s, and finally an ore throughput of 18.4 Mtpa which could consider either size of loader (i.e. 17 t or 21 t) loaders which is more common in South America. The 18.4 Mtpa schedule was ultimately selected as the final underground mining schedule and used for the financial evaluation as it is considered the more likely scenario given local loader experience.

Future work could also consider the 19.4 Mtpa schedule pending local availability and acceptability for using 21 t loaders, providing potential further upside. The difference in relative NPV as reported in PCBC is within the accuracy of the study and the final decision will therefore be more likely a strategic decision than a financial decision. The final production scheduling is discussed in more detail in Section 1.18.2 Underground scheduling.

Figure 16.48 : Final production schedules at 18.4 Mtpa and 19.4 Mtpa maximum rate for 400 m max height

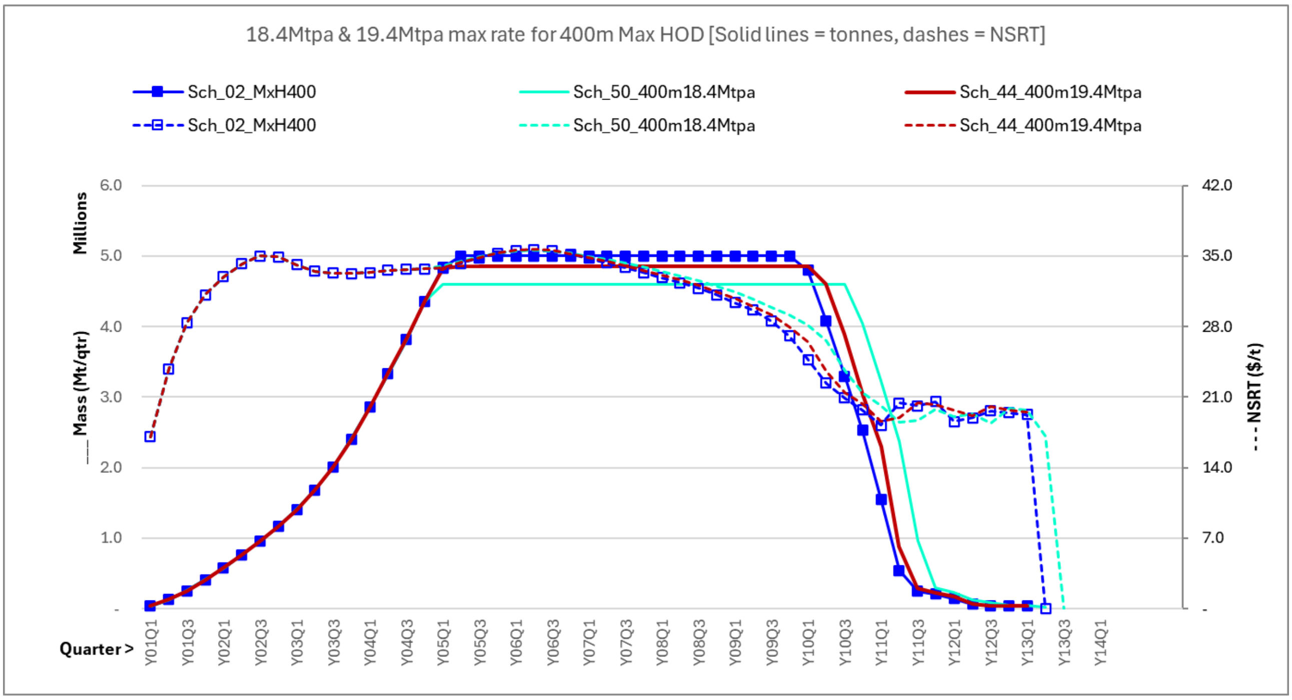


Figure 16.49 : Sensitivity Summary – All 400 m maximum height of draw results

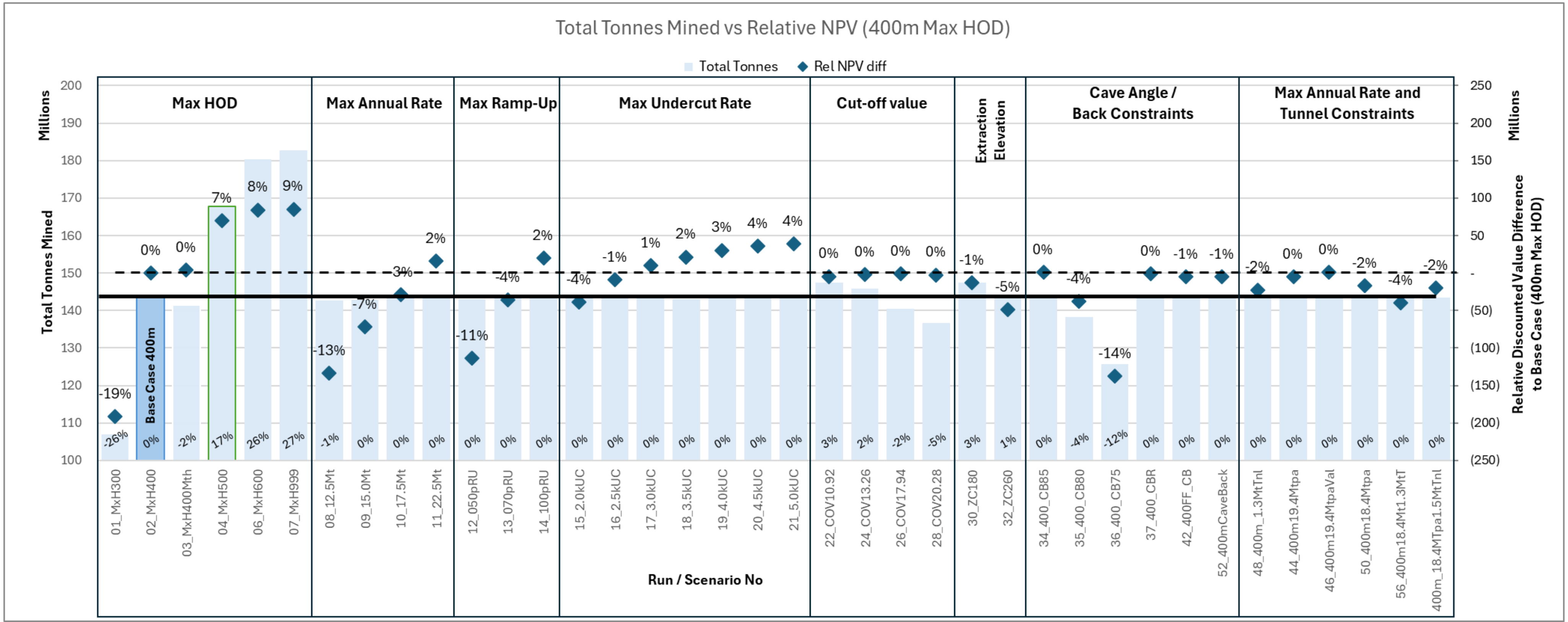
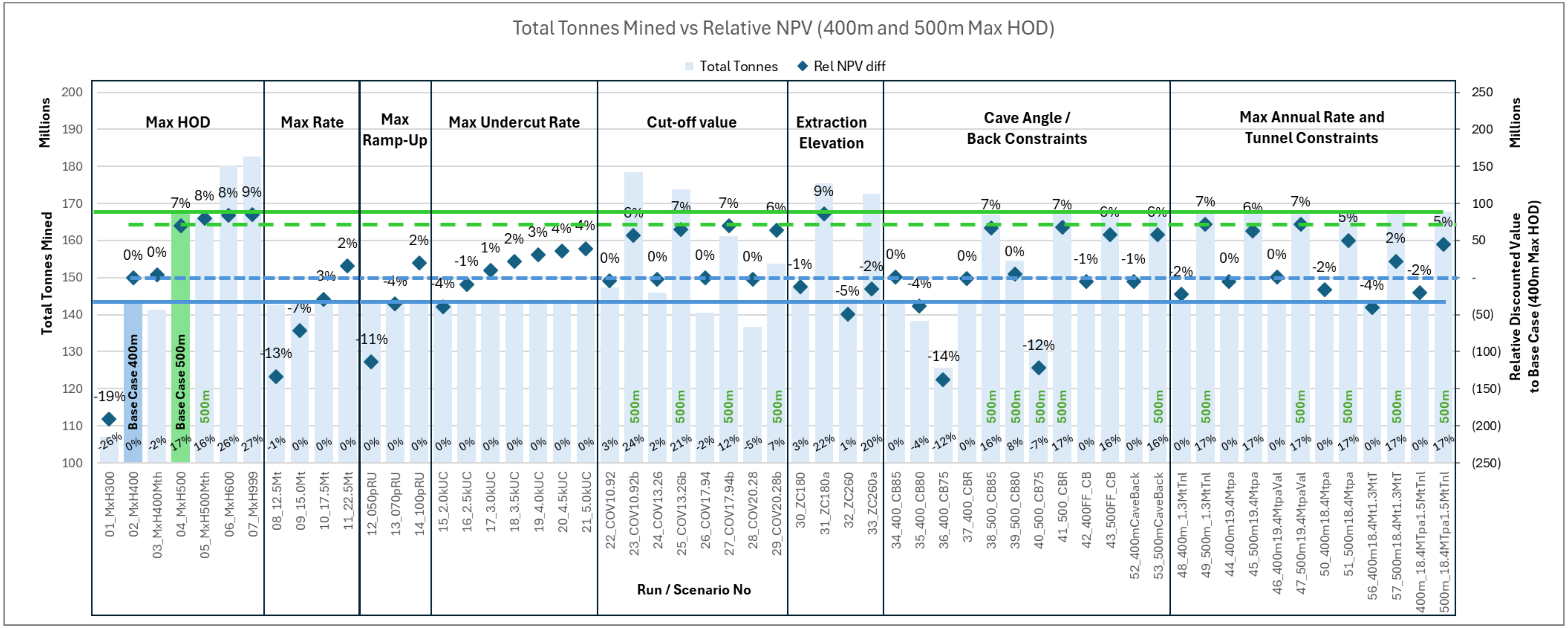


Figure 16.50 : Sensitivity Summary – All 400 m and 500 m maximum height of draw results



16.4.9 Mine Design

The underground geotechnical assessment performed by GMT as described in Section 1.16.1 was used to inform the underground mine design. The following section describes the mine design requirements used in the block cave design.

The PCBC results were used to define the extraction level and undercut level elevations, as well as the footprint maximum boundary. From this the perimeter drives and major infrastructure, such as the crushers, were positioned to allow stable geotechnical stand-off distances and to suit materials handling from the cave. Access from surface via declines, cave preconditioning development and primary ventilation circuits were then designed to suit the position of the major infrastructure, allowing for suitable stand-off to remain outside the cave influence zone.

16.4.10 Geotechnical recommendations for major infrastructure stand-offs

Geotechnical stand-off distance were included in the design as described in Section 16.4.3 and 16.4.5, and includes items such as crusher stand-off from the undercut, cave break angles in influence on access development, ventilation rises and preconditioning levels.

16.4.11 Tunnel profiles

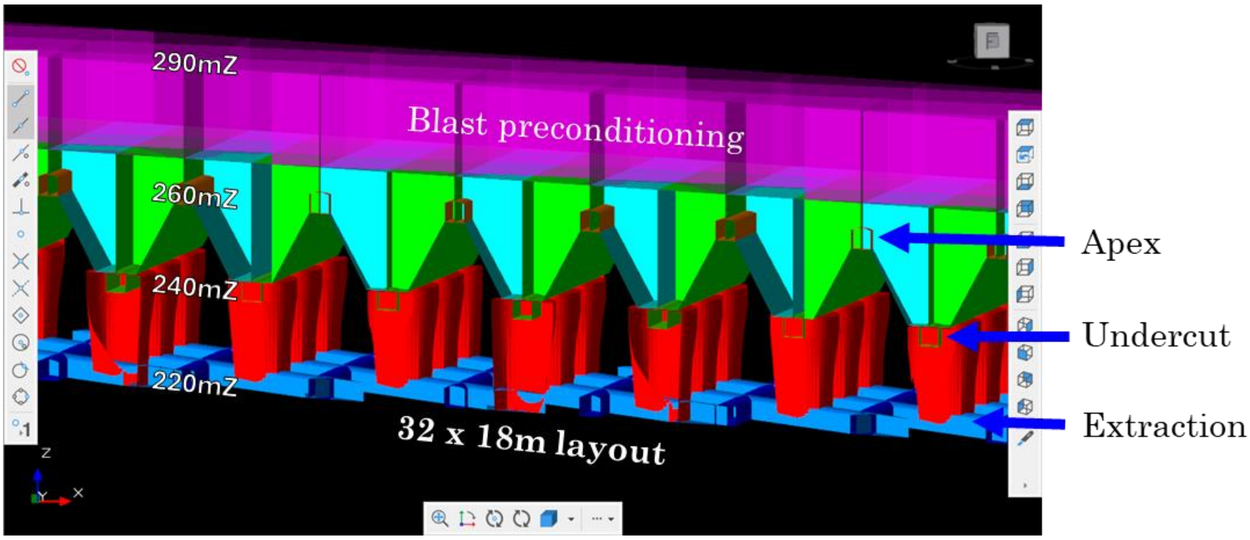
Standard excavation profiles were utilised based on typical cave operations. Larger extraction drives are implemented around the world in competent ground conditions, allowing the use of larger 21 t range LHDs. The decision was made for this study to use the more common 17 t LHD size and drive sizes, especially common for South American operations. This also provides marginally larger and more stable pillars on the extraction level layout due to smaller tunnel requirements for a given Drawpoint layout.

16.4.12 Undercutting

Undercutting is a critical part of block caving and is required to create an initial void large enough to initiate and sustain caving of the intact rock above the intended extraction footprint. The sequence in which the undercut rings are blasted relative to the drawbells connecting the extraction level to the undercut level vary based on geotechnical and operational conditions and can influence the success of the cave, especially during initial production ramp-up.

As the blasted undercut area expands, increased abutment stress loading is experienced ahead of the advancing undercut front. This can cause major damage to the extraction level if not managed appropriately. Given relatively shallow depth and manageable geotechnical and stress conditions, and the addition of hydro-fracture preconditioning, a post undercut method was selected for this study. To reduce risk, bit an apex and undercut level was added in such a location relative to the drawbells that the undercut method can be switch to a post undercut at a later stage. This is not a typical approach as it adds an extra undercut level, but it reduce risk at this early study stage by increasing the blasted undercut height, allowing high blast preconditioning to be completed from the Apex level during undercutting, and allows the undercutting sequence to be switched from post to pre undercutting. This layout is shown in Figure 16.51.

Figure 16.51 : Block Cave layout



16.4.12.1 Undercut Methodology

The ground conditions and expected stress environment at the planed Cortadera block cave allows for a post-undercutting sequence to be applied. For the planned Cortadera block cave, provision was made in the PFS for both an apex level and undercut level, and the undercut level tunnels are located on top of the drawbells to allows switching from a post-undercut to a pre-undercut sequence if required. This will be further optimised in future studies and provides flexibility to reduce undercut development by removing the apex level and relocating the undercut tunnels above the major apex should it be required.

For the planned Cortadera block cave, a straight undercut face was selected. The PEA considered a chevron shape starting in the middle of the deposit, but a straight undercut front moving across the entire width of the deposit was considered easier to manage given the revised starting position in the north of the footprint in the PFS.

16.4.12.2 Undercut Ring Geometry

For pre-undercutting and advance undercutting, the size of the undercut rings are generally made smaller to reduce the volume of swell material to be dealt with on the undercut level. Blasting of the undercut rings also occurs in a more "choked blast" environment as the only free breaking volume for the undercut rings is created by loading swell of the previous ring and the volume of the undercut tunnel. This typically results in lower undercut heights.

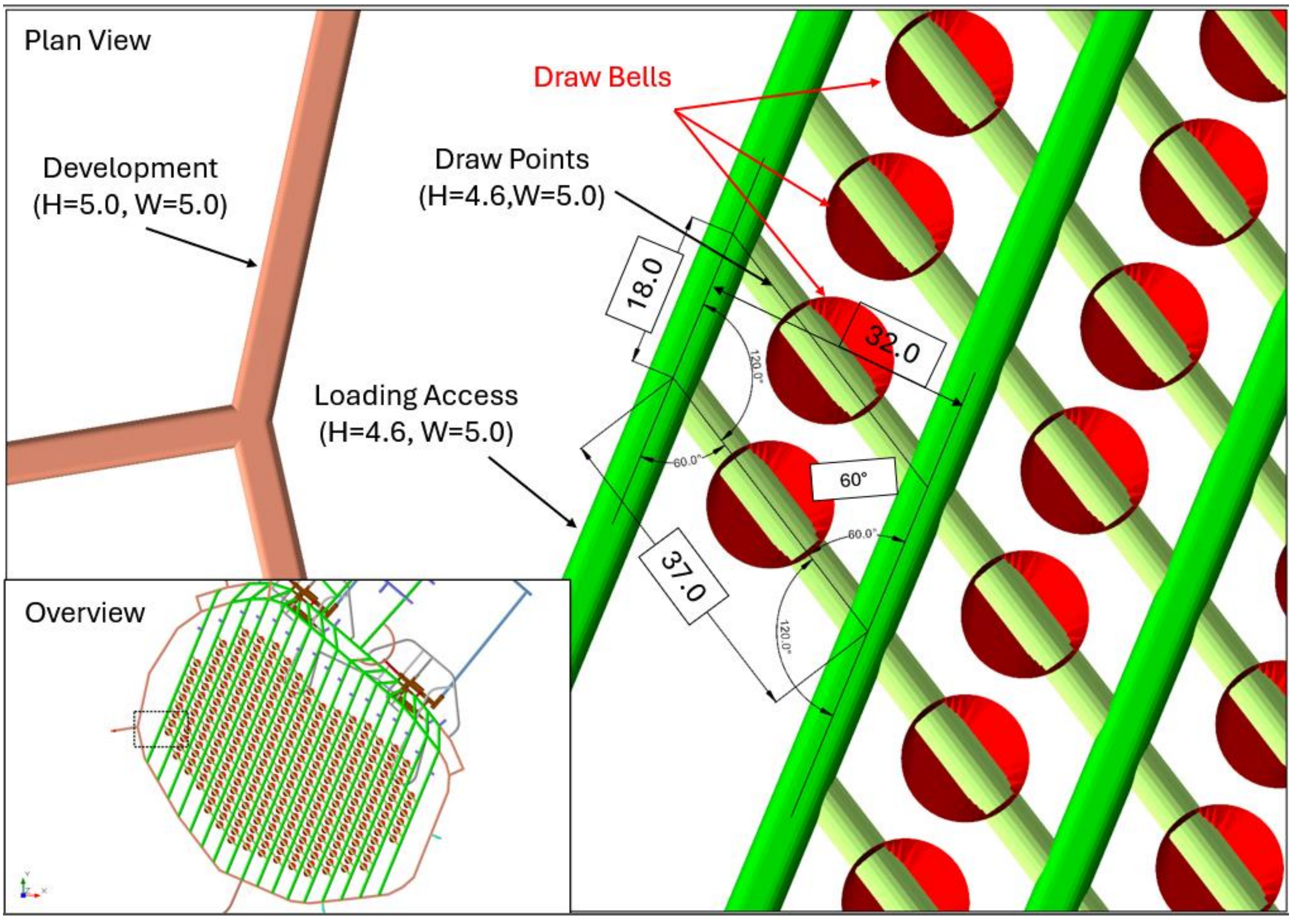
Post-undercutting has the advantage of breaking the undercut rings into the drawbell voids below, providing larger free breaking volume and therefore typically allows for larger rings and higher undercut height to be blasted. This creates more blasted material that can be loaded during production ramp-up, allowing for less stoppages due to oversize and faster production ramp-up.

For the planned Cortadera block cave, two undercut levels were designed – an undercut level 20 m vertically above the extraction level, and a second apex level a further 20 m above the undercut level. The drawbells will be blasted into the undercut level, allowing visual inspection and confirmation that the drawbells have been blasted successfully from the undercut level. The undercut rings, and blast preconditioning rings will then be fired from the apex level into the void created by the drawbells. Should a change from a post-undercut to an advance undercut be required, or should the apex tunnels become inaccessible, the undercut tunnels can be used to assist in recovering the undercut in that area.

16.4.13 Footprint layout

Large modern block caves generally either use the Herringbone layout or the El Teniente layout. The El Teniente layout is generally selected for large footprint caves with multiple tipping locations along the extraction drive. Production loaders are required to travel in both directions to access drawpoints on either side of the extraction drive. Given the geometry of this block cave footprint, both crushers were located on the northern side, with the undercut front moving away from the crushers towards the south. To compensate for crushers being located in the north only, turning bays have been designed in the northern side of each extraction drive between the first drawpoint and the crusher.

Figure 16.52 : 32 x 18m El Teniente layout



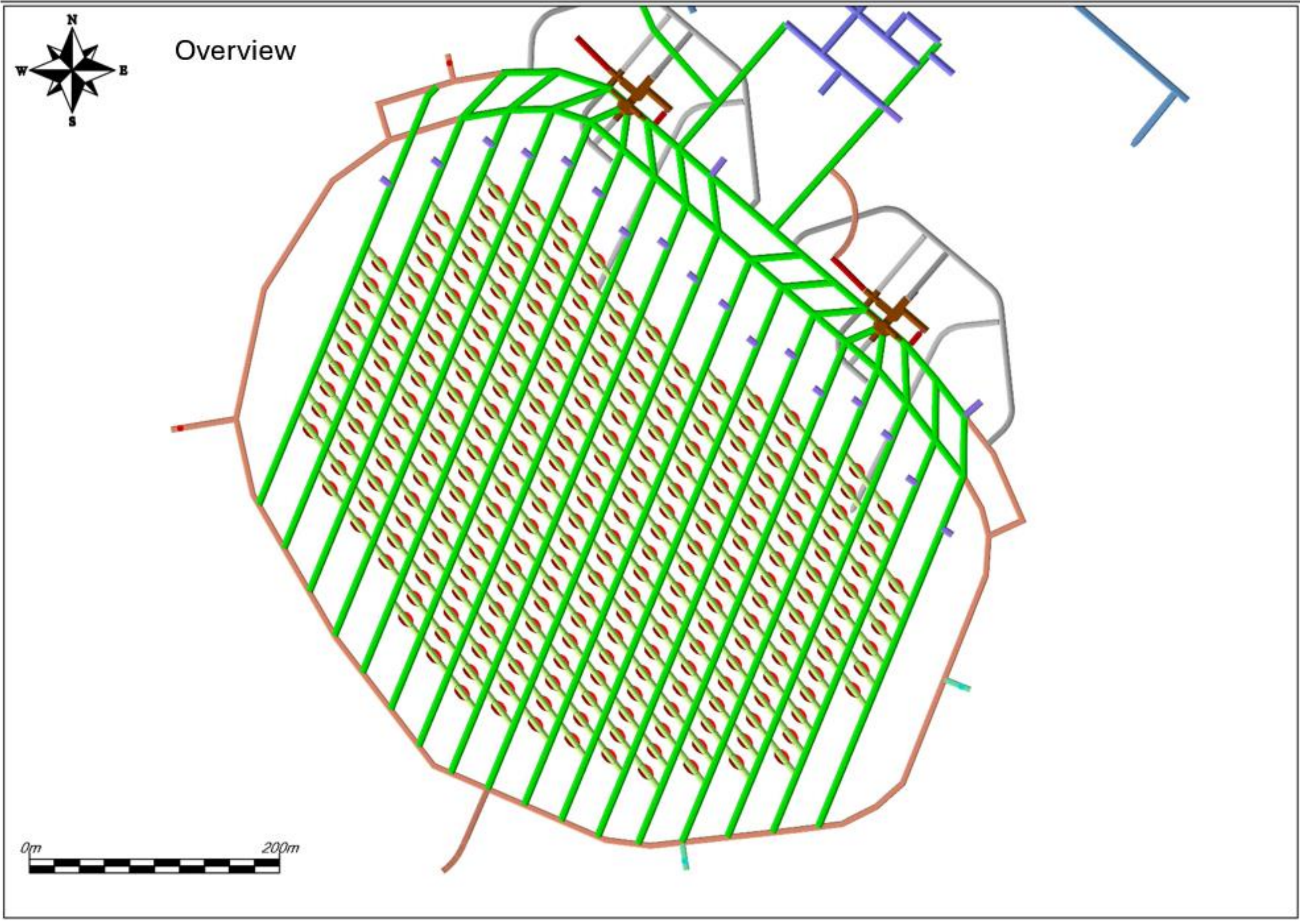
16.4.14 Materials handling drawpoint to crusher

The underground materials handling design considered two optimally located underground crushers each with four access/tipping entry points for the LHD, shown in Figure 16.54. LHD's will load in the production drives in a forward position, reverse out of the drawpoint, into a reverse/rehandle bay and then drive forward towards the crusher entry points.

Crusher location and number of crushers analysis was undertaken for the selected footprint. Consideration was given to having crushers on both sides of the footprint versus crushers on only one side, as shown in Figure 16.53. The benefit of having crushers on both sides, or an alternative materials handling system on the southern side, is that the loader tramming distance is greatly reduced, resulting in potentially higher productivities. The total production rate capacity of 20 Mtpa through the surface materials handling system, loader maximum tramming distance and capital infrastructure requirements were key considerations in this analysis. The options of a second materials handling system, either through extra crushers or a truck haulage system on the southern side were discounted for the following reasons:

- Maximum loader tramming distance along the extraction drives (north-south) for most of the cave is within a single loader tramming capability
- Given the northwest – southeast width of the deposit (parallel between extraction drives), two crushers are required to avoid excessive loader tramming from the extraction drive entrance to the crusher (as shown for the northern crushers in Figure 16.53.
- The cost of duplicating the materials handling system on the southern side would therefore either require two additional crushers (four total), or a truck load and haul system to bring ore from the southern side to the crushers in the north. The capital cost for this could not be offset by a potential increased production rate from the cave footprint as the constraint was determined by downstream materials handling and not by loader capacity from the footprint

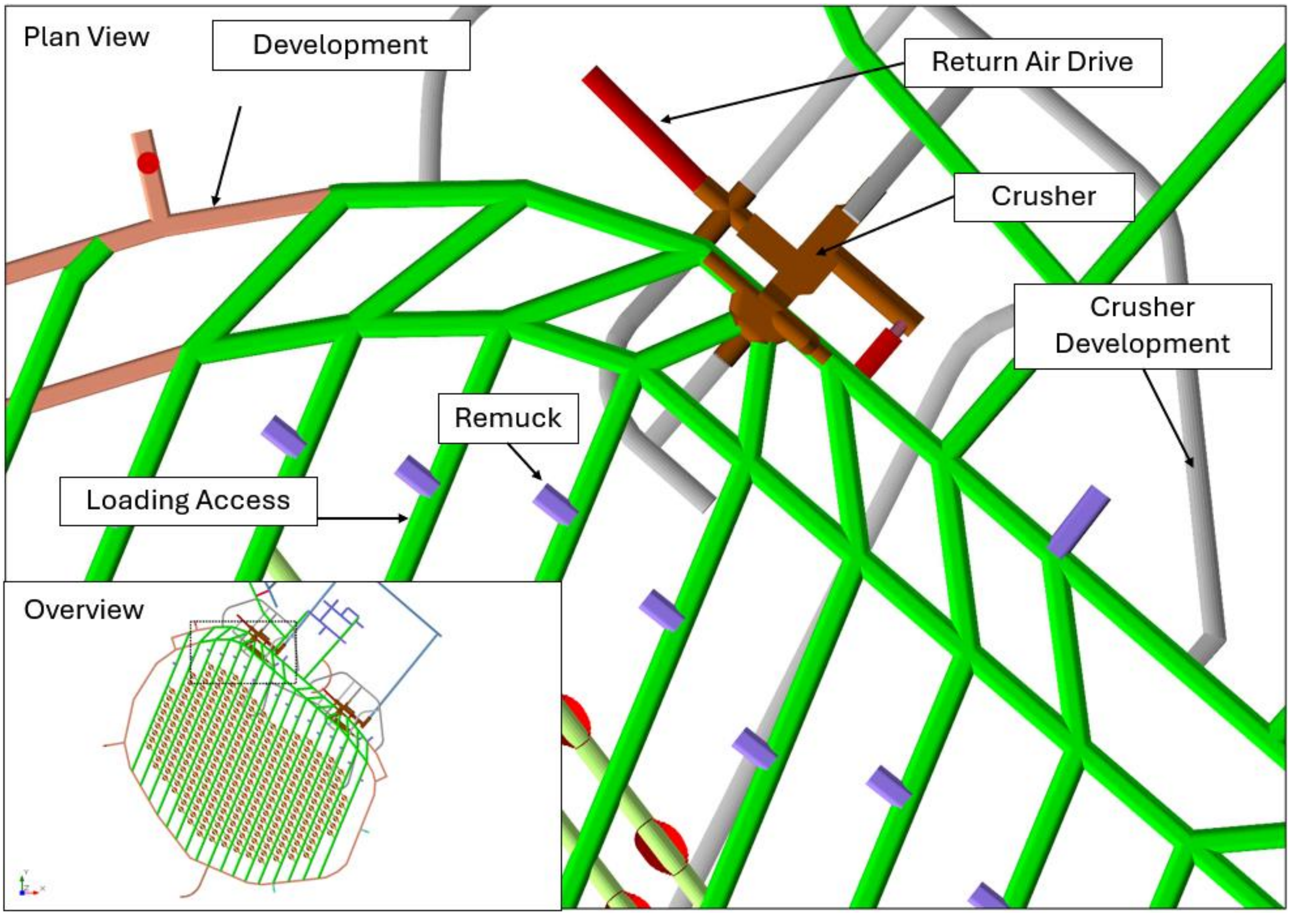
Figure 16.53 : Footprint general arrangement



Crusher tipping points are only accessed by LHD's and there is a grizzly and hydraulic rock breaker located at the tipping point. The LHD distances from loading points to the crusher tipping points change over time and initially only one crusher is utilised for a period. The maximum LHD tramming distances do reach 400m, but the average one-way distances are managed through LHD tramming schedules and the introduction of the second crusher when the draw profiles move closer to the centre of the footprint midpoint. This is the obvious challenge with the underground block cave production (the longer than ideal LHD tramming distances).

The underground rock handling layout and designs might benefit from an East and West crusher tipping point where the LHD tramming distances could be reduced and optimised, alternatively if the crushers remain on the one side (per the current design) an alternate tipping point with trucking to the crushers may improve the LHD productivities. This may produce a slightly more complicated equipment flow model and entry to crushers may require a dedicated truck entry, yet these comments require further design and simulation study work.

Figure 16.54 : Materials handling drawpoint to crusher



16.4.15 Underground Design

To gauge the timing for the planned Cortadera block cave extraction within the overall Costa Fuego Project, and to determine infrastructure requirements outside the cave footprint, a development design was completed within Datamine Studio UG software. The design utilises a twin decline layout, keeping access near long term vertical ventilation rises to facilitate practical development down to the 150mRL silo discharge level. One decline will accommodate a conveyor belt to transport process feed to surface and serve as a second means of egress, while the other will serve as primary access for mobile plant, personnel, and material.

Cave optimisation results indicated that the block cave extraction level should be located on the 220 m RL level, and the undercut level and apex level located on the 240 m RL and 260 m RL levels respectively. The block cave footprint design makes allowance for a larger footprint by placing long term infrastructure outside the cave abutment zone and ensuring the designed development quantities allow expansion flexibility if the Mineral Resource increases. It also ensures sufficient time is allowed to establish the initial capital infrastructure, with a level of conservatism built in.

The undercut levels will utilise a set of perimeter drives that connect the mine workings with two (2) Fresh Air Rise and (one/two) Return Air Rise to surface. A large fleet of load haul dumpers (LHDs) will haul process feed from the respective draw points to the crusher feeder arrangement on the extraction level. This fleet considers conventional diesel propulsion as the base case and can be modified to allow fully electrical solutions between the time of study and the time the access declines reach the extraction level. If the LHD fleet is fully electric, and process feed is transported to surface via a conventional conveyor installation, the total ventilation requirement will be significantly reduced compared to a conventional diesel fleet solution. This will require minor extra development for battery charge and swap bays (if using battery operated LHDs) or connection of the extraction tunnels to the crusher tipple to allow electric LHD trailing cable connection points, and for traffic management between electric LHDs to avoid vehicle interactions. The footprint is at the maximum limit of current electric LHD trailing capacity, and a dual materials handling system may be required should electric LHDs with trailing cables be employed.

The workshop design will account for the resultant fleet specified by the next phase of study, considering spatial requirements, total fleet size, service intervals and storage requirements. The conceptual design allows for a generic workshop layout to account for overall development metres, with a reasonable location away from the block cave influence zone.

The extraction level design feeds two crushers located on the northern side of the footprint. The current production rate is determined by the overland RopeCon belt capacity and the sulphide process plant capacity, which were sized for the open pit operations. The ultimate cave footprint has the capacity to provide higher throughput rates but will require revision of the material handling system to a central ore pass, or a dual system with LHD tipping points on either side of the extraction tunnel to allow shorter hauls. This option has not been designed in this study as the single sided layout provided the required process capacity and provides a simple solution with less transfer points.

Figure 16.55 to Figure 16.56 depicts the complete underground mine design as developed for the underground study.

Figure 16.55 Plan view of Underground Design

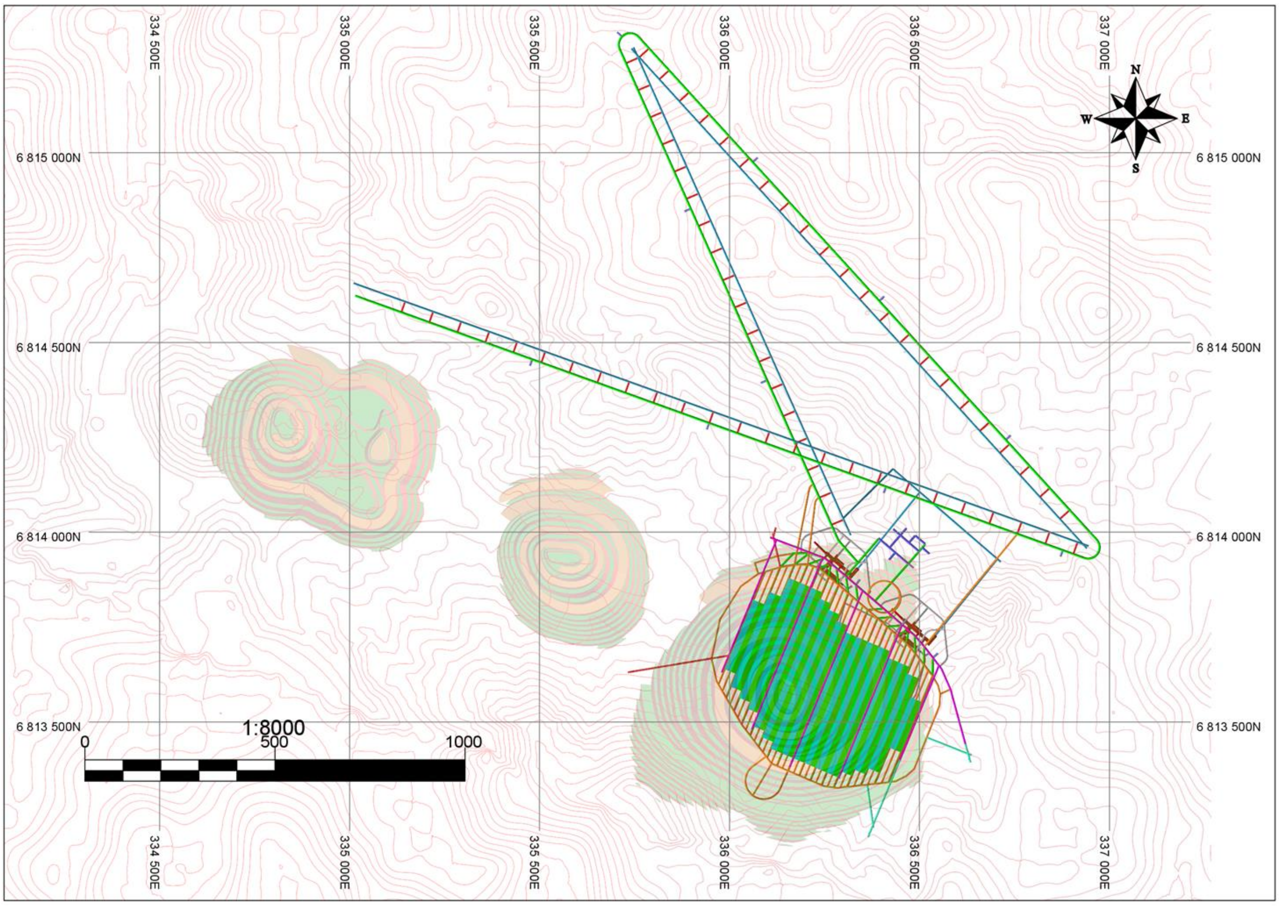


Figure 16.56: Isometric view of Underground Design

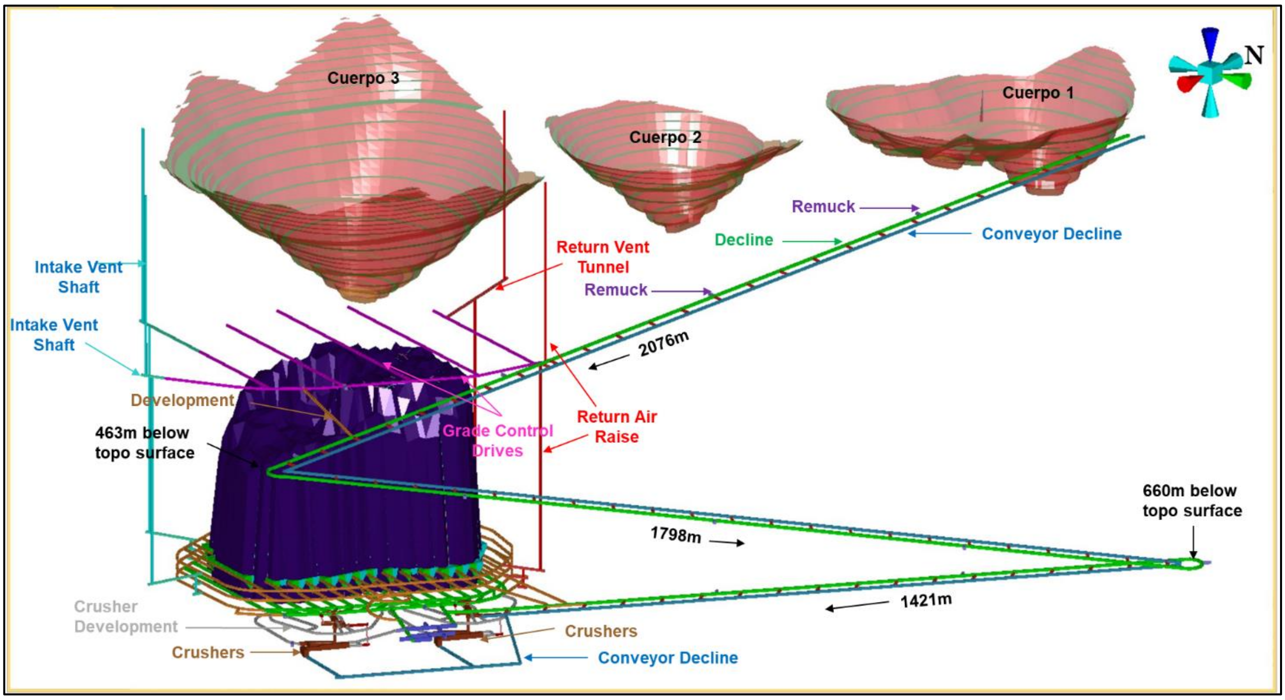
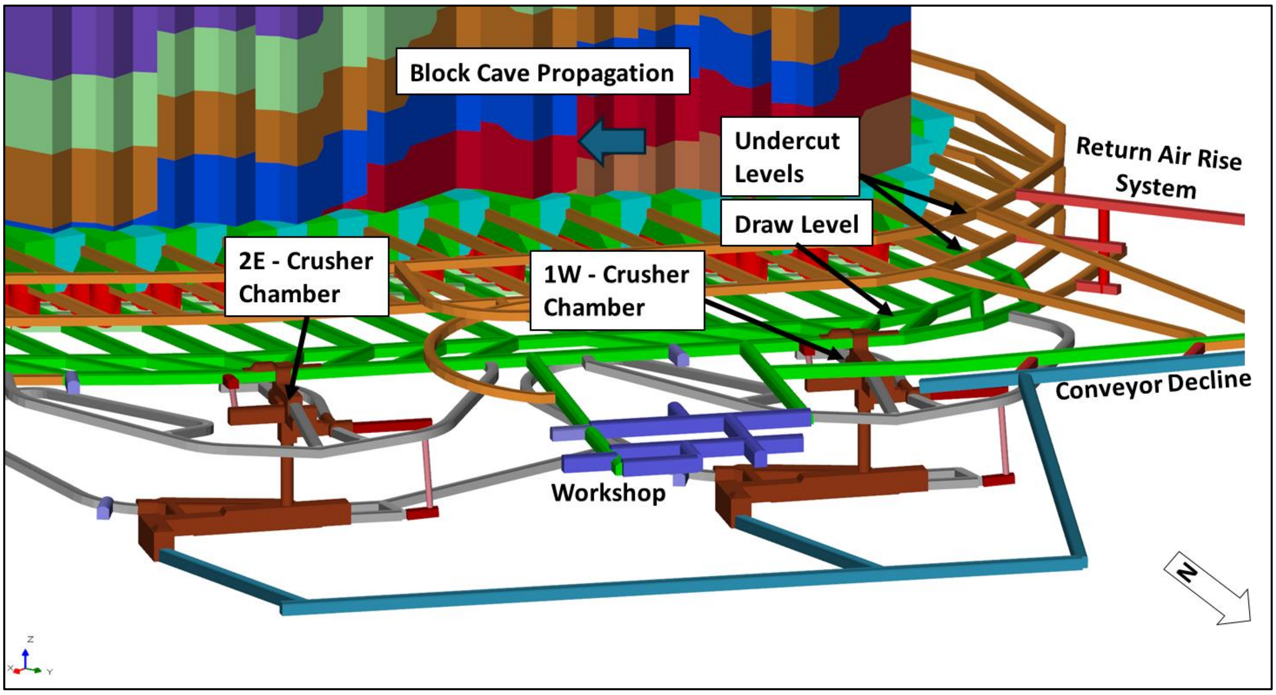


Figure 16.57 shows an isometric view of the design, looking in a south-westerly direction. Processing feed will be drawn from a series of draw points located on the extraction level. Half of the processing feed will be handled via the 1W crusher system and the other by the 2E crusher system.

Figure 16.57: PFS Block Cave Primary Infrastructure - isometric view (looking south-west)



16.4.16 Underground Design quantities

Table 16.21 summarises the underground mine design quantities.

Table 16.21: Underground mine design quantities

Description	Unit	Totals
Portal volume Development	m^3	7,200
Portal Support	m^2	600
Decline_m	m	5,518
Decline Vol	m^3	173,155
Decline_Connect_m	m	2,187
Decline_Connect_Vol	m^3	68,797
Conveyor_Decline_m	m	6,318
Conveyor_Decline_vol	m^3	221,829
Remuck_m	m	429
Remuck_Vol	m^3	14,959
Development_m	m	6,747
Development Vol	m^3	165,601
Precondition LVL m	m	5,092
Precondition LVL Vol	m^3	125,033
Vent_Drives_m	m	2,403
Vent_Drives_vol	m^3	58,676
Crusher_acc_ramp_m	m	2,232
Crusher_acc_ramp_vol	m^3	49,115
Crusher Vol	m^3	39,122
WSHOP_m	m	371

Description	Unit	Totals
WSHOP_vol	m^3	16,801
Apex_Access_m	m	7,244
Apex_Access_vol	m^3	159,599
Undercut_Access_m	m	6,938
Undercut_Access_vol	m^3	152,863
Loading_Access_m	m	9,307
Loading_Access_vol	m^3	242,071
Draw_Point_m	m	9,570
Draw_Point_vol	m^3	221,903
Crusher Venthole m	m	104
Crusher Venthole vol	m^3	721
Raisebore m	m	3,504
Raisebore Vol	m^3	73,200
Ore Development m	m	38,151
Ore Development Vol	m^3	910,761
Ore Develop ore (Incl Undercut ore)	kt	2,300
Ore Develop ore (Incl Undercut ore)	m^3	825,189
Ore Develop Cu	%	30.77%
Ore Develop Au	ppm	0.09
Ore Develop Mo	ppm	100.62
Ore Develop Ag	ppm	0.59
Undercut + Bell tonnes Drilled & Blasted	kt	10,132
Total Block Cave Ore Drawn (Inc. Bells)	kt	143,475
Cave Ore Cu	%	0.44
Cave Ore Au	ppm	0.16
Cave Ore Mo	ppm	92.54
Cave Ore Ag	ppm	0.79

16.4.17 Materials Handling Design - Crushers

Due to the direction of the cave propagation, Crusher 1W will ramp-up to full production first, followed by Crusher 2E, with their respective production profiles shown in Figure 16.58. The crusher type and model are based on the processing feed properties, coupled with the throughput requirements. Both crushers are expected to reach a maximum throughput rate of 3.2 Mt per quarter. This translates to a rate of nearly 2,200 tph. This calculation is based on a system utilisation of 6,132 hours per year, but de-rated by an additional 6% due to a coarser particle size distribution of un-blasted rock. This translates to an effective 5,760 hours required at the stated 2,200 tph to achieve this peak throughput rate. It also represents a more aggressive view of each crushers' throughput requirements, ensuring that the chosen crusher should be able to handle the required duty.

Considering the production profile in Figure 16.58, three points of interest are chosen to facilitate a reasonable crusher selection, and labelled Scenario A through Scenario C. Scenario A depicts a position early in the block cave ramp-up, where the Height of Draw (HOD) is low for both the western and eastern feeds, and thus producing the coarsest material required to be handled by each crusher. At this time, the average draw points HOD for the western feed material is roughly 90 m above the apex level, while the average HOD for the eastern feed material is closer to 50 m.

Scenario B represents the peak production throughput for Crusher 1W, with an average HOD of ~165 m, and thus well advanced. At the same time Crusher 2E has an average HOD of ~100 m but is still in its ramp-up phase. Crusher 1W handles a larger proportion of the feed over most of the life of the block cave, with Crusher 2E only overtaking it after nine years in the cave's life.

At this point Crusher 2E briefly ramps up to the previously stated 3.2 Mt per quarter throughput rate as the feed profile for Crusher 1W is busy declining and is the third position tested for crusher selection purposes (Scenario C).

If the selected crusher can handle the material type and throughput requirements at these points, the selected crushers are deemed appropriate for the entire production profile. The two primary factors considered for the appropriate crusher selection includes the expected Particle Size Distribution (PSD) and the Feed Opening on the respective crushers. Though some of the manufacturers may indicate a minimum throughput at the smallest Open Side Setting (OSS) for the gyratory crushers performing the duty, the true performance is a combination of the PSD and Bond Work Index (BWi) of the feed material. If a significant proportion of the feed are larger particles, the crusher throughput is significantly reduced, and the supplier will not be able to warrant the required throughput.

The most effective way is to alter the expected rock PSD is through consideration of pre-conditioning, coupled with the secondary breakage as the cave's HOD increases. Rocks larger than 2 m^3 cannot be loaded with the chosen LHDs and will require drilling and blasting at the draw points to reduce their size. Operationally the largest rocks that can successfully be handled on the grizzly will be balanced with draw point productivity. A twin primary crusher solution is considered practical considering the high throughput rate required for this application. Two primary crusher types were considered, which includes gyratory (PG) crushers and jaw gyratory (PJG) crushers. The PJG accepts a larger feed particle and is fed from one feed direction. The PG type is well suited to direct feed applications and frequently fed from more than one feed location. Both have a reduction ratio of up to 4.5:1.

Another strategy of reducing the amount of oversize feed material is to install a fixed grizzly at the LHD tipple and employ a fixed rock breaker to reduce the maximum particle size. This is less effective than the pre-conditioning and secondary breakage technique as described above.

Figure 16.58: Quarterly Production Profile per Crusher Complex

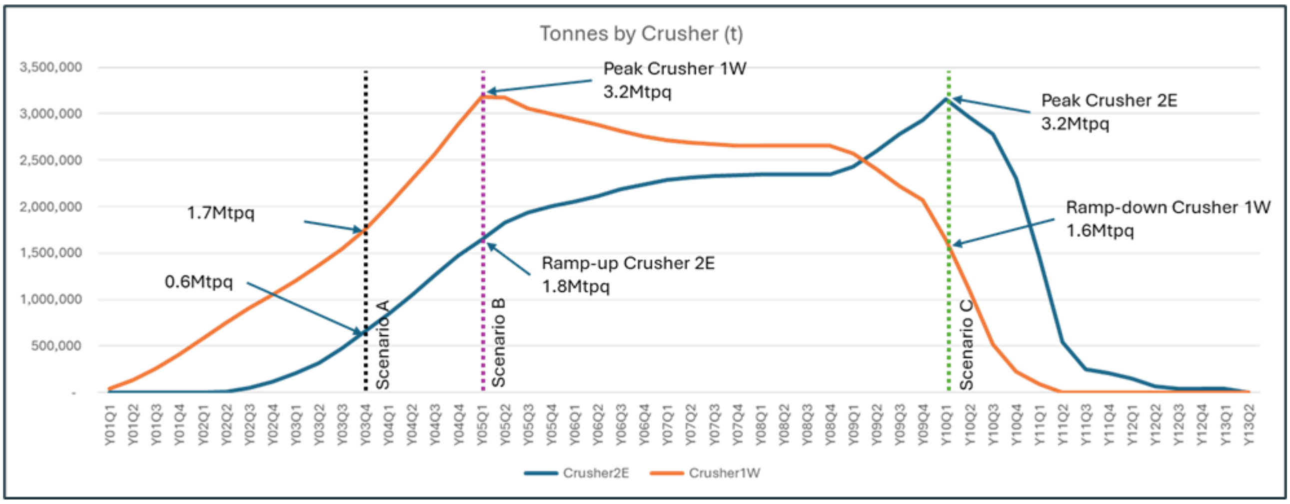


Table 16.22 provides a detailed summary of the respective tonnage rates and expected particle distribution for each of the scenarios discussed above per lithological composition. Lithological compositions are described in more detail in Section 14. It also shows the average HOD for each scenario. It is noteworthy that for Scenario A, Crusher 2E expects the coarsest feed, but the required crusher feed rate at that point is less than 500 t/hr. The hardest lithology is the "20-Intramineral Porphyry" with a Bond Work Index of ~17 kWh/t, though most lithologies are reasonably close to this value. The most abrasive lithology is the "10-Early Porphyry". This information was supplied to three crusher suppliers to obtain October 2024 budget pricing estimates and each crusher supplier's recommended crusher solution.

Table 16.22: Production Profile Material Composition per Scenario at Respective HOD

Scenario	Item	Unit	Value	BWi	Ai	HOD (m)	P50 (mm)	P80 (mm)
Scenario A - Crusher 1W	Tonnage	t/hr	1,216	16.67	0.17	90	552	790
	5-Hornfels	t/hr	139	16.95	0.11		669	932
	10-Early porphyry	t/hr	417	16.32	0.21		585	819
	20-Intra- porphyry	t/hr	451	17.11	0.17		431	654
	Other	t/hr	209	16.25	0.14		669	932
Scenario A - Crusher 2E	Tonnage	t/hr	454	16.66	0.18	54	661	884
	5-Hornfels	t/hr	77	16.95	0.11		778	1,016
	10-Early porphyry	t/hr	211	16.32	0.21		669	909
	20-Intra- porphyry	t/hr	134	17.11	0.17		554	737
	Other	t/hr	33	16.25	0.14		778	1,016
Scenario B - Crusher 1W	Tonnage	t/hr	2,203	16.70	0.17	167	551	789
	5-Hornfels	t/hr	334	16.95	0.11		669	932
	10-Early porphyry	t/hr	780	16.32	0.21		585	819
	20-Intra- porphyry	t/hr	816	17.11	0.17		431	654
	Other	t/hr	272	16.25	0.14		669	932
Scenario B -	Tonnage	t/hr	1,270	16.62	0.18	100	562	801

	5-Hornfels	t/hr	170	16.95	0.11		669	932
	10-Early porphyry	t/hr	579	16.32	0.21		585	819
	20-Intra- porphyry	t/hr	365	17.11	0.17		431	654
	Other	t/hr	155	16.25	0.14		669	932
Scenario C - Crusher 1W	Tonnage	t/hr	1,140	16.68	0.15	379	589	837
	5-Hornfels	t/hr	266	16.95	0.11		669	932
	10-Early porphyry	t/hr	110	16.32	0.21		585	819
	20-Intra- porphyry	t/hr	345	17.11	0.17		431	654
	Other	t/hr	419	16.25	0.14		669	932
Scenario C - Crusher 2E	Tonnage	t/hr	2,194	16.66	0.16	329	574	817
	5-Hornfels	t/hr	444	16.95	0.11		669	932
	10-Early porphyry	t/hr	704	16.32	0.21		585	819
	20-Intra- porphyry	t/hr	626	17.11	0.17		431	654
	Other	t/hr	420	16.25	0.14		669	932

The lithology and tonnage estimates are based on the initial Footprint Finder production profile, which had sufficient accuracy to facilitate the appropriate crusher selection. Though the table shows the expected size of perfect cubes, blocks typically present underground at a ratio of 1 : 0.67 : 0.45 in the longest length to primary width to secondary width dimensions.

The largest expected particle that would be fed to the PG would pass through a nominal grizzly aperture of 1,170 mm. It is expected larger particles would be handled at the draw points with explosives, and those that do report to the tipple handled via rock breaker at the fixed grizzly installation. Using the generic rule of thumb dimensions highlighted above, the rock will have a second largest dimension (D) of almost 1,100 mm. This results in a dimension to feed opening ratio of roughly 78% for a 5,475-class crusher, and 84% for a 5,065-class crusher.

Gyratory crushers are designed to accommodate an 80% - 85% ratio, which indicates that the grizzly would be appropriate for the crusher combination. Even at a HOD of around 50 m above the draw level at lower throughput rates, nearly 90% of the particles presenting to the draw points will have a second largest dimension of less than 1,100 mm. As such the design criteria looks to be sufficient, but once the PSD is better understood, a full study is required to confirm these results. This study will trade-off lower productivity at draw points (due to lower draw point availability) versus a larger gyratory crusher and grizzly with increased apertures. In discussion with the equipment suppliers, a 1,080 mm static grizzly for the 5,065-class crusher is recommended, along with a 1,170 mm static grizzly for the 5,475-class crusher for sustained uninterrupted performance.

GMT presented a fragmentation analysis document in September 2024 and supplied the following PSD for the 10-Early porphyry and 20-Intramineral porphyry lithologies at a HOD of 50 m and beyond. At 50 m HOD, nearly 90% of the distribution is less than 1.0 m^3 (1,000 mm^3) as shown in Figure 16.59 and Figure 16.60. which represents the south-east region of Scenario A as referenced in Table 16.22.

A 1.0 m^3 rock would represent a typical rock with nominal dimensions of roughly 1.5 m (L) x 1 m (D) x 0.67 m (d), which a 5,475-class crusher is able to handle.

Figure 16.59: Cumulative Mass Distribution – GM-10 (early-mineral porphyry) at HOD of 50m in the SE Region

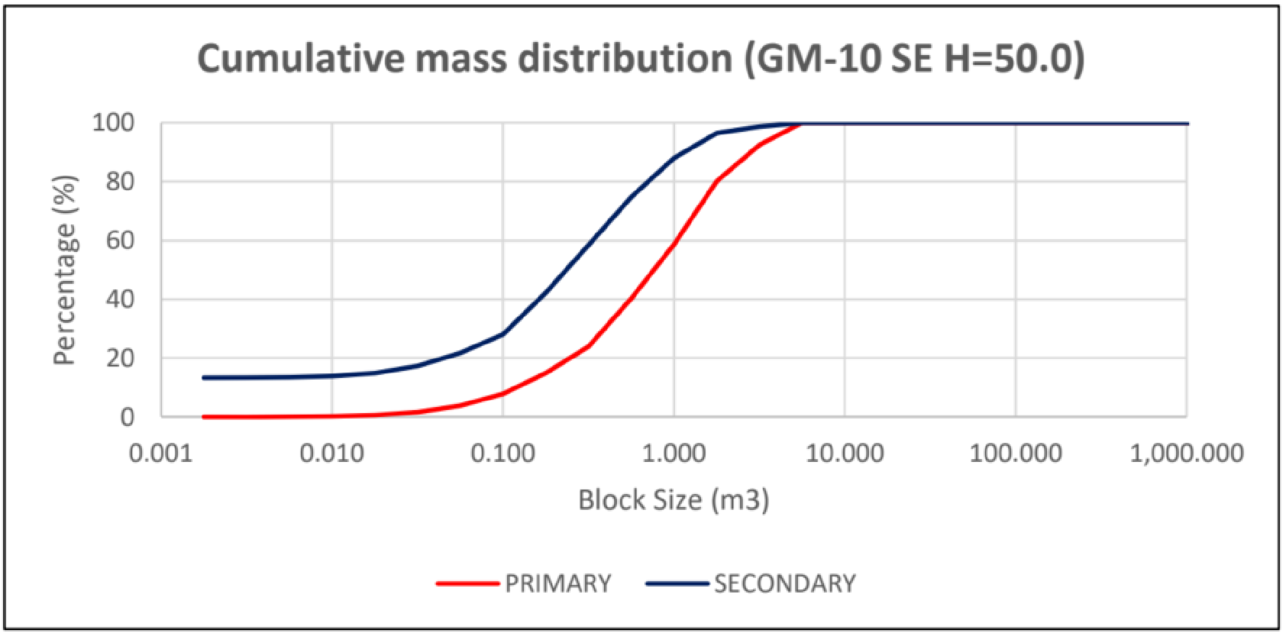
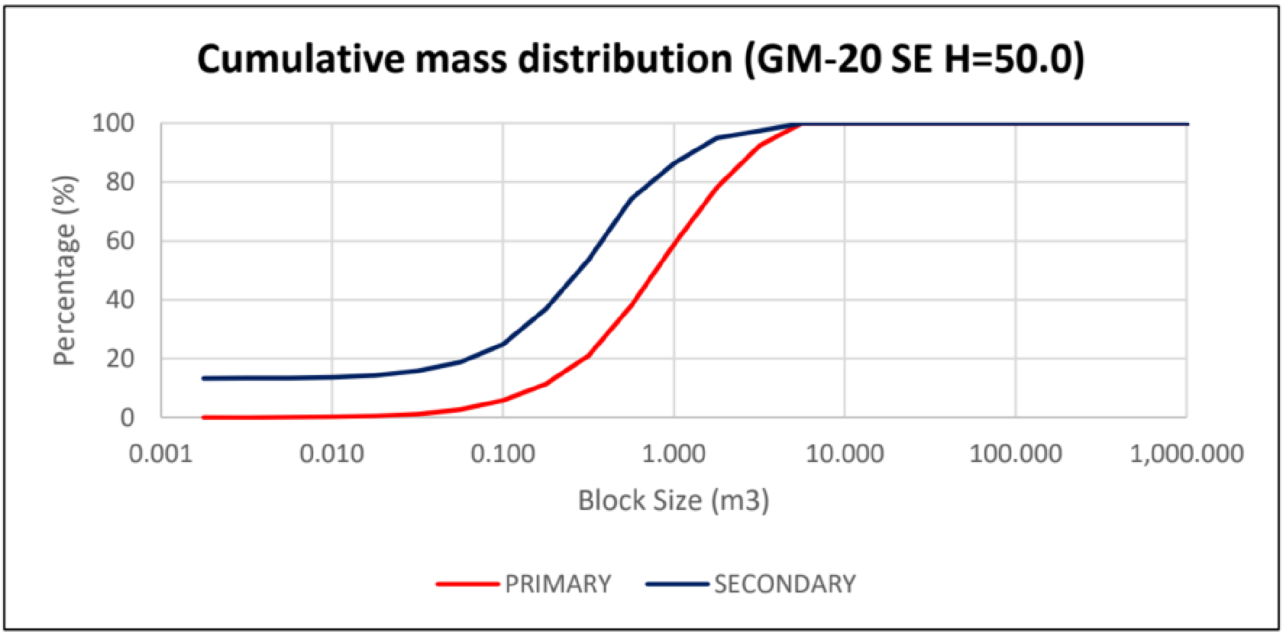


Figure 16.60: Cumulative Mass Distribution – GM-20 (intramineral porphyry) at HOD of 50 m in the SE Region



The north-west region for Scenario A has a HOD of roughly 90 m and presents a better situation than the south-east region, with about 98% of the secondary mass distribution at less than 1.0 m3, as highlighted in Figure 16.61 and Figure 16.62. The first scenario chosen was based on the worst-case rock properties presented in the south-east region at a low HOD of only 50 m. As the HOD increases, the PSD becomes more favourable.

Figure 16.61: Cumulative Mass Distribution – GM-10 (early-mineral porphyry) at HOD of 90 m in the NW Region

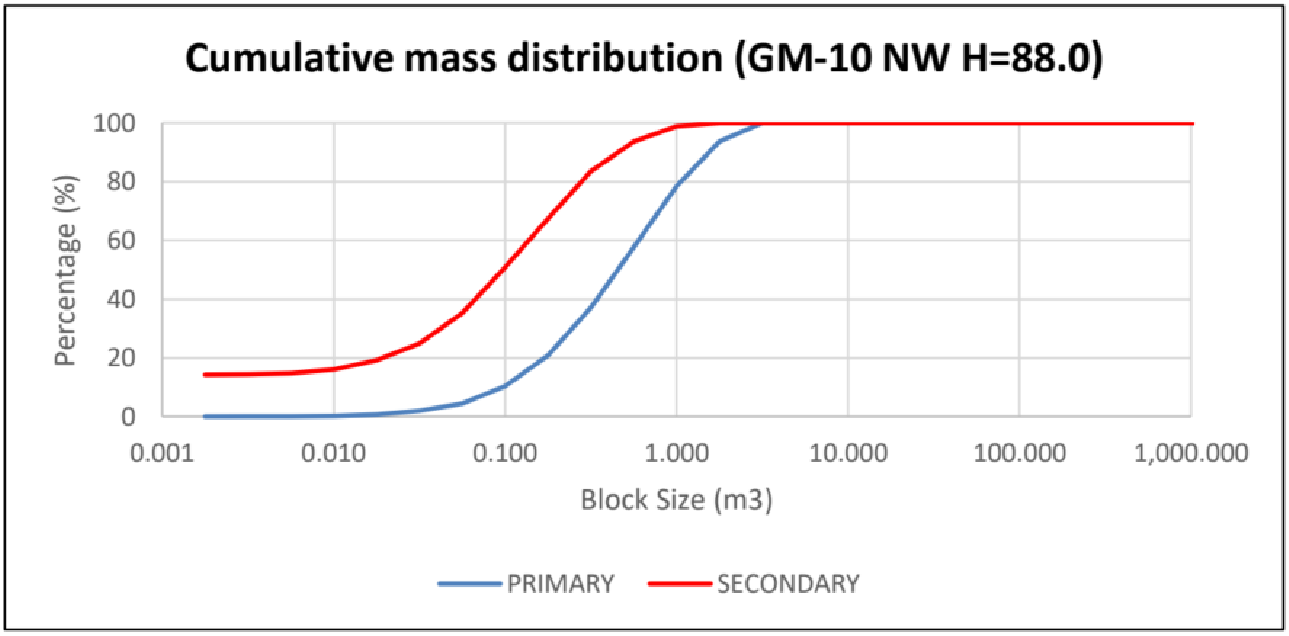
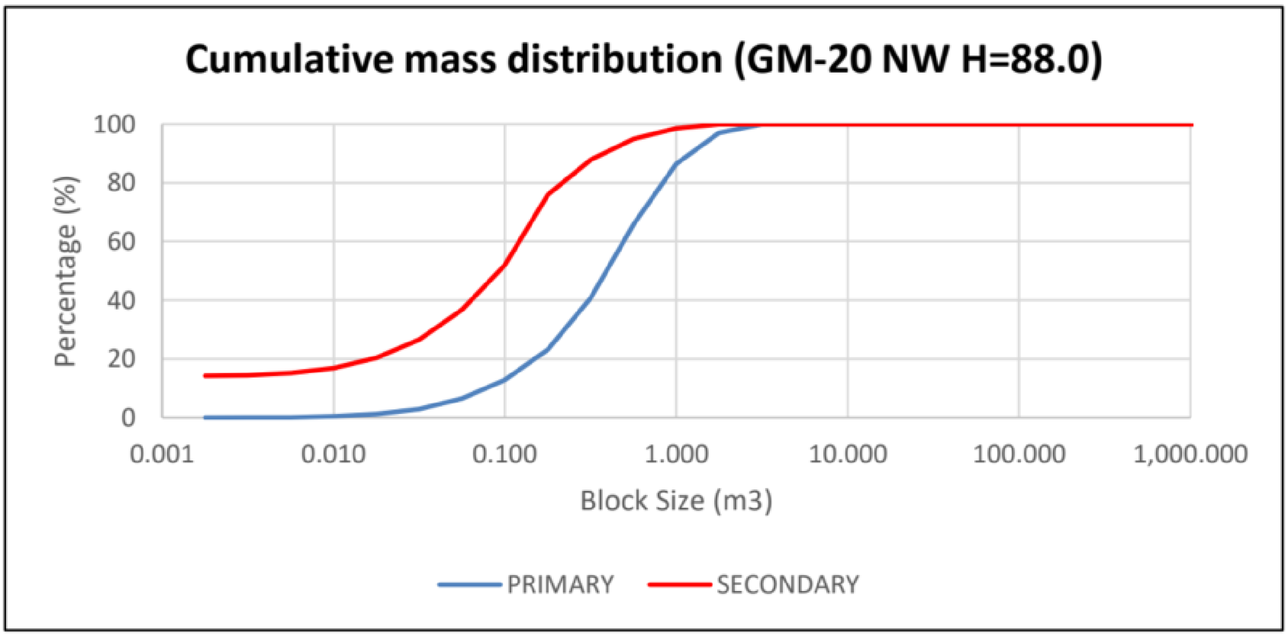


Figure 16.62: Cumulative Mass Distribution – GM-20 (intramineral porphyry) at HOD of 90 m in the NW Region



For Scenario B, the HOD in the south-east region is approximately 100 m and is shown in Figure 16.63 and Figure 16.64. The data was only presented up to a HOD of 88 m, with the assumption made that secondary fragmentation would continue to improve at higher HODs. At this point in time the HOD for the north-west region is already over 160 m, and still well represented by Figure 16.61 and Figure 16.62 above. Scenario C thus utilised the same parameters as Scenario B, with the only difference being that Crusher 2E now crushes the peak throughput of 3.2 Mt per quarter rather than Crusher 1W.

Figure 16.63: Cumulative Mass Distribution – GM-10 (early-mineral porphyry) at HOD of 88 m in the SE Region

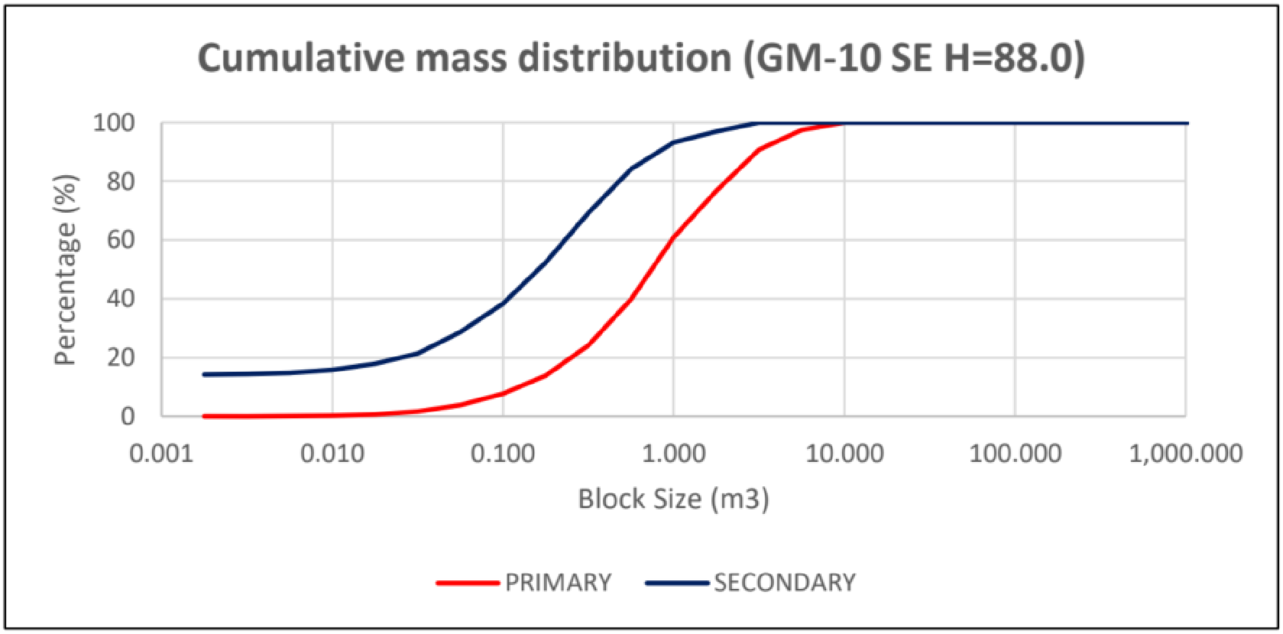
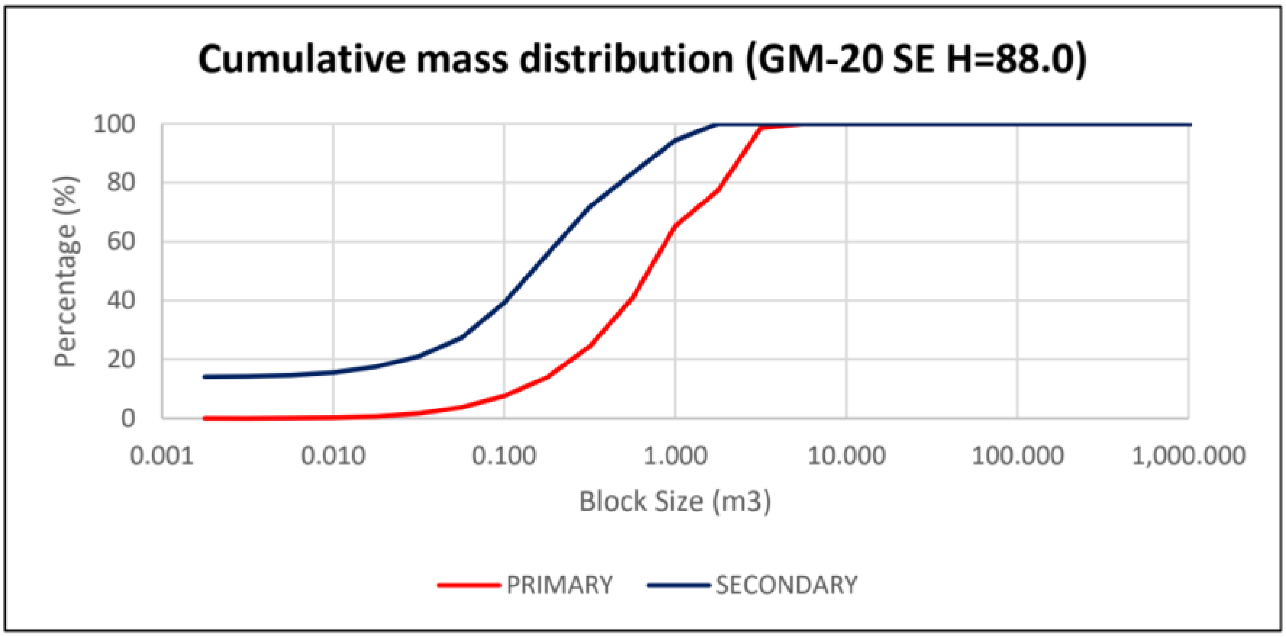


Figure 16.64 Cumulative Mass Distribution – GM-20 (intramineral porphyry) at HOD of 88 m in the SE Region



The tonnage per lithology changes over the life of the cave, and the percentage splits in Table 16.23 are accordingly estimated for the three scenarios. Lithology 05, 10 and 20 contributes approximately 86% of the mining inventory and governs the rock characteristics expected at the crusher during the Life of Mine.

Table 16.23: Composition of Lithologies

Lithology	Percentage
LT01	5.0%
LT02	3.8%
LT05	15.1%
LT06	2.3%
LT10	35.0%
LT20	35.5%
LT30	2.3%
LT31	0.1%
LT32	0.0%
LT40	0.9%
Unclassified	
Total	**100%**

Lithology	Percentage
LT05	15%
LT10	35%
LT20	35%
Total	**86%**

Figure 16.65 and Figure 16.66 show how the lithology ranges change over time for Crusher 1W and Crusher 2E, respectively, showing the three primary lithologies, with the remainder grouped under “other”.

Figure 16.65: Variable Lithology over the Life of the Block Cave – Crusher 1W

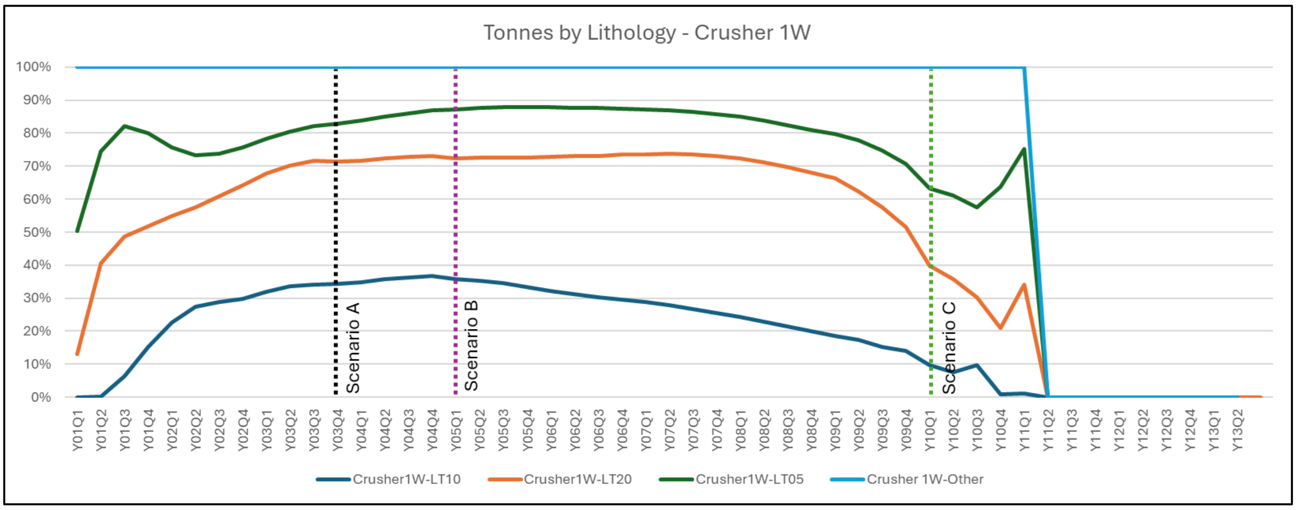
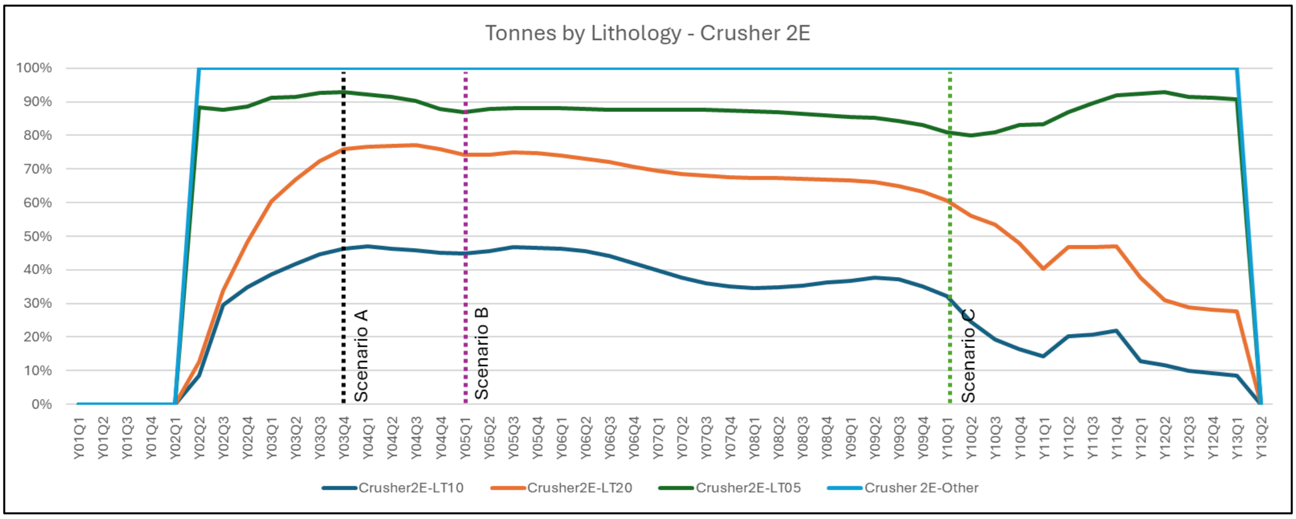
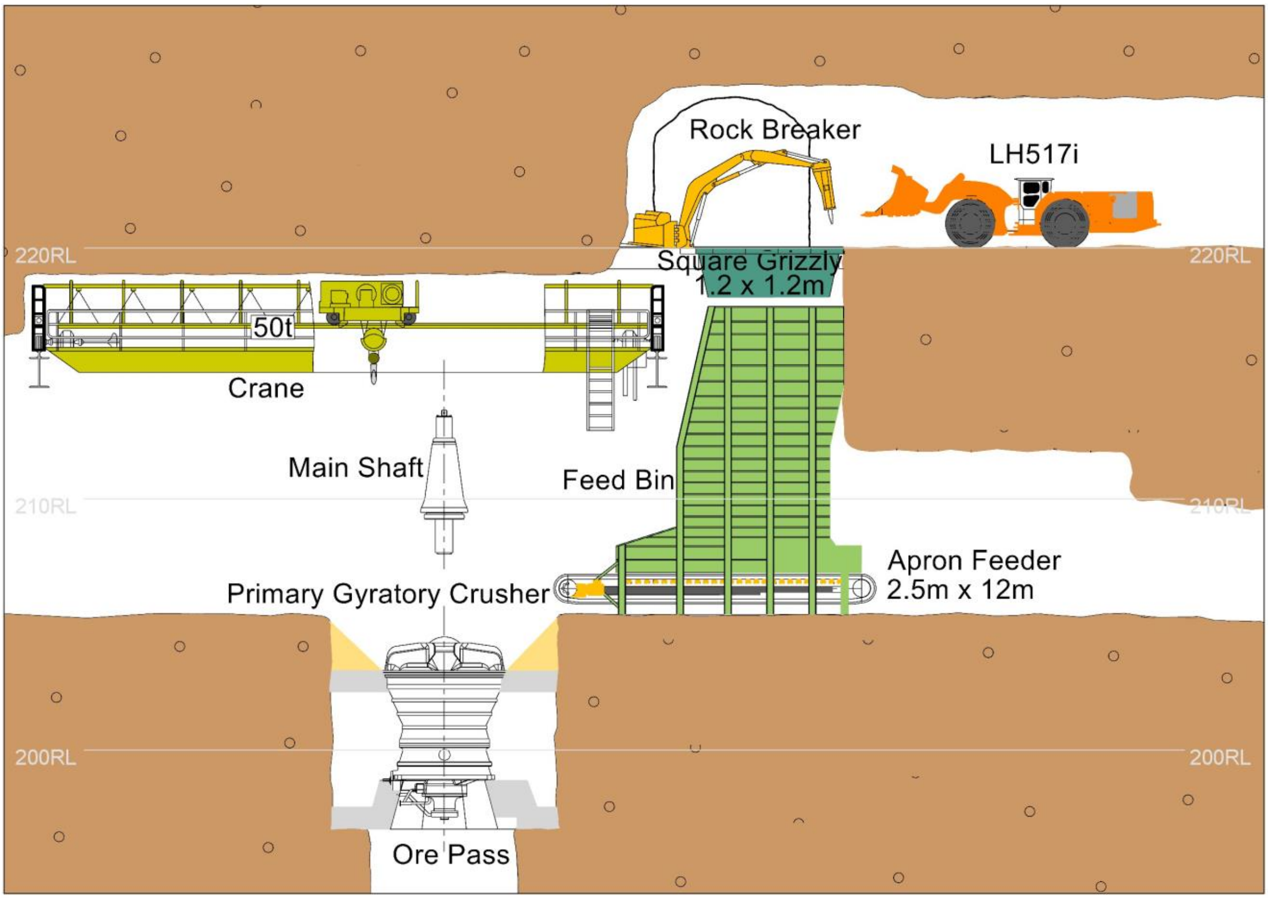


Figure 16.66: Variable Lithology over the Life of the Block Cave – Crusher 2E



The crusher feeder arrangement is shown in Figure 16.67. Three LHDs will be able to tip ore into one feeder bin, with a static grizzly installed above the bin.

Figure 16.67: Loading and Crushing Layout – Section View



In the figure above, a 5,475-class PG crusher is shown for scale, and the mode of processing feed delivery from the LHD tipple position to the PG is outlined. The layout decouples the loading on the extraction level from the operation of the PG and is advantageous if any of the downstream components break down. In such a situation, the Apron feeder can be stopped, the PG can complete crushing the material contained within and later start up under no-load conditions. This also reduces the need for a massive surge capacity beyond the PG, though this is advantageous for other reasons.

In the proposed design within the PFS layout, a 6 m diameter by 30 m high orepass is utilised to decouple production activities on the extraction level with the continuous operation of the conveyor belt transporting processing feed to surface. The ore pass will have a maximum live surge capacity of roughly 1,300 tonnes, which translates to just over half an hour of operation at maximum throughput of each sub-system. This surge is important when considering that the two alternative processing feed streams will need to be combined onto a single trunk line, capable of transporting up to 4,000 tph.

Three crusher suppliers were contacted to provide the Project with budget estimate pricing and to recommend their best class of PG suitable for the Project. The three suppliers contacted are Metso Outotec, FLS and Sandvik. Technical specifications and pricing is highlighted in Table 16.24.

Metso Outotec provided practical design guidance and recommended the MKIII 54-75 PG as the most suitable solution. Each of these PGs are roughly US$1M more expensive than the smaller MKIII 50-65 but have a larger feed opening and are able to crush a larger proportion of the total feed, while reducing the downside risk on productivity of the block cave. It was indicated that the equipment, installation, and commissioning of the supporting equipment normally ranges between 4 to 5 times the CAPEX of the crusher unit alone. Two 54-75 PGs would cost a total of US$6.6M, with the expected total cost of the two crushing complexes equating to US$26.5M.

FLS provided guidance that their BK 54-67 jaw gyratory crusher would be suitable and have not submitted an estimate for the KB 54-67 PG. Seeing as these two crushers are very comparable in size and class, the same CAPEX price is listed here for their PG as their jaw gyratory. The jaw gyratory has a larger feed opening than the PG and is indeed the preferred solution from FLS. FLS also claim that there are significant safety benefits with their offering as all material maintenance is done from above. The set of BK 54-67 jaw gyratory crushers would cost a total of US$5.9M, with the expected total cost of the two crushing complexes equating to US$25.5M.

Sandvik indicated that their CG820i PG would be suitable if the F100 particle size could be reduced, and if this could not be done then it would be best to use the CG830i PG. They indicated that the production throughput rate would underutilise the CG830i, and as such the application falls between their two crushers and it would be best to try and alter the feed material to better suit the CG820i. Both sizes are presented as an equivalent comparison. The set of CG830i PG crushers will cost a total of US$10.8M, with the expected total cost of the two crushing complexes equating to US$32.5M.

Table 16.24: Crusher Supplier Budget Pricing Estimate – October 2024

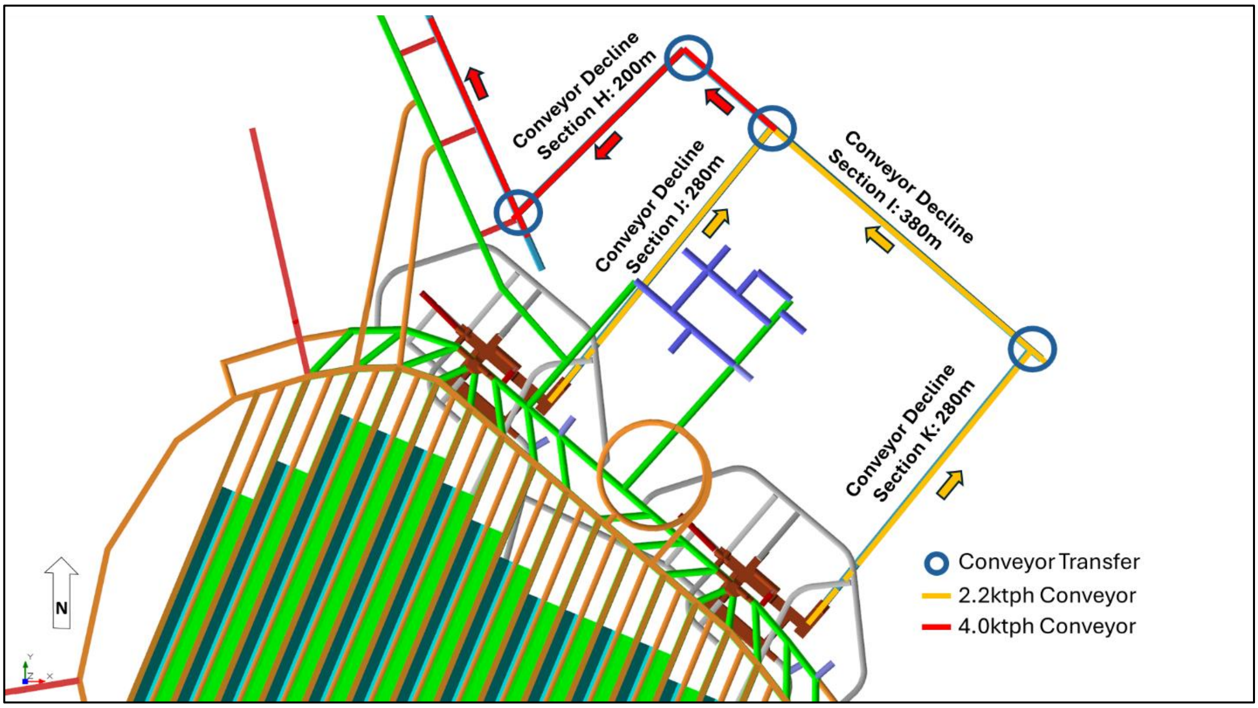
Supplier		FLS		Metso		Sandvik	
Crusher Type	Unit	KB 54-67 Pro	BK 54-67	MKIII 5065	MKIII 5475	CG820	CG830
Date of Estimate		29/10/2024	29/10/2024	9/10/2024	9/10/2024	18/10/2024	18/10/2024
Main Characteristics							
Crusher Feed Opening	mm	1,220	1,350	1,270	1,370	1,372	1,525
F100 Max Feed size	mm	1,037	1,148	1,080	1,165	1,166	1,296
Installed Power	kW	500	500	450	600	525	660
Max OSS	mm	185	200	178	203	228	241
Min OSS	mm	130	130	150	152	127	153
Throughput at Max OSS	t/hr	4,300	4,000	2,958	3,513		
Throughput at Min OSS	t/hr	2500?	2500?	2,414	2,908		
Equipment form factor							
Width	m	4.43	3.67	4.77	4.95	4.76	5.56
Height	m	7.65	7.90	7.03	7.28	7.90	8.94
Weight (Total installed)	t	180	175	153.3	242.1	244	380
Heaviest (Transport)	t	37.1	46	25.8	67.6	50.5	80.5
Heaviest (Maintenance/install)	t	32.5	34	29.6	67.6	50.5	80.5
Delivery Schedule	months						
Capital cost							
Cost	$M (USD)	5.9	5.9	4.6	6.6	8.4	10.8
Additional equipment required		Yes	Yes	Yes	Yes		
Main Apron Feeder (Feed PG)	$M (USD)	2.7	2.7	2.7	2.7	2.7	2.7
Apron Feeder (below Pass)	$M (USD)	1.0	1.0	1.0	1.0	1.0	1.0
Total Mechanical Capex	$M (USD)						
Civils		4.0	4.0	4.0	4.0	4.0	4.0
Structural		5.0	5.0	5.0	5.0	5.0	5.0
Electrical		1.0	1.0	0.8	1.0	1.2	1.5
Install		5.9	5.9	5.5	6.1	6.7	7.5
Excavations							
Excavation Size (Estimate)	$M (USD)	Covered in mining Contract estimate ($25M each) - Not included here					
Cost of Excavation	$M (USD)						
Total CAPEX	**$M (USD)**	**25.5**	**25.5**	**23.6**	**26.5**	**29.1**	**32.5**
Crushing Operating cost	$/t (USD)	0.08	0.08	0.08	0.08	0.08	0.08

It is evident that the FLS and Metso Outotec solutions would both work from a practical perspective, and the total cost of the crushing system differs by less than 5%, which is well within error margins of this study. We consider either solution as practical or sufficient over the life of the block cave. It is recommended that a full trade-off be completed at the next phase of study to determine the most optimal crushing solution versus the productivity of the block cave.

16.4.18 Materials Handling Design – Conveyors

Figure 16.68 shows the plan view of the individual conveyor systems as they combine the block cave feed onto the main conveyor trunk line system.

Figure 16.68: Conveyor System Feeding Main Trunk Line – Plan View



This twin barrel decline system consists of three primary sections of variable lengths. Individual conveyor flights were initially broken into 1,000 m lengths (or less) and aimed to standardise the conveyor components as far as practical. Figure 16.69 highlights those section lengths within the primary decline system. Subsequently some original equipment manufacturers (OEMs) were contacted to obtain feedback and budget pricing for the conveyor sections. One of these OEMs indicated that the three primary decline sections could likely be serviced by single flights of conveyor and thus reduce the number of drive units, and in-line transfers required.

The decline system was designed to align with the surface RopeCon layout that will transfer the crushed processing feed to the process plant located at Productora. This initial leg also provides access to a grade control drilling (GCD) platform level. It also provides easy access to the Return Air Rise (RAR) raisebore hole locations to surface, which provide through-ventilation that would keep the development advance rates efficient as the decline advances.

Figure 16.69: Initial Primary Conveyor System Design to Surface – Plan View

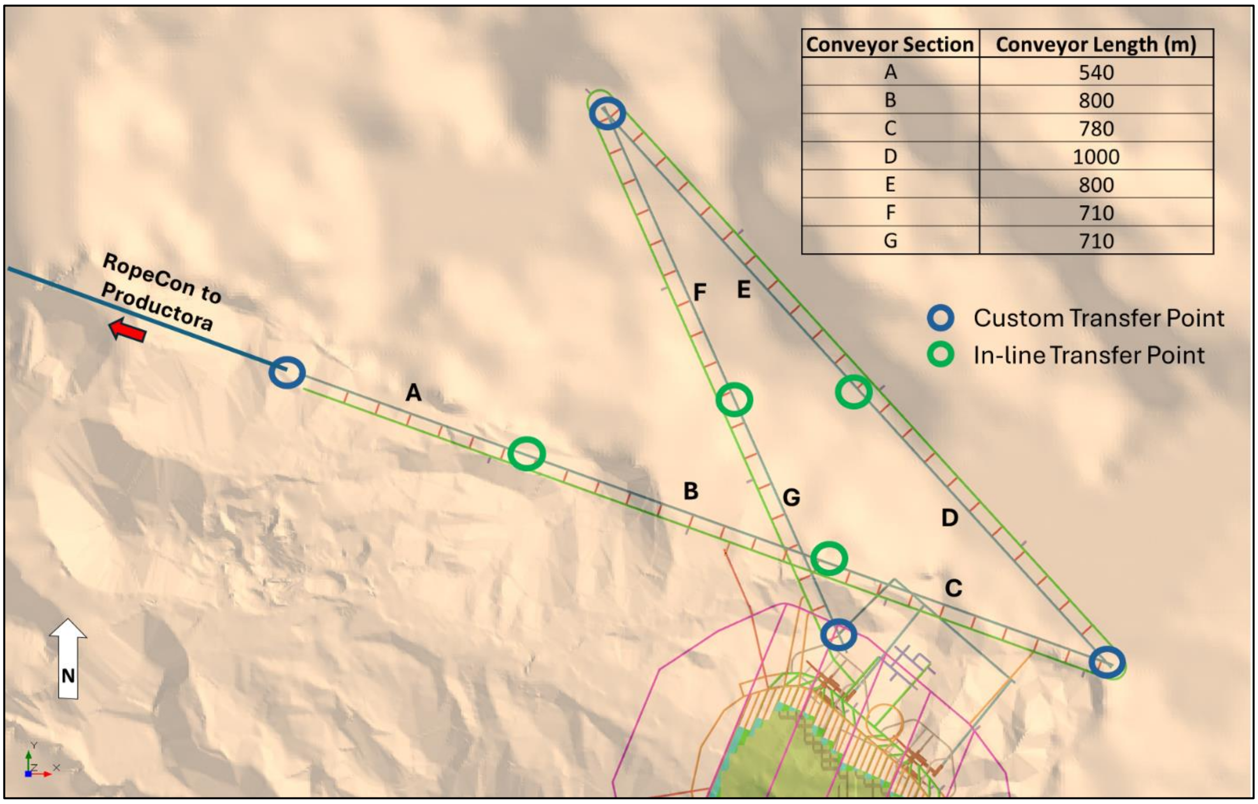


Table 16.25 highlights the initial conveyor specifications for the layout shown in Figure 16.69, and was obtained from the Helix software suite. It is calculated that nearly 8.5 MW of installed power will be required to convey the processing feed produced from the block cave.

Table 16.25: Initial Conveyor and Drive Specifications

Conveyor Drive Motors	Installed Power (kW)	Power Draw - Percentage of Installed (%)	Conveyor Belt Width (mm)	Belt Velocity (m/s)
Motor - A	710	77.9%	1,800	2.5
Motor - B	1,000	89.5%	1,800	2.5
Motor - C	1,000	88.9%	1,800	2.5
Motor - D	1,400	78.1%	1,800	2.5
Motor - E	1,000	89.5%	1,800	2.5
Motor - F	1,000	77.0%	1,800	2.5
Motor - G	1,000	77.0%	1,800	2.5
Motor - H	335	75.6%	1,800	2.5
Motor - I	500	85.5%	1,800	2.5
Motor - J	250	65.9%	1,500	2.0
Motor - K	250	65.9%	1,500	2.0
Total Installed	8,445			

Table 16.26 shows the conveyor design parameters required to sustain the annual production throughputs. Conveyor suppliers were contacted to supply October 2024 budget pricing on the underground conveyor design and also shown in Table 16.26. Four Conveyor suppliers were contacted for budget pricing: FLS, Fenner, Takraf, and CPS.

Only Takraf responded with a rough estimate of EUR6,100/lm, indicating that this should be sufficient to cover the conveyor installation for a +25% estimate. ABGM assessed the conveyor requirements through the Helix software package and costing the conveyor sub-components individually. The relevant Capex sub-sections are shown in Table 16.26, with the electrical components estimated at 10% of the fixed elements sub-section. Earthworks estimated at 25% of the fixed elements sub-section, and lastly a fixed civil allowance of US$10,000,000 to allow for major works at the head end and tail end of conveyor sections, transfer arrangements and significant concrete along the belt length. It is recommended that this scope should be well packaged and studied in the next phase of study, utilising a EPCM company with the relevant engineering disciplines to complete a first principles cost estimate.

Table 16.26: Conveyor Supplier Budget Pricing Estimate – October 2024

Source of Estimate		ABGM (Helix)	Takraf	CPS	Fenner
Date of Estimate		Oct-24	Oct-24	Oct-24	Oct-24
Confidence		+-35%	+-35%	+-35%	+-35%
Design Aspects					
Design belt angle	deg	9	9	9	9
Length of total installation	m	6,676	6,676	6,676	6,676
Annual rate (dry)	tpa	24,528,000	24,528,000	24,528,000	24,528,000
Hours of operation	hrs/yr	6,132	6,132	6,132	6,132
Maximum Throughput (system)	t/h	4,000	4,000	4,000	4,000
P95 Lump size	mm	225	225	225	225
Volumetric Density (structural calc)	t/m3	1.9-2.2	1.9-2.2	1.9-2.2	1.9-2.2
Elevation variance	m	900.0	900.0	900.0	900.0
Belt Width	mm	1500-1800	Unspecified	Unspecified	Unspecified
Belt Speed	m/s				
Installed Power (Total)	kW				
Running Power	kW				
CAPEX	**Unit**				
Fixed Elements (head-, tailend, drives,etc)	USD (M)	13.3	Single unit estimate - EUR6,100/lm	No estimate given.	No estimate given.
Ground Modules	USD (M)	1.7			
Transfer Chutes (Custom)	USD (M)	0.7			
Electrical	USD (M)	1.3			
Commisioning spares	USD (M)	0.6			
Earthworks	USD (M)	3.3			
Concrete	USD (M)	10.0			
Install	USD (M)	4.0			
Delivery to site	USD (M)	0.6			
Contingency	USD (M)	8.8			
Total CAPEX	**USD (M)**	**44.2**	**44.8**		
OPEX					
Energy cost	$/kWhr	0.075	0.075	0.075	0.075
Local maintenance personnel rate	$/hr	18.52	18.52	18.52	18.52
Local maintenance personnel rate	$/annum	40,000	40,000	40,000	40,000
Total OPEX	**USD/t**	**0.137**	**0.137**	**0.137**	**0.137**
Total OPEX	**USD/tkm**	**0.021**	**0.021**	**0.021**	**0.021**
OPEX Total over 10 years	**USD (M)**	**$ 34**	**$ 34**	**$ 34**	**$ 34**
CAPEX per linear metre	**$/m**	**$ 6,619**	**$ 6,710**	**$ -**	**$ -**

16.4.19 Materials Handling Design – Fixed Plant Construction Schedule

Figure 16.70 shows the proposed fixed plant construction schedule, with the civil and structural components of each flight of conveyor implemented in parallel with the conveyor decline face advance. This lags the face by three months to separate development and construction activities. Once the next decline section starts, the previous section of conveyor is completed and commissioned. The commissioned conveyor then serves to reduce waste trucking distance and thus improves the decline development advance rate. The conveyor can be loaded with development waste through the use of a stacker conveyor but will require double ventilation doors to prevent short circuiting of the decline ventilation air. The time it will take between starting the next decline section and a commissioned conveyor section is roughly five months. During this period, waste will still be trucked to surface or the last commissioned conveyor length.

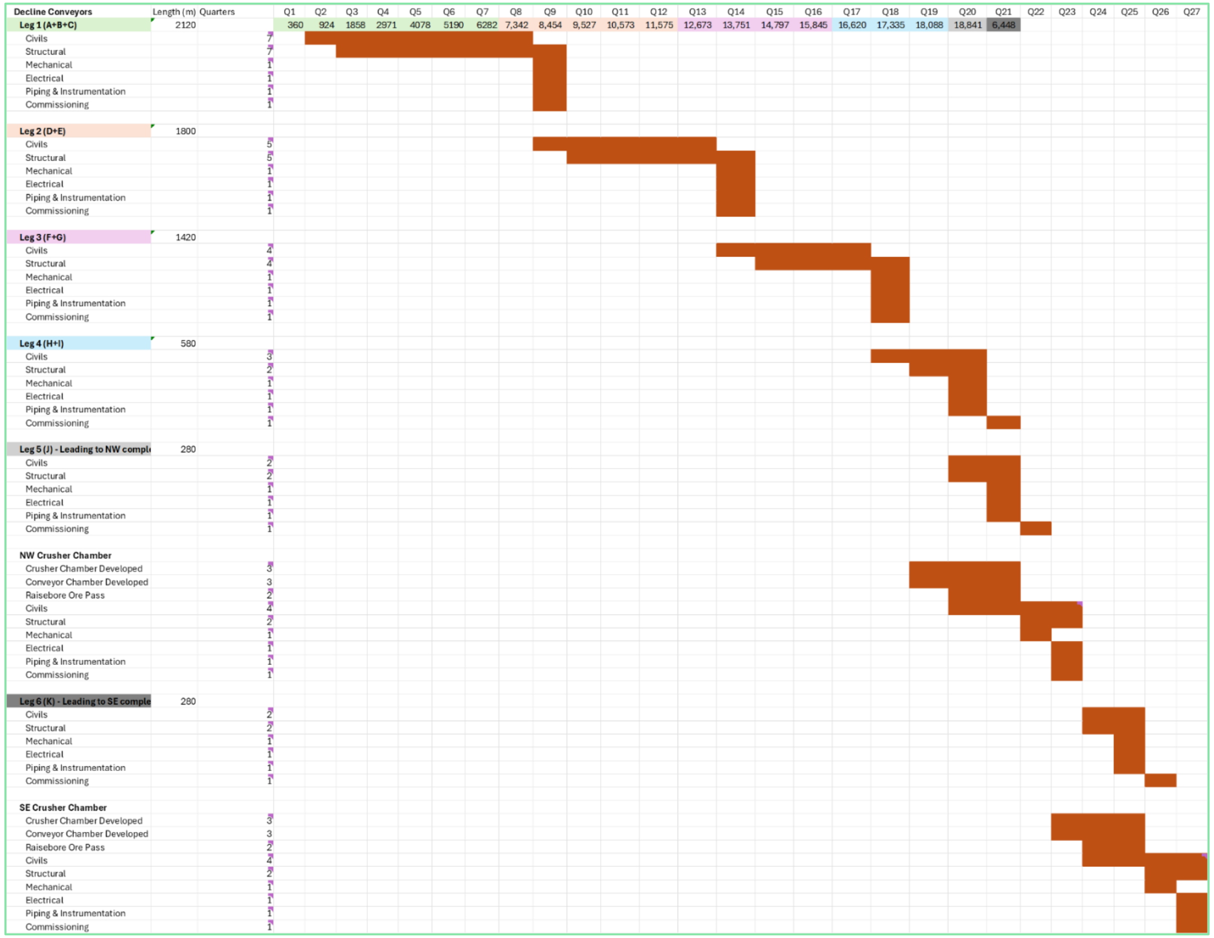
Each of the large chambers for the crusher installation and the conveyor feed installation is expected to take six months to develop and support. These large chambers can be developed simultaneously, followed by the raisebore hole that connects the two excavations. Each of the raisebores will be completed within a two-month window. Civil construction may start in parallel with the raisebore excavation. Structural construction starts two months after civil construction starts. The first critical piece of mechanised equipment that needs to be installed is the 50 t overhead (OH) gantry crane, which will be used to safely construct the apron feeder and PG crusher. If the MKIII 54-75 PG is chosen, the OH gantry crane capacity will need to increase to 70 t due to the heavier sub-components of this significantly larger crusher. Once the OH crane is in place and commissioned, the PG installation can commence. The Installation of the apron feeder and feed bin will follow the PG installation.

A much smaller installation will be required on the conveyor feed level, with a pentice structure below the ore pass, utilising a shorter apron feeder with chute work to feed the individual conveyor sections. The crushing system can be commissioned with development waste, that should pose little problems to the system, and once commissioned, the grizzly and rock breaker system at the LHD tipple location can be constructed and commissioned. This only needs to be in place before caved material is introduced to the crusher complex.

The same construction sequence is then employed for the SE crusher complex, and thus reduces the risk associated with all major construction as they are not required simultaneously.

It is recommended that this scope should be well packaged and studied in the next phase of study, utilising a EPCM company with the relevant engineering disciplines to complete a first principles cost estimate, and develop a detailed construction schedule.

Figure 16.70: Fixed Plant Construction Schedule



16.5 Waste Rock Dumps and Stockpiles

Dumps were designed in 15 m lifts. Each lift is constructed at an approximate angle of repose of 37°. A 15 m setback between each lift maintains the overall angle at 25° to facilitate reclamation and long-term stability. A constant 2.0 t/m^3 loose density was assumed. All the dumps avoid the quebrada boundaries, except for the Cortadera Cuerpo pit valley's where there are approvals to mine and disturb that area.

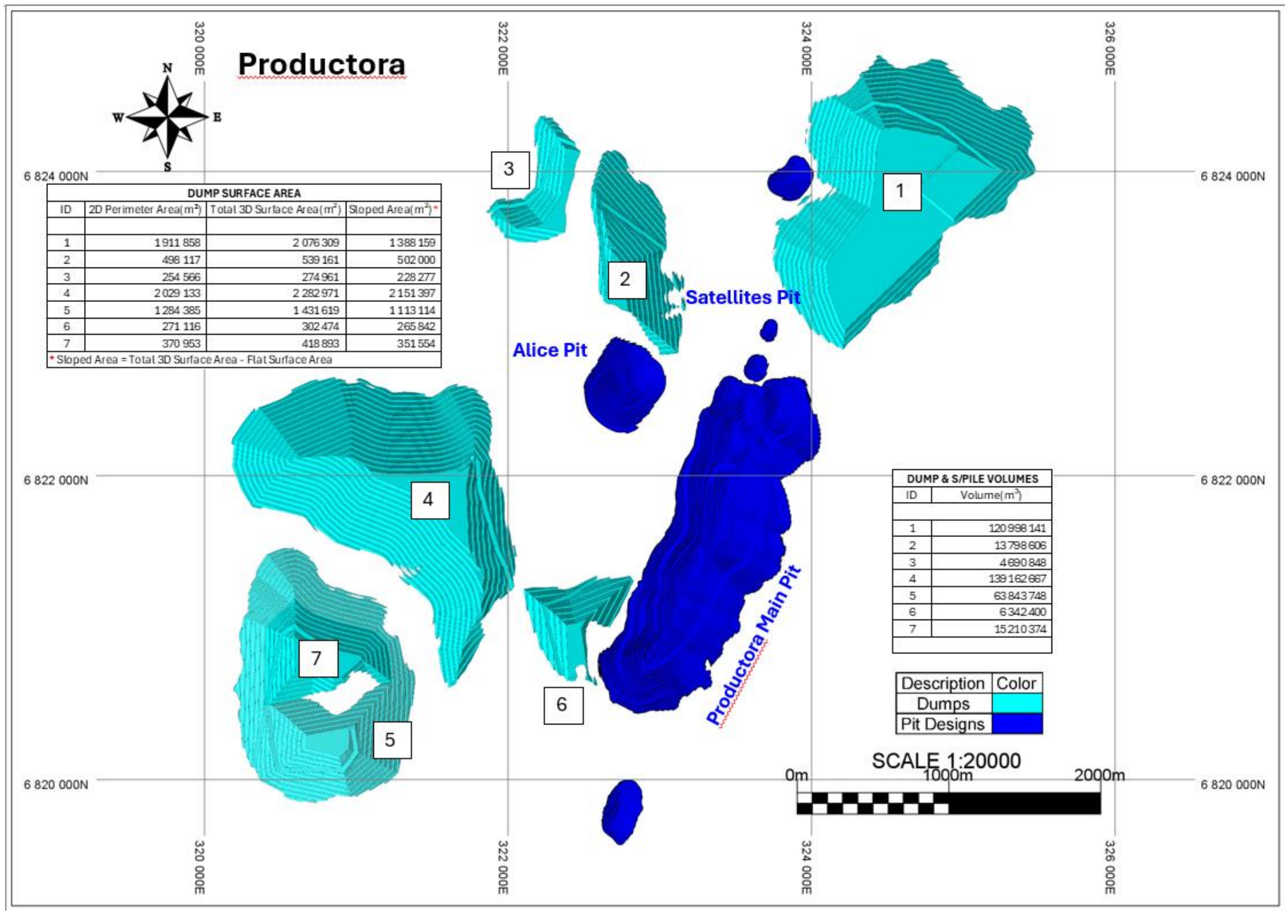
The main ROM pad is located in a strategic location reasonably close to the Productora and Alice open pits, where the main ROM primary crusher will feed a transfer (overland) conveyor belt to the Productora plant comminution circuit. There is also a ROM pad design with a primary crusher at Cortadera. The ROM pad at Cortadera was optimally located to connect the underground block cave ore via the Cortadera surface primary crusher to the RopeCon loading point.

All stockpiles (SP) and Waste Rock Dumps (WRD) were specifically designed at an angle of repose of 36 degrees with sufficient berms to provide for stable impoundment designs. The geotechnical team reviewed the waste dumps and stockpile designs and simulated the factors of safety to ensure they are deemed reasonable with lower risk of failures.

16.5.1 Productora and Alice Waste Rock Dumps

Productora and Alice utilises a combination of Waste Rock Dumps, Heap Leach Dumps and Stockpiles, aiming to minimise the haulage costs associated with transporting and placing production inventory. Figure 16.71 shows the placement and relative size of these dumps and stockpiles.

Figure 16.71 : Plan View of the Productora Waste Rock Dumps and Stockpile Layouts



The calculated capacities of the respective dumps and stockpiles are shown in Table 16.28.

Table 16-27: Productora Rock Dump and Stockpile Capacities		
DUMP ID	**TYPE**	**Volume Loose (Mm³)**
1	WRD	121.0
2	S/PILE SULP	13.8
3	S/PILE OX	4.7
4	WRD	139.2
5	WRD	63.8
6	S/PILE SULP	6.3
7	WRD	15.2
Total Waste Capacity		**364.0**

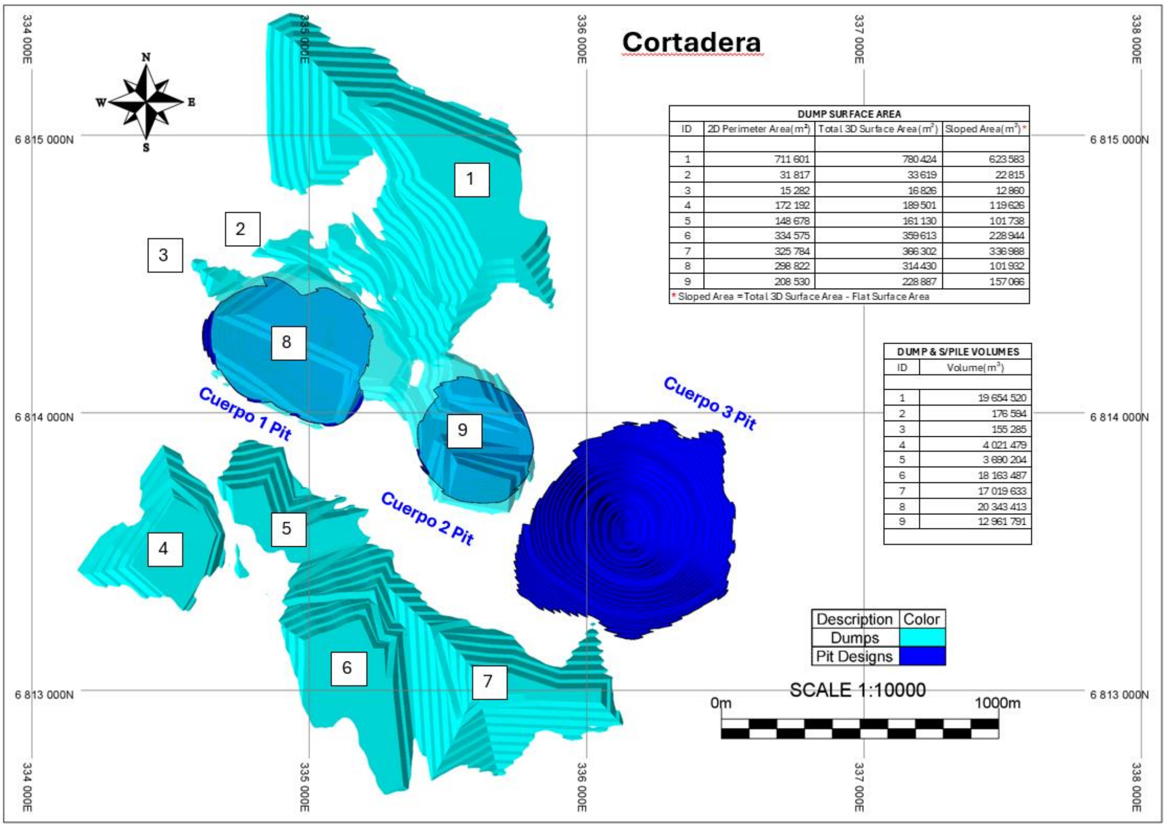
The Productora Pit and Alice Pits produce approximately 171 Mm^3 in-situ waste rock and 230 Mm^3 loose (swelled) waste rock needing impoundment at/around the Productora and Alice surface areas. The waste dump designs clearly provide sufficient/excess impoundment volumes as required for these two open pit areas/operations as can be seen in the designed capacity of 364Mm^3 value.

16.5.2 Cortadera Waste Rock Dumps

Cortadera utilises a combination of WRDs, aiming to minimise the haulage costs associated with transporting and placing production inventory. Figure 16.72 shows the placement and relative size of these dumps.
In Figure 16.72, WRD 1 will be constructed first using Cuerpo 1 and 2's open pit waste to establish the Rope-con loading area for ore to be transported to the concentrator near Productora. WRD 1 will also serve as the platform for the underground portal development for the Cuerpo 3 block cave mine. The waste from the Cuerpo 3 open pit and any waste from Cuerpo 3 the underground development will be backfilled into/ deposited onto the Cuerpo 1 and 2 open pit's and the associated WRD's (8/9).

SP 5 is a contingency ore stockpile for Cuerpo 3 pit. This stockpile holds approximately 5% of the total Cuerpo 3 concentrate ore.

Figure 16.72 : Plan View of the Cortadera Waste Rock Dumps and Stockpile Layouts



The calculated capacities of the respective waste dumps and heap leach pads are shown in Table 16.29.

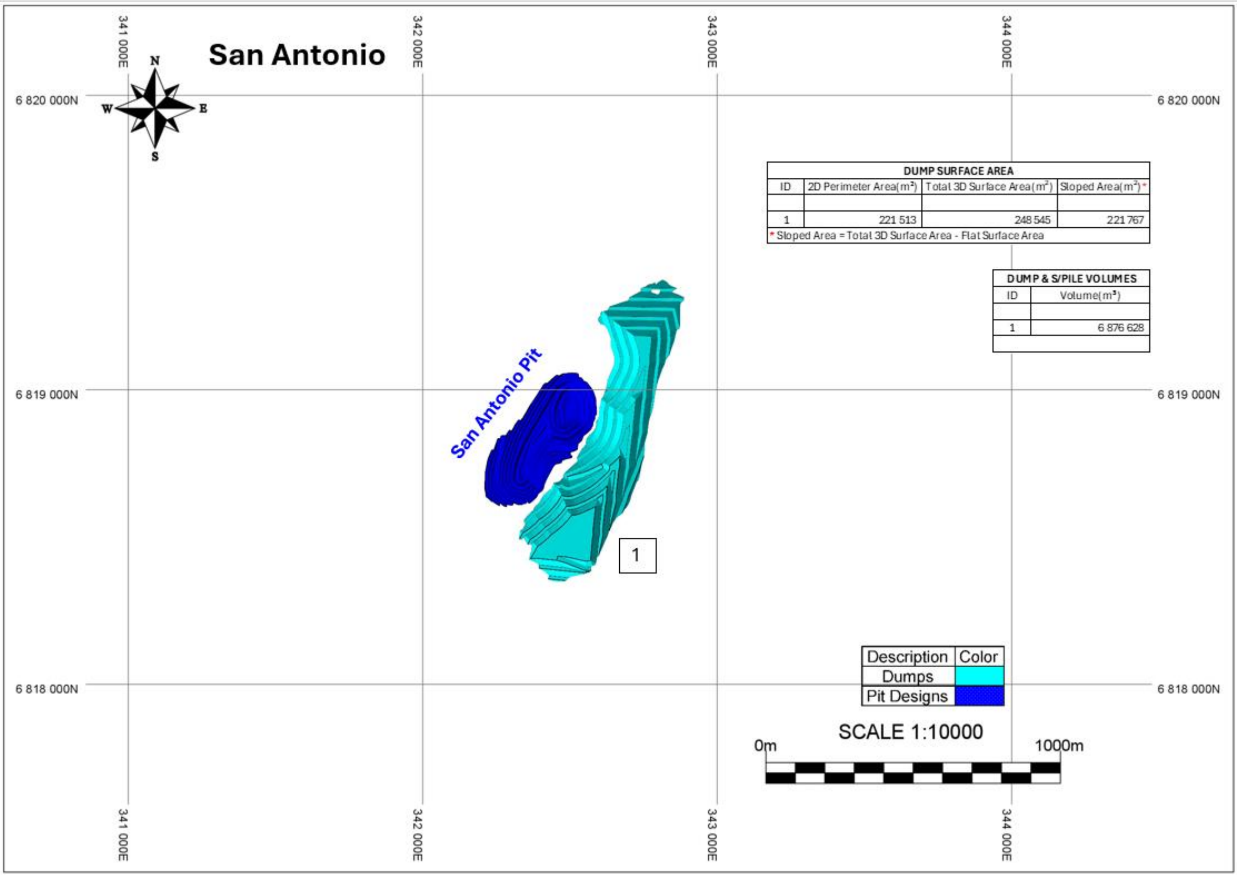
Table 16-28 : Cortadera Waste Rock Dump and Stockpile Capacity		
DUMP ID	**TYPE**	**Volume Loose (Mm^3)**
1	WRD	19.7
2	WRD	0.2
3	WRD	0.2
4	WRD	4.0
5	WRD	3.7
6	S/PILE SULP	18.2
7	S/PILE OX	17.0
8	WRD	20.3
9	WRD	13.0
Total WRD Capacity		**96.3**

The Cortadera Pit waste volume that will require placement being roughly 47.4 M lcm with a further 1.5 M lcm associated with UG waste development.

16.5.3 San Antonio Waste Rock Dumps

San Antonio utilises a single WRD located to the east of the pit, shown in Figure 16.73.

Figure 16.73 : Plan View of the San Antonio Waste Dump Layouts



The calculated capacity of the waste dump is shown in Table 16.27.

Table 16.27 : San Antonio Waste Rock Dump Capacity	
Name	**Volume (lcm)**
WRD	6.8
Total Waste	**6.8**

The San Antonio Pit waste volume is approximately 4 M lcm and the San Antonio dump area also allows for some ore stockpiling if required.

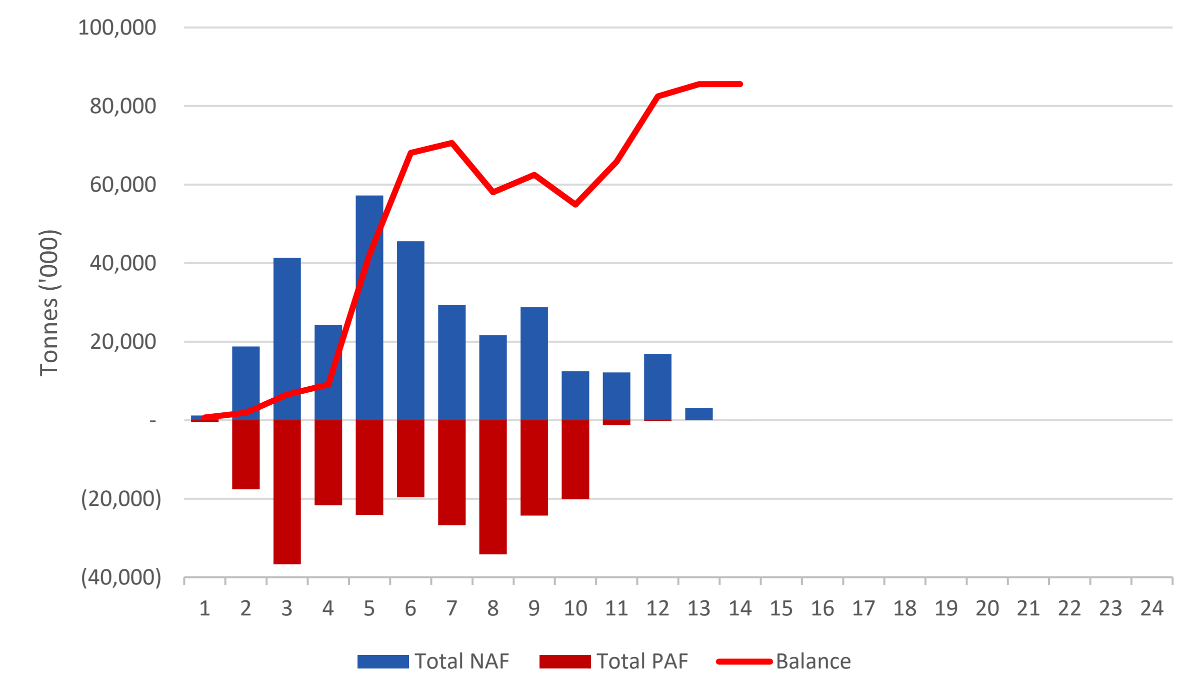
16.5.4 Acid Rock Drainage

Geochemical assessment was completed to assess for potential acid generation and neutralisation.

- Neutralization potential (NP) = (CA_PCT/40.08) * 100.09*10
- Acid potential (AP) = S_PCT*31.25
- Net result (NPP) = NP-AP

Using these conditions, waste rock was categorised accordingly with the following delivery schedule:

Figure 16.74 : PAF/NAF Schedule



16.5.5 Waste Dumps and Stockpile Slope Stability Evaluation

This section summarises the findings from the Geotechnical analysis developed in the report, *'GEO-AN-WDSP-CF-PFS-HCH-01_rev1'*.

Stability analysis for waste dump and stockpile designs has considered the maximum capacity option of each structure, under dry substratum.

A conservative approach was adopted for the shear resistance of the deposited material, acceptance criteria, and horizontal seismic coefficient. These conservative measures, particularly in cases where data gaps exist, help to reduce uncertainty and ensure a more robust and reliable analysis.

The stability of waste dumps and stockpiles was assessed using Limit Equilibrium Analysis, following the acceptability criteria recommended by Sernageomin (2023[1]). The results indicate that the designs for waste dumps and stockpiles meet acceptability criteria during both the operational and post-closure phases.

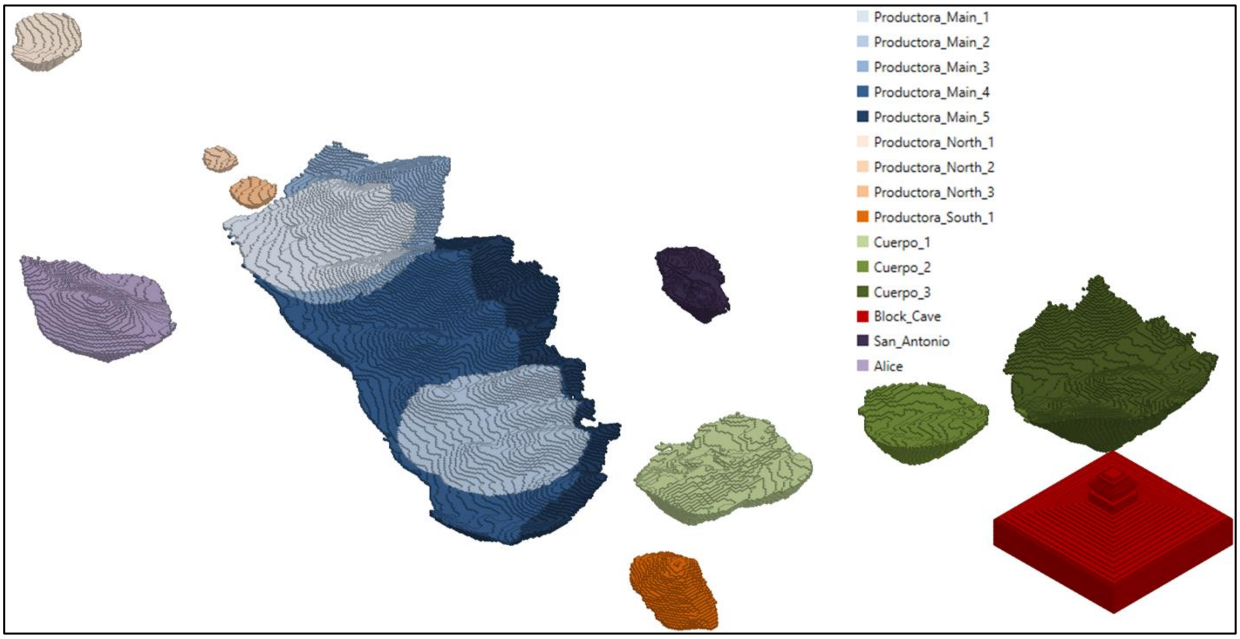
[1] Sernageomin (2023). Guía de Presentación de Proyectos de Botaderos de Estériles en Faenas Mineras. IT-SEGMIN-001.

16.6 Strategic Mine Schedule

Strategic mine scheduling is an industry best practice that enhances the long-term value of mining operations and supply chains. Optimisation technologies—such as mixed-integer linear programming (MILP)—are used to identify mathematically optimal solutions for scheduling models that define available source material, processing options, and the constraints governing material access and processing.

For the Costa Fuego Project, scheduling was conducted using the strategic scheduling software Minemax. The following sections outline the Minemax model's key elements, including battery limits, objectives, operational mechanics, and outcomes.

Figure 16.75 : Schedule Stages



16.6.1 Block Model

The block model comprises four deposits with unique characteristics, spatially grouped into two mining regions:

- Productora Region: Contains the Productora and Alice deposits.
- Cortadera Region: Contains the Cuerpo 1, 2 & 3 open pits, block cave and San Antonio deposits.

Table 16.28 : Resource Category and Weathering Fields

VALUE	RESCAT	WTCODE
1	-	Oxide
2	Indicated	Transitional
3	Inferred	Fresh
4	Unclassified	-

Tonnage summaries for these deposits are provided in Table 16.29. Materials are categorised as follows:

- Oxide: WTCODE = 1, RESCAT = 2, CU_PCT > 0.2
- Transitional: WTCODE = 2, RESCAT = 2, CU_PCT > 0.2
- Fresh: WTCODE = 3, RESCAT = 2, CU_PCT > 0.2
- Waste: All remaining material

Table 16.29 : Pit-Limited Block Models and Tonnage Summary

Region	Block Model		Oxide	Transitional	Fresh	Waste	Total
Productora	prod_202412_pfs.csv	Mt	17.1	24.5	111.5	511.4	**664.5**
	alice_202406_pfs_eng.csv	Mt	2.9	0.7	5.2	37.4	**46.1**
Cortadera	cort_c123_202412_pfs.csv	Mt	5.5	6.7	42.1	118.4	**172.7**
	sa_202406_pfs_eng.csv	Mt	0.3	2.2	0.9	9.0	**12.4**
Total (pit-limited)		**Mt**	**25.7**	**34.1**	**159.7**	**676.2**	**895.7**

Numerous fields within the block model inform the schedule. Details on these fields are provided in Table 16.28 through Table 16.35.

Table 16.30 : Metallurgical Recovery Fields and Block Model Calculations by Mine Area

Block Model Field	Calculation
Productora	
CU HEAP LEACH OXIDE	CUHLOX = (58.209 * CU_PCT) + (-7.5857 * NA_PCT) + 51.5024 *(Maximum 95%)*
CU DUMP LEACH OXIDE	CUDLOX = 0.51 * (-2.614 * LI_PPM + 95.97) *(Minimum 10%)*
CU DUMP LEACH TRANS	CUDLTR = 0.51 * (-2.614 * LI_PPM + 95.97) *(Minimum 10%)*
CU DUMP LEACH FRESH	CUDLFR = 0.51 * (-2.614 * LI_PPM + 95.97) *(Minimum 10%)*

CU CONCENTRATOR TRANS	IF(CU_PCT>0.05) CUCONTR = (19.609 *CU_PCT) + 63.443
	IF(CU_PCT<0.05) CUCONTR = 0
	(Maximum 90%)

AU CONCENTRATOR TRANS	IF(AU_PPM>0.02) AUCONTR = (145.4 * AU_PPM) + 38.549
	IF(AU_PPM<0.02) AUCONTR = 0
	(Maximum 80%)

MO CONCENTRATOR TRANS	MOCONTR = 56
AG CONCENTRATOR TRANS	AGCONTR = 40

CU CONCENTRATOR FRESH	IF(CU_PCT>0.05) CUCONTR = (9.072 * CU_PCT + 83.66)
	IF(CU_PCT<0.05) CUCONTR = 0
	(Maximum 95%)

AU CONCENTRATOR FRESH	IF(AU_PPM>0.02) AUCONTR = (145.4 * AU_PPM) + 38.549
	IF(AU_PPM<0.02) AUCONTR = 0
	(Maximum 80%)

MO CONCENTRATOR FRESH	MOCONTR = 0.9 * (5.676 * LOGN(MO_PPM) + 51.191)

Table 16.30 : Metallurgical Recovery Fields and Block Model Calculations by Mine Area	
Block Model Field	**Calculation**
	(Maximum 95%)
AG CONCENTRATOR FRESH	AGCONFR = 40
Cortadera	
CU HEAP LEACH OXIDE	Cuerpo 1: CUHLOX = 36.3 Cuerpo 2&3: CUHLOX = (-2.2209 * CO_PPM) + 93.58 *(Maximum 95%, Minimum 10%)*
CU DUMP LEACH OXIDE	Cuerpo 1: CUHLOX = 36.3 Cuerpo 2&3: CUDLOX = 0.65 * (-2.2209 * CO_PPM) + 93.58 *(Minimum 10%)*
CU DUMP LEACH TRANS	Cuerpo 1: CUHLOX = 36.3 Cuerpo 2&3: CUDLTR = 0.65 * (-2.2209 * CO_PPM) + 93.58 *(Minimum 10%)*
CU DUMP LEACH FRESH	Cuerpo 1: CUHLOX = 36.3 Cuerpo 2&3: CUDLFR = 0.65 * (-2.2209 * CO_PPM) + 93.58 *(Minimum 10%)*
CU CONCENTRATOR TRANS	CUCONTR = 17.016*LOGN(CU_PCT)+86.378 *(Maximum 80%, Minimum 8%)*
AU CONCENTRATOR TRANS	AUCONTR = 104.74*AU_PPM+29.42 *(Maximum 90%)*
MO CONCENTRATOR TRANS	MOCONTR = 21.8
AG CONCENTRATOR TRANS	AGCONTR = 27
CU CONCENTRATOR FRESH	CUCONFR = 17.016*LOGN(CU_PCT)+96.378 *(Maximum = 90%, Minimum = 18%)*
AU CONCENTRATOR FRESH	AUCONFR = 104.74*AU_PPM+29.42 *(Maximum 90%)*
MO CONCENTRATOR FRESH	MOCONFR = 0.9 * (12.563 * LOGN(MO_PPM) + 21.88) *(Maximum 92%)*
AG CONCENTRATOR FRESH	AGCONFR = 27
Block Cave	
CU CONCENTRATOR FRESH	8.615*LOGN(CU_PCT)+96.122 *(Maximum = 95%)*
AU CONCENTRATOR FRESH	AUCONFR = 30.368*AU_PPM+51.637 *(Maximum 90%)*
MO CONCENTRATOR FRESH	MOCONFR = 0.9 * (12.563 * LOGN(MO_PPM) + 21.88) *(Maximum 92%)*
AG CONCENTRATOR FRESH	AGCONFR = 38
Alice	
CU HEAP LEACH OXIDE	46 (Fixed for all oxide blocks)
San Antonio	
CU HEAP LEACH OXIDE	50 (Fixed for all oxide blocks)

Table 16.31 : Mill Throughput Calculations		
FIELD	**Description**	**Calculation**
TPUTBM	Ball Mill throughput	37905/(BWI * 0.7881 - 2.453)
TPUTSAG	SAG Mill throughput	23230/(0.4184 * (5.1113 * ($DWI^{0.7059}$) - 0.4305))
TPH	Final Mill throughput	Minimum value of Ball Mill and SAG Mill throughputs

Table 16.32 : Acid Consumption – Heap Leach		
FIELD	**Domain (LGCCODE)**	**Calculation/Value**
Productora		
ACID_HL	All Domains	0.267 * LOG10(CA_PCT)) + 1.3755
Cortadera		
ACID_HL	Calcsilicate #1 (1)	36 (kg/t H2SO4)
	Andesitic volcaniclastics #1 (2)	21 (kg/t H2SO4)
	High-apatite calcsilicate (3)	20 (kg/t H2SO4)
	Andesitic volcaniclastics #2 (4)	21 (kg/t H2SO4)
	Moderate-apatite calcsilicate (5)	36 (kg/t H2SO4)
	Andesitic volcaniclastics #3 (6)	21 (kg/t H2SO4)
	Apatite carbonate (7)	56 (kg/t H2SO4)
	Carbonate (8)	56 (kg/t H2SO4)
	Andesitic volcaniclastics #4 (9)	21 (kg/t H2SO4)
	Calcsilicate #2 (10)	36 (kg/t H2SO4)
	Andesitic volcaniclastics #5 (11)	21 (kg/t H2SO4)
	Porphyry Intrusions (30)	25 (kg/t H2SO4)
Alice		
ACID_HL	All Domains	6 (kg/t H2SO4)
San Antonio		
ACID_HL	All Domains	10 (kg/t H2SO4)

Table 16.33 : Acid Consumption – Dump Leach (Productora Only)		
LGCCODE	**Domain**	**Acid Consumption Value (kg/t H2SO4)**
1	Albite	7
2	Background	15
3	Kaolinite	26
4	Kspar	3
5	Magnetite Amphibole	26
6	Sericite Albite	3
7	Sericite	6
8	Sodic Calcic	16
9	Not Classified	12
-99	Outside of Model Area	99

Table 16.34 : Royalties (RYLTCODE)		
FIELD	**Royalty**	**Description**
0	No Royalty	

Table 16.34 : Royalties (RYLTCODE)		
FIELD	**Royalty**	**Description**
1	CCHEN	Uranio 1 al 70 : Subject to 2% non-gold NSR royalty, 4% gold NSR royalty, and 5% non-metallic NSR royalty
2	Zapa	Zapa 1 Al 6 : Subject to 1% Gross royalty on all products
3	Montosa	Montosa 1 al 4 : Subject to 3% NSR royalty on all products

Table 16.35 : Variable Mining Cost (VCOST_MI)			
Mine Area	**Base Value**	**Adjustment Factor**	**Base Level**
Productora	$2.03/t	(+US$0.01/t per 5m bench above base, +US$0.0125/t per 5m bench below base	810 mRL
Cortadera	$2.03/t		800 mRL
Alice	$2.23/t		1,010 mRL
San Antonio	$2.23/t		1,095 mRL

16.6.2 Material Destination Pathways

The schedule model is configured with material pathways dictated by material characteristics such as resource classification and economic viability. The available pathways include:

- To Concentrator (Primary)
- To Heap Leach
- To Concentrator (Secondary – Pre-Crush then fed to Concentrator)
- To Dump Leach
- To Stockpile
- To Waste Dump

Only economic material with an Indicated resource classification is considered for processing routes. The schedule results inform final capital and operating cost estimates, using interim cost assumptions that are validated against the financial model.

16.6.2.1 To Concentrate (Primary)

The primary concentrate pathway, processed at the Productora concentrator, is the most profitable option. Revenues are derived not only from copper but also from molybdenum, gold, and silver.
Throughput is calculated using the following formula:

*Throughput = TPH * Operating hours*

Here, the TPH (tonnes per hour) is determined as the minimum value of TPUTBM (ball mill throughput) and TPUTSAG (SAG mill throughput), based on regressions using comminution data (see Table 16.31 and Section 16 for further details). The concentrator has a nameplate operating capacity of 8,000 hours per year (approximately 20.7 million tonnes per annum).

Commissioning occurs in Year 3, Quarter 1 (Y3Q1) with a ramp-up as follows:

- Y3Q1: 40% capacity (800 hours/quarter)
- Y3Q2: 80% capacity (1,600 hours/quarter)
- Y3Q3: 100% capacity (2,000 hours/quarter, continuing to the life of mine)

Throughput rates vary with material type; for example, the Productora open pit can be fed at a higher rate than the Cortadera block cave due to more favourable comminution properties.

The concentrator accepts transitional and fresh rock with an Indicated resource classification and applies the cost assumptions detailed in Table 16.36.

Table 16.36 : Concentrate Material Destination Cost Assumptions

Ref.	Ref. Description	Unit	Value	Comment
A	Operating Hours	hrs/yr	8,000	
B	Tonner per Hour	t/hr	2,400	example for calculation
C	Copper Grade	%	0.2	example for calculation
D	Copper Recovery	%	85	example for calculation
E	Copper Concentrate	%	25	
F	Cu_Conc_TonnesRatio	x	0.0068	= ([C] / 100) * [D] / [E]

	Molybdenum Float		Productora	Cortadera	San Antonio	Comment
G	Annual	M$/yr		1.05		Recalculate as variable
H	Power	$/t cucon		0.36		
I	Consumables	$/t cucon		0.60		

	Concentrator		Productora	Cortadera	San Antonio	Comment
J	Concentrator Variable	$/t		5.17		
K	Molybdenum Fixed	$/t		0.05		= [G] / ([A] * [B])
L	Molybdenum Variable	$/t		0.01		= ([H] + [I]) * [F]
M	Rope Conveyor	$/t	-	0.10	0.10	
N	Road Transport	$/t	-	-	3.40	
O	Concentrator Fixed	M$/q		10.85		

	Comparison to Final		Productora	Cortadera	San Antonio	Comment
P	Scheduled	$/t	5.23	5.33	8.74	= [J] + [K] + [L]+ [M]+ [N]
Q	Final	$/t	5.20	5.41	8.81	
R	Delta	%	101%	99%	99%	= [Q] / [R]

A base variable cost of $5.17 per tonne is applied across all feed sources. Additional transport costs include $0.10 per tonne for Cortadera and San Antonio material (via rope conveyor) and a further $3.40 per tonne for San Antonio (via road transport over 17 km at $0.20 per tonne-km). The resulting average processing variable cost from the schedule is $5.31 per tonne.

16.6.2.2 To Heap Leach

The heap leach pathway is available exclusively to oxide material (WTCODE = 1) with an Indicated resource classification. Material processed via this route is crushed by jaw crushers with a throughput of 4.0 Mt/yr, then undergoes acid treatment on designated pads. The resulting pregnant leach solution is processed through a 12,000 t/yr solvent extraction and electrowinning (SX-EW) circuit to recover copper.

Commissioning for heap leaching occurs in Y3Q1 with the following ramp-up:

- Y3Q1: 50% capacity (500,000 tonnes/quarter)
- Y3Q2: 90% capacity (900,000 tonnes/quarter)
- Y3Q3: 100% capacity (1,000,000 tonnes/quarter, continuing to the life of mine)

Cost assumptions are detailed in Table 16.37.

Table 16.37 : Heap Leach Material Destination Cost Assumptions

Ref.	Ref. Description	Unit	Value	Comment
A	Copper Grade	%	0.35	example for calculation
B	Copper Recovery	%	65	example for calculation
C	Metal Ratio	x	0.0023	= ([A] / 100) * ([B] / 100)

	Backend		Productora	Cortadera	San Antonio	Comment
D	Backend Cost	$/t	555.82			

	Heap Leach Total		Productora	Cortadera	San Antonio	Comment
E	Variable	$/t	5.26			
F	Backend	$/t	1.26			= [D] * [C]
G	Rope Conveyor	$/t	-	0.10	0.10	
H	Road Transport	$/t	-	-	3.40	
I	Fixed	M$/y	6.48			

	Comparison to Final		Productora	Cortadera	San Antonio	Comment
J	Scheduled	$/t	6.52	6.62	10.02	= [E] + [F] + [G]+ [H]
K	Final	$/t	6.17	6.21	9.61	
L	Delta	%	106%	107%	104%	= [J] / [K]

A base variable cost of $5.26 per tonne is applied, with additional transport expenses similar to the primary concentrator pathway. The schedule yields an average heap leach variable cost of $6.58 per tonne.

16.6.2.3 To Concentrate (Secondary) – Pre-Crush with Oxide Crusher then fed to Concentrator

When the oxide crushing circuit (designed primarily for heap leaching) operates below full capacity, its surplus can pre-crush material for the concentrator. This is particularly advantageous when the concentrator's performance is constrained by the SAG mill due to coarser material from sources such as the Cuerpo open pits or Block Cave. Pre-crushing enables material to bypass the SAG mill and feed directly into the ball mill, thereby increasing throughput.

Costing for this process is under review. During schedule optimisation, it is costed similarly to the primary concentrator pathway plus an additional US$0.72 per tonne. Metallurgical assumptions remain unchanged, and the fixed concentrator cost is shared between both the primary and secondary streams. The schedule indicates an average processing variable cost of $6.00 per tonne.

16.6.2.4 Dump Leach

Dump leach processing is applied to mined material with low-grade content that is uneconomical for concentration or heap leaching. This pathway requires additional drilling and blasting for finer fragmentation before transporting the material to a dump leach pad. Acid treatment liberates metals from the fragmented material, with copper subsequently recovered via an SX-EW circuit.

While evaluated for both the Productora and Cortadera regions, iterative scheduling excluded dump leach material from Cortadera due to lower economic viability. Cost assumptions are detailed in Table 16.38.

Table 16.38 : Dump Leach Material Destination Cost Assumptions

Ref.	Ref. Description	Unit	Value	Comment
A	Copper Grade	%	0.13	example for calculation
B	Copper Recovery	%	40	example for calculation
C	Metal Ratio	x	0.0005	= ([A] / 100) * ([B] / 100)

	Backend		Productora /Alice	Comment
D	Backend Cost	$/t	555.82	

	Heap Leach Total		Productora / Alice	Comment
E	Variable	$/t	5.00	
F	Backend	$/t	0.28	= [D] * [C]
G	Fixed	M$/y	2.31	

	Comparison to Final		Productora / Alice	Comment
H	Scheduled	$/t	5.28	= [E] + [F]
I	Final	$/t	5.02	
J	Delta	%	105%	= [H] / [I]

From the schedule, the resulting average variable cost is $5.28 per tonne.

16.6.2.5 To Stockpile

All material may be stockpiled for later reclamation, which incurs a rehandle cost of $0.97 per tonne. The schedule optimisation accounts for these costs whilst managing material movement.

16.6.2.6 To Waste Dump

Material classified as waste is deposited onto a waste dump, incurring only mining costs with no additional processing expenses.

16.6.3 Financial Parameters

16.6.3.1 Metal Price

Based on CIBC consensus pricing, the following metal prices have been adopted:

- Copper: US$4.20 per lb
- Gold: US$2,000 per oz
- Silver: US$25.00 per oz
- Molybdenum: US$17.50 per lb

16.6.3.2 Discounting

A quarterly discount factor of 1.94% (equivalent to an annual rate of 8%) is applied.

16.6.3.3 Delayed Recoveries

Metal realised through leaching occurs over multiple periods.

- Heap Leaching: Full recovery over 240 days.
- Dump Leaching: Full recovery over two years (8 quarters).

Quarterly delayed recoveries are applied as follows:

Table 16.39 : Delayed Leach Recovery								
Material	**Q1**	**Q2**	**Q3**	**Q4**	**Q5**	**Q6**	**Q7**	**Q8**
Heap Leach Rec %	38	38	24					
Dump Leach Rec %	12.5	12.5	12.5	12.5	12.5	12.5	12.5	12.5

16.6.4 Grade Reblocking

To generate a mine schedule from the block model, material aggregation is often necessary to reduce the number of blocks to a manageable level for optimisation. This approach simplifies the scheduling process while preserving critical geometry to ensure correct mining sequences, such as mining upper benches before lower ones and adhering to location precedence with bench lags. Aggregation is applied within benches and across pit stages to balance operational detail with computational efficiency.
Location specific, grade specific schedule bins were used to determine processing routes. An equivalent recovered copper grade is calculated for each pathway using block model recovery fields. For example, for concentrate feed:

- Concentrator Cu recovery (con_rec_cu) = if(WTCODE==2,CUCONTR, if (WTCODE==3,CUCONFR, 0))

The result is then applied in the full expression:

- ((CU_PCT/100) * (price_cu*2204.62) * con_rec_cu +AU_PPM * (price_au/31.10348) * con_rec_au + AG_PPM * (price_ag/31.10348) * con_rec_ag + (MO_PPM/1000000) * (price_mo*2204.62) * con_rec_mo**)** / (price_cu*2204.62)

For heap and dump leach pathways, only copper is considered Table 16.40 summarises the schedule bins, associated tonnages, and average grades, including the raw recovered material values.

Table 16.40 : Schedule Bins used to determine material processing routes							
Bin	**Bin**	**Mt**	**Cu %**	**Au g/t**	**Ag g/t**	**Mo ppm**	**Raw Value $/t**
Productora							
HG	0.220	134.3	0.50	0.10	0.38	174.80	48.42
MG	0.185	17.3	0.18	0.04	0.27	123.67	18.56
LG	0.122	63.4	0.14	0.03	0.25	82.42	13.91
Subtotal Concentrate		**215.0**	**0.37**	**0.07**	**0.33**	**143.45**	**35.84**
HL	0.060	24.1	0.42	-	-	-	27.95
DL	0.044	20.7	0.13	-	-	-	8.96
Alice							
HG	0.295	4.6	0.44	0.04	0.19	57.06	40.00
MG	0.170	2.0	0.22	0.03	0.16	40.94	20.62
LG	0.118	2.1	0.14	0.03	0.20	24.90	13.05
Subtotal Concentrate		**8.6**	**0.32**	**0.03**	**0.18**	**45.63**	**29.08**
HL	0.070	3.8	0.33	-	-	-	13.92

Table 16.40 : Schedule Bins used to determine material processing routes							
Bin	**Bin**	**Mt**	**Cu %**	**Au g/t**	**Ag g/t**	**Mo ppm**	**Raw Value $/t**
DL	0.044	1.7	0.13	-	-	-	6.84
Cuerpo							
HG	0.291	21.6	0.46	0.14	0.75	20.97	39.94
MG	0.160	27.6	0.26	0.09	0.41	29.80	20.15
LG	0.112	17.1	0.17	0.05	0.28	37.24	12.56
Subtotal Concentrate		**66.3**	**0.30**	**0.10**	**0.49**	**28.85**	**24.62**
HL	0.066	12.6	0.20	-	-	-	9.69
DL	-	-	-	-	-	-	-
San Antonio							
HG	0.200	3.1	0.86	0.01	1.28	1.79	75.38
MG	0.130	0.9	0.17	0.01	0.38	2.52	14.84
LG	0.070	0.8	0.10	0.01	0.36	2.17	9.18
Subtotal Concentrate		**4.7**	**0.61**	**0.01**	**0.96**	**1.99**	**53.35**
HL	0.066	0.4	0.66	-	-	-	30.45
DL	-	-	-	-	-	-	-
Total Open Pits							
HG	--	163.5	0.50	0.10	0.44	147.97	47.57
MG	--	47.8	0.23	0.07	0.35	63.79	19.50
LG	--	83.3	0.15	0.04	0.26	70.97	13.57
Subtotal Concentrate		**294.6**	**0.35**	**0.08**	**0.37**	**112.53**	**33.40**
HL	--	40.8	0.35	-	-	-	21.03
DL	--	22.4	0.13	-	-	-	8.80

Where;
HG/MG/LG = High/Medium/Low grade material to concentrator
HL = To heap leach
DL = To dump leach

Grade reblocking is performed using these schedule bins for each bench in each stage.

16.6.5 Stockpiles

Stockpile management follows the same aggregation logic as reblocking. Available stockpiles are detailed in Table 16.41.

Table 16.41 : Stockpiles			
Name	**Description**	**Stages**	
PR_HG	Productora High Grade Concentrate	Productora Main Pit_1	Productora North Pit_1
PR_MG	Productora Medium Grade Concentrate	Productora Main Pit_2	Productora North Pit_2
PR_LG	Productora Low Grade Concentrate	Productora Main Pit_3	Productora North Pit_3
PR_HL	Productora Heap Leach	Productora Main Pit_4	Productora South Pit_1
PR_DL	Productora Dump Leach	Productora Main Pit_5	
AL_HG	Alice High Grade Concentrate	Alice Pit	
AL_MG	Alice Medium Grade Concentrate		
AL_LG	Alice Low Grade Concentrate		
AL_HL	Alice Heap Leach		
AL_DL	Alice Dump Leach		
CU_HG	Cuerpo High Grade Concentrate	Cuerpo Pit_1	
CU_MG	Cuerpo Medium Grade Concentrate	Cuerpo Pit_2	
CU_LG	Cuerpo Low Grade Concentrate	Cuerpo Pit_3	
CU_HL	Cuerpo Heap Leach		
CU_DL	Cuerpo Dump Leach		

Table 16.41 : Stockpiles		
Name	**Description**	**Stages**
SA_HG	San Antonio High Grade Concentrate	San Antonio Pit
SA_MG	San Antonio Medium Grade Concentrate	
SA_LG	San Antonio Low Grade Concentrate	
SA_HL	San Antonio Heap Leach	
SA_DL	San Antonio Dump Leach	
BC_HG	Block Cave High Grade Concentrate	Block Cave - Underground
BC_MG	Block Cave Medium Grade Concentrate	
BC_LG	Block Cave Low Grade Concentrate	

All material permitted for stockpiling incurs a rehandle cost of US$0.97 per tonne, which is factored into the schedule optimisation.

16.6.6 Schedule Costs

Processing costs have been discussed in Sections 16.6.2.1 to 16.6.2.6. Additional cost items included in the strategic schedule are described below.

16.6.6.1 Mining Costs

A mining cost field (VCOST_MI) is applied to the block model and used in the schedule. Details of this field are in Table 16.42.

Table 16.42 : Variable Mining Cost (VCOST_MI)			
Mine Area	**Base Value**	**Adjustment Factor**	**Base Level**
Productora	$2.03/t	(+US$0.01/t per 5m lift above base, +US$0.0125/t per 5m lift below base	810 mRL
Cortadera	$2.03/t		800 mRL
Alice	$2.23/t		1,010 mRL
San Antonio	$2.23/t		1,095 mRL
Block Cave	$6.55/t	--	--

The block cave adopts a single rate of $6.55/t of rock mined for the purposes of the multi-source MineMax scheduling process.

16.6.6.2 Fixed Costs

In addition to fixed processing costs, a general and administrative expense of M$13.11 per year is incurred from the concentrator commissioning in Y3Q1.

16.6.6.3 Capital Costs

Capital costs were initially modelled whilst using an annual resolution to optimise sequencing. However, transitioning to a quarterly resolution while maintaining the same lookahead window resulted in an exponential increase in computational demand, significantly extending solve times. Consequently, capital costs were removed from the model, although the insights derived from this preliminary work informed the final outcomes.

16.6.7 Constraints

Following establishment of available pathways, the scheduler was configured with constraints. These form the targets and conditions that must be honoured when optimising the sequence.

16.6.7.1 Precedence

Location-based precedence's are established to ensure the schedule is executable and adheres to design constraints. For example:

- Productora: Sequencing logic prevents stages from undercutting one another, maintaining pit design integrity.
- Cortadera: Cuerpo 1 and Cuerpo 2 open pits serve as waste rock storage for Cuerpo 3. Consequently, mining in Cuerpo 1 and 2 must be completed before operations commence in Cuerpo 3 to provide adequate waste deposition capacity.

Table 16.43 : Open Pit Precedence

Location	Preceding Location	Description
Productora Main Pit_3	Productora Main Pit_1	Pit 'Productora Main 1' will be mined ahead of Pit 'Productora Main 3'
Productora Main Pit_4	Productora Main Pit_2	Pit 'Productora Main 2' will be mined ahead of Pit 'Productora Main 4'
Productora Main Pit_4	Productora Main Pit_3	Pit 'Productora Main 3' will be mined ahead of Pit 'Productora Main 4'
Productora Main Pit_5	Productora Main Pit_4	Pit 'Productora Main 4' will be mined ahead of Pit 'Productora Main 5'
Cuerpo 3	Cuerpo 1	Pit 'Cuerpo 1' will be mined ahead of Pit 'Cuerpo 3'
Cuerpo 3	Cuerpo 2	Pit 'Cuerpo 2' will be mined ahead of Pit 'Cuerpo 3'

16.6.7.2 Bench Advance

Bench advance constraints are critical, particularly given the steep terrain and the number of benches that must be mined concurrently. Effective staging considers:

- The initiation of new stages.
- The progression and completion of existing stages.

Larger stages require slower bench advancement, which is accounted for in the scheduling process.
Based on a 5-metre lift (flitch) height, applied quarterly, bench advance constraints are as follows:

Table 16.44 : Bench Advance

Location	Maximum Benches Mined Per Quarter	Maximum Vertical Advance Per Year
Productora Main_1	4	80 m
Productora Main_2	4	80 m
Productora Main_3	5	100 m
Productora Main_4	5	100 m
Productora Main_5	5	100 m
Productora North_1	4	80 m
Productora North_2	4	80 m
Productora North_3	4	80 m
Productora South_1	4	80 m
Alice	4	80 m
Cuerpo_1	5	100 m
Cuerpo_2	5	100 m
Cuerpo_3	5	100 m
San Antonio	4	80 m

16.6.7.3 Material Movement Constraints

Two discrete mining fleets are scheduled for the Costa Fuego Project, each with specific total material movement (TMM) constraints:

- Productora OP (group): 27,000,000 tonnes per quarter (108 Mt/yr) (Figure 16.76)
- Cortadera OP (group): 10,500,000 tonnes per quarter (42 Mt/yr) (Figure 16.77)

Figure 16.76 : Schedule Rock Tonnes – Productora Region

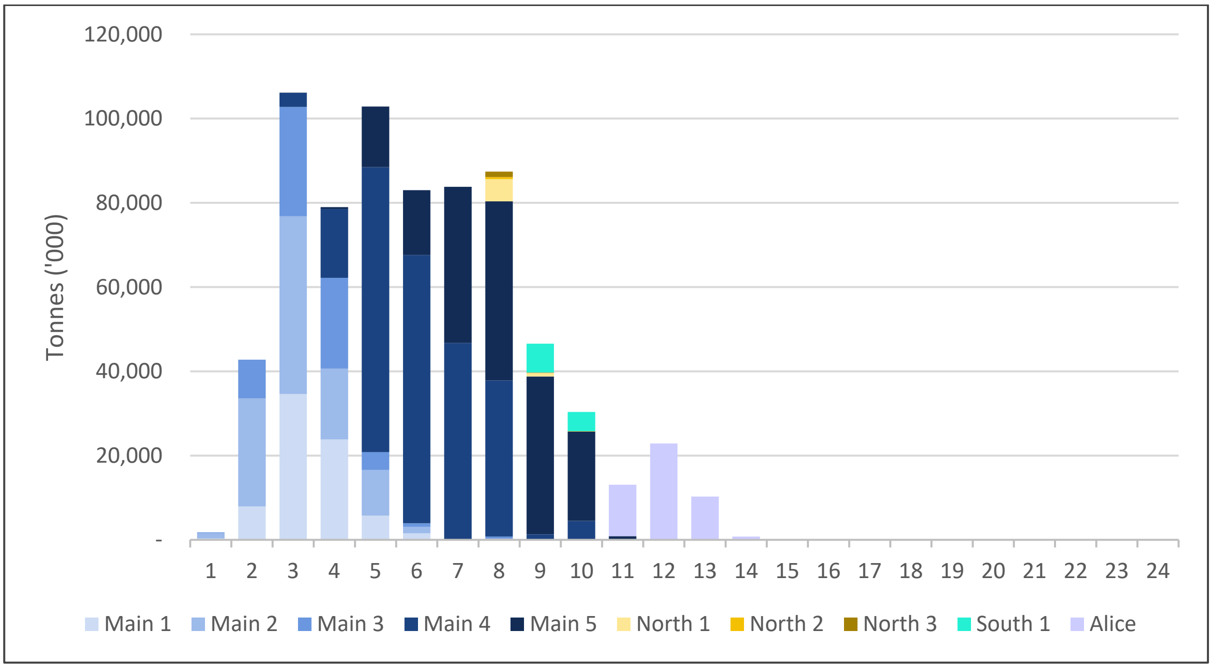
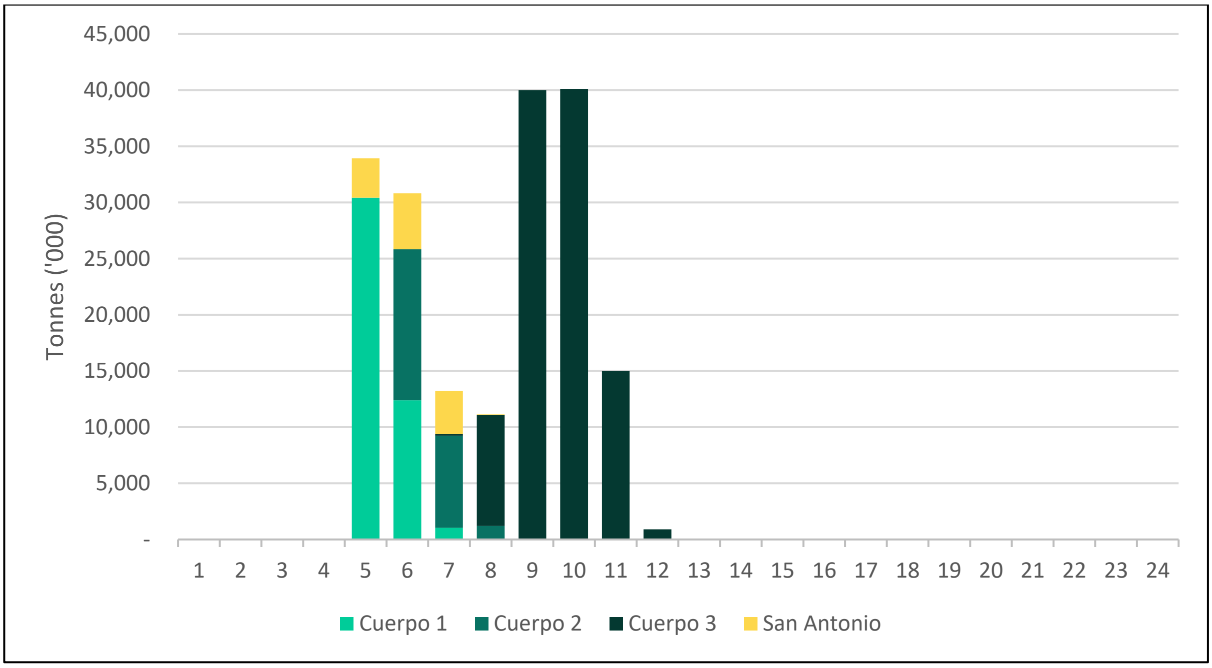
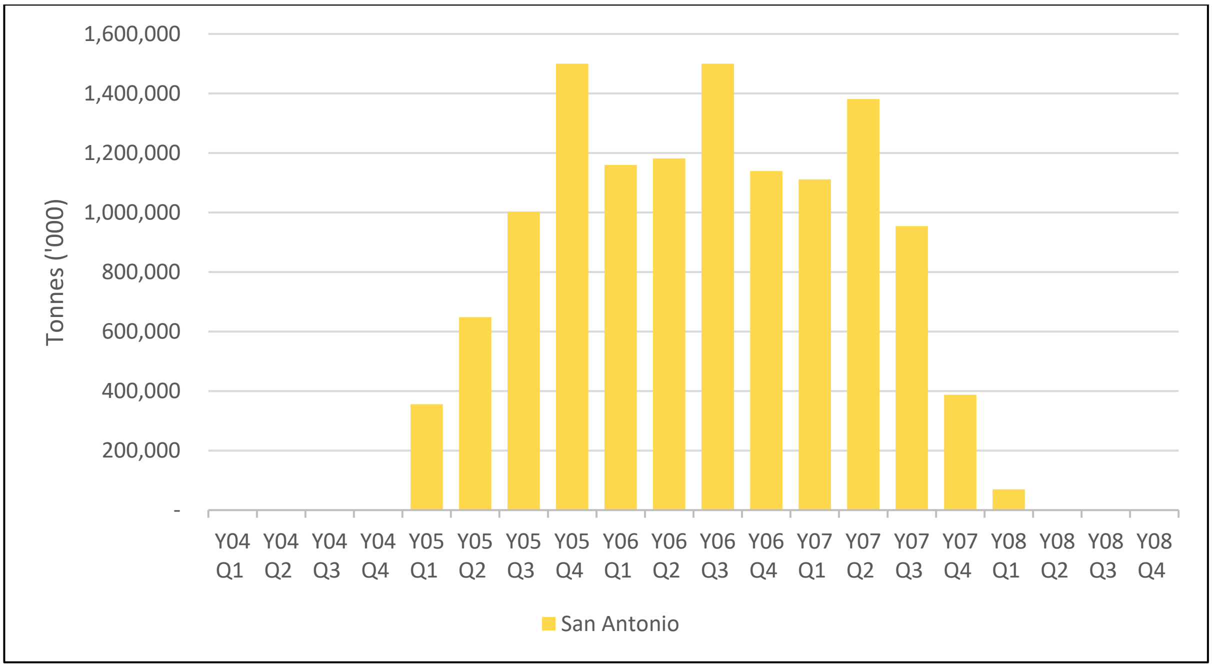


Figure 16.77 : Schedule Rock Tonnes – Cortadera Region



To control pre-production expenditures, a cumulative maximum movement constraint of 50 Mt is imposed for the pre-production period (through Year 2, Quarter 4). Additionally, a ceiling on ore production at San Antonio is set at 250,000 tonnes per quarter, reflecting the maximum throughput achievable on the road between San Antonio and Cortadera.

Figure 16.78 : Schedule Rock Tonnes – San Antonio



16.6.7.4 Infrastructure Constraints

16.6.7.4.1 Concentrator (Primary)

The concentrator has a nameplate operating capacity of 8,000 hours per year (approximately 20.7 million tonnes per annum). Commissioning occurs in Year 3, Quarter 1 (Y3Q1) with a ramp-up as follows:

- Y3Q1: 40% capacity (800 hours/quarter)
- Y3Q2: 80% capacity (1,600 hours/quarter)
- Y3Q3: 100% capacity (2,000 hours/quarter, continuing to the life of mine)

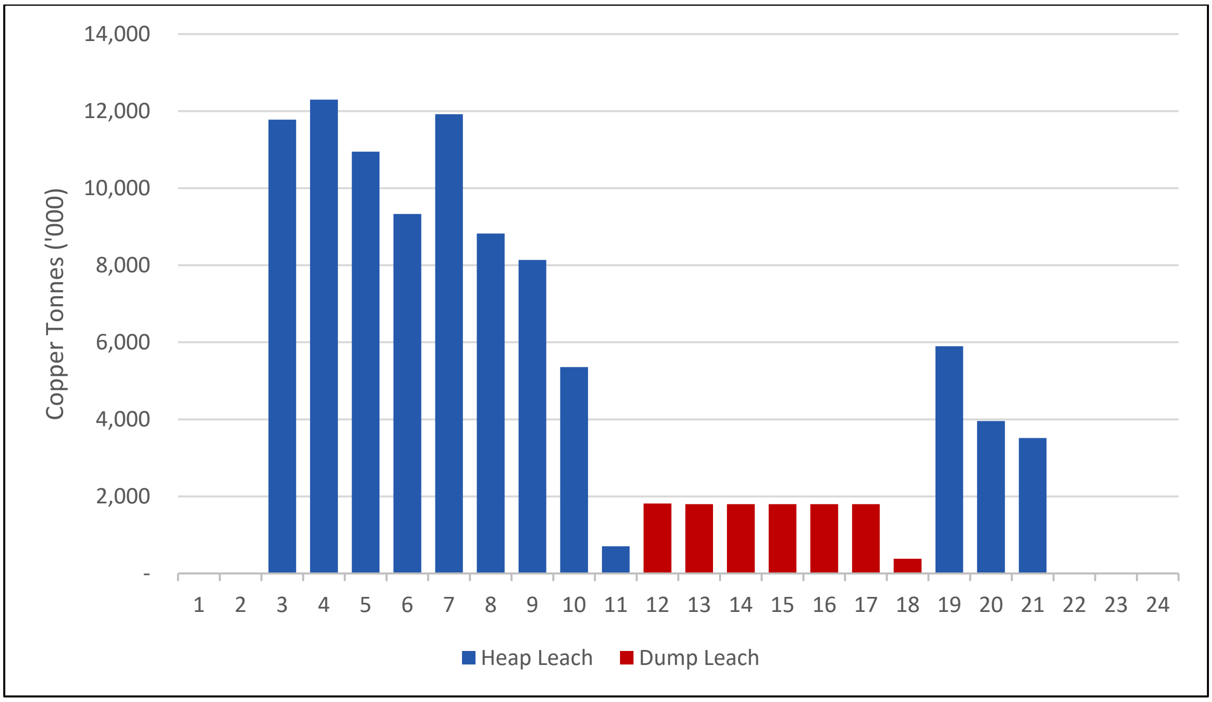
16.6.7.4.2 Rope Conveyor

The rope conveyor, which links Cortadera and Productora, is a critical infrastructure milestone. Material transfer between regions is not possible until the conveyor is commissioned. To defer associated capital expenditure, its commissioning is targeted for Year 5, Quarter 1 (Y5Q1). Consequently, no mining activity occurs in Cortadera before this date.

16.6.7.4.3 SX-EW Circuit

The SX-EW circuit is limited to a copper production capacity of 12,000 tonnes per year and becomes available from Y3Q1. It receives copper from both the heap leach and dump leach pathways and may serve as a bottleneck if copper production exceeds this rate.

Figure 16.79 : Schedule Copper – SXEW

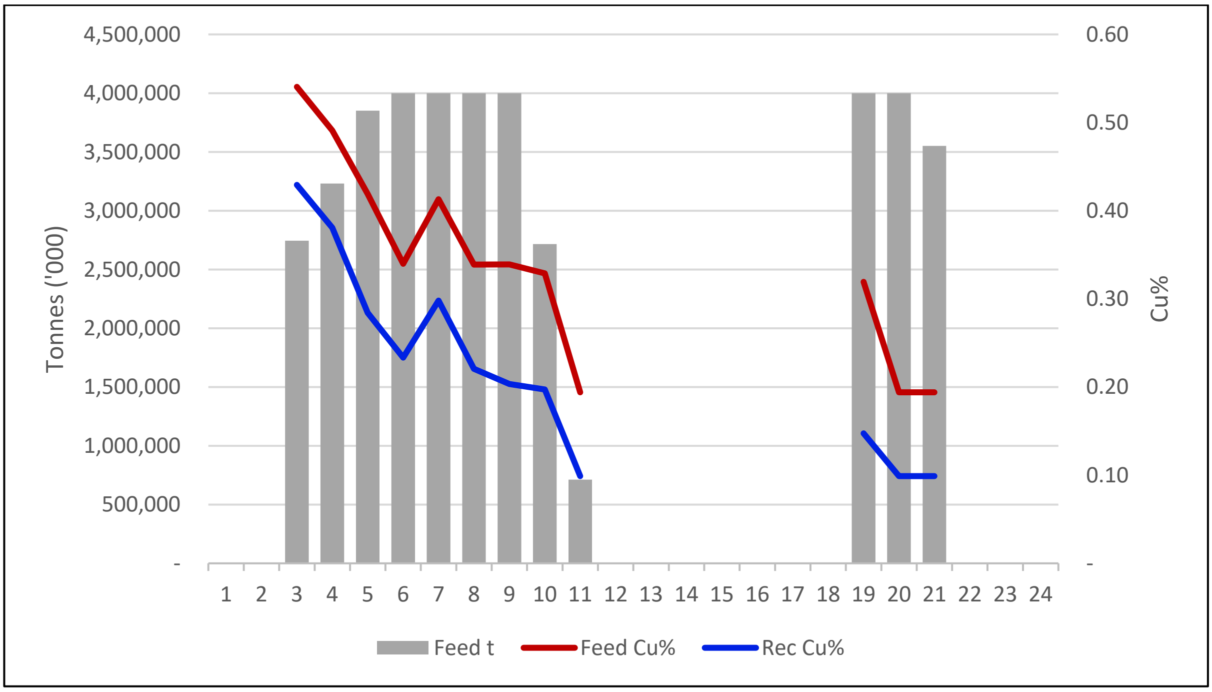


16.6.7.4.4 Heap Leach (Oxide Crusher)

The oxide crusher capacity is set at 4 Mtpa. Commissioning occurs in Y3Q1 with the following ramp-up:

- Y3Q1: 50% capacity (500,000 tonnes/quarter)
- Y3Q2: 90% capacity (900,000 tonnes/quarter)
- Y3Q3: 100% capacity (1,000,000 tonnes/quarter, continuing to the life of mine)

Figure 16.80 : Schedule Feed – Heap Leach



16.6.7.4.5 Oxide ROM Concentrator Assist (ORCA)

When the oxide crushing circuit (designed primarily for heap leaching) operates below full capacity, its surplus can be used to pre-crush material for the concentrator. This process is particularly advantageous when the concentrator's performance is constrained by the SAG mill due to the coarser nature of material from sources such as the Cuerpo open pits or Block Cave. Pre-crushing allows material to bypass the SAG mill and feed directly into the ball mill, thereby increasing throughput.

Two constraints are observed to resolve the available capacity:

- A concentrate ceiling throughput of 2,970 TPH for the combined feed.
- An oxide crusher ceiling throughput of 4.0 Mt/yr for oxide material and 5.0 Mt/yr for other feed.

The resulting feed schedule is presented in the following charts:

Figure 16.81 : Schedule Feed – ORCA (source)

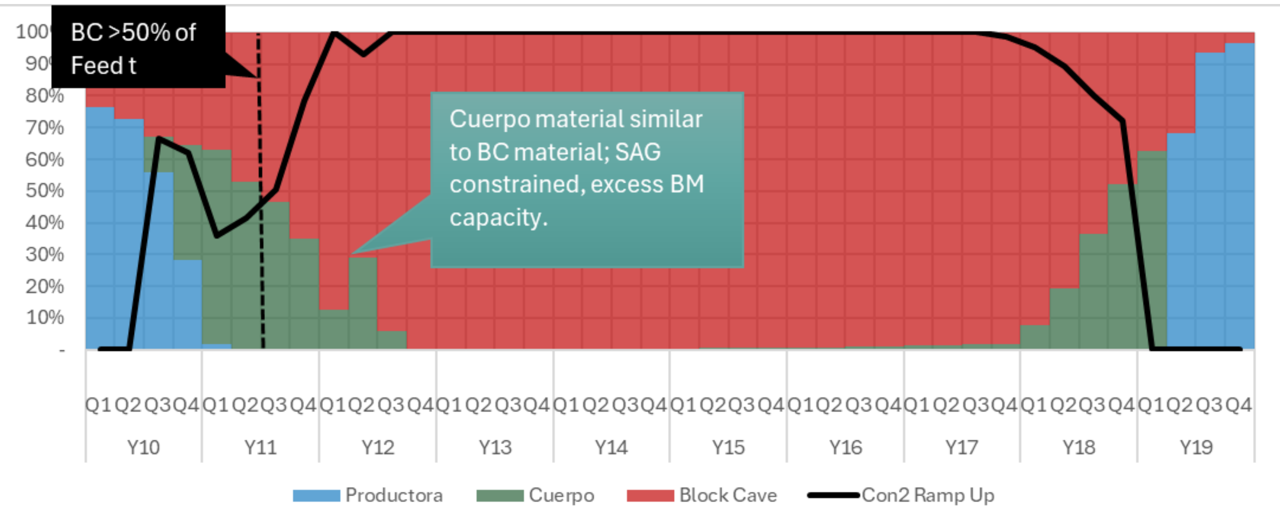
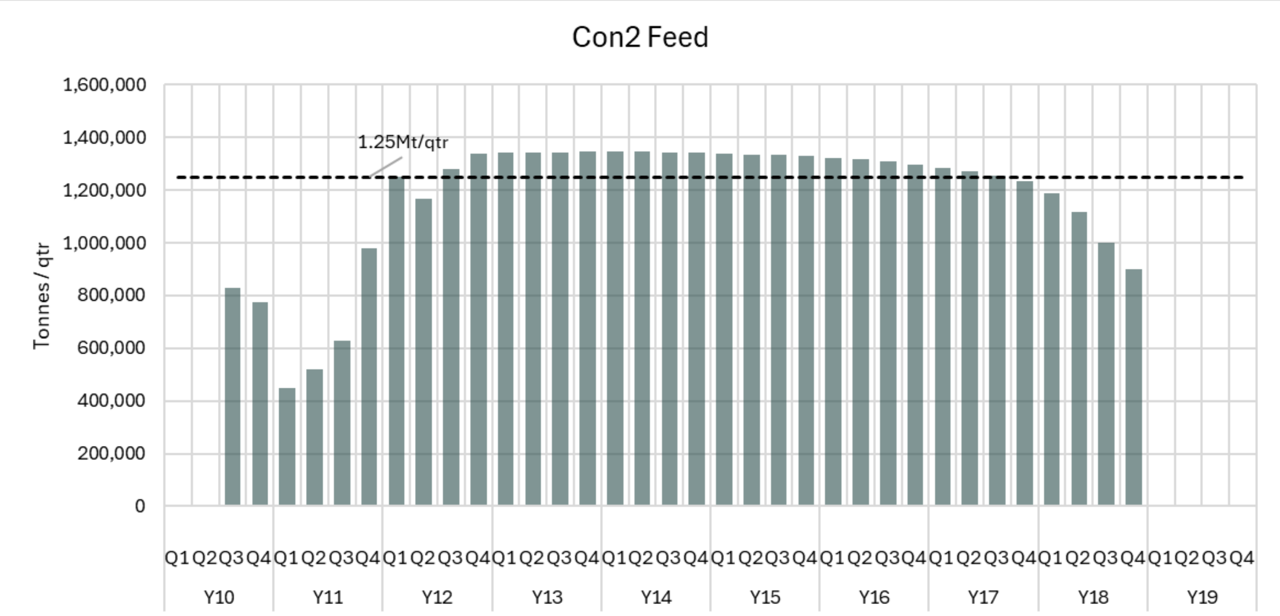


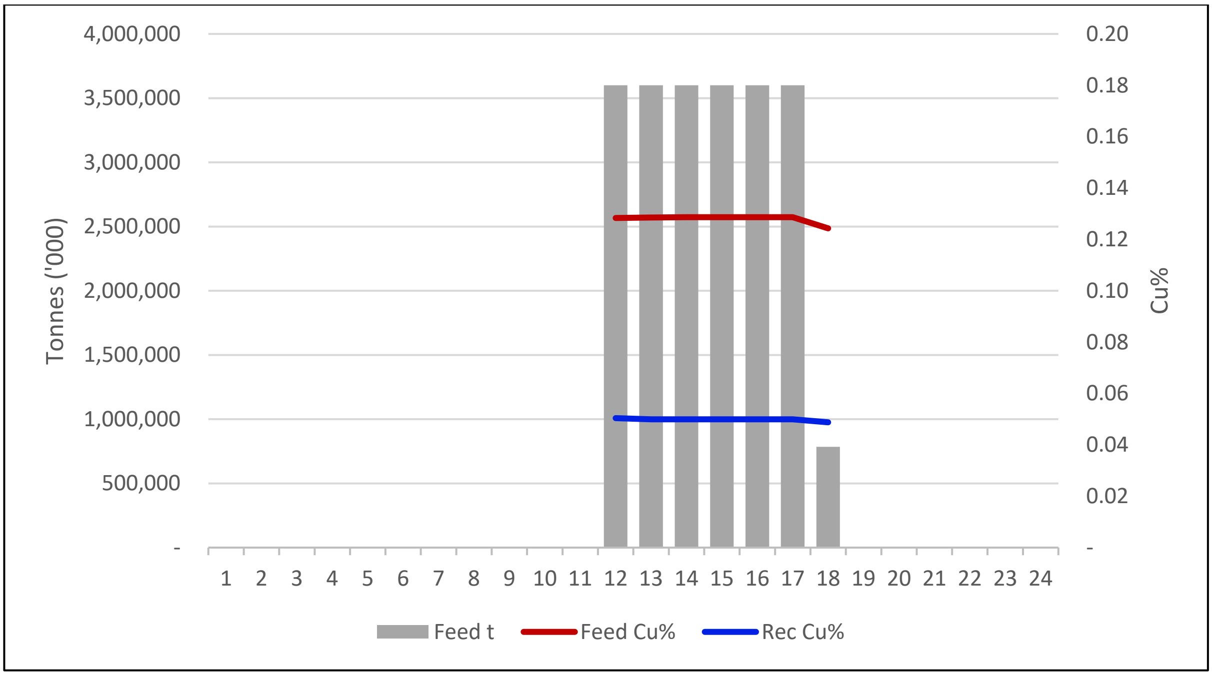
Figure 16.82 : Schedule Feed – ORCA (tonnes)



16.6.7.4.6 Dump Leach

Dump leaching uses the existing heap leach infrastructure when the oxide crusher is operating to pre-crush concentrate feed. Dump leach pads are limited by area requirements, given that leaching occurs over an extended period (2 years). Dump leach material is treated at a rate of 3.6 Mt/yr.

Figure 16.83 : Schedule Feed – Dump Leach



16.6.7.5 Block Cave Constraints

16.6.7.5.1 Underground Development Schedule

The block cave development and production schedule is generated using an underground mine scheduling package. The output is organized into "benches," with each bench representing the results of one quarter. Minemax advances one bench per period, replicating the underground schedule. The block cave follows a defined development schedule spanning 28 quarters, commencing in Year 3, Quarter 1 (Y3Q1).

16.6.7.5.2 Interaction with Open Pit Operations

The block cave schedule is interdependent with the Cuerpo 3 open pit, which is located directly above the cave. To ensure geotechnical stability, a minimum standoff distance of 300 metres must be maintained between the advancing cave and the base of the open pit. Consequently, operations at Cuerpo 3 must conclude before the cumulative extraction of 46 Mt of cave ore is reached, which is scheduled for completion by Year 13, Quarter 2 (Y13Q2).

16.6.8 Results

Schedule outcomes are presented in the following charts.

Figure 16.84 : Concentrator Stockpile – Productora Region

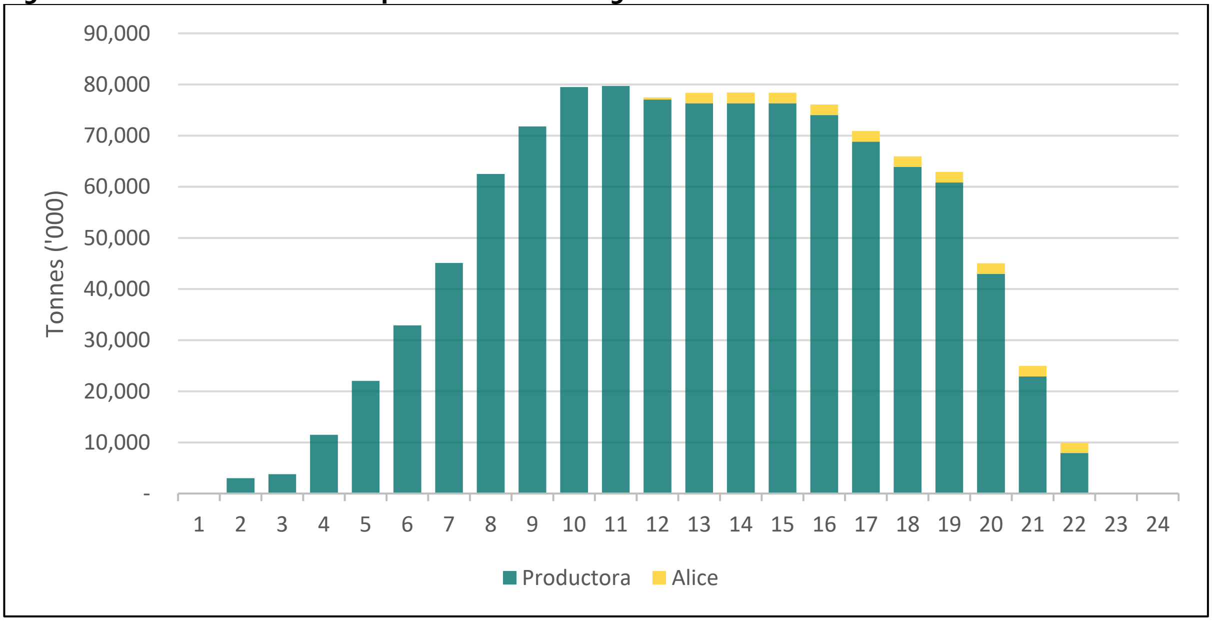


Figure 16.85 : Concentrator Stockpile – Cortadera Region

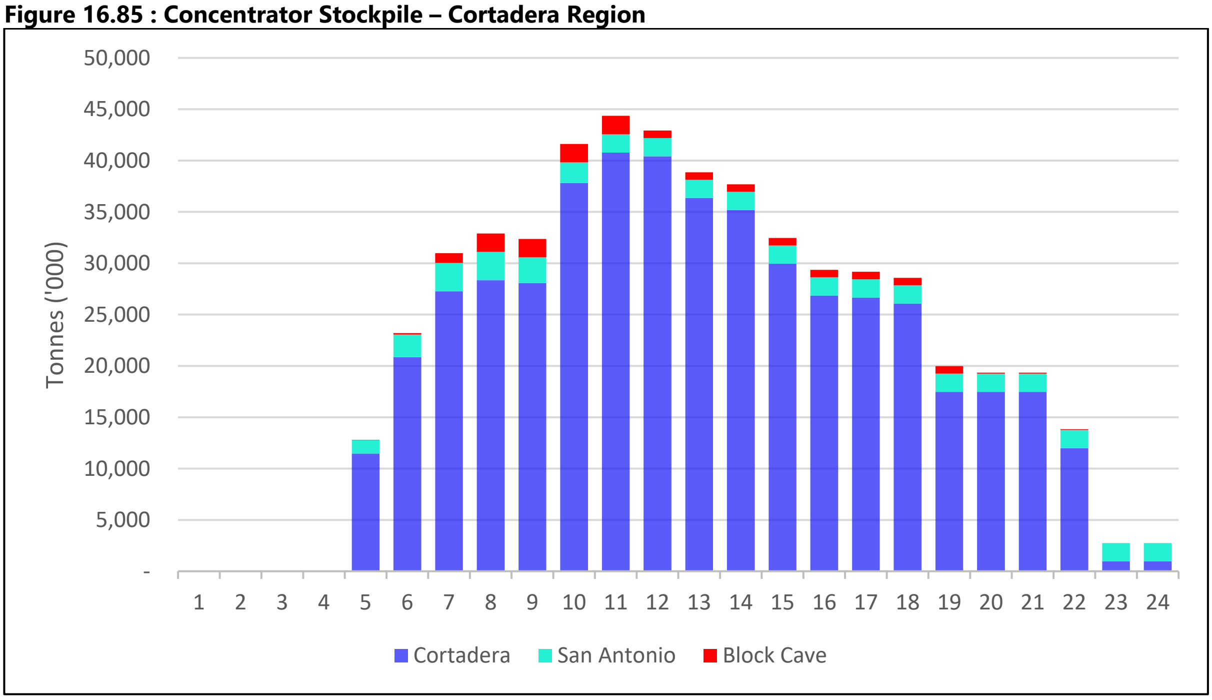


Figure 16.86: Concentrator Feed Tonnes and Grade

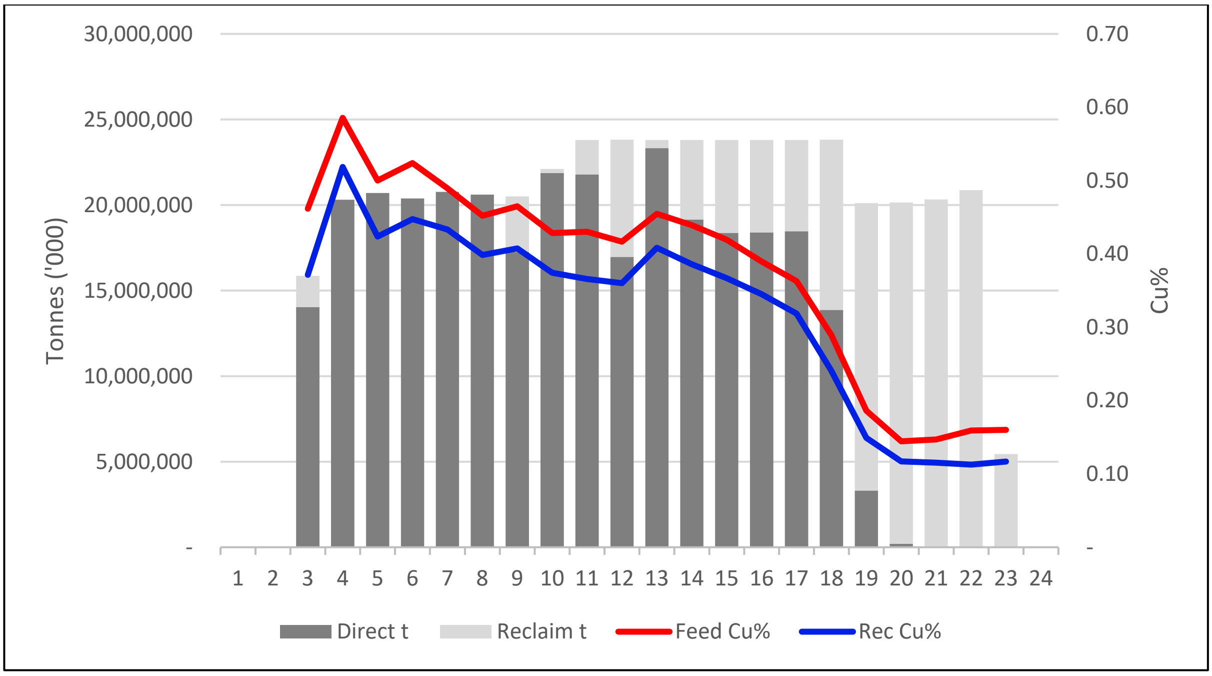


Figure 16.87: Concentrator Feed Tonnes by Source

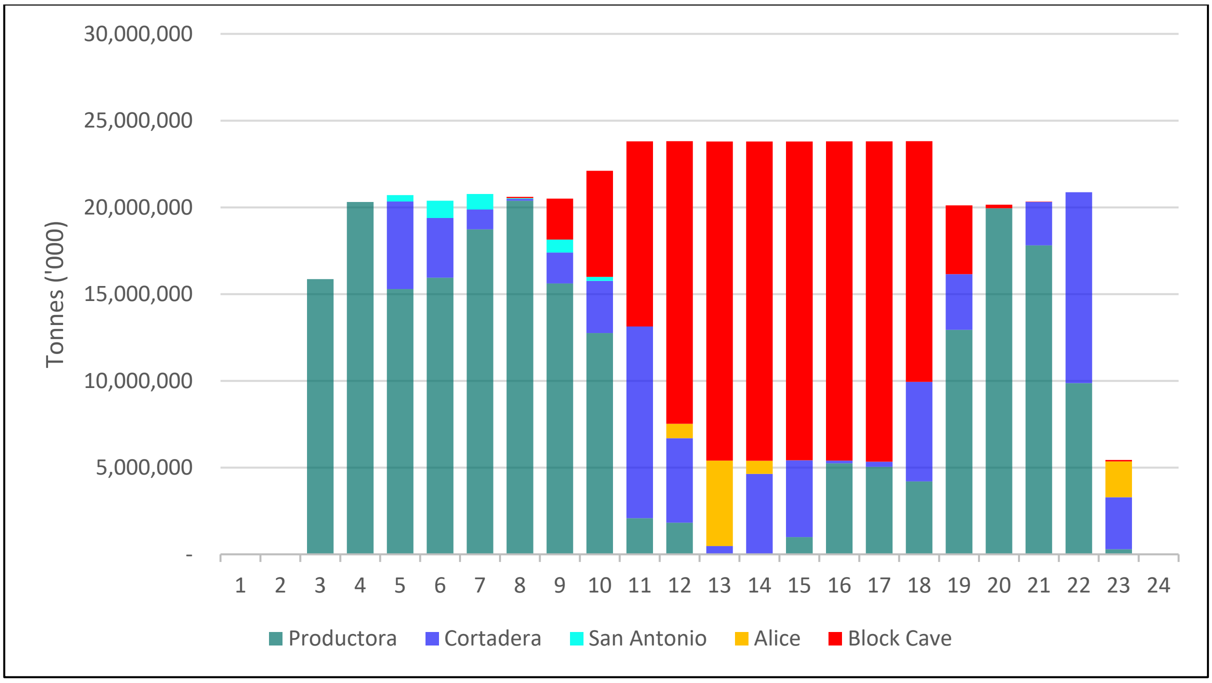
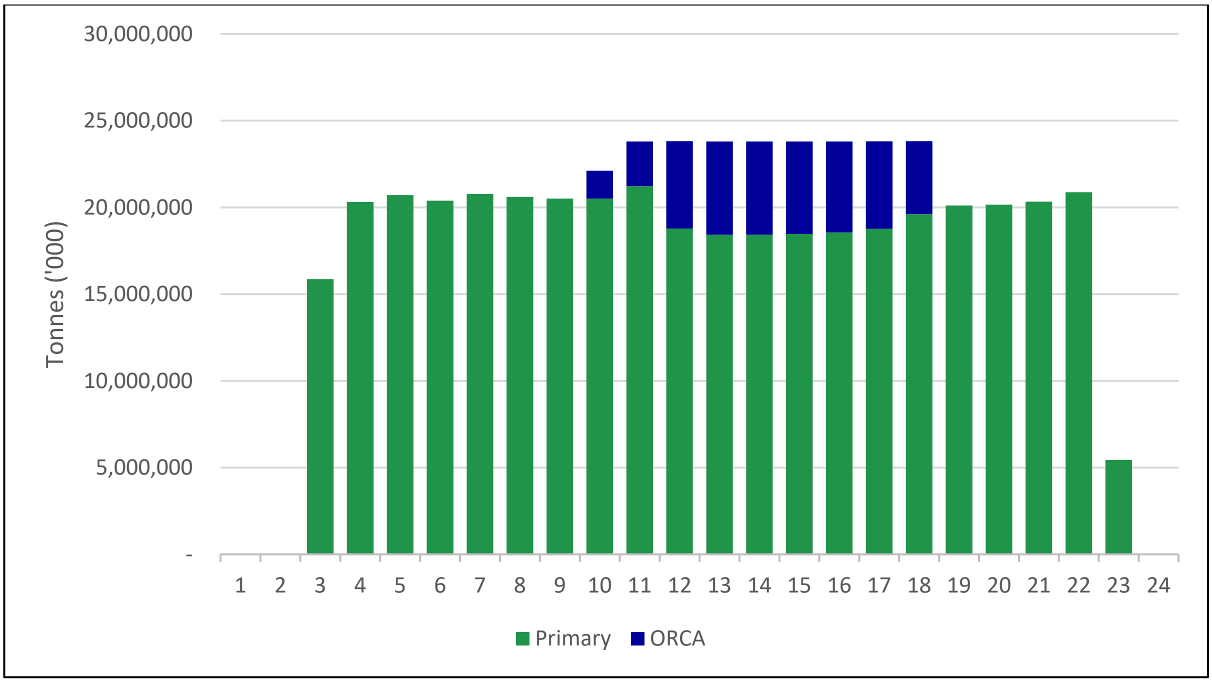


Figure 16.88: Concentrator Feed Tonnes by Type



16.6.9 Underground Mine Schedule

16.6.9.1 Mining Sequence

The underground development scheduling/planning used Datamine Studio UG and EPS software whilst merging schedules with the PCBC block cave/cave draw schedules. The mine development and underground infrastructure scheduling inclusive of the proposed underground materials handling system contains multiple activities and design aspects which are complex to plan.

This study specifically identified the typical learnings as identified during the commissioning of other block cave operations (be it a new block cave or simply a new lift in and existing operation). The learnings are mostly around the underground materials handling system development and installation and ultimate commissioning timeframes. This study considered a detailed activity breakdown of the design, development, procurement and installation of the underground materials handling system for the block cave mine plan. This activity/construction schedule was then further delayed/slowed down for unplanned delays, typically making the construction and commissioning timeframe twice as long as scheduled.

The development schedule and the key dates/times when specific underground design locations are reached were scheduled and tested with the mine planning scheduler software, however, these activities were also built in an activity simulator (SimMine) to further test impacts of significant unplanned events and delays on these specific locations becoming available in the underground development timelines. The slower/later date (Studio UG/EPS vs SimMine) of reaching the undercut, underground crusher chambers and production level positions were assumed and adopted in the final development schedule, but alternate options were also tested in SimMine to identify if "lost" time could be made up in this specific mine design and schedule with reasonably positive results indicating that it will be possible. This is explained in the mine simulation section in more detail.

The study team assumed post undercutting as a strategy for the proposed block cave operation, however, design and schedule allowances were made for advanced undercutting also. Post undercutting allows undercut extraction tunnels, undercut tunnels and apex tunnels to be developed while infrastructure is established, and ahead of firing draw bells and undercut rings.

The firing of the draw bells is predicated on several key development designs/activities and the underground materials handling infrastructure. Establishing good primary ventilation, having sufficient rock handling optionality, and establishing access on both ends of a production drive, can keep development of the draw bells ahead of the undercutting advancement.

The mining/cave establishment schedule are further guided by the geotechnical sequence from the northwestern corner to the southeastern corner, at an azimuth of 22.5 degrees east of south. Draw bells were scheduled to lead the advancing undercut rings by one to two draw bells. Draw point tunnel development in turn lead the draw bell development by at least one to two draw points, allowing sufficient time for draw point development and the installation of arches and concrete roadways prior to commencing with draw bell mining.

16.6.9.2 Mining Schedule

16.6.9.2.1 Open pit interaction

The proposed block cave operation is located underneath the planned Cuerpo 3 open pit at Cortadera. The finished block cave draw heights (400 m from the undercut level) is planned to finish approximately 120 m below the bottom of the Cuerpo 3 open pit, however, the block cave will break through to the Cuerpo 3 open pit eventually. The most important constraint to consider with the underground scheduling, particularly in relation to when block cave mining can commence, and advance (processing feed production from the proposed block cave operation) is the completion timeframe of the Cuerpo 3 open pit.

Geotechnical analyses determined that there are a few key design/schedule constraints to consider, allowing the block cave mining to commence whilst there is still mining activities taking place in the Cuerpo 3 open pit. These constraints dictate the appropriate start and advancement timeframes of the proposed underground operation. These constraints/factors are:

- maintenance of a safe pillar between the block cave propagation and the open pit. The minimum safe pillar/distance calculated is 300 m. This is 300 m from the top of the cave propagation (breaking zone) and the bottom of the open pit. It is important to note that the cave propagation/damage zone is well in advance of the actual cave draw height (when considering the two cave schedules).
- determination of early starts for undercutting operations therefore enabling some understanding in the mining schedules as to when undercutting could commence, and therefore when primary materials handling is required
- back engineering the underground schedule timeline, enabling understanding of when the portals and access development could commence.

16.6.9.2.2 Development schedule

The following table depicts the underground mine development timelines.

Underground Mining Key Dates	**UG Yr1**	**UG Yr2**	**UG Yr3**	**UG Yr4**	**UG Yr5**	**UG Yr6**	**UG Yr7**	**UG Yr8**	**UG Yr9**	**UG Yr10**	**UG Yr11**
Portal Development											
Declines developed											
UG Access reaching Pre-Condition level											
UG Access reaching Crusher Chamber 1											
Undercut Level Accessible											
Crusher 1 installation											
UG Access reaching Crusher Chamber 2											
Bell Blasting commence											
Crusher 2 Installation											
Production Level Access reached											
Initial Draw bell ore tipped onto temporary loading point onto main conveyor bin											
Tipping into Crusher 1											
Cave drawing commence - feeding Crusher 1											

Underground Mining Key Dates	UG Yr1	UG Yr2	UG Yr3	UG Yr4	UG Yr5	UG Yr6	UG Yr7	UG Yr8	UG Yr9	UG Yr10	UG Yr11
Cave Steady State Mining Reached											

16.6.9.2.3 Undercut rate

In the post undercut sequence selected for this cave the drawbells lead the undercut face by one to two bells, allowing the drawbells to be blasted prior to undercutting, and the undercut rings to be blasted shortly after into the void created by the drawbells. This allows for a faster ramp-up rate in the schedule as swell from blasting the undercut rings are loaded on the extraction level with the rest of the production tonnes, and the undercut rings can generally be blasted at a faster rate due to the larger free space created by the drawbells beneath.

A test schedule was completed in both monthly and quarterly increments and given the study stage and total life of the block cave was found to immaterially different in outcomes. Quarterly schedule increment were used in all remaining schedules.

The key strategy was to start the decline development "just-in-time" where the block cave production ramp-up would be timed optimally to increase the processed grades especially when the higher-grade process feed at Productora is depleted and the better process feed grades at Cortadera open pit operations see a grade decline. There are other sequential challenges honoured with the Cortadera open pit schedules and the Cortadera underground block cave schedule. Of primary concern is retaining a safe pillar between the block cave and the open pit mining within Cuerpo 3 (which falls within the future block cave zone of influence).

Decline development starts at the portal position near the Cuerpo 1 pit location, utilising a twin decline configuration enabling optimal face utilisation to advance the decline system as efficiently as possible. The decline system utilises a gradient of 1:7 in this design but will be optimised in the next stage of study.

The first leg of the decline system accesses the 690 RL level that sets up the first Return Air Rise leg to surface, providing through ventilation to this point in the decline and thus shortening re-entry times after blasting. This level will also provide a grade control drilling (GCD) platform to firm up on grade control definition drilling and enable preparation for the installation of a seismic monitoring system to track cave propagation over time.

Each individual decline face advances at a maximum advance rate of 120 m per month and any in-ore development tunnelling was limited to 80 linear meters per month to enable/allow for grade control functions. The development advance rates are deemed to be achievable when considering that both declines are not consistently advanced at this instantaneous advance rate, and that there are three development faces located near each other at any given point in time and that a development jumbo drill can typically achieve 260 m advance per month, however during development scheduling the total monthly jumbo development rate was limited to 240m/mth/jumbo crew.

The block cave undercut will be developed at roughly three to six draw bells per month during ramp-up, with six drawbells per month during steady state, or 25 000 m^2/yr to 30 000 m^2/yr. This will translate into a ramp-up period of three to four years to reach steady state production from the cave. This rate is considered conservative given the long undercut face length and post undercutting method selected but was selected as a risk mitigating strategy given the study stage and potential for switching to pre-undercutting and slower advance rates in the next study phase.

These assumptions will be revisited should further work result in any changes to either the open pit or underground due to potential interaction between the two mining methods.

16.6.9.2.4 Material mixing

PCBC considers all material from the extraction elevation to the best economical height of draw above the extraction level within a selected footprint. A minimum height of draw of 200 m and a maximum height of draw of 400 m was used for all draw columns – material above the 400 m height of draw was allowed to cave as per geotechnical recommendation and contribute to material mixing and dilution. Once the cave breaks through to the open pit, rilling of surface material was allowed.

Pre-vertical mixing for static analysis vs dynamic mixing in schedule. Tested monthly versus quarterly – immaterial impact, continued to use quarterly for speed of processing and allowing more options to be analysed to understand sensitivity.

PCBC standard Template Mixing algorithm was used to determine the impact of the non-linear material flow nature in a cave scenario. Table 16.45 shows the PCBC Template Mixing settings used. Vertical mixing, horizontal mixing, toppling, rilling, initial immobile / frozen material at the draw cone's base and erosion over time as well as differential movement probability of course and fines component of cave material were modelled. Cone shrinkage was not used as this is typically only used in caves with a material change expected from coarse to very fine fragmentation material in the draw columns, resulting in a material reduction in cone size, impacting material recovery and vertical mining rate.

Table 16.45: PCBC Template Mixing Settings

Setting	Value	Description
HMIX_FACTOR	1	Horizontal mix adjustment (0 – 1)
VMIX_FACTOR	1	Vertical mix adjustment (0 – 1)
TOPPLE_SLOPE	45	Topple slope angle (degrees) – on surface, in pit
RILL_SLOPE	45	Rill slope angle (degrees) – internal, against cave back
VMIX1	0.6	Vmix control 1
VMIX2	0.3	Vmix control 2
VMIX3	0	Vmix control 3
HMIX1	0.02	Hmix control 1
HMIX2	0.005	Hmix control 2
HMIX3	0	Hmix control 3
MIX_HORIZON	180	Mixing horizon
DEF_DENS	2.2	Default density of caved material
EROSION	0.02	Rate of erosion
FINES	0.6	Fines probability (0.5 to 0.7)
BASE_FROZEN	0.5	Fraction of base frozen at start (0-1)
HT_FROZEN	100	Height in column where frozen stops
ISOLATED	0	Switch for Cone shrinkage – not used
RILL_MINCAVE	50	Min cave height for rilling – above high undercut height
RANGE_2D	1	Range for Neighbours 1=2D, 0=3D

16.6.9.2.5 Drawpoint maturity curve

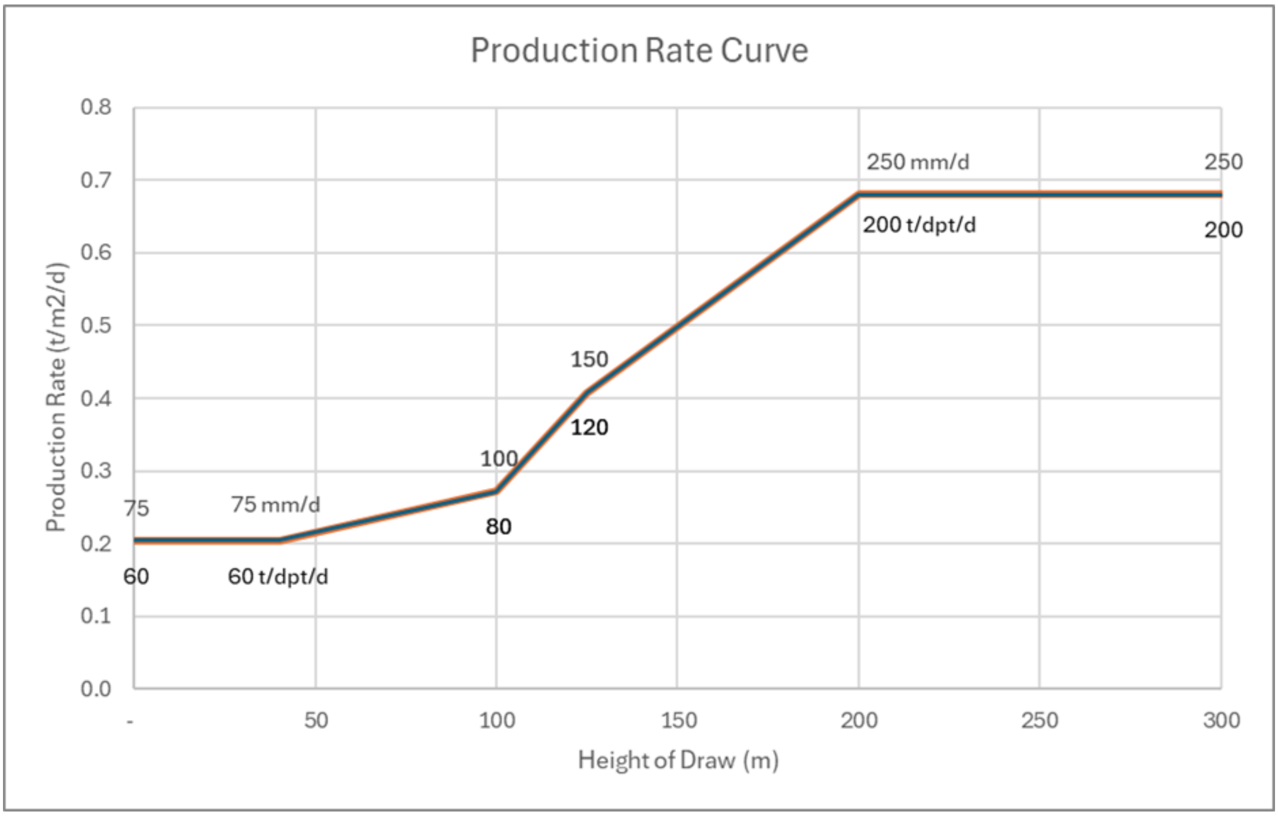
PCBC uses a drawpoint maturity curve as shown in Table 16.46 and Figure 16.89 to calculate the maximum allowable draw rate from each drawpoint at any given time in a drawpoint's life and in the overall production schedule, based on that drawpoint's maturity, or cumulative height of draw mined. This was done in three stages:

1. Prior to 100% hydraulic radius: PCBC draw is restricted to 0.2 t/m2/d, or about 75mm/d vertical draw rate, which equals about 60 t/dpt/day in the schedule. The reduced draw is used to restrict the schedule to allow for increased activity while development and drawbell construction is taking place, but also to account for potential reduced loading rates due to oversize early in the cave's life from primary fragmentation. As the cave mature and produce secondary fragmentation the oversize occurrences generally reduce, and loading rate can increase.
2. Post 100% hydraulic radius: PCBC determines each drawpoints maturity based on its cumulative height drawn at each step in the schedule – for the schedule a height of 200 m (100,000 t) was selected to represent full maturity for production rate calculation purposes. Based on the maturity a maximum production rated is calculated as per Table 16.46. It is expected that through hydro fracturing and intensive blast precondition up to 70 m above the extraction level floor will allow a faster ramp-up in future iterations.
3. Last three years of schedule during ramp-down: during the last three years of the schedule the cave is fully matured, but drawpoints start to deplete, in turn limiting the available tonnes that can be mined in each period. It is common practice to increase the draw rate from the remaining drawpoints at this stage, for which an equivalent rate of 350 mm/d was selected (280 t/dpt/d).

Table 16.46 : PCBC Production Rate Curve Table

Cumm Tonnes	HOD (m)	% Mature	mm/d	t/m2/d	t/dpt/d
-	-	Start	75	0.204	60
UC/Bell Swell	40	6%	75	0.204	60
40,000	100	40%	100	0.272	80
60,000	125	60%	150	0.408	120
100,000	**200**	**100%**	**250**	**0.680**	**200**

Figure 16.89 : Production Rate Curve (mm/d and t/dpt/d versus drawpoint maturity)

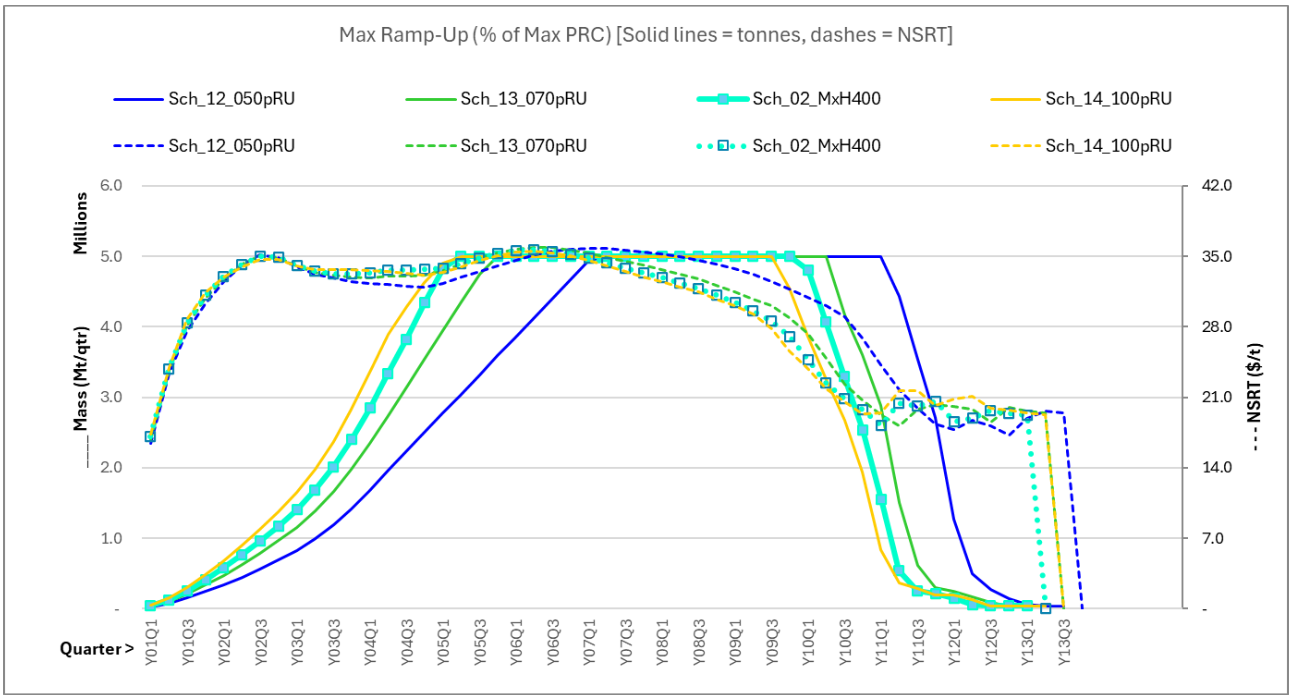


16.6.9.2.6 Production ramp-up

Production ramp-up is a combination of adding new drawpoints to the active production area as the undercutting process continues, and existing drawpoints increasing draw rates as the cave matures. The maximum production rate was constrained in the final schedules to 85% of this maximum ramp-up rate to account for overall system availability and total system throughput capability. This should be confirmed through further simulation work.

Figure 16.90 shows ramp-up sensitivities for maximum rate, base case rate (85%), 70% and 50%. The Base Case schedule ramps up to the target maximum rate over a period of about four years (16 quarters). Restricting the ramp-up rate to 50% of maximum capacity extends this to approximately seven years.

Figure 16.90 : Production ramp-up sensitivity



16.6.9.2.7 Production total throughput constraints

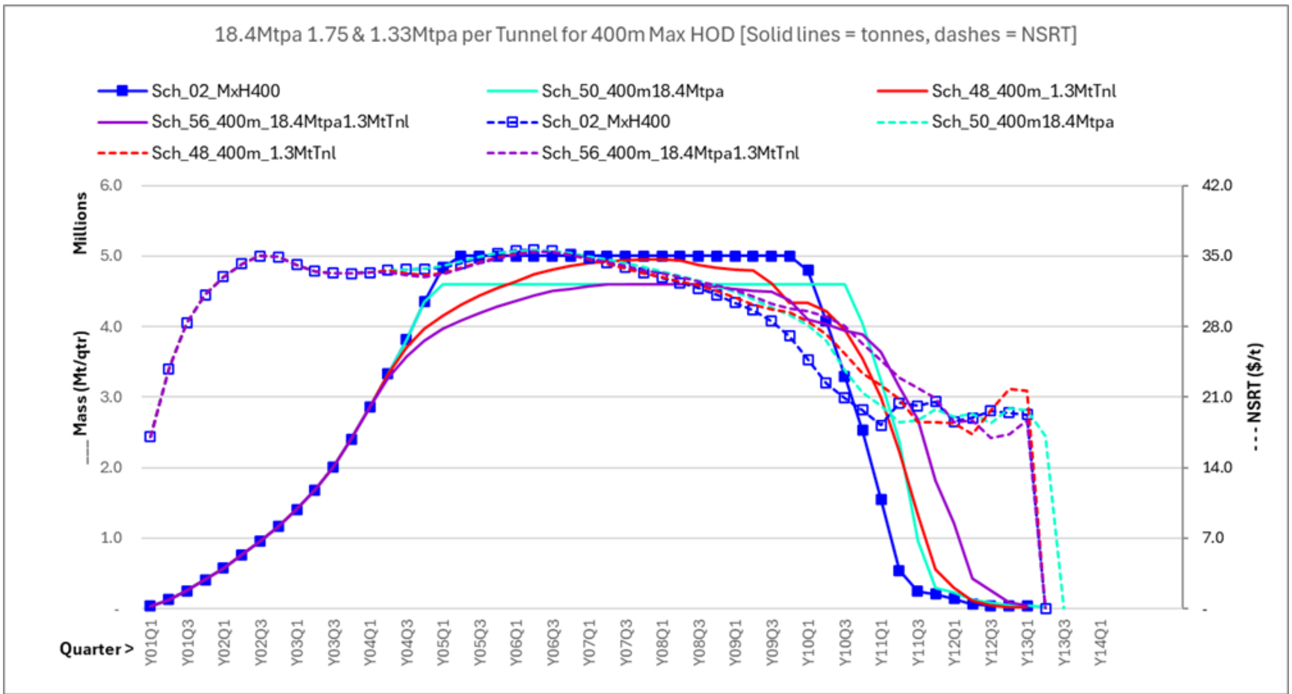
PCBC can model various production capacity constraints. PRC was discussed in the previous section. Further production constraints applied in the schedule were:

- Individual production tunnel constraints
  - This limit is typically defined by the lower value of the sum of the drawpoints in any tunnel (cave's capacity for that tunnel) and the maximum capacity a loader can achieve given the tunnel length and material handling arrangement. The cave footprint results in relative long production tunnels in some instances, hence production tunnel constraints were used to limit the throughput to a single loaders capacity. Constraints of 1.0 Mtpa, 1.3 Mtpa, 1.5 Mtpa and 1.75Mtpa per loader per tunnel were modelled. Capacity of a loader in a tunnel considers general productivity parameters such as, tramming distances and non-productivity %.
  - A rate of 1.0 Mtpa/tunnel/year constrain the overall production rate to 15 Mtpa, which is well below the targeted rate of 18.4 Mtpa to 20.0 Mtpa.
  - A rate of 1.3 Mtpa/tunnel is the average annual rate spread equally over the available 15 tunnels as a long-term steady state average rate.
  - A rate of 1.5 Mtpa/tunnel represents two tunnels out of production at all times, thus only producing from 13 tunnels, and represent the short-term instantaneous rate per tunnel.
  - A rate of 1.75 Mtpa/tunnel is based on some of the high production rates seen in Australian purpose build high production rate footprints and only require 11.5 of the 15 tunnels at any point in time to achieve the required throughput.

- Three total tonnes produced constraints were scheduled:
  - A high annual rate of 20 Mtpa to match the maximum process plant and materials handling rate put in placed based on the open pit operations.
  - A medium rate of 19.4 Mtpa was evaluated as a likely maximum rate but requires larger 18 t or 21 t loaders to achieve high throughput rates in the longer production tunnels.
  - Base Case rate of 18.4 Mtpa was used representing typical 17 t loaders.

Future work may impose further production constraints based on the crushers' capacity, or potential production zone constraints, such as ventilation restrictions or logistical limitations.
Figure 16.91 shows the base case ramp-up to 18.4 Mtpa.

Figure 16.91 : Production total capacity constraints



16.6.9.2.8 Production objective or priority

When more tonnes are requested in any scheduling period than what the footprint can produce, PCBC will only schedule what is available. If, however, there are more tonnes available from the footprint than what the maximum demand is in any period, PCBC has the option to distribute the requested tonnes over the available drawpoints. For example, all drawpoints can be scaled proportionally to their remaining tonnes to ensure a typical block cave profile where all drawpoints are completed at a similar time. Or newer or older drawpoints can be allocated maximum priority. This block cave's geometry lends itself to a typical block cave schedule, where all drawpoints are scheduled to close at a similar point (PCBC method QREMAIN).

Maximum remaining value (PCBC method QVALUE) was also tested, despite this not being recommended in most block caves as it can lead to "high grading" which can have detrimental impact on the overall cave recovery over time. The difference between the two methods was immaterial though, giving almost identical results. This is due to the highest value order for drawpoints being very similar to maximum remaining tonnes, producing a very similar schedule output.

16.6.10 Underground Activity Simulations

The key questions always raised with underground mine schedules are typically:

- Do you allow sufficient time for learnings, ramp up and building up to achievable steady state mining or advance rates (in development)
- Have you simulated potential interactions and bottlenecks in the mine schedule which could impact the mine development schedule or mining productivities
- How were potential bottlenecks identified, and the risk thereof mitigated or eliminated

To answer these key questions, it is imperative to consider an activity/flow simulator (software) to model these activities and all possible interactions and test the potential outcomes of unplanned events and its impact on the proposed mine schedule. SimMine was selected as a suitable software platform to enable the study team to answer these questions and to facilitate derisking strategies through design or schedule rate adjustments.

16.6.10.1 Simulating the proposed development Schedule

The access development associated with the block cave is the critical path to allow for the commencement of the block cave, therefore the timing of the development must be fully understood. It is important to understand the potential development schedule risks and opportunities that may exist.

16.6.10.1.1 SimMine setup

The following process was followed to allow SimMine to evaluate the development schedule:

- The Block Cave Mine design/layout was imported into SimMine and set up for use by the software.
- SimMine required the setup of the envisaged:
  - Face profiles for the development (width, height, Area)
  - Shift schedules
  - Activities (Drill, Charge, Blast, Load and Haul and bolting) and the envisaged time delays/dependences associated with each activity
  - Fleet – with the associated envisaged availabilities, utilisations and capacities
  - Rock properties (Equipment fill factors, densities)

All inputs into SimMine were based on the current understanding of industry standards, adjusted to suit regional/project specific conditions.

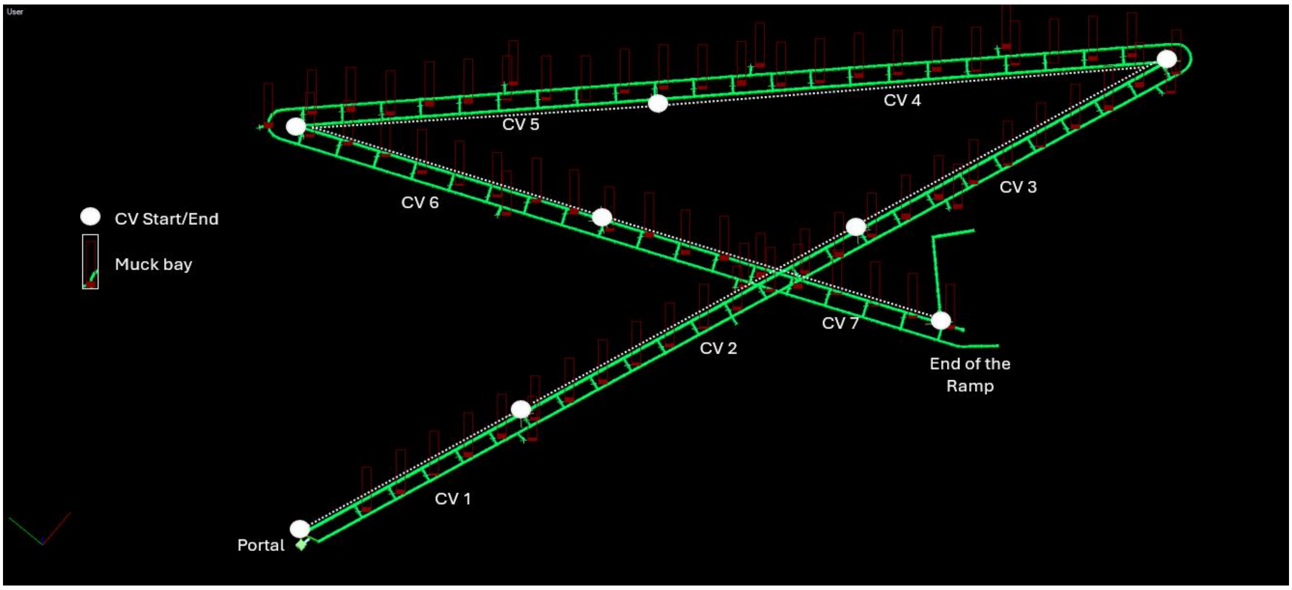
16.6.10.1.2 Scenarios

Initially a base case scenario was run in SimMine to determine if the schedule produced in Datamine UG – EPS is practical and achievable. Post the confirmation of the Base case several further scenarios were run to determine the risks and opportunities associated with the development schedule.

The base case consisted of the following, with the additional scenarios are summarised in Figure 16.46 :

- Shift schedules – 2 shifts
- Fleet
    - 1 x Drill rig
    - 1 x Bolter
    - 1 x Charger
    - 1 x LHD (17t)
    - 1 x Truck (51t)
- Conveyor used once available.

Figure 16.92: Decline design with conveyor and muck bay locations



16.6.10.1.3 Outcomes

The base case simulation indicated that the Datamine UG – EPS schedule is practical and achievable, as similar timeframes were determined by SimMine.

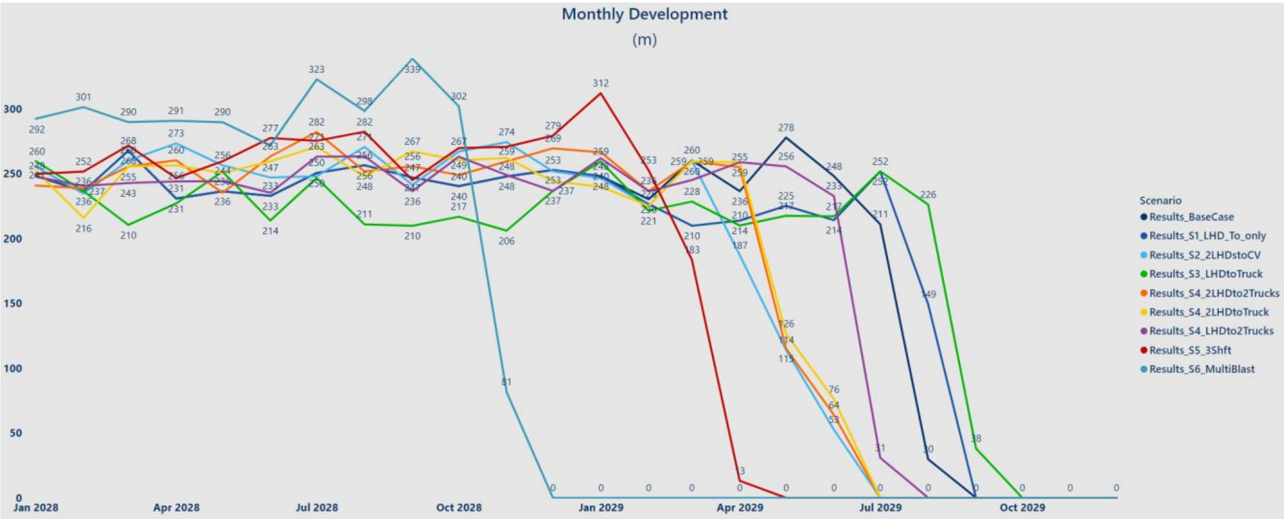
The simulations also showed that the introduction of additional equipment had a minimal impact on the overall schedule.

Once all scenarios were run, it was determined that the main constraint was face availability, therefore the three shift schedules and multi-blasting scenarios had the highest advance rates and therefore shortened the overall development schedule.

Table 16.47: Summary of Scenario Completion dates

Scenario	Completion data	Variance in months
Base Case	Aug-29	
LHD Only	Aug-29	
2 LHDs to CV	Jun-29	-2
LHD to Truck	Sep-29	1
2 LHDs to 2 Trucks	Jun-29	-2
2 LHDs to 1 Truck	Jun-29	-2
LHD to 2 Trucks	Jul-29	-1
3 Shifts	Apr-29	-4
Multi-blast	Nov-28	-9

Figure 16.93: Simulated advances and completion dates for the scenarios



16.6.10.1.4 Risks

Although a similar schedule to the base case can be achieved without the purchase of a truck, it was determined that this scenario introduced risk, as no alternative haulage method would be available if a major conveyor breakdown was experienced.

16.6.10.1.5 Opportunities

The introduction of multi-blasting conditions showed the best opportunity to improve the overall schedule, however it should be understood this opportunity will have additional costs associated with it. The mine design and ventilation layouts developed for this study enables the possibility of multi-blast development conditions as the initial air flows have a dedicated intake decline (access/man and material/truck decline) where air can be forced into the development ends and a dedicated return airway (the conveyor decline). If access or development progress are impacted, there is the possibility to "catch-up" by introducing periods of multi-blast operations.

16.6.10.1.6 Testing the Caving Schedule

In respect to the Block caving production schedule, SimMine was used to firstly test if the ramp-up and steady state production profiles are achievable through the mine design/layout developed and also to determine the optimal equipment fleet size (number of units and load capacities) and the associated procurement schedule required, to meet the ramp up schedule and to determine if the envisaged maximum peak production of 18.4mtpa is achievable.

Higher production rates were also tested (19.4 and 20 Mtpa ore cave production rates). This was to satisfy the study team that sufficient understanding can be developed in relation to operational risk and equipment capacities needed to achieve these production rates or to outline if alternate rock/material handling systems should be considered.

The key equipment and simulation consideration is the loading units (LHD) utilisation and efficiencies with the given underground mine design and loading layout. This Block Cave operation has a large footprint, and the current mine design has both crusher feeding points on the one side of the footprint. SimMine was used to test the potential productivities that the LHD's could be run at and if these would satisfy the PCBC Block Cave schedule. The SimMine simulation furthermore provided support for the PCBC scheduling where specific production drive loading limits were imposed during the mine scheduling.

The key outputs of the SimMine simulations were related to decline rates which were proven to be practical and achievable and also outlined the potential for upside on faster multi-blast development conditions improving the advance rates.

Another important outcome from SimMine was the confirmation of LHD productivities and at which mining schedule rates, either a 17 t or 21 t LHD would be required. The SimMine input specifications considered the typical LHD speeds and capacities, but it is understood that the initial cave fragmentation might be larger and therefore it is expected to have lower than expected bucket fill factors. The bucket fill factors considered were 70%, 80% and 85%. It is recommended that the bucket fill factor ramps-up as the cave matures and fragmentation improves.

LHD operating speeds were factored to 90% of the operating specifications. Additional unproductive times and longer waiting times were also introduced, particularly at the crusher entry points and at the LHD turn bays. The LHD will load front-in and then reverse into the turn bay and drive front on into the crusher tipping points. This impacts LHD productivity and, with longer one way haulage and turning/waiting times at a congested layout, it was deemed appropriate to downgrade the estimated capacities by 15%.

SimMine simulations further confirmed that 15x operating LHD's with one spare LHD (17t class) would be required to achieve 18.4 Mtpa mining rates. 13x 21 t LHD's with one additional LHD will achieve the same. The turn bays were designed at lengths that could consider both the 21 t and 17 t LHD's and it is suggested that the production drives could be widened a to better accommodate the 21 t loader option. The designs and schedules did assume additional overbreak on waste and the PFS could therefore use either loader option at the selected underground ore mining rate of 18.4 Mtpa. If the production rate is increased above 19.4 Mtpa, the 21 t loaders are suggested as the appropriate loader size.

Base-case estimated productivities for the 17 t LHD (Table 16.48) and the 21 t LHD (Table 16.49) are shown below. These are available on several Equipment Manufacturer specifications lists/websites.

Table 16.48 : 17 t LHD operating specifications considered in SimMine (but reduced in SimMine by 15%)

Grade performance										
Volvo TAD1382VE, Stage V and Volvo TAD1372VE, Tier 4f (3 % rolling resistance, with lock-up)										
Empty										
Percent grade	0.0	2.0	4.0	6.0	8.0	10.0	12.5	14.3	17.0	20.0
Ratio					1:12	1:10	1:8	1:7	1:6	1:5
1st gear (km/h)	5,4	5,4	5,4	5,4	5,3	5,3	5,3	5,3	5,2	5,2
2nd gear (km(h)	9,7	9,6	9,5	9,5	9,4	9,3	9,2	9,1	8,2	7,3
3rd gear (km/h)	16,9	16,7	16,5	16,3	14,6	12,8				
4th gear (km/h)	30,1	29,4	23,6							
Loaded										
Percent grade	0.0	2.0	4.0	6.0	8.0	10.0	12.5	14.3	17.0	20.0
Ratio					1:12	1:10	1:8	1:7	1:6	1:5
1st gear (km/h)	5,4	5,4	5,4	5,3	5,3	5,3	5,2	5,2	5,2	5,1
2nd gear (km(h)	9,6	9,5	9,5	9,4	9,3	8,9	7,8	7,2		
3rd gear (km/h)	16,8	16,5	16,2	13,5						
4th gear (km/h)	29,7	24,3								

Table 16.49: 21 t LHD operating specifications considered in SimMine (but reduced in SimMine by 15%)

GRADE PERFORMANCE									
Volvo TAD1344VE									
Empty									
Percent grade	0	2.0	4.0	6.0	8.0	10.0	12.5	14.3	17
Ratio					1:12	1:10	1:8	1:7	
1st gear (km/h)	6.1	6.1	6.0	6.0	6.0	6.0	5.9	5.9	5.9
2nd gear (km(h)	10.9	10.9	10.8	10.7	10.6	10.5	10.1	9.4	8.3
3rd gear (km/h)	19.1	18.8	18.5	18.1	15.4	13.3			
4th gear (km/h)	34.2	33.4	26.2						
Loaded									
Percent grade	0	2.0	4.0	6.0	8.0	10.0	12.5	14.3	17
Ratio					1:12	1:10	1:8	1:7	
1st gear (km/h)	6.1	6.1	6.0	6.0	5.9	5.9	5.9	5.8	5.7
2nd gear (km(h)	10.9	10.8	10.7	10.6	10.5	9.3	8.0		
3rd gear (km/h)	19.0	18.6	17.8	14.4					
4th gear (km/h)	33.9	28.1							

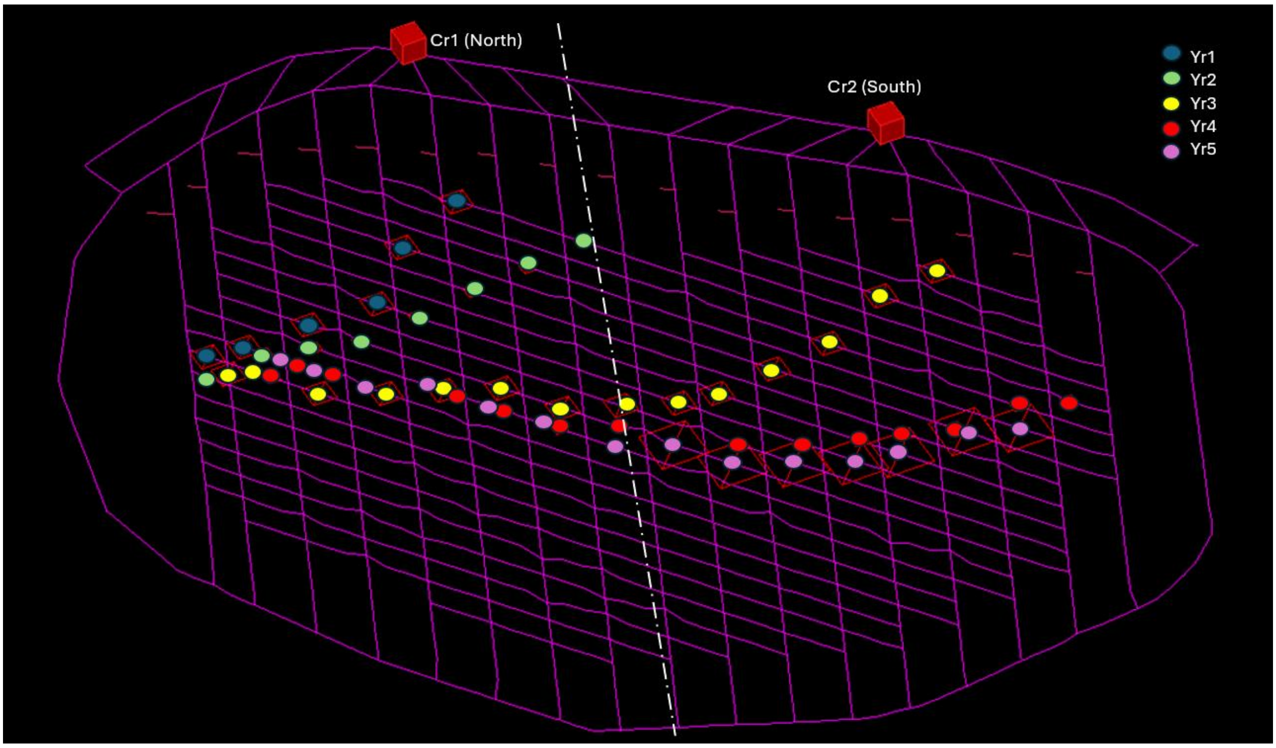
16.6.10.1.7 SimMine setup

The following process was followed to allow SimMine to evaluate the development schedule:

- The Block Cave Mine design/layout was imported into SimMine and set up for use by the software.
- SimMine required the setup of the envisaged:
  - Face profiles for the development (width, height, Area)
  - Shift schedules
  - Crushers were added into the design with appropriate start dates and limitations
  - Fleet – with the associated envisaged availabilities, utilisations and capacities
  - Rock properties (Equipment fill factors, densities)
  - The PCBC ramp-up schedule year 1-5 was re-created within SimMine
  - Equipment was allocated to the two crushers with appropriate start dates to achieve the annualised production targets.

All inputs into SimMine were based on industry standards, as of the date of this Report, and adjusted to suit the regional/project specific conditions.

Figure 16.94: Crusher and Production plan mid points Year 1-5.



16.6.10.1.8 Outcomes

The simulation indicated that the ramp-up to steady state production schedule is achievable, with the implementation of additional LHD's during the ramp-up period. The simulations confirmed that a total of 14 LHD's would be required to achieve steady state production.

The following equipment procurement and implementation schedule was determined (Table 16.50).

Table 16.50: Equipment requirements

LHD No.	StartDate Year	Crusher
1	1	1 North
2	2	
3	2	
4	2	
5	3	
6	3	
7	3	
8	3	2 South
9	3	
10	3	
11	3	
12	4	
13	4	
14	4	

16.6.10.1.9 Risks

Although the simulation indicated that the production ramp-up schedule and peak production is achievable, it also highlighted a risk associated with achieving the 20 Mtpa schedule when the LHD bucket capacities are limited to 17 t. A scenario was tested with LHD having a bucket capacity of 21 t, and this resulted in the achievement of the 20 Mtpa with limited issues. The selected PFS mining rate of 18.4 Mtpa however can use the 17 t LHD or 21 t LHD options, but the 17 t LHD is more widely used in Chile. The 18.4 Mtpa block cave mining schedule will still require skilled operators with very good overall mine productivity if utilising 17 t LHD's.

16.6.10.1.10 Opportunities

The Block caves ramp-up to steady state production schedule can be de-risked by selecting equipment that conforms to the geotechnical limitations associated with the tunnel dimensions while also allowing for a LHD with a 21 t bucket capacity. Therefore, higher ramp-up productivities could be achieved with a larger LHD, and higher overall mining rates could be achieved.

16.7 Recommendations

16.7.1 Geotechnical - Open Pits

The proposed mine design for the Costa Fuego Project's open pits are acceptable for the PFS. The following general recommendations should be considered in the next phase of the study:

- Improve the quality and comprehensiveness of the geotechnical database by conducting new drilling campaigns and regularly updating the Geotechnical Model for each deposit
- Update and refine the structural models with expanded information on the boundaries of each structural domain. Additionally, enhance data quality regarding the persistence and spacing of identified joint sets
- Conduct direct shear tests to assess the strength properties of joints and principal structures
- Update the Hydrogeological Model with the latest piezometric data and assess potential water tables for each open pit, considering base, optimistic, and pessimistic water levels to account for uncertainty in slope stability
- Perform on-site validation of principal structures to confirm their precise location, orientation, and extent, as well as to identify new structures to keep the structural model up to date.
- Conduct a seismic hazard assessment specifically for the open pit analyses. This assessment should determine peak ground acceleration (PGA) values for earthquake return periods that correspond to the project's lifespan under operational and post-closure conditions.
- Reassess and optimize the open pit designs based on any updated geotechnical, structural, and hydrogeological data, including conducting slope stability analyses at the bench-berm, inter-ramp, and overall scales.

In addition to the general recommendations, specific recommendations have been provided by mine area, as detailed below:

16.7.1.1 Productora

- In Section 06, stability is influenced by two subparallel joint systems. However, there is uncertainty regarding the presence of these structures within this section's location due to its structural domain definition, where lithological boundaries establish the domain limits (GeoEkun, 2024a). It is therefore recommended to assess whether these two joints coexist within this structural domain
- While these findings do not necessitate changes to the Productora pit design, an updated structural model and direct shear tests to evaluate joint shear strength properties will be required in the next project stage.

16.7.1.2 Alice

- No specific recommendations are needed for the Alice open pit.

16.7.1.3 Cortadera

- Assess the structural characteristics of the Skarn domains under oxide, transitional, and fresh weathering conditions. The evaluation should include joint set characteristics such as persistence, spacing, and shear strength properties.

- The fault structure labelled "076_IS_COR24_V1_0_EXTENDED" in the structural model (GeoEkun, 2024b) should be validated on-site for its location, orientation, and extent in the upcoming project stages, as it is critical to inter-ramp stability in Sections S01 and S05 of Cuerpo 2 under pseudo-static conditions.

16.7.1.4 San Antonio

- Develop a new structural model that delineates the boundaries of structural domains and describes their respective joint sets and characteristics, including spacing, persistence, and shear strength properties
- This structural model must also evaluate the existence, orientations, and extent of primary faults to enhance the quality of inter-ramp and overall stability analyses.

16.7.2 Geotechnical - Waste Rock Dumps

The acceptability criteria are defined based on the criteria suggested in the "Guideline for Mine Waste Dump and Stockpile Design" (Hawley & Cunning (2017)) based on current industry practices, as of the date of this Report, and are also recommended by SERNAGEOMIN (2023). However, these criteria are conservative and valid for PFS stages where data confidence is low. It is recommended that these values are re-evaluated in accordance with the guidelines provided by Hawley & Cunning (2017), where lower Safety Factor (SF) values are acceptable depending on a higher data confidence and the degree of consequence if a failure occurs.

The horizontal seismic coefficient used for the pseudo-static analysis, with a value of 0.2, is relatively high compared to typical values used in geographically similar open-pit projects. This coefficient, defined for long return periods, should be re-evaluated and validated through a seismic hazard report in future stages of the Project to ensure it reflects site-specific conditions more accurately.

In terms of the shear resistance parameters for deposited material, a characterization of the deposited materials is recommended. Due to the large maximum particle size of some dump materials (up to 2 m), a large-scale testing is unfeasible, so an alternative would be to conduct laboratory testing on a finer gradation sample of the material, such as the matrix method (Siddiqi, 1984), scalping method (Al-Hussaini, 1983), scalping-replacing method (Donaghe and Torrey, 1979) and parallel gradation method (Lowe, 1964). This characterization could also provide variability to the shear resistance of the deposited material and allow calculating the Probability of Failure (PoF) in future analyses.

16.7.3 Geotechnical - Underground

In addition to updating the geotechnical analysis in this report as new information becomes available, further work is needed to ensure data sufficiency for a feasibility study aligned with block caving standards. The following tasks should be completed prior to or during the next study phase:

16.7.3.1 Data collection:

In the next stage of studies, additional geotechnical drilling must be prioritised to ensure comprehensive coverage within the planned cave footprint and extraction levels. Drillholes should be designed with extended trace lengths that intersect the undercut and extraction levels. Dedicated geotechnical drillholes will also be essential for evaluating areas designated for critical infrastructure, such as access and conveyor declines, ventilation shafts, and crusher chambers.

In the next stages of the Project, it is important to implement more reliable stress determination methods, such as overcoring in drillholes and within future developments, to obtain more accurate data. In the early stages, acoustic emission techniques were used, which, while in most cases if effective in identifying stress orientation, often provide inaccurate measurements of stress magnitude.

Intact rock testing should be conducted within the hanging wall sediments host-unit, to gain a clearer understanding of mechanical properties. Additionally, conduct direct shear testing on joints and veins to assess behaviour under stress, which will provide essential data to enhance stability analysis.

Although limited in scope, Point Load Testing (PLT) should be conducted on core samples from future campaigns to quantitatively identify zones of weakness within the rock mass. Where possible, correlations with Uniaxial Compressive Strength (UCS) should also be established to enhance data coverage and capture the spatial variability of the rock mass.

16.7.3.2 Geotechnical hazards:

In addition to refining the current analysis on fragmentation, cave propagation, subsidence, and stability as more data becomes available, several geotechnical hazards and risks must be thoroughly investigated and addressed in the next project study stage. This proactive approach is standard practice in block caving projects, even when certain risks are considered unlikely or absent. Key risks to assess include:

- Air blasts
- Mud rushes
- Rock bursting
- Mine collapses
- Dilution
- Stability of vertical developments.

16.7.3.3 Numerical modelling:

16.7.3.3.1 Cave Flow and Recovery

It is recommended that flow modelling is incorporated into the FLAC3D simulations. This model should include essential parameters such as drawpoint spacing, material fragmentation, and height of draw to evaluate ore recovery and dilution. The results can then be compared against PCBC forecasts for verification, ensuring alignment with expected outcomes and validating the reliability of the PCBC model.

16.7.3.3.2 Numerical Modelling of Extraction and Undercut Levels

A large-scale 3D model of the extraction and undercut levels should be developed to assess rock mass damage resulting from stress abutment caused by the advancing undercut front and drawbell incorporation. This model will support the validation of the mining sequence, emphasizing the stability of the crown pillar (the rock mass between the undercut and extraction levels) and extraction level pillars. Additionally, it will evaluate key factors such as drawbell positioning ahead of the undercut front and the length of undercut lead-lags.

16.7.3.3.3 Crusher Chamber Modelling

It is recommended that 3D models of the crusher chambers are developed to evaluate their stability and assess ground support requirement.

16.7.4 Mining open Pit

The recommendations for the open pits can be subdivided into six main categories:

- Pit Design Optimisation
  - The pit designs were split into practical mining pushback designs. This was done to better stage the mining (particularly in the larger pit perimeters like Productora and Cortadera), however, due to time constraints, it is possible that the pit stages could be optimised to enable better material scheduling. This work has not been done as yet but should be considered. As the pit stage designs might evolve, the total pit designs could potentially be optimised further through a combination of ramp locations and optimising geotechnical berm placements.
- Pit Ramp location optimisation
  - During the mine cost modelling it became evident that the pit ramp locations and therefore the stage pit ramp locations were not optimising haulage requirements. Further work has been undertaken to optimise stage design ramp locations and in particular, ramp exit positions and ramp switch back reductions where possible. This is an ongoing study improvement requirement.
- Mine Schedule refinements/improvements
  - The study duration and complexity of ore sources and optimising value makes mine production scheduling and the sequencing of mining entities incredibly challenging. Even after optimising perceived project value through mine production scheduling, there are some minor practical constraints that always require attention. Total rock smoothing, smoothing of ore mining, sequential pit stage depletion, mining practical mining periods and sustaining reasonable mining periods within mining stages are important and is necessary for practical mining contract management. There are therefore a lot more mine scheduling options to be considered and evaluated post this study. The smoothing of total rock and ore and enabling more practical pit stage depletion strategies (not to bounce around in pit stages or preventing a lot of pit stage stop-start mining schedules). The current mine schedules are reasonable but can benefit from further improvements and optimisation.
- Mining Equipment
  - The current study considered a larger mining equipment unit/fleet option for bulk mining activities (bulk waste and some bulk ore mining). The pit designs considered changing from wide double lane ramps for the bulk of the upper pit benches changing over to single lane ramps in the deeper benches (with some reduction in mining truck productivity parameters). The very pit bottom sections then reduced the ramp widths further so the bottom most benches are mined with smaller mining equipment (100t trucks and matching excavators). There is certainly a case to upsize the mining ramps in the designs and to enable mining of large mining equipment to the very bottom of the pits. The wider ramps may create ore loss or add additional waste rock, however, these parameters and potential impacts need to be studies to absolutely understand the impacts these may have on mining cost vs. more waste rock or

reduced ore. It is unclear how the Project economics may change/be affected with mining equipment and pit design sensitivities.

- Surface mining infrastructure
  - Several waste dumps and stockpile were designed during the study and although the location and size of these dumps and stockpiles were reviewed in detail, it is possible that further dump and stockpile location optimisations/refinements could also improve/optimise haulage distances and therefore reduce mine haulage costs. It is therefore important to review and refine the surface designs to optimise haulage and mining cost. It may also reduce equipment requirements (if location and size optimisation is possible).
  - The crusher location and interaction of the open pit ore and the RopeCon loading point at Cortadera could also benefit from additional reviews and refinement design work. It is not clear how much improvement (if any) could be facilitated with a further design review and refinement, yet it is believed that this design review could be important.
- Geotechnical Design Criteria
  - The Geotechnical work for this study was done to a high level of detail but the mine designs only moved through one sequence of design-to-geotechnical reviews. With further geotechnical reviews and evaluations it may be possible to optimise pit design which could reduce waste rock or improve/increase ore mining. These potential improvements are not necessarily definitive and further study work is required to determine if there are potential optimisation upside.

16.7.5 Mining Block cave

Alternative crushers and block cave layouts may be considered for a full trade-off but were not included in the scope of this study. The description above provides a practical solution that has been used at other block cave operations, ensuring that the crushing and materials handling is practical and forms a good basis for future trade-offs in subsequent study work.

Crushers that may perform well in such a trade-off includes:

- Jaw crushers (single or double toggle)
- Gyratory crushers
- Jaw Gyratory crushers
- Rotary Breakers

Each of the types of crushers has advantages and disadvantages that may impact the fixed plant design for the planned Cortadera block cave. Larger gyratory crushers could be amenable to direct tipping without the static grizzly and apron feeder combination. Jaw crushers typically have an apron feeder and vibrating grizzly feeder combination to limit fines through the crusher. There has been much improvement to rotary breakers, and these are worth consideration due to their increased application in hard rock environments. They also have a much lower form factor but also requires a controlled feed.

17 Recovery Methods

17.1 Introduction

The Costa Fuego Project will mine and process fresh and oxidised ores to produce copper concentrate, copper metal and molybdenum concentrate. The copper concentrate will also contain gold and silver.

The Project spans approximately 30 km from east to west, either side of the Panamericana Norte Highway, and consists of four copper/gold deposits. The two main Project areas, Productora and Cortadera, are joined by a 20 km rope conveyor taking ore from Cortadera in the east to the flotation concentrator at Productora in the west.

The Productora and Alice deposits lie in the west of the Project area. These deposits are adjacent to each other, are close to the processing facilities and both contain a mix of oxide, fresh and transition ores. Fresh ore is direct tipped at the Productora primary crusher or is stockpiled. Oxide ore is transported to the oxide ROM pad or is stockpiled. Stockpiled oxide ore will be delivered to the oxide ROM pad as required. Transition ore will typically go to the concentrator but will be present to a lesser extent in the oxide stream.

All primary crushed fresh ore from Productora and Alice is conveyed a relatively short distance overland to the concentrator crushed ore stockpile. All ROM pad oxide ore from Productora and Alice is fed by loader to the oxide crushing circuit to prepare it for heap leaching.

The San Antonio deposit is a small satellite deposit located at the eastern margin of the Project. Ore from San Antonio is trucked to Cortadera for primary crushing and the crushed ore is loaded onto the rope conveyor for transport to Productora.

Surface ore from Cortadera is a combination of oxide (destined for leaching), fresh (destined for flotation in the concentrator) and transition. Transition ore is variable by definition and can be altered fresh material (supergene) or it can also be a mix of oxide and fresh at the alteration interface. Transition is typically sent to the concentrator, but relatively fresh transition ores may be sent to leaching when mined in close proximity to oxide ore.

All Cortadera oxide ore is stockpiled, and batches are reclaimed for processing on a regular basis by loader and truck. The oxide ore batch is primary crushed and sent by rope conveyor to be offloaded at the oxide Run of Mine (ROM) pad at Productora. Batch oxide crushing ceases when it is predicted that the Productora ROM pad will be filled to capacity by the ore in transit on the rope conveyor. Alternatively, it will revert to fresh ore crushing at a time that prevents the concentrator from running out of feed.

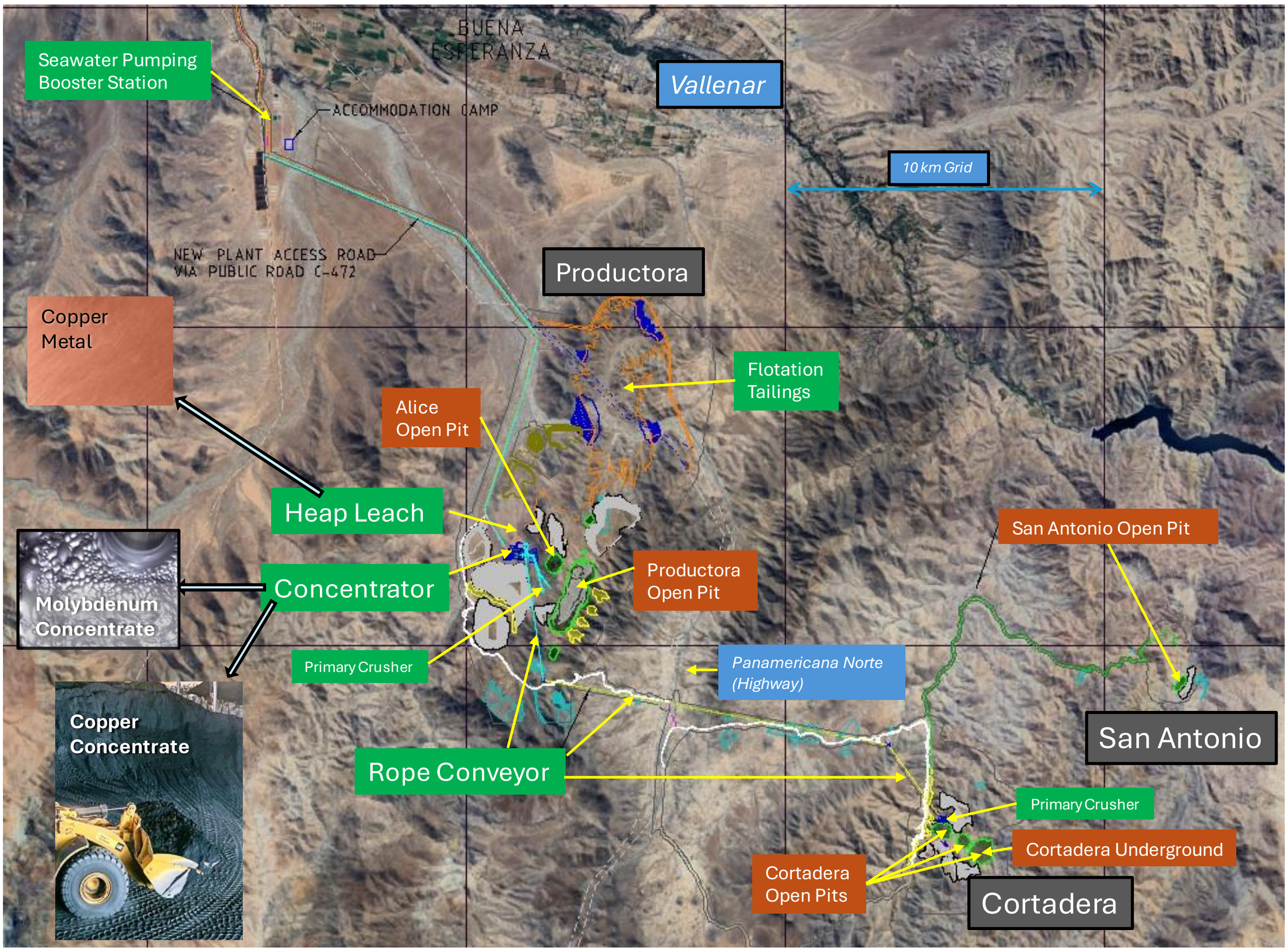
Fresh ore is normally direct tipped at the Cortadera primary crusher and sent by the rope conveyor to be offloaded at the crushed ore stockpile. Fresh ore will also be stockpiled at Cortadera, and, after reclaiming and primary crushing, it is loaded on the rope conveyor. During heap leach operations, all fresh Cortadera ore (direct tipped and reclaimed) is offloaded from the rope conveyor directly onto the concentrator crushed ore stockpile. When heap leaching ore preparation ceases later in the Project, the oxide crushing plant will be reconfigured and repurposed to prepare fine-crushed fresh ore as supplemental concentrator feed. Batches of fresh Cortadera ore (direct tipped or reclaimed ore) will then be offloaded from the Rope Conveyor at what was the oxide ROM pad, to provide part of the feed to the fine crushing circuit (the remaining fine-crush feed is provided from Productora and Alice).

Ore from the Cortadera Cuerpo 3 Block Cave is crushed underground and conveyed to the surface. It is continuously loaded onto the rope conveyor and all underground ore will be offloaded at the concentrator crushed ore stockpile.

The concentrator receives sulphide ores from all four Costa Fuego Project deposits, and it will make both copper and molybdenum concentrates. The liquors from heap leaching and dump leaching will be upgraded in a single solvent extraction (SX) facility followed by copper metal recovery using electrowinning (EW).

The geographic layout of the process components of the Costa Fuego Project are shown in Figure 17.1.

Figure 17.1: Geographic Layout of the Process Components of the Costa Fuego Project (HCH, 2025)



17.2 Production Plan

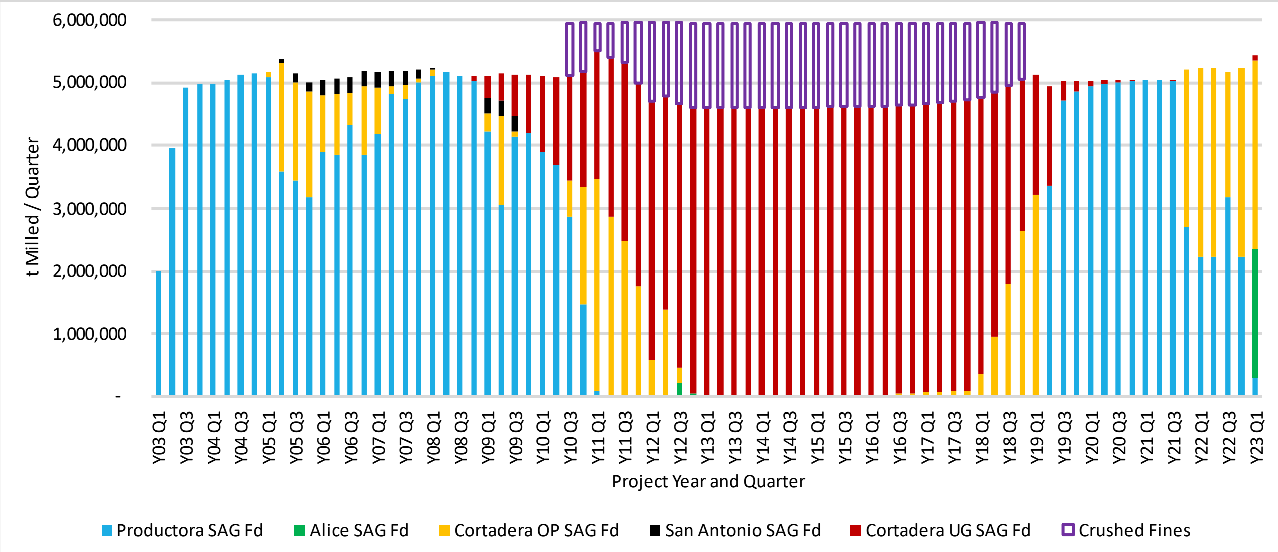
The Project has a processing ramp-up time of one year during which commissioning will occur of both the concentrator and oxide heap leach. The current mine schedule indicates LOM average annualised feed to the concentrator plant of 21.7 Mt/a, as summarised in Figure 17.2.

The LOM average feed rate includes coarse and fine feed, with the fine feed opportunistically fed in the middle of the Project. Primary crushed sulphide ore is represented by all of the solid columns in Figure 17.2.

The purple unfilled columns at the top of the graph represent crushed sulphide ore fines (P_{80} of 12 mm) that is co-fed with the coarse ore, but has no effect upon the SAG milling capacity of the circuit.

The red solid columns represent Cortadera underground ore (Cortadera UG) which is high competence and cannot be processed faster than indicated in Figure 17.2 as it is limited in the SAG mill.

Figure 17.2: Quarterly Ore Delivery to Concentrator by Deposit and by Supplemental Crushed Fines



The amount of crushed fines that can be co-processed with Cortadera UG is limited based on the volumetric capacity of the classification and flotation circuits and by the spare ball mill grinding opportunity that arises when treating Cortadera UG ore. The plant volumetric capacity has been limited to the equivalent of just under 6 Mt/Q (24 Mt/a), which is the PDC design capacity assigned to the comminution and flotation circuits.

The spare ball milling capacity results because the power required from the ball mill is well below the installed capacity. This is due to the low plant feed rate and the lower BWi associated with the Cortadera UG SAG feed. If the ball mill power was maintained at maximum under these conditions (and no fine ore was co-fed), the flotation feed would be severely overground. The maximum amount of fine crushed ore that can be co-fed then depends on the spare ball milling capacity and the fine crushed ore's grinding work index.

From Figure 17.2 the overall average feed rate and the average rates for each feed crush type are:

- All Milling Circuit Feed- 21.7 Mt/a (5% above the PDC Feed Rate of 20.7 Mt/a)
- Primary Crushed Feed - 19.7 Mt/a (full LOM)
- Fine crushed Feed - 4.7 Mt/a (part of LOM)

To demonstrate the comminution differences between the deposits it has been assumed that the average ores from each of the deposits are being treated alone in the grinding circuit. The resulting throughput rates are compared in Table 17.1. The throughput rates are calculated using the comminution models presented in Section 13. It must be recognised that at most times the concentrator feed will consist of a blend of ores from the various deposits, as shown above in Figure 17.2 and feeding of one ore by itself is uncommon.

Table 17.1: LOM Concentrator Feed Contributions and Stand-alone Average Throughput Rate by Deposit

Deposit	LOM Concentrator Feed from Deposit (Mt)	Relative SAG-Limited Feed Rates of Primary Crushed Ores (Mt/a) [1]	Basis # Test Samples	Concentrator Individual Deposit Coarse Feeds and Mixed Deposit Fine Feed (Mt) [2]	LOM Concentrator Feed Proportion (%)
Productora	215	21.2	36	195	44
Alice	8.6	23.2	2	2.4	0.5
Cortadera Open Pit	66	24.2	23	53	12
Cortadera Block Cave	146	20.1	22	146	33
San Antonio [2]	**3.3**	**20.4**	**1**	**3.0**	**1**
Crushed Fine Ore (all deposits)	**-**	**-**	**(81)**	**40**	**9**
Concentrator Feed	**439**	**-**	**83**	**439**	**100**

[1] Assuming 8,000 hours per annum and deposit ore is treated alone as SAG mill feed (only for comparative purposes)

[2] Taken from HCH_PFS_Schedule_Output_RevE (Final PFS mine schedule)

The deposit with the lowest average throughput rate is confirmed as Cortadera UG. It has the hardest average characteristics for SAG milling (i.e. the highest competence) resulting in the lowest average feed rate. The 80th percentile Cortadera UG ore can only be processed in isolation at 18.7 Mt/a.

Cortadera UG will be the sole source of primary crushed concentrator feed for a number of years in the middle phase of the Project, and while treating it the throughput bottleneck will dominantly be the SAG mill. Fine crushing to a P_{80} of 10 to 12 mm is small enough that it will pass through the SAG mill without impacting its capacity. However, at this size the fines are also coarse enough to demand significant grinding work by the ball mill. To facilitate fine crushing of hard sulphide ore, crushing of heap leach feed ceases and modifications are made to the oxide crushing circuit to suit the new duty. Fine crushed feed can be produced from direct tip sulphide ores or reclaimed sulphide ores, regardless of grade.

By management of the proportions of coarse and fine ores in SAG feed, both the SAG mill and the ball mills will be utilised at close to maximum available power during this Project phase. The grind P_{80} to flotation will generally be maintained at 125 µm.

The oxide heap leach circuit will treat up to 4 Mt/a and will average 3.7 Mt/a over the first 6.5 years. The PFS Heap Leach feed schedule is shown in Figure 17.3 and the overall deposit tonnages contributing to the feed are given in Table 17.2. The period where the oxide ore crushing circuit will be re-purposed is clearly evident in both Figure 17.2 and Figure 17.3.

Figure 17.3: Quarterly Ore Delivery Schedule (RevE) to Heap Leach by Deposit

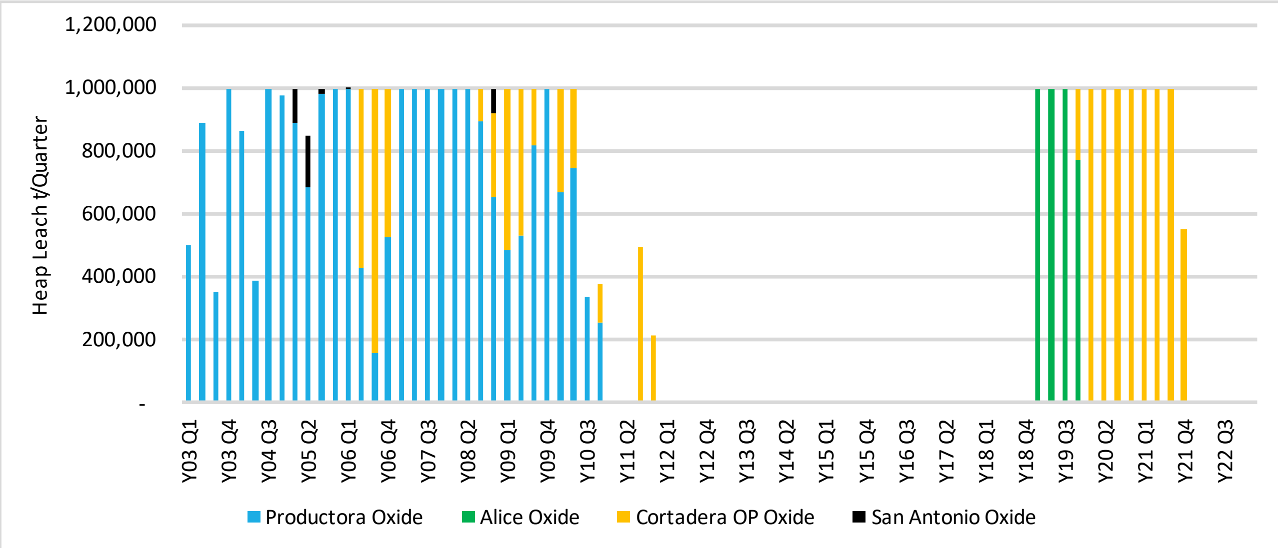


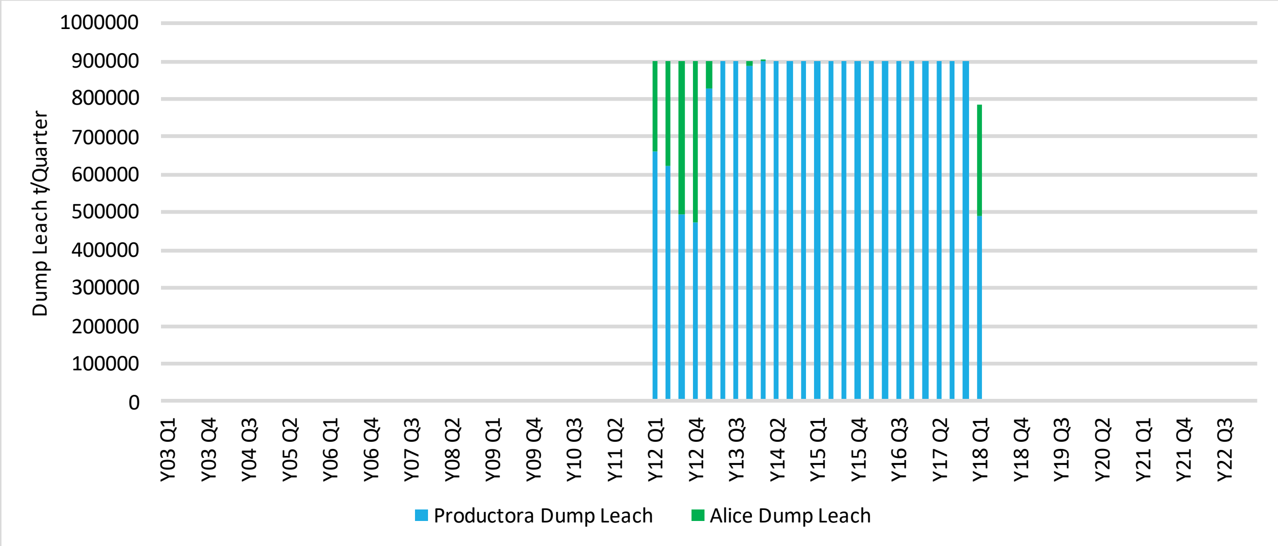
Table 17.2: LOM Heap Leach Feed Contributions by Deposit

Deposit	LOM Heap Leach Feed (Mt)	Heap Leach Feed (%)
Productora	24	59
Alice	3.8	9.2
Cortadera Open Pit	12.6	31
San Antonio	0.6	0.8
Heap Leach Feed	**41**	**100**

As Cortadera UG is all below the base of oxidation there is no oxide ore (heap leach feed) originating from the block cave.

Productora and Alice sulphide material below the variable mill-feed cut-off grade is stockpiled to be processed via a low-grade sulphide dump leach. The dump leach output replaces tapering oxide leach production and helps maintain consistent copper metal production through the SX-EW plant across project life. The Dump Leach schedule is shown in Figure 17.4. The operational Dump leach feed rate is 3.6 Mt/a.

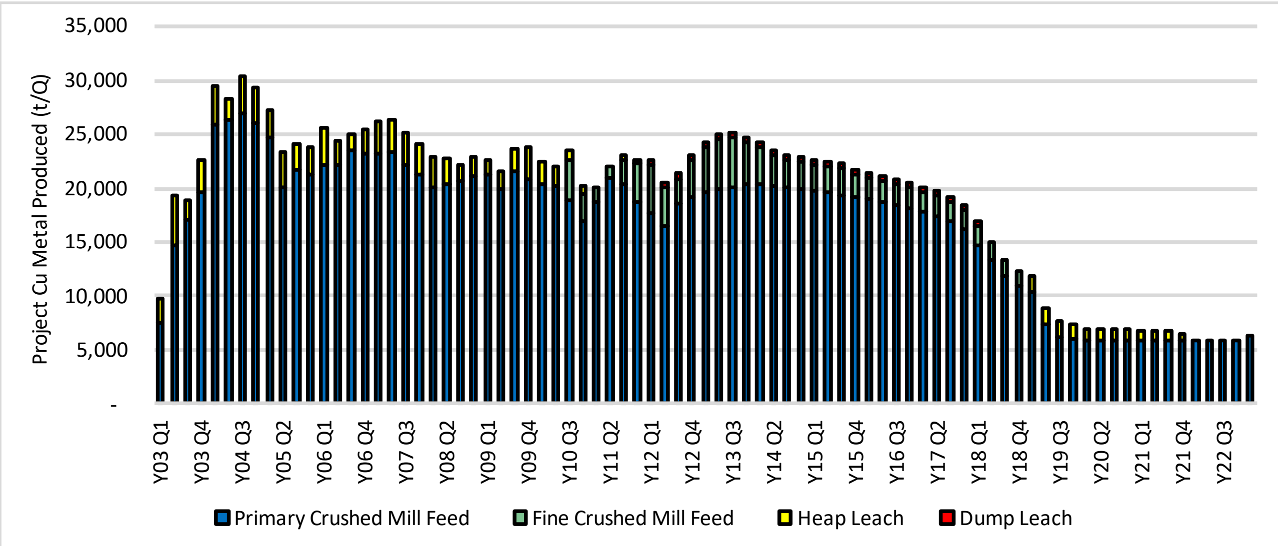
Figure 17.4: Quarterly Ore Delivery Schedule (RevE) to Dump Leach by Deposit



Note that due to their long residence times, both the heap and dump leach continue to produce for a number of quarters after the delivery of final ore from the mine.

From years 3 to 18 under current mine schedule, the average aggregate metal production across the three processing pathways of the Costa Fuego Project is 90.4 kt/a Cu, 46 koz/a Au, 153 koz/a Ag and 4,000 klb/a Mo. Quarterly copper production for the Project from all sources and for LOM is given in Figure 17.5. LOM metal production across the 20-year Project, including ramp up and ramp down and residual leaching, is 1,557 kt Cu, 786 koz Au, 2,561 koz Ag and 73,000 klb Mo. Note that in the latter part of the Project the concentrator and the leaching circuits will be treating stockpiled low-grade ores, hence the production drop-off.

Figure 17.5: Quarterly Copper Production for Project



17.3 Overall Project Layout

The Project layout (Figure 17.1) reflects the dominance of the two main orebodies, Productora and Cortadera, the lack of flat ground for processing facilities at Cortadera, the downhill aspect between Cortadera and Productora and the abundance of reasonably level ground at Productora for a concentrator, a heap leach facility and tailings storage. The requirement to transport ore efficiently from Cortadera to Productora for processing, and the problematic terrain between the two deposit areas, has led to the specification of a rope conveyor to minimise both cost and environmental impact.

Having Productora as the epicentre of Project activity provides easy access from local communities for employees and contractors. Productora is also adjacent to Chile's primary north-south Highway, Panamericana Norte, allowing easy road access for trucks delivering goods inwards and taking products to customers. In addition, Productora has good access to the port at Huasco for its seawater supply and for exporting copper concentrate. The Project's connection to Chile's national power grid is also located between Productora and Huasco.

Compared to Productora and Cortadera, relatively minor activity occurs at San Antonio, which is located at the eastern extent of the Project. All transport of goods, water, people and ore in and out of San Antonio occurs by truck. Ore is transported through challenging topography to Cortadera, where it is crushed then transported by rope conveyor to Productora concentrator.

17.4 Copper Sulphide Concentrator

The Productora Concentrator accepts primary crushed ores from Productora, Alice, Cortadera Open Pit, Cortadera Underground and San Antonio. A SAG and Ball mill circuit grinds the ore to achieve economic liberation levels ahead of rougher and scavenger flotation. Rougher and scavenger Cu/Mo concentrate is reground and then floated in the cleaner section. Separate Cu and Mo concentrates are produced for sale, with Au and Ag reporting to the Cu Concentrate. Thickened flotation tailings are stored in a conventional TSF.

17.4.1 History: Copper Sulphide 2016 PFS Comminution Circuit

The 2016 PFS comminution circuit was designed to grind 14 Mt/a of feed, significantly smaller than the nominal 21.7 Mt/a circuit in this study. The 2016 flowsheet consisted of SAG and Ball mills grinding to 150 µm P_{80} for Productora and 180 µm P_{80} for Alice. This compares to 125 µm P_{80} for all deposits in the current study. A single pebble crusher was selected to operate in closed circuit with the SAG mill. The flowsheet was designed to process Productora and Alice ores as Cortadera was yet to be discovered.

The 2016 circuit consisted of a 20 MW SAG mill followed by two 9.5 MW ball mills. The selected 2016 SAG mill size is consistent with the mill selected in this 20 Mt/a study. The ball mill power selection in 2016 (totalling 19 MW) is half the installed ball milling power selected in this study, which means that the power difference is higher (proportionally) than the tonnage difference. However, when compared on both a tonnage increase basis and incorporation of a finer grind size, the ball mill scaling between the two studies is entirely consistent.

17.4.2 Comminution Circuit Design for PFS

Comminution modelling was performed to confirm mill selections and select crusher and screen sizes. The inputs to modelling included the mill feed rate, the feed size distribution, the ore comminution properties, the flowsheet setup and the grind size target. Power based modelling has been performed for this study, utilising the Morrell Power Method.

Modelling was also performed using JKSimMet to cross-check the power-based calculations and to provide estimates of key stream properties, such as the ball mill cyclone feed.

17.4.3 Copper Concentrator Design Basis

The comminution design criteria are summarised in Table 17.3. Note that the design conditions were selected for the critical early mine life, before the introduction of fine crushed mill feed.

Table 17.3: Comminution Circuit Summary Process Design Criteria

Criteria	Units	Value	Source
Annual capacity (Years 3 to 10)	Mt/a (dry basis)	20.7	Hot Chili
Crushing utilisation	%	70	Wood
Crushing circuit operating hours	h/a	6,132	Calculated
Crushing circuit throughput	Dry t/h	3,376	Calculated
Primary Grinding circuit utilisation	%	91.3	Wood
Primary Grinding circuit operating hours	h/a	8,000	Calculated
Primary Grinding circuit throughput	Dry t/h	2,588	Calculated
Primary Grinding circuit product P_{80}	µm	125	Testwork*
Feed proportion to SAG pebble crusher	%	15	Wood
DWi (Average SAG mill competency)			
Productora	kWh/m^3	7.5	Testwork
Alice	kWh/m^3	7.0	Testwork
Cortadera OP	kWh/m^3	6.6	Testwork
Cortadera UG	kWh/m^3	8.7	Testwork
BWi (Average ball mill grindability)			
Productora	kWh/t	18.4	Testwork
Alice	kWh/t	17.4	Testwork
Cortadera OP	kWh/t	14.2	Testwork
Cortadera UG	kWh/t	16.9	Testwork

*: Also see section 13.3.2.9

The specific energy requirements (SE, power at pinion in kWh/t) and the throughput requirements are used in combination to arrive at the design pinion power requirements for each equipment item. The design power requirements are then escalated using factors (operating headroom to allow for media wear cycles and drive efficiency through air gaps, pinions and gearboxes) to arrive at the maximum drawn power and subsequently allow the selection of the installed motor power from a set of standard motor sizes. All these values are summarised in Table 17.4.

Table 17.4: Major Equipment Power Selection

Units	Design Specific Energy	Pinion Power	Operating Headroom	Max Pinion Power	Drive Efficiency	Max Motor Output	Installed Motor
	kWh/t	MW	%	MW	%	MW	MW
Primary Crusher	0.1	0.44	35	0.68	93	0.73	0.75
SAG Mill	8.6	23.6	13	27.1	97	28.0	28
Pebble Crusher	0.2	0.61	20	0.77	95	0.81	0.94
Ball Mill	10.8	31.1	5	32.7	95	34.5	38
Total	19.7	55.8		61.3		64.0	67.7

A SAG mill size of 28 MW was selected because it is the largest installed gearless mill drive (GMD) size globally. The 38 MW of Ball mill power is supplied by two 19 MW mills, the largest twin pinion drive arrangement Wood recommends. It is these installed and available (pinion) milling powers that set the plant capacity, as described above.

The selected comminution equipment is described in Table 1.10.

Table 17.5: Major Comminution Circuit Equipment Selections

Element	Units	Model	Installed Power (MW)	Set (mm)	Diameter (ft)	Length (ft)	Diameter (m)	Length (m)
Productora Sulphide Primary Crusher	1	Superior 60-89 MK III Gyratory	0.60	175 (open side)				
Cortadera Surface Primary Crusher	1	FLS KB 130-75 DM Pro Jaw Gyratory	0.65	152 (open side)				
Cortadera Underground Primary Crusher	2	Superior 54-75 MKIII-UG	0.6	152 (open side)				
SAG Mill	1	28 MW GMD	28		40	28	12.2	8.53
Pebble Crusher	1	Metso MP 1250	0.94	13 (closed side)				
Ball Mills	2	19 MW Twin Pinion	38		28	40	8.53	12.2

The FLS KB (Jaw Gyratory) unit shown is the double-sided mouth version (DM) suitable for direct ROM tipping installations.

17.4.4 Comminution circuit Throughput Predictions

The equations used to predict milling throughput rates in mine planning are indicated in Table 17.6. Note that for additional conservatism the available pinion powers have been set about 5% lower than the available pinion kW based on the installed motor power values.

Table 17.6 : Variable Throughput Calculations

SAG Mill Throughput (TPH)	TputSAG = kWAS /(0.4184 * (5.1113 * DWi ^ .7059) - 0.4305) where kWAS (net SAG mill available power) = 23 MW
Ball Mill Throughput (TPH)	TputBM = kWAB / (BWi * 0.7881 - 2.453) where kWAB (net Ball mill available power) = 34 MW

The throughput calculations have the general form for a given mill (X):

Tput(X) = KWA(X) / SE(X)

Where SE(X) is the specific energy, in kWh/t required for the milling stage. The specific energy is the power consumed at the mill shell per tonne of feed.

The prediction of DWi and BWi is covered in Section 13.

The result of the calculation of SAG mill (TputSAG) throughput and Ball mill throughput (TputBM) is derived for each block. The block throughput value (Tput) is determined by the mill, which is the bottleneck, which is the lowest of the SAG mill and Ball mill throughput values. The average estimated throughput values for each milling stage and for the circuit as a whole are shown in Table 17.7.

Table 17.7 : Average Throughput for SAG and Ball Mills and Grinding Circuit by Deposit in Mine Plan

Variable	Productora	Alice	Cortadera OP	Cortadera UG	San Antonio
TputBM (tph)	3,880	3,366	3,744	3,419	2,546
TputSAG (tph)	2,631	2,810	2,961	2,348	3,141
Tput (tph)	2,464	2,810	2,840	2348	2,546

Note that Tput is a theoretical calculation and can result in a throughput value that is greater than may be possible under any circumstances across the full plant. Where the minimum of the TputSAG and TputBM for a given feed exceeds a nominated plant throughput upper limit then Tput may be capped to this upper limit.

17.4.5 Ore Properties, Mill Selections and Plant Throughput

Through this process there are many different ore properties and throughput rates generated. Apparent inconsistencies exist between values reported here, but that is because some values are first chosen to frame the project and then later the same values are estimated on a broader basis using methods developed in the PFS. The flow of this procedure is as follows.

- Measure ore properties on metallurgical variability samples
  - Develop averages and 80th percentile values for each deposit and for the overall sample set

- Propose mills based on global installed precedent
  - 28 MW GMD SAG Mill
  - 2 x 19 MW Twin Pinion Ball Mills

- Conduct preliminary comminution modelling to estimate the plant throughput "nameplate"
  - Estimated at 20.7 Mt/a for the early project life
  - This value becomes the basis for the Process Design Criteria and Mass Balance
  - From that point on the "nameplate" becomes a reference point for assessing deposit treatment rate response and for setting up throughput models for mine planning
  - Once mine planning has been performed the resulting ore delivery schedule takes precedence over the nameplate

- Conduct milling circuit throughput calculations using the overall averages and 80th percent values
  - Ensure that the average ore properties result in an annual plant throughput rate equal to or greater than the nominated feed rate of 20.7 Mt/a (22 Mt/a was achieved).
  - Ensure that the 80th percentile ore properties result in an annual plant throughput rate that is approximately equal to the feed rate of 20.7 Mt/a (20.3 Mt/a was achieved).

- Conduct milling circuit design calculations using each deposit average and 80th percent values
  - Confirm that the selected mills accommodate all deposits at reasonable throughput rates

	Average Properties (Mt/a)	80th Percentile Properties (Mt/a)
Productora	21.2	18.5
Alice	23.2	22.1
Cortadera OP	24.2	20.1
Cortadera UG	20.1	18.7
San Antonio	20.4	20.4
Weighted Average	21.2	18.8

- Develop geometallurgical prediction methods
  - For the BWi
  - For the DWi
  - For determining a conservative limiting plant throughput rate on a block basis

- Embed the throughput prediction methods in the block model
  - Develop a mining plan
  - Develop and ore delivery schedule

- Interrogate the mine plan
  - Extract the blended feed throughput rates on a quarterly basis
  - Determine the predicted average rates for LOM
  - Determine the predicted average rates for important operating periods (such as the first 7 years)

- Optimise Mine Plan
  - Constrain and schedule to produce consistent annual metal output (applying recovery models described below)
  - Implement stockpile working strategies
  - Optimise ore destinations
  - Investigate opportunities for taking advantage of unused milling power (such as fine crushing sulphide ore and co-feeding it when spare ball milling capacity is available).

Once the mills have been selected then the throughput rates are driven by ore properties and the selected mill powers, not by the values assumed as "nameplate".

17.4.6 Copper Sulphide Flotation Circuit Design Criteria and Process Flow Diagrams

The copper sulphide concentrator is designed to process open pit and underground mined sulphide ores. Other criteria have been updated in line with reassessments of data, such as the significant correction to Mo recovery described in Section 12. Other criteria were selected to ensure that extended quarterly periods of high values were accounted for. It must be noted that if high values persist for a number of quarters then the day to day fluctuations of grade and recovery will result in even higher values that must be managed by sections of the plant such as cleaning and concentrate handling.

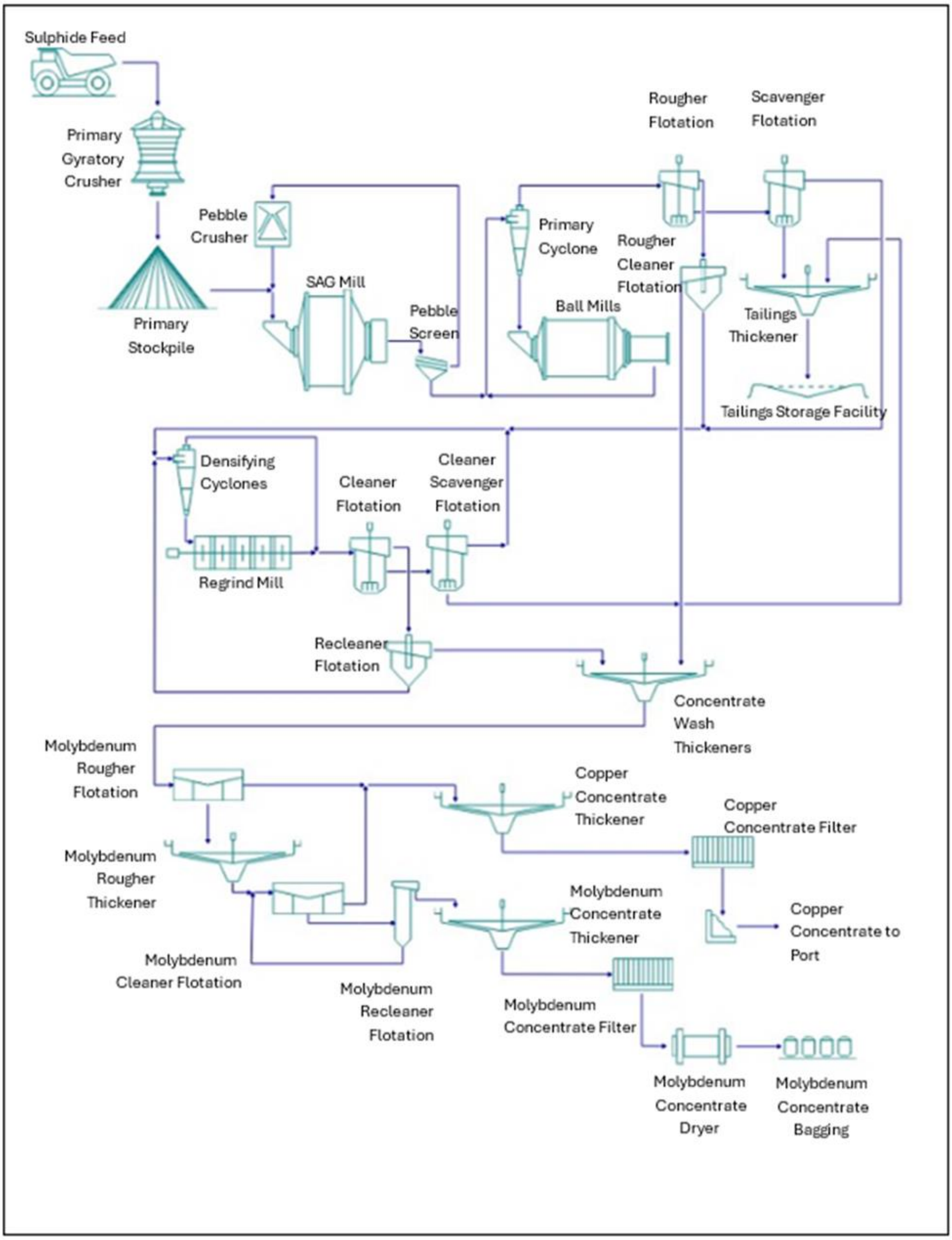
The principal criteria underpinning the sulphide concentrator design are:

- Annual capacity (Project Years 3 to 10): 20.7 Mt/a
- Concentrator throughput rate (nominal): 2588 t/h
- Design production feed Cu head grade: 0.53%
  High grade period Y4Q1 to Y7Q3
- Cu recovery to Cu concentrate: 87.2%
  (Avg post commissioning to Y18)
- Final concentrate grade: ≥25% Cu.

Regrind milling will occur using an IsaMill, selected for its ability to produce a sharp product size distribution and because it has internal classification. A densifier cyclone is still required ahead of IsaMIlling to ensure the mill feed density is controlled. The design requirements for regrinding are given below. Note that a signature plot will be required for IsaMill selection and warranty purposes. This is recommended to be performed in FS as the Glencore online IsaMill Selection calculation is approximate.

- Regrind Mill Feed F_{80} (densified) 150 µm
- Regrind Mill Feed F_{80} 25 µm
- Regrind mill new feed t/h (densified) 88 t/h (from Mass Balance)
- Regrind mill new feed t/h (densified) 143 t/h (High feed rate and mass pull 6% of feed)
- Regrind Mill Specific Energy (kWh/t) 12 kWh/t (reversing Glencore IsaMill selection online calculation)

Figure 17.6: Simplified Sulphide Process Plant Flowsheet



Source: Wood, January 2025.

The concentrator feed primary stockpile will also receive feed from Cortadera open pit and underground via rope conveyor. In addition, at year 10 (which is processing year 8, because mining commences in year 1 and processing in Year 3) the oxide circuit crushers will be modified and used to crush additional sulphide ores which will then be conveyed to the SAG Feed stockpile.

17.4.7 Copper Sulphide Process Plant Description

17.4.7.1 Primary Crusher Productora

Run of mine (ROM) process feed from Productora pit and Alice pit is direct tipped to the Productora primary crusher or stockpiled remotely from the ROM pad. Stockpiled fresh ore will be loaded into trucks for delivery to the primary crusher when required. The primary crusher dump pocket has a minimum capacity of 2.5 truckloads to allow trucks to direct dump into double-sided tip points above the crusher. Plant design assumes a large majority of the ore is direct feed from trucks to the crusher. Production feed that arrives at the ROM pad but is unable to be dumped directly will be temporarily tipped onto the ROM working stockpile. A loader will be used to feed ROM working stockpile ore to the ROM bin as required.

17.4.7.2 Primary Crusher Cortadera Open Pit

The Primary crushing duty at Cortadera is much less than at Productora. For this reason, a small jaw-gyratory crusher and single sided tipping has been recommended. Although the tipping is single sided, the jaw-gyratory has wide Jaw style mouths on both sides of the spider arm to facilitate direct tipping.

ROM ores from the Cortadera pit are transported by truck to the Cortadera ROM area or to a remote oxide or sulphide stockpile. A ROM pad working stockpile is also available if trucks are unable to tip. Remote stockpiled Cortadera ores are reclaimed by loader and delivered to the ROM bin by truck as necessary. During normal operation, sulphide ore is the main feed to the crusher, and it will be direct tipped into the ROM bin. Campaigns of oxide ore will occur periodically using loaders or excavators filling trucks at the oxide remote stockpile. Typically, Cortadera sulphide ore mined during these oxide campaigns will be remote stockpiled.

The surge chamber apron feeder under the crusher controls the feed onto the sacrificial loading conveyor. Cortadera block cave ore is conveyed from underground crushing to the surface and is also loaded onto the sacrificial loading conveyor. The loading conveyor feeds the first section of the rope conveyor which takes the ore to Productora. Due to the terrain the rope conveyor will be in three constant operating sections followed by a fourth section which allows switching between two destinations. The third rope conveyor section delivers onto a reversible section of rope conveyor which will either deliver to the concentrator crushed ore stockpile or, when reversed, deliver to the oxide ROM pad.

17.4.7.3 Underground Crusher(s) Cortadera

Two underground crushers are part of the block cave design, and each is located at close to the deepest levels in the mine, below the block cave drawpoint level. Underground crusher feeding arrangements are described in Section 16. An apron feeder under each crusher discharge surge chamber controls feed onto the first incline conveyor that brings block cave ore to the surface. A number of straight inclined conveyors are needed to navigate the path to the surface. The last incline conveyor emerges at the surface and transfers directly onto the sacrificial conveyor taking primary crushed surface ore to the rope conveyor. Underground ore can only be received onto the loading conveyor when concentrator feed is being crushed on the surface or when the loading conveyor is empty. Underground ore crushing and conveying ceases when oxide ore is being crushed at Cortadera.

17.4.7.4 Rope Conveyor

The Rope Conveyor has been designed for a maximum capacity of 3640 t/h (29 Mt/a) and a nominal capacity of 3140 t/h (25 Mt/a). The limit of 3140 will normally be adhered to with the maximum allowances being provided for short term surges.

The Cortadera primary crusher sacrificial conveyor delivers all feed onto the rope conveyor. Two weightometers will control the output from the apron feeder on the Cortadera primary crusher discharge chamber. The first weightometer will control the apron feeder speed to deliver to its setpoint. The second weightometer measures the combined underground ore from the portal plus the Cortadera primary crusher product and controls the loading of the rope conveyor. A further weightometer on the underground conveyor measures the Block Cave ore produced for accounting purposes and provides a redundant feed-forward measure for rope conveyor loading control.

17.4.7.5 Concentrator Primary Stockpile

The concentrator primary stockpile is fed by the Productora overland conveyor and the rope conveyor from Cortadera. Late in the Project it will also be fed fine crushed sulphide ore from the repurposed oxide crushing circuit.

The crushed ore stockpile live capacity is 42,700 t (16 hours). The design has an additional hour of live emergency capacity in the event it is necessary to empty the rope conveyor onto an already full stockpile.

Three reclaim apron feeders, mounted underneath the stockpile will control the flow of the material from the stockpile onto the SAG mill feed conveyor. Any two of the three feeders can be used to provide the full SAG mill feed and this allows for maintenance to be conducted on the third. It is also possible to use all three feeders simultaneously, especially as this flexibility may be needed to provide SAG feed particle size distribution management.

An emergency apron feeder also loads onto the SAG mill feed conveyor and is fed by the emergency feed bin. This emergency facility is used to keep the concentrator running or partially running through primary stockpile ore shortages, during any events in the reclaim tunnel, simultaneous downtime on two out of three reclaim feeders or during major upstream maintenance activities. Note that the emergency feed will usually be sourced from the dead part of the outer stockpile, which is coarse due to natural ore segregation. Typically, this will lead to operation from the emergency feeder being SAG mill limited at rates well under 20 Mt/a.

17.4.7.6 Primary Grinding

The SAG mill feed conveyor discharges into the SAG feed chute where high saline process water is also added. The SAG mill feed conveyor is fitted with a weightometer to monitor and control the ore and water feed rates to the SAG mill. The mill will be driven by a gearless motor which is inherently a Variable Speed Drive (VSD) and is highly instrumented for reliability and controllability.

A grate at the discharge end of the SAG mill allows slurry to exit along with pebbles as coarse as 75 mm. The pebble-slurry mix is delivered to the central discharge trunnion by pulp lifters. The SAG mill trunnion discharges onto a single large vibrating screen and an operationally ready spare is always available on a roll-out roll-in system or a rapid lift-out lift-in system. Screen oversize pebbles are loaded onto the pebble crusher feed conveyor while the screen undersize slurry mixes with ball mill discharge and is pumped the ball mill cyclones.

Additional fine feed is added to the SAG mill feed conveyor from the repurposed oxide crushing circuit beyond year 10 (which is processing year 8) of the Project. The feed rate of fine ore is controlled by the responses of the ball mill and cyclone circuit in terms of grind P_{80} and recirculating load.

17.4.7.7 Pebble Crushing

Pebbles from the SAG mill discharge screen are crushed and then returned to the SAG mill via the SAG mill feed conveyor. The pebble feed conveyor has an overhead magnet to remove partial or whole grinding balls, a weightometer to monitor the pebble production rate and a metal detector to further protect the pebble crushers. The pebble conveyor feeds a bin which feeds the crusher, which discharges onto the product conveyor and then to the SAG mill feed conveyor.

17.4.7.8 Primary Classification

The mill discharge hopper has two pairs of duty standby pumps and each feeds a cyclone cluster, one for each of the ball mills. The cyclone underflow is returned to the ball mills. The cyclone overflow passes through trash screens before feeding flotation. Cyclones are operated to target a grind P_{80} of 125 µm.

17.4.7.9 Secondary Grinding

Two twin-pinion ball mills operate in parallel, and each is in closed circuit with its own dedicated cyclone cluster. The mills are overflow discharge with trommel screens fitted to capture ball scats and any rock scats. The mills are fitted with variable speed drives to minimise startup power impact on the supply grid and provide operational flexibility.

17.4.7.10 Grinding Media Charging

Automatic ball feeding is provided to all mills to maintain optimal charge levels at all times. For the SAG mill, the balls are fed onto the (loaded) SAG feed conveyor by a ball feeder. For the ball mills, a bucket elevator lifts balls to a launder for distribution to both mills. The ball addition rate is controlled to maintain power draw setpoint levels in the ball mills.

Crane-based fast loading facilities are used for first media fill and in the rare event a mill needs to be emptied and then refilled after major maintenance. These facilities are not permanently installed and are stored elsewhere on site when not required.

17.4.7.11 Cu/Mo Rougher and Scavenger Flotation

Trash screen undersize reports to the copper rougher flotation feed conditioner. Collector is added to the conditioning tank and frother is added at the tank overflow discharge. Overflow feeds a bank of rougher/scavenger tank flotation cells operating in series. The first two cells of the bank are roughers and the last four cells are scavengers. Rougher and scavenger concentrates are collected and pumped separately.

The destination of the rougher concentrate is flexible. It can be sent to regrind feed with the scavenger concentrate or it can be sent to a Jameson cell for cleaning without regrinding. Rougher-cleaner concentrate at final concentrate grade is sent to the Cu/Mo concentrate thickener. When the rougher-cleaner cell cannot produce final concentrate grade or when the combined concentrate has not achieved target grade, the rougher concentrate is redirected to the regrind mill densifying cyclone and the Jameson cleaner cell is bypassed (or it can be repurposed as the start of the main cleaning circuit). Cu/Mo rougher concentrate destination is at the operator's discretion and will be guided by continuous concentrate grade monitoring.

Scavenger concentrate is always sent to regrinding before cleaning.

17.4.7.12 Regrind Mill

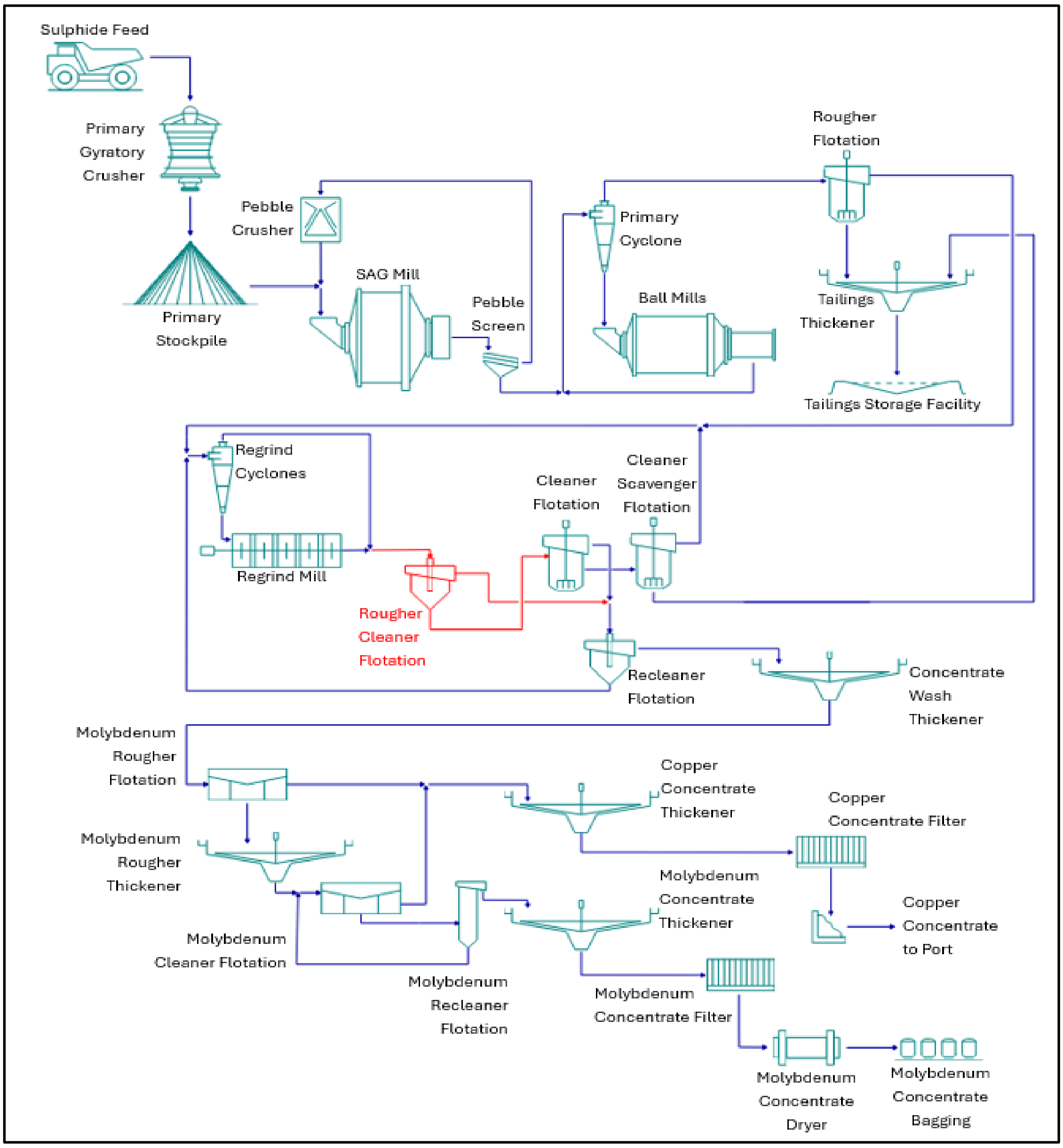
The regrind IsaMill is fed at 45 to 50% solids (densifying cyclone underflow) and the discharge of the IsaMill is internally classified to the target P_{80} (nominally 25 µm). Mill discharge slurry is combined with the densifying cyclone overflow (also nominally 25 µm P80) before pumping to the cleaner flotation circuit. Ceramic media will be added to the IsaMill feed slurry as required. Management of dumping and reclaim of IsaMill ceramic media is incorporated in the vendor design.

17.4.7.13 Cu/Mo Concentrate Cleaning

The rougher cleaner will be a Jameson cell with froth washing producing, in many circumstances, a final grade concentrate in excess of 25% Cu (Figure 17.6). If the cell is unable to make a 25% Cu concentrate or the combined final concentrate is less than 25% Cu, then the rougher concentrate will be diverted to densifying cyclone feed and to regrind. The circuit with regrinding of all the Rougher-Scavenger concentrate is shown in Figure 17.7.

Both circuits (Figure 17.6 and Figure 17.7) use the same equipment in different modes. The switch between modes is made by changing pumping destinations.

Figure 17.7: Cu/Mo Flotation with Rougher-Scavenger Regrinding



Source: Wood, January 2025.

The rougher and scavenger banks are large tank cells. The cleaner and cleaner scavenger banks are smaller tank cells and all final concentrate is produced by Jameson Cells. The IsaMill grinding duty will be variable due to multiple factors including changes due to ore source (deposit), varying Cu grade in concentrator feed and the regrinding (or not) of rougher concentrate.

Although already reground, the middlings from cleaning are best recirculated in a way that they have another chance for liberation. This is achieved by recirculating middlings direct to IsaMill feed, bypassing the densification step. As this is a low density stream it will lower the dilution water requirement needed to adjust the densifying cyclone underflow to the IsaMill feed density.

Scavenger tailings and cleaner scavenger tailings are both rejected to final tailings.

17.4.7.14 Copper/Molybdenum Concentrate Washing

The Cu/Mo concentrate will be separated into copper and molybdenum concentrates. The molybdenum flotation circuit will be described separately, later in this section. However, molybdenum flotation performs poorly in seawater and must be carried out in reverse osmosis (RO) water. In addition, chloride removal is essential to ensure copper concentrate quality.

Most of the seawater is removed from the Cu/Mo concentrate (serving both purposes above) using two counter-current decantation (CCD) washing thickeners. Concentrate is fed to the first wash thickener and combined with overflow water from the second. The second wash thickener receives underflow from the first and RO water as a dilutant. The second thickener underflow, at 66% solids, is repulped in RO water and then fed to the molybdenum flotation conditioner. After molybdenum rougher flotation the final copper concentrate is sent to the copper concentrate thickener.

17.4.7.15 Copper Concentrate Thickening

The final copper concentrates returning from the molybdenum rougher and cleaner scavenger flotation circuit are thickened in a high-rate thickener to 66% solids. The thickener underflow is pumped to the copper concentrate filtration circuit. The thickener overflow water is returned to the low saline process water tank for reuse in the flotation plant.

17.4.7.16 Copper Concentrate Filtering

The copper concentrate thickener underflow is pumped to an agitated filter feed tank for storage and feeding to a concentrate dewatering filter. Slurry from the concentrate filter feed tank is fed to one of two horizontal pressure filters to produce a filter cake with a target moisture content of less than 10% by weight. It is important that the moisture content of the copper concentrate be safely below the transportable moisture limit (TML). Both filters are designed for full plant throughput allowing for maintenance, but they are also able to be used concurrently.

Filter cake from the filter presses is discharged under gravity directly to a concrete bunker before being transferred to stockpiles by front-end loader (FEL). Concentrate for export is loaded on to trucks by FEL.

17.4.7.17 Tailings Disposal

Cleaner scavenger tailings and rougher scavenger tailings are pumped to two duty high-rate thickeners operating in parallel. The tailings thickener underflows, at about 70% solids, are pumped to a tailings discharge tank before being pumped to the tailings storage facility (TSF) using two-stage pumping. The second pumping stage is located at the toe of the TSF embankment as it will only be required when TSF walls reach the single stage pumping height limit. A detailed analysis of the tailings pumping system is recommended during the Feasibility Study as it may show that the second pump station can be delayed to later in the mine life.

The first stage of the tailings pumping system is located near the tailing thickeners. Both stages of the tailings pumping system are comprised of duty and standby pumps. The tailings thickener overflow water reports via a gravity pipeline to the high salinity process water pond for reuse in the concentrator.

Decant return water from the TSF is pumped to the high salinity process water pond using a mobile decant pontoon pump and a fixed diesel (or solar) powered decant return water pump station. The two pumps are operated on a duty-standby fashion.

17.4.8 Copper Sulphide Concentrate Handling

The filtered copper concentrate stacked on a concrete bunker is reclaimed using a loader and either transported to an undercover storage area or loaded directly on to trucks for transport to the port concentrate storage shed and ship-loading facility.

An undercover storage area at the port stores up to 1.5 ship-loads of filtered cake.

17.4.9 Copper Sulphide Flotation Reagent Handling and Storage

The reagents that will be used in the copper sulphide process plant include:

- Frother
- Collector
- Flocculant.

Reagent mixing occurs in designated areas within the process plant. The design of these areas considers requirements such as section bunding, with dedicated sump pumps for individual reagents, segregated ventilation, distribution within the required plant area(s) and dust and fume control around reagents with potential for dust or fume release. The layout and general arrangement of the reagent area accounts for the separation between incompatible reagent types.

17.4.10 Copper Sulphide Utilities

17.4.10.1 Introduction

The utilities include equipment and facilities to store and distribute the following water services:

- Raw (sea) water
- Desalinated (RO) water
- Fire water
- Filtered water
- Potable water (including safety shower/eye wash)
- Gland water
- Cooling water
- Process water (high and low salinity)

- Concentrated Brine (200 g/l NaCl)
- Tailings decant water
- Underground mine dewatering water (treated).

Air services for the concentrator are:

- Compressed air
- Instrument air
- Low pressure flotation air.

17.4.10.2 Raw Water

Seawater is used as raw water in the processing plants and at the mines. Seawater may be delivered via the infrastructure described in Section 18.0 as makeup water to the seawater pond at the heap leach SX/EW plant and will form the sole makeup water to the seawater pond located adjacent to the sulphide concentrator plant.

Two raw water pumps, operating in duty and standby mode, supply raw water to the concentrator, the RO treatment plant, and the fire water tank.

The fire water tank has three skid mounted pumps. An electric jockey pump to keep the line primed, a main electric pump and a separate diesel pump in case of an electrical outage during a fire.

17.4.10.3 High Salinity Process Water

High salinity (HS) process water is recovered from concentrate and tailings for re-use in the concentrator. HS process water is sourced from the following:

- Tailings dam decant return water pond
- Tailings thickener overflows
- Copper/molybdenum concentrate CCD1 overflow
- Raw water from site raw water pond.

Process water pumps, operating in two duty and one standby mode, supply process water to the concentrator. The make-up water for the process water pond is from the raw water pond.

17.4.10.4 Low Salinity Process Water

Low salinity (LS) process water is recovered from concentrate for re-use in the concentrator. LS process water is sourced from the following:

- Cu concentrate thickener
- Mo concentrate thickener.

RO water is used within the molybdenum cleaner flotation circuit and exits this circuit as low saline process water, which is stored separately in a low saline process water tank.

LS Process water is used in molybdenum rougher flotation.

17.4.10.5 RO Water

A reverse osmosis (RO) water plant treats seawater to provide low chloride water. RO water is used to wash the copper concentrate and in the molybdenum cleaner flotation circuit. Also, RO water feeds a demineralisation plant which supplies SX-EW. The RO water treatment plant is a vendor supplied package. The RO plant will also supply a treatment plant to produce potable quality water for use across the Project.

17.4.10.6 Concentrated Brine

Brine from the RO treatment plant is sent to an Electrodialysis plant to produce concentrated brine (200 g/l NaCl) for use in the heap and dump leaching processes. Excess concentrated brine is added to the leaching raffinate pond.

Electrodialysis (ED) is a process mostly known as a medical treatment for diabetes, but it is also commonly used in water purification. Often electrodialysis is applied to brackish water (low saline) as a competing technology to RO. However, another established application is the manufacture of highly concentrated brine (~200 g/l NaCl), typically from RO brine (50 to 70 g/l NaCl).

All of the RO brine produced at Productora will feed ED which will produce 200 g/l concentrated brine and a complementary flow of slightly saline water (~5 g/l) which can be sent to the process water pond. Salt is essential for the leaching process and the concentrated brine eliminates the need to import dry salt by truck. The substantial transport cost for trucking salt over 1000 km from the north of Chile makes it uncompetitive compared to producing concentrated brine on-site by electrodialysis using what is considered an RO waste stream.

The concentrated brine is used in the heap leach agglomerator and in the initial irrigation of the dump leach. Any excess concentrated brine is sent to the leaching raffinate pond to increase its salinity and its effectiveness during ongoing leach irrigation. At 200 g/l, the concentrated brine is higher than the required irrigation concentration of about 120 g/l chloride. The concentrated brine is an ideal makeup stream for the leaching water systems.

The process water also benefits from the addition of the ED brackish reject water, as it is much lower salinity than seawater. The salinity of the concentrator process water supply is lowered, and this reduces the washing duty for Cu/Mo concentrate (to remove chlorides) and, in turn, reduces the RO water demand.

The result is an integrated RO and ED arrangement that produces RO for concentrate washing (and all other site uses), concentrated brine from ED as a low-cost source of salt for leaching and a supply of brackish water that helpfully reduces the salinity of the concentrator process water.

17.4.10.7 Potable Water

RO water is chlorinated, and UV sterilised to produce potable water for the Project. It is supplied to the buildings and support facilities as drinking and domestic water and supplies emergency safety showers and eye wash stations in the concentrator, oxide plant and in the mining contractor’s area. Potable water storage tanks are provided at key Project locations with associated potable water distribution pumps.

17.4.10.8 Air Services

Compressors and blowers will provide air to the systems within the Process Plant:

- Plant air
- Instrument air
- Flotation air.

All compressed air passes through dryers before been fed into the main receiver. All major areas have a local receiver fed from the main receiver. Instrument air will be fed from a dedicated receiver which is also fed off the main receiver but has further cleaning, such as oil removal.

Two separate sets of blowers will be used to supply flotation air. The primary sulphide float will be fed from three blowers set up in a lead lag standby arrangement with the pressure setpoints for the lag unit adjusted during commissioning. The molybdenum float will be fed by a duty standby blower setup. A nitrogen supply is also provided for molybdenum flotation.

17.5 Molybdenum Circuit

17.5.1 Introduction

Molybdenum is floated selectively from the copper/molybdenum concentrate leaving the final copper concentrate essentially molybdenum free. A dry saleable molybdenum concentrate is produced.

17.5.2 Molybdenum Circuit Design Criteria and Process Flow Diagrams

The molybdenum recovery circuit is designed to process all sulphide (Cu/Mo) concentrate as depicted in Figure 17.6. The principal criteria dictating the molybdenum plant design are:

- Fresh production feed head grade (design) 200 ppm Mo (typical 160 ppm).
- Molybdenum plant feed grade (Cu/Mo concentrate) is nominally 0.65% Mo and 25% Cu
- Molybdenum recovery from plant feed to Cu/Mo concentrate is 81% with typical recoveries ranging from 25% to 95%
- Molybdenum recovery from Cu/Mo concentrate to final Mo concentrate is 88% and this results in a 71% overall average Mo recovery
- Final molybdenum concentrate grade is 50% Mo minimum.

17.5.3 Molybdenum Recovery Circuit Plant Description

17.5.3.1 Cu/Mo Concentrate Wash

The high saline process water used in Cu/Mo flotation (>10,000 mg/L chloride) is removed by washing in a two stage CCD as described previously. The washing serves two purposes, one is to ensure that the Cu concentrate meets customer specification and has <500 ppm Cl^- after filtration. The other purpose is to ensure that the molybdenum flotation circuit operates as designed. The molybdenum rougher flotation slurry water must have less than ~1000 ppm chloride and the cleaners less than 500 ppm. Conductivity measurements are used to measure and control the chloride levels in water in sensitive parts of the circuit. The detrimental effect of seawater on molybdenum flotation is described in Section 13.

17.5.3.2 Molybdenum Rougher Flotation

Cu/Mo concentrate is conditioned for five minutes with reductant (sodium hydrosulphide, NaSH), which lowers the oxidation potential to less than -400 mV and depresses copper flotation. The conditioning tank overflow discharges to the molybdenum rougher flotation circuit. Kerosene (collector) and frother (only if necessary) are added to the molybdenum rougher feed. Nitrogen is available to replace flotation air if it is necessary to minimise reductant consumption.

The molybdenum rougher flotation circuit consists of conventional flotation cells operating in series. The rougher concentrate is collected in a hopper for pumping to the head of the molybdenum cleaning circuit. The tails from the molybdenum rougher flotation circuit flows, by gravity, to the molybdenum scavengers.

While dilution up to this point in the circuit has used low saline process water (supplemented by fresh RO water as necessary), all water in the remainder of the molybdenum circuit is fresh RO water. The low saline and RO water is monitored for conductivity to ensure it is always suitable for molybdenum flotation.

17.5.3.3 Molybdenum Scavenger Flotation

The molybdenum scavenger flotation circuit consists of conventional cells operating in series. The scavenger concentrate is pumped to the head of the molybdenum rougher flotation circuit. The scavenger tails are the final copper concentrate for the plant and are pumped to the copper concentrate thickener ahead of filtration.

17.5.3.4 Molybdenum Cleaner Flotation

Concentrates from the molybdenum rougher circuit are upgraded in a four-stage cleaning circuit as detailed below:

- Molybdenum rougher concentrate is pumped to the molybdenum rougher concentrate thickener. The thickener underflow is delivered to a concentrate surge tank that can hold up to eight hours of rougher concentrate. The surge tank volume is utilised to stabilise the feed to the molybdenum cleaning circuit in the face of changing head grades and yields for both molybdenum and copper.
- The high-density conditioner product is diluted with RO water and then flows to the first molybdenum cleaners. Stage 1a of the molybdenum cleaning flotation circuit consists of two conventional cells operating in series. Stage 1b consists of one flotation cell and acts as a cleaner scavenger with concentrate from 1b being pumped back to the surge tank.
- Concentrates from the cleaner 1 circuit are sent to 3 stages (stages 2 to 4) of cleaner columns with the fourth column generating final concentrate grade. Concentrate from each column is pumped to the next with the tails being pumped to the previous stage. Concentrate from cleaner stage 4 (column 3) is pumped to the molybdenum concentrate thickener.

17.5.3.5 Molybdenum Concentrate Thickening

The molybdenum concentrate thickener produces 65% w/w solids in the underflow. The overflow water will report to the low saline process water pond. The molybdenum concentrate thickener underflow is pumped to a molybdenum concentrate storage tank ahead of filtration.

17.5.3.6 Molybdenum Filtration and Drying Circuits

The molybdenum concentrate storage tank is designed to hold 24 hours of production capacity. The thickened slurry is pumped from the storage tank to a molybdenum plate pressure filter for further dewatering. The pressure filter produces a filter cake at a nominal 8% moisture.

The filter cakes discharge from the pressure filter and drops under gravity into a hopper. The hopper feeds the molybdenum concentrate dryer to further reduce its moisture to 4.5%, a level chosen to minimise dusting. The dried molybdenum concentrate is loaded into 1.9 tonne bulk bags ahead of transport to the customer.

17.5.3.7 Molybdenum Concentrate Handling and Transport

The bags containing dried molybdenum concentrate are loaded onto trucks for transport by road to a smelter in Santiago.

17.5.4 Molybdenum Plant Reagent Handling and Storage

The reagents that are used in the molybdenum flotation plant include:

- Collector (kerosene)
- Reductant and Cu depressant, sodium hydrosulphide (NaSH)
- Flocculant (same as used in the Cu circuit)
- Nitrogen gas.

Reagent mixing is done in designated areas within the process plant. The design of these areas considers requirements such as section bunding coupled with dedicated sump pumps for individual reagents, segregated ventilation, distribution within the required plant area(s) and dust and fume control around reagents with potential for dust or fume release. The layout and general arrangement of the reagent area accounts for the separation between incompatible reagent types.

17.5.5 Molybdenum Recovery Circuit Plant Services

17.5.5.1 Low Saline Process Water

Low saline water is used in the rougher and scavenger stages of the molybdenum flotation circuit. The low saline water is stored in a low saline water tank, separate from the high saline process water.

The low saline process water tank is supplemented with freshly desalinated water from the RO plant when required to provide additional volume or when it is necessary to lower its conductivity.

RO water is used in the cleaner and recleaner plant sections and for concentrate dilution prior to thickening.

17.5.5.2 Compressed Air

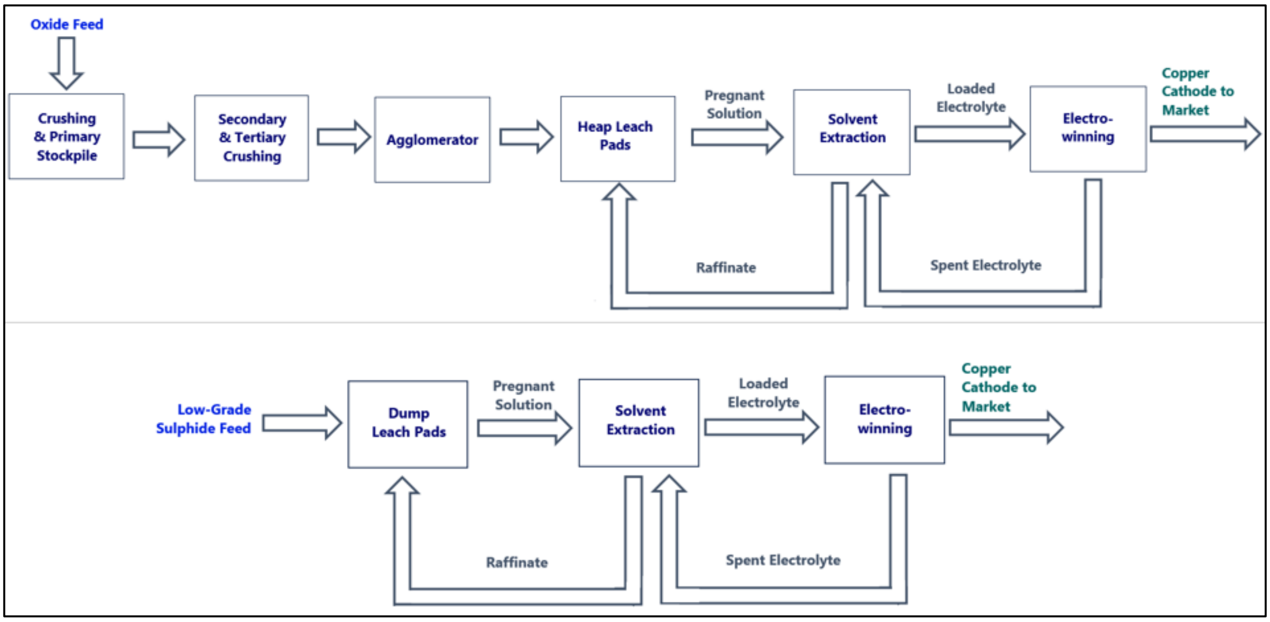
Refer to 17.4.8.8 for details.

17.6 Leaching Plant – Copper Oxide and Sulphide

17.6.1 General

The leaching plant encompasses a complex of a heap leach facility for oxidised ore and a dump leach facility for low-grade sulphide ore, both located at Productora. The copper solutions obtained from these plants will feed a Solvent Extraction and Electrowinning (SX-EW) plant to produce 12,000 t/yr of copper cathodes.

Figure 17.8: Leaching Process Flow Diagram



Top: Heap leaching. Bottom: Dump leaching. Both stand-alone operations (common SX-EW facilities). Source: PMC, 2025

17.6.2 Heap Leaching Process Description

17.6.2.1 Oxide Crushing Plant

Oxide ore from the Productora and Alice open pits is delivered to the ROM pad, or to the oxide stockpile, in 100 t capacity rear dump haul trucks. Ore from the oxide stockpile is reclaimed by loader and trucked to the ROM pad. The target maximum particle size of ore on the ROM pad will be 1000 mm. ROM pad ore is stored in separate fingers of varying types and grades to facilitate blending of the feed into the crushing plant. The ores from the fingers are fed to the ROM bin using a front-end loader (FEL) according to a blend determined by process operations. The ROM bin tip point is not sized to accept dumping trucks. Dust suppression fogging sprays are located around the ROM bin and activate during ore dumping.

Dumping at the ROM bin is controlled by dump/no dump traffic signals mounted at the ROM pad adjacent to the ROM bin. The lights are primarily controlled by a level sensor mounted at the front of the bin and are also controlled by the stockpile level indication. The ROM bin has a sloped parallel bar static grizzly with 900 mm spacing at the top to capture oversize ahead of the crusher. Oversize is removed by loader and put aside. Ore that passes through the static grizzly is fed to the crusher by a vibrating grizzly feeder. The vibrating grizzly oversize reports to the primary jaw crusher. The vibrating grizzly undersize is combined with the primary crushed oversize on the crushed ore stockpile feed conveyor. A fixed magnet will remove any metal from the primary crushed ore stream.

17.6.2.2 Screening Plant

The crushed ore is reclaimed from under the stockpile by three vibrating feeders loading onto the sizing screen bin feed conveyor. Reclaim rate is monitored and controlled by a weightometer. A fourth vibrating feeder is installed at the outer perimeter of the stockpile to be used as emergency feeder when stockpile level is low. The two parallel double deck sizing screens are each fed from the bin by vibrating feeders.

- The sizing screen upper decks have a nominal panel aperture of 50 mm. Lower decks have a nominal panel aperture of 15 mm. The sizing screens will produce three products: +50 mm, -50+15 mm and –15 mm.
- The coarse product from the top deck (+50 mm) is transported by the sizing screen oversize conveyor to the secondary crusher feed bin.
- The intermediate size material (-50+15.0 mm) is transported by the sizing screen intermediate conveyor to the tertiary crusher feed bin.
- The undersize material from the lower deck (-15.0 mm) is the final product and is transferred to an agglomeration feed bin via an agglomeration feed bin conveyor. An automatic sampler is installed on the agglomeration feed bin conveyor for heap leach metallurgical accounting.

17.6.2.3 Secondary and Tertiary Crushing

The sizing screen top deck oversize is secondary crusher feed. The secondary cone crusher operates at a closed side setting of 30 mm generating nominally -60 mm crushed product. The secondary crushed material is recycled by conveyor to the sizing screen feed.

The intermediate sizing screen product (-50 mm +15 mm) is tertiary crusher feed. The tertiary cone crusher operates at closed side settings of 14 to 16 mm. The tertiary crushed material is recycled by conveyor to the sizing screen feed.

17.6.2.4 Pre-conditioning, Agglomeration and Stacking

The overflow chute drops the ore in a controlled manner onto the stockpile feed conveyor. A hopper and feeder belt are provided to allow reclaim of the stockpiled heap leach feed to agglomeration.

A vibrating pan feeder transfers the heap leach feed from the feed bin to the agglomerator feed conveyor. The drum agglomerator will be fitted with a VSD to regulate the residence time in the agglomerator. Sulphuric acid and concentrated brine will be added to pre-condition the heap leach feed prior to stacking.

An overland conveyor will transfer the agglomerated heap leach feed to the head of a series of grasshopper conveyors, which feed a mobile radial stacker conveyor for constructing the heap pads.

17.6.2.5 Deferred Sulphide Ore to Oxide Crushing Circuit

Once the crushing of oxide ore for Heap Leach is complete, deferred sulphide ROM ore from Productora and Cortadera will be fed from various stockpiles to the oxide crushing circuit via the ROM pad at a nominal rate of 4 Mtpa. The crushing circuit will require some upgrading to accommodate the sulphide ore at the required rate and conveyors are required to deliver the crushed ore to the concentrator crushed ore stockpile.

17.6.2.6 Heap Leaching

According to the current production schedule presented in the PFS, three processing stages are observed:

- Stage 1: from year 3 to 10, Heap Leaching stand alone.
- Stage 2: from year 12 to 17, Dump Leaching stand-alone.
- Stage 3: from year 19 to 21, Heap Leaching stand-alone.

In Stages 1 and 3 of heap leaching, the production of the cathode achieves 12 kt/y, while in stage 2 of dump leaching, the production is 2.3 kt/y

The Productora and Cortadera oxide and transition ore, crushed and agglomerated with sulfuric acid and salt equivalent in concentrated brine, is deposited in the 6 m high heap leach pads and left to rest.

After the agglomeration and resting stage, intermittent irrigation and resting cycles occur. Cycles of one week of irrigation followed by three weeks of resting continue for the entire leaching timeframe, which spans 240 days.

Irrigation of the piles is counter current, which means watering the new (freshly loaded) piles with intermediate leach solution (ILS) from the more depleted piles. This way, the resulting pregnant leach solution (PLS) reaches 2.0 g/L Cu and 120 g/L Cl, suitable for feeding the SX-EW plant.

Once the new piles begin to decrease their Cu concentration, a second cycle of irrigation is initiated with refining solution (Raffinate), which is the spent solution from the SX plant which is recycled to the heaps.

As the leaching cycle is four weeks, the heaps are organized in four sectors so that there is a continuous flow of irrigation and continuous production of PLS to feed SX-EW.
The water make-up is due to the water consumption in evaporation, and residual moisture is supplied by concentrated brine that is fed in the agglomeration stages and in the raffinate pond.

Four process ponds and one emergency pond are required for solution handling:

- A dedicated seawater pond for leaching
- Raffinate pond
- ILS pond
- PLS pond
- Emergency pond.

17.6.2.7 Solvent Extraction (SX) and Electrowinning (EW)

17.6.2.7.1 Solvent Extraction

The SX plant was designed to process 750 m^3/h of PLS with seawater containing 20 g/L of chloride. Therefore, a washing stage with demineralised water was included in the earlier PEA study. For this PFS study a second stage of washing is included due to the application of hypersaline leaching solution containing 120 g/L of chloride.

The configuration of the SX plant has the following stages:

E1 - E2 - W1 - W2 – S

Where the stages are defined as follows:

E1-E2: Extraction stages
W1-W2: Washing stages
S: Stripping stage.

The purpose of the SX plant is to purify the PLS solution coming from the heap leach pads. This solution contains, in addition to Cu^{2+}, other ions such as Cl^{-}, Fe^{3+}, Mg^{2+}, SO_4^{2-}, etc. Using solvents, which are reagents with the ability to selectively capture Cu and discard other ions, it is then possible to transfer the copper to an electrolyte solution with a high Cu concentration of 50 g/L.

The solvents comprise the organic phase, which is sent to two mixer-settlers (E1 and E2) along with PLS, where the transfer of Cu from the PLS (aqueous phase) to the organic phase takes place.

The separation of the aqueous-organic phases in the settlers is not perfect, such that the organic phase is contaminated by the entrainment of 500 ppm of the aqueous phase, which has a high chloride concentration (120 g/L). Thus, two washing stages (W1 and W2) with demineralised water will be executed to remove the salt from the organic phase.

The next unit stage is stripping (S1), where the Cu from the clean organic phase is transferred to the depleted electrolyte from the EW stage. This electrolyte still contains 35 g/L Cu, which will then generate a rich electrolyte with a Cu content of 50 g/L Cu once the Cu from the organic phase is added.

At this stage there is also entrainment of 100 ppm of the organic phase to the rich electrolyte as well as a cooling of the electrolyte. Consequently, the rich electrolyte is processed in the Tank Farm before is sent to EW. In the Tank Farm the rich electrolyte is cleaned by means of various settlers and filters, and the temperature is increased to 40 C°, by means of a heat exchanger, to feed the EW stage.

17.6.2.7.2 EW (Electro-Winning)

The electro-winning plant is designed to produce 12 kt/year of copper cathodes.

The EW stage starts in the recirculation tank that receives the rich electrolyte. It then moves on to the first LME Grade A quality cathode production stage, which is obtained in the commercial cells. Two electrolyte conditioning reagents are added, one being guar, which is a flocculant to decant suspended solids, and cobalt sulphate, which acts as a catalyst to smooth the cathode surface. Titanium cathodes will be used due to the chloride content, measuring either 1.0x1.0 m or 1.0x1.2 m.

The semi-depleted electrolyte then passes to the polishing cells to continue with the cathode production stage, while the depleted electrolyte is recycled to the SX stage, leaving a fraction of the flow to be recycled to the rich electrolyte tank.

At the end of the deposition cycle, which lasts approximately seven days, the cathodes are harvested and sent to the automatic stripping machine, washed with demineralised water and packaged for export.

17.6.3 Heap Leaching Design Basis

The main design basis for the Heap Leaching is shown in Table 17.8.

Table 17.8: Heap Leaching Design Basis

Item	Unit	Quantity
Annual Heap Leaching Feed	t/y	4,000,000
Design Factor	1/1	1.15
Cu Head Grade (Design)	%	0.40
Cu Recovery (Design)	%	75.0
Heap Leaching Cycle	d	240
Irrigation percentage	%	0.25
Rest time percentage	%	0.75
Acid Consumption	kg/t	20 (23)[1]
Salt Consumption (as concentrated brine 200 g/l NaCl)	kg/t	15
Heap Leaching Lift Height	m	6
Heap Leaching Utilisation	%	94
Cu Cathode Production	t/y	12,000

[1]Weighted average from the Geometallurgical model (OPEX)

17.6.4 Dump Leaching Process Description

According to the Mine Plan defined for the PFS stage, there will be three separate operating periods for heap and dump leaching. This means that both plants will operate in stand-alone mode. (See 17.6.2.6)

Since the heap facilities will be available during the dump operation period and both installations are close, pumps designed for the heap, which are in the same irrigation flow range, will be used, considering the recirculation of raffinate and ILS. In the case of ponds, since the raffinate flow decreases by half (375 m^3/h dump versus 750 m^3/h dump), the residence time of the ponds increases by two times.

Fresh low grade ROM ore from Productora and Alice open pits (sub concentrator feed cutoff grade) is sent to dump leaching, either direct from the open pits or from a dump leach ROM stockpile. Once the truckload is dumped the ore is then irrigated with sulphuric acid and high chloride concentrated brine.

The initial stage of the leach is the conditioning or curing of the fresh ore with salt as concentrated brine and sulfuric acid. The ore is then left to rest. Unlike the Heap, which will mainly process oxides and transition mineral, the dump will mainly process low-grade primary sulfide mineral, for which the oxidation process during rest is more energetic using air injected by blowers.

During the resting period, the following phenomena occur:

1. Activation
   - Dry auto-catalytic transformation is achieved through air blow and temperature increase.
   - Washing and re-humidification.
   - Two physicochemical phenomena involved: haloclasty and criptoflorescence.
2. Breakdown and physical destruction of the passivating layer by handling chloride solutions.
3. Generation of access routes so that the chloride process solutions can enter the interior of the thickest ore particles.

The dump leaching cycle is two years and two dump leach facilities should operate simultaneously every year.

Considering a 4-week cycle of irrigation and rest, and the 2-year leaching period definition, the annual processing of low-grade ore has been divided into two sectors or modules of 5 Mt each implemented biannually so that four modules per year are available for processing. This allows for the implementation of irrigation every four weeks.

The aeration stage will be provided by 20 blowers with 20,000 Nm^3/h capacity each through HDP pipes installed across the dump leaching pad.

As a design criterion, the PLS feed flow to the SX-EW plant of 750 m^3/h (previously defined) is decrease to the half. Consequently, the raffinate discharge flow from the SX-EW plant will be 375 m^3/h.

17.6.5 Dump Leaching Design Basis

The main design basis for the Dump Leaching is shown in Table 17.9.

Table 17.9: Dump Leaching Design Basis

Item	Unit	Quantity
Annual Dump Leaching Feed	t/yr	3,600,000
Design Factor	1/1	1.15
Cu Head Grade (Design)	%	0.16
Cu Recovery (Design)	%	40.0
Dump Leaching Cycle	d	730
Irrigation percentage	%	0.25
Rest time percentage	%	0.75
Dump Leaching Lift Height	m	20
Acid Consumption	kg/t	30 (design) 15-25 (consumption) (9)[1]
Salt Consumption (as concentrated brine 200 g/l NaCl)	kg/t	15
Dump Leaching Utilisation (trucks operation)	%	70

[1]Weighted average from the Geometallurgical model (OPEX)

17.6.6 Heap and Dump Leaching Reagent Handling and Storage

The Costa Fuego Project hydrometallurgical plant uses sulphuric acid and concentrated brine produced by an Electrodialysis Plant (200 g/l NaCl) as primary reagents in the heap and dump leaching area of Productora. The following products are used as auxiliary reagents in the SX-EW area: guar, activated clay, cobalt sulphate, and ferrous sulphate.

17.6.6.1 Heap and Dump leaching reagents

a) Sulfuric acid

The Main Acid Storage Tank, with a capacity of 1,000 m^3, will be installed at the Productora site. This tank is designed to hold a seven-day supply of acid for the Heap Leaching and Productora Dump operations. These storage volumes consider an acid dosage of 20 kg/t for 4.0 Mt/year heap leaching ore and 25 kg/t for 3.6 Mt/year for dump leaching ore. An industrial grade acid with a concentration of 98% and a density of 1.83 tonnes/m^3 is used. A design factor of 1.15 has been applied.

b) Salt

Typical installations for the hypersaline process use bulk industrial salt (NaCl) stored in a stockpile equipped with a dosing system. In the Costa Fuego Project process, given the availability of brine from the RO Plant, an electrodialysis plant will be installed to produce concentrated brine with a high chloride concentration (200 g/l NaCl), which will be the source of the salt supply. From the concentrated brine pond, a flow of 70 m^3/h will be sent to heaps and dumps, which is equivalent to a dosage of 15 kg/t of salt.

17.6.6.2 SX-EW reagents

a) Guar plant

The Guar plant is sized to meet the requirement of the strong electrolyte solution, using a dosage of 250 g/t of cathode. The nominal daily total is 9.8 tonnes at a concentration of 0.5 % w/w. A 30-hour storage tank is required.

b) Cobalt sulphate plant

The cobalt sulphate plant is sized to meet the requirement of the strong electrolyte in the solution, with a dosage of 150 ppm, considering recirculation and electrolyte bleed losses. Total $CoSO_4$ consumption is 38.2 kg/day, with a concentration of 21% w/w. One bag of 25 kg is used to prepare 1.4 m^3 of cobalt sulphate solution for storage.

c) Ferrous sulphate plant

The ferrous sulphate plant is sized to meet the requirement of the strong electrolyte in the solution and to maintain a concentration of 303 ppm. The total consumption is 81 kg/day, and the 72-hour storage tank capacity is 1.4 m^3. A maxi bag dosing system feeds solids into the replenishment tank.

d) Activated clay

Bentonite is used as an activated clay for the organic regeneration stage. Regular consumption is 2.0 t/month, and the reagent is supplied in 1.0 t bulk bags. Activated clay is stored in a facility with a capacity of 2.0 tons, ensuring a sufficient supply of reagents for one month.

These reagents consumptions based on flow streams have been determined in previous studies: a PLS feed of 750 m^3/h and a strong electrolyte of 116 m^3/h.

17.6.7 Heap and Dump Leaching Plant Utilities

The principal utilities and services necessary for the Productora heap and dump leaching process are electric power, seawater, treated water and blown air.

a) Electric Energy (EE)

The EE is used in the crushing plant and the pumping systems for heap leaching and dump leaching solution irrigation. A major consumer of EE in the SX-EW plant is the production of copper cathodes in the electrolytic cells.

The Productora main substation will supply EE to the leaching area. The power required for leaching is 10 MW.

b) Seawater

The main seawater pond that supplies the concentrator also feeds the leaching seawater pond, which has 36 hours of storage.

c) RO Treated water (Reverse Osmosis Plant)

In the two stages of chloride washing from the organic phase, the demineralised water used in the SX-EW plant is obtained from the Productora RO Plant. Also, the RO plant supplies drinking water for operations personnel.

The Productora RO plant operates at 48% efficiency, with the remaining 52% of the input being brine, pumped to the Electrodialysis Plant.

d) Blown air

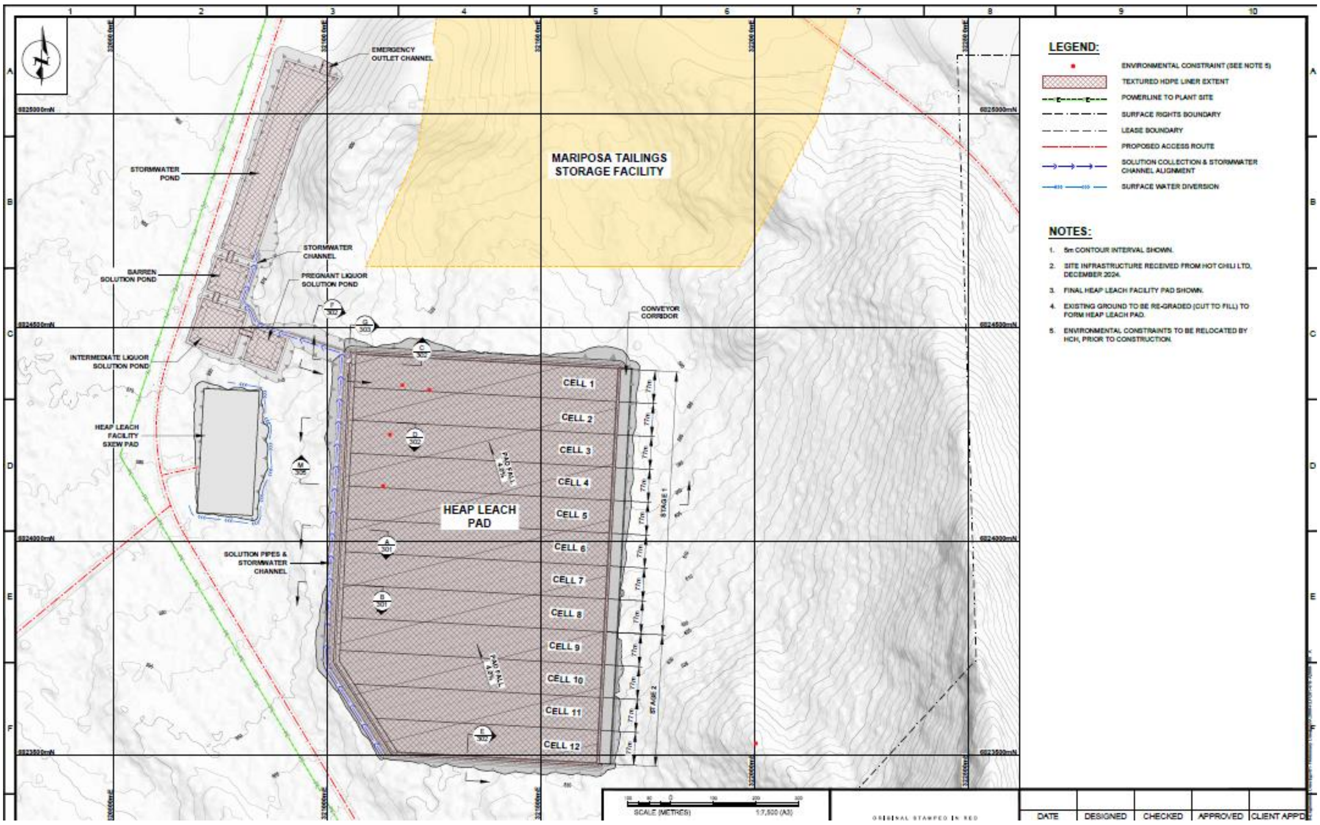
Blowing air is used during the resting stage of dump leaching to promote oxidation and increase the temperature within the ore. This plant utilises high-flow, low-pressure blowers with a capacity of 20,000 m^3/h each. Blown air is supplied with an approximate flow rate of 375,000 m^3/h for dump leaching using 20 blowers.

17.6.8 Heap Leach Construction Sequence

17.6.8.1 Pad Construction Sequence

A phased development of the heap leach pad (HLP) was developed to defer capital expenditure. Based on the current stacking requirements, a two-phase pad construction is proposed. The initial development phase is the construction of 8 leach cells for ore placement in Year 1, followed by an additional four leach cells for ore placement in Year 2, as shown in Figure 17.9.

Figure 17.9: Heap Leach Facility General Arrangement



The Heap Leach Facility will comprise the following:

- HLP including perimeter/internal berms and bunds (including HDPE liner)
- Solution Collection System
- Process Ponds (Pregnant Liquor Solution Pond, Intermediate Liquor Solution Pond, Raffinate Liquor Solution Pond and Stormwater Pond)
- Leak Collection and Recovery System (LCRS).

Construction of the Phase 1 HLP and ponds can be divided into several distinct activities, and it is envisaged that construction will be carried out broadly in this order.

Activity 1 – Site Preparation

The site preparation will be carried out as follows:

- Clear vegetation and strip topsoil within the disturbed footprint.
- Construct surface water run-off diversion channels and bunds.
- Construct temporary access roads.

Activity 2 – Pad Earthworks

Following the completion of site preparation, the following earthworks will be carried out:

- Cut to fill the pad surface to provide overall falls in accordance with the design intent; ensure slopes are not steeper than 1V:3H for liner placement; and construct cell divider berms.
- Cut to fill from the pad and pond area and construct the perimeter embankments and berms.
- Scarify and compact the subgrade to form the leach pad subbase layer, importing low-permeability fill from the pond excavations (if required) to complete the layer.
- Excavate to form a solution collection trench.
- Import low-permeability fill to form the subbase for the solution channels and operating ponds (if required).
- Scarify, rock rake, moisture condition and compact suitable in-situ material or tailings (if suitable) as a sub-base for HDPE liner installation.

Activity 3 – Pond Earthworks

Following the completion of site preparation, the following earthworks will be carried out:

- Excavate the ponds with material arising hauled to pad construction or stockpile as appropriate.
- Scarify and compact the subgrade in each pond to form the sub-base layer with import of low-permeability fill (if required) to complete the layer.
- Scarify, rock rake, moisture condition, and compact suitable in-situ material or tailings (if suitable) as a subbase for HDPE liner installation.
- Excavate to form LCRS and underdrainage sumps.

Activity 4 – Placement of Liners

A specialist sub-contractor will carry out the supply and placement of geo-synthetic liners:

- Install liners on prepared subgrade.
- Secure liners along the upper edges in anchor trenches.
- Concurrent with the lining of the ponds, install LCRS and underdrainage sumps and associated riser pipes.
- Place wearing course to crests of pond embankments.

Activity 5 – Placement of Solution Collection and Distribution System

- Supply and place secondary collection pipes, starting with connections to the launder boxes and then laying pipes upstream.
- Install primary collection pipes at 25 m intervals across each cell. The pipes will be surrounded with Zone F1 bedding.
- Load, haul and place Zone F2 drainage and protection layer across the pad (500 mm thick).

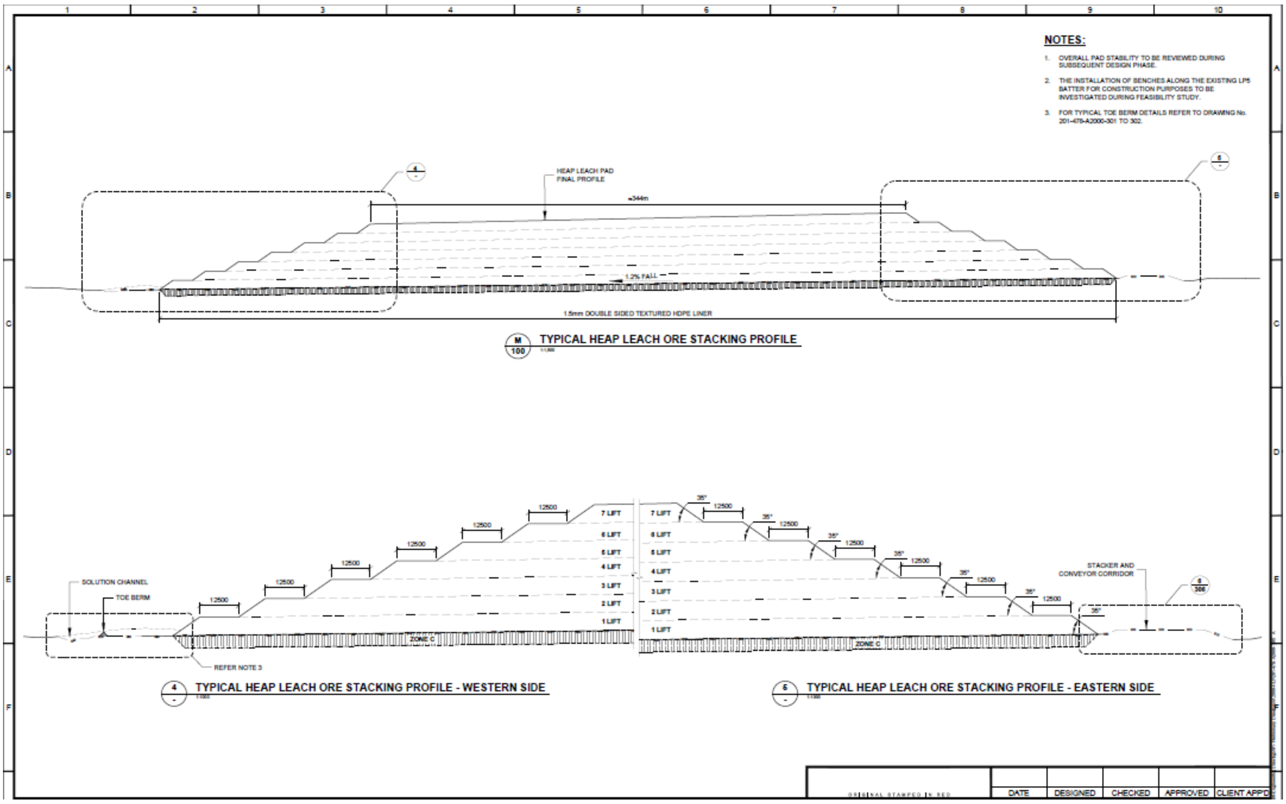
Activity 6 – Phase 2 Construction

Phase 2 construction will comprise similar activities as outlined above. However, Activity 3 will not be required as these will be constructed as part of Phase 1 construction.

17.6.8.2 Stacking Sequence

The heap construction sequence is set by the stacking rate and the layout and size of the proposed 12 heap leach cells. The overall stacking plan is based on seven vertical 6 m lifts over the facility life, as shown in Figure 17.10.

Figure 17.10: Heap Leach Facility Typical Pad Sections and Details



The cells will be stacked in sequence, starting with Cell 1. In Year 1, cells 1, 2, 3, 4, 5, 6, 7 and 8 will be stacked sequentially. In Year 2, cells 8, 9, 10, 11, 12,1, 2, 3 and 4 will be stacked sequentially.

On completion of each leach cycle, the drip irrigation will be removed, the upper surface of each lift will be reworked and compacted, and interlift drainage will be installed. An additional layer of geomembrane at each lift has been added as a provisional item in case spent ore needs to be physically isolated from subsequent lifts to maintain pH control. The irrigation system will then be reinstated for the next lift.

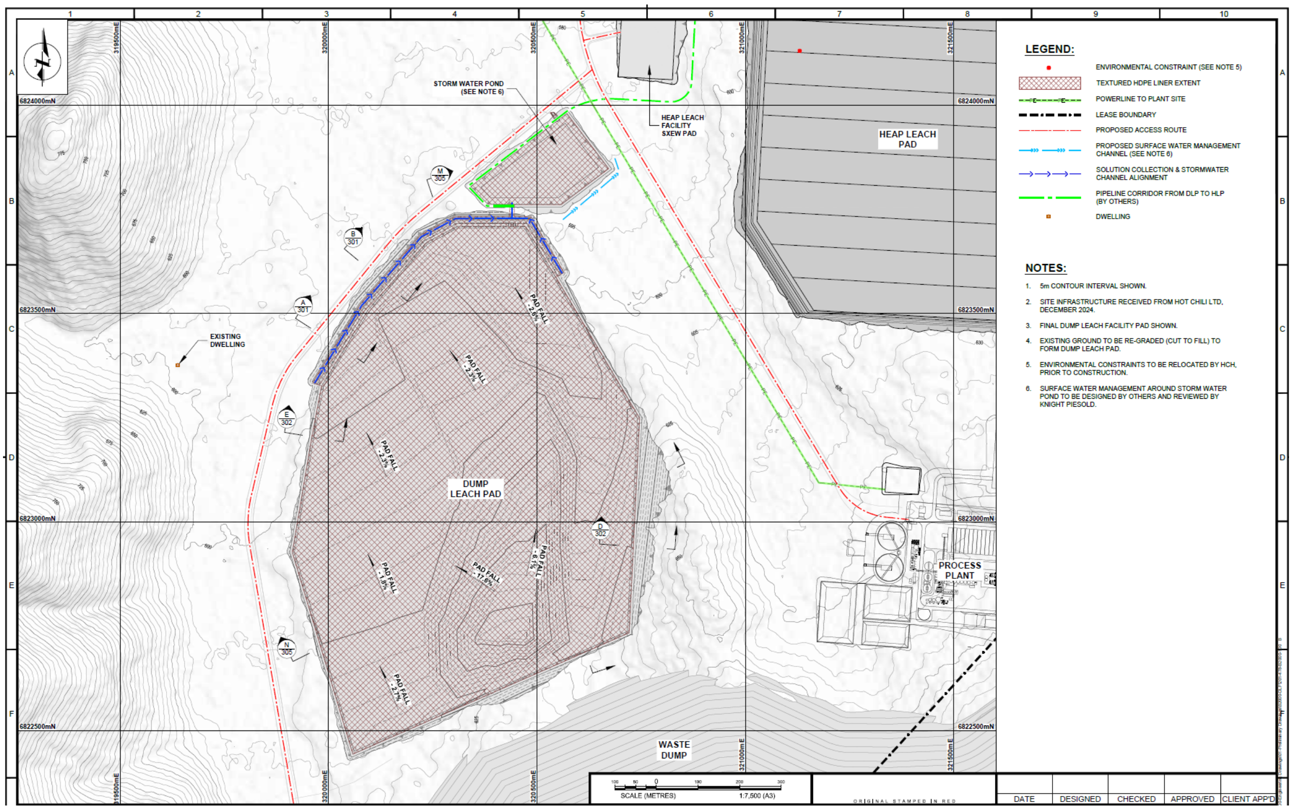
The stacking sequence will then continue similarly for the remainder of the Heap Leach LOM.

17.6.9 Dump Leach Construction Sequence

17.6.9.1 Pad Construction Sequence

A phased development of the Dump Leach Pad (DLP) was not considered at this stage. However, it may be possible to phase the construction to defer some initial capital expenditure, which will be assessed as part of the next design phase. No discrete cells have been incorporated, and the DLP will be constructed as a single cell, as shown in Figure 17.11.

Figure 17.11: Dump Leach Facility General Arrangement



The Dump Leach Facility will comprise the following:

- Dump Leach Pad, including perimeter/internal berms and bunds (including HDPE liner)
- Solution Collection System
- Process Ponds (Intermediate Liquor Solution Pond, Raffinate Liquor Solution Pond and Stormwater Pond)
- Leak Collection and Recovery System (LCRS).

Construction of the Dump Leach Pad and ponds can be divided into several distinct activities, and it is envisaged that construction will be carried out broadly in this order.

<u>Activity 1 – Site Preparation</u>

The site preparation will be carried out as follows:

- Clear vegetation and strip topsoil within the disturbed footprint.
- Construct surface water run-off diversion channels and bunds.
- Construct temporary access roads.

Activity 2 – Pad Earthworks

Following the completion of site preparation, the following earthworks will be carried out:

- Cut to fill the pad surface to provide overall falls in accordance with the design intent; ensure slopes are not steeper than 1V:3H for liner placement.
- Cut to fill from the pad and pond area and construct the perimeter embankments and berms.
- Scarify and compact the subgrade to form the leach pad subbase layer, importing low-permeability fill from the pond excavations (if required) to complete the layer.
- Excavate to form a solution collection trench.
- Import low-permeability fill to form the subbase for the solution channels and operating ponds (if required).
- Scarify, rock rake, moisture condition and compact, suitable in-situ material or tailings (if suitable) as a sub base for HDPE liner installation.

Activity 3 – Pond Earthworks

Following the completion of site preparation, the following earthworks will be carried out:

- Excavate the ponds with material arising hauled to pad construction or stockpile as appropriate.
- Scarify and compact the subgrade in each pond to form the sub-base layer with import of low permeability fill (if required) to complete the layer.
- Scarify, rock rake, moisture condition, and compact suitable in-situ material or tailings (if suitable) as a subbase for HDPE liner installation.
- Excavate to form LCRS and underdrainage sumps.

Activity 4 – Placement of Liners

A specialist sub-contractor will carry out the supply and placement of geo-synthetic liners:

- Install liners on prepared subgrade.
- Secure liners along the upper edges in anchor trenches.
- Concurrent with the lining of the ponds, install LCRS and underdrainage sumps and associated riser pipes.
- Place wearing course to crests of pond embankments.

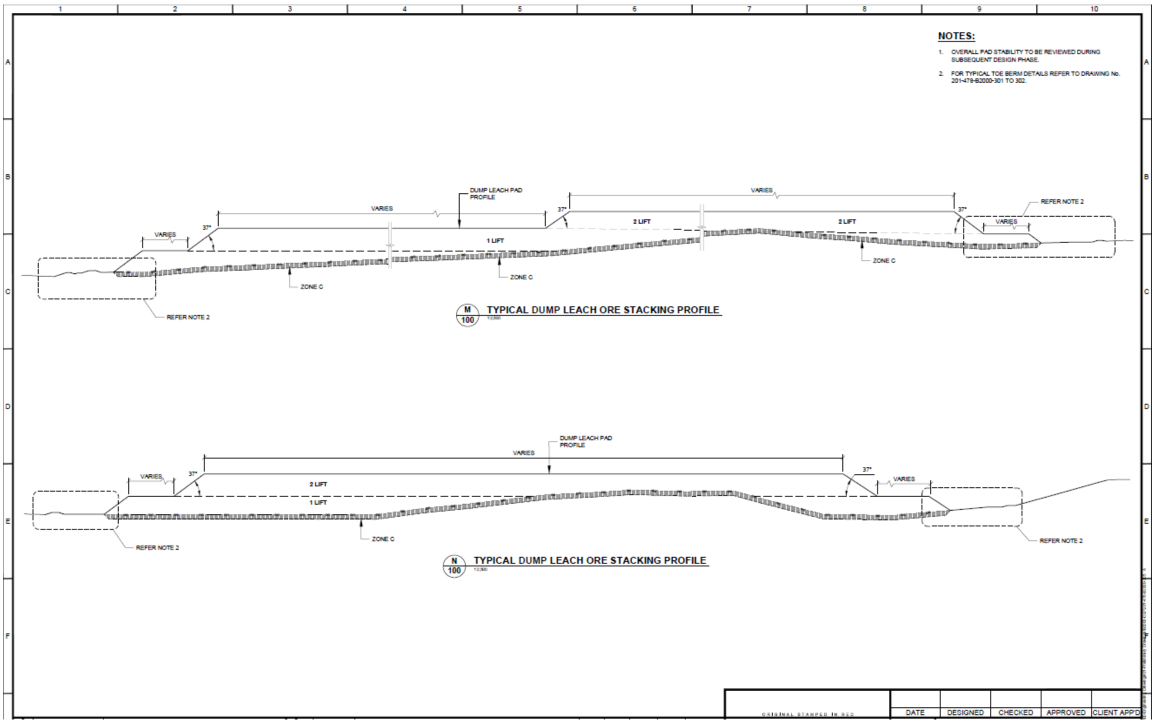
Activity 5 – Placement of Solution Collection and Distribution System

- Supply and place secondary collection pipes on the HDPE liner, starting with connections to the launder boxes and then laying pipes upstream.
- Install primary collection pipes at 25 m intervals across each cell. The pipes will be surrounded with Zone F1 bedding.
- Load, haul and place Zone F2 drainage and protection layer across the pad (500 mm thick).

17.6.9.2 Stacking Sequence

The dump construction sequence is set by the stacking rate and the layout and size of the proposed dump leach cells. The overall stacking plan is based on two vertical 20 m lifts over the facility's life, as shown in Figure 17.12.

Figure 17.12: Dump Leach Facility Typical Pad Sections and Details



The cell will be stacked from the lowest to the highest pad elevation, commencing at the northern extent of the pad and progressively moving to the South. Due to the pad gradient and the stacking requirements, the raises will be terraced as part of the lift designs to accommodate the storage requirements.

On completion of each leach cycle, the drip irrigation will be removed until the next raise is placed, after which it will be reinstated. No interlift drainage is currently proposed; however, this can be installed if required.

The stacking sequence continues similarly for the remainder of the Dump Leach LOM.

17.7 Comments on Section 17

The concentrator QP is satisfied that PFS standard requirements have been met in regard to geometallurgy, comminution, flotation, product and tailings handling and in process design. Recommendations have been made with respect to additional testwork to support an FS standard process design.

The hydrometallurgy QP notes that the work completed for this PFS has been developed based on a robust flowsheet. This flowsheet integrates the heap leaching process of high-grade oxide and transition ores from the Productora and Cortadera mines with the dump leaching process of low-grade sulphide ores from Productora, utilising seawater for both processes. The process design is grounded in pilot tests conducted on heap and dump column leaching using a hypersaline leaching technology, based on NaCl salt addition as concentrated in the agglomeration stage. Additionally, industry standards for leaching plants and Solvent Extraction and Electrowinning (SX-EW) have been applied.

The QP also highlights benefits related to improved project efficiency, as discussed earlier. Potential improvements include optimising blasting techniques to better particle size distribution for low-grade sulphide ROM recovery and exploring other elements recovery.

18 Project Infrastructure

18.1 Introduction

18.1.1 Existing Infrastructure and Services

The Project will have access to and utilise existing infrastructure and services in the Vallenar/Huasco region. The township of Vallenar (17 km from the mine site) would provide accommodation and services to support the Project. Figure 1.18 confirms the existing and planned regional infrastructure for the Project footprint.

General infrastructure around Vallenar includes the following:

- Aerodrome (3 km south of Vallenar)
- Pan American Highway (5 km east of the Productora mine site)
- Access roads from the Pan American Highway (C486 or Algarrobo route) and from Maitencillo (C472) will provide partial access to the mine site
- Main road (C-46) from Vallenar to Huasco
- A 220 kV electrical substation located at Maitencillo, connected to the Chilean electrical grid

18.1.2 Site Development

The process plant will be sited at Productora to the west of both the Alice and Productora pits. The ROM pad and primary crusher would be located adjacent to the main haul road.

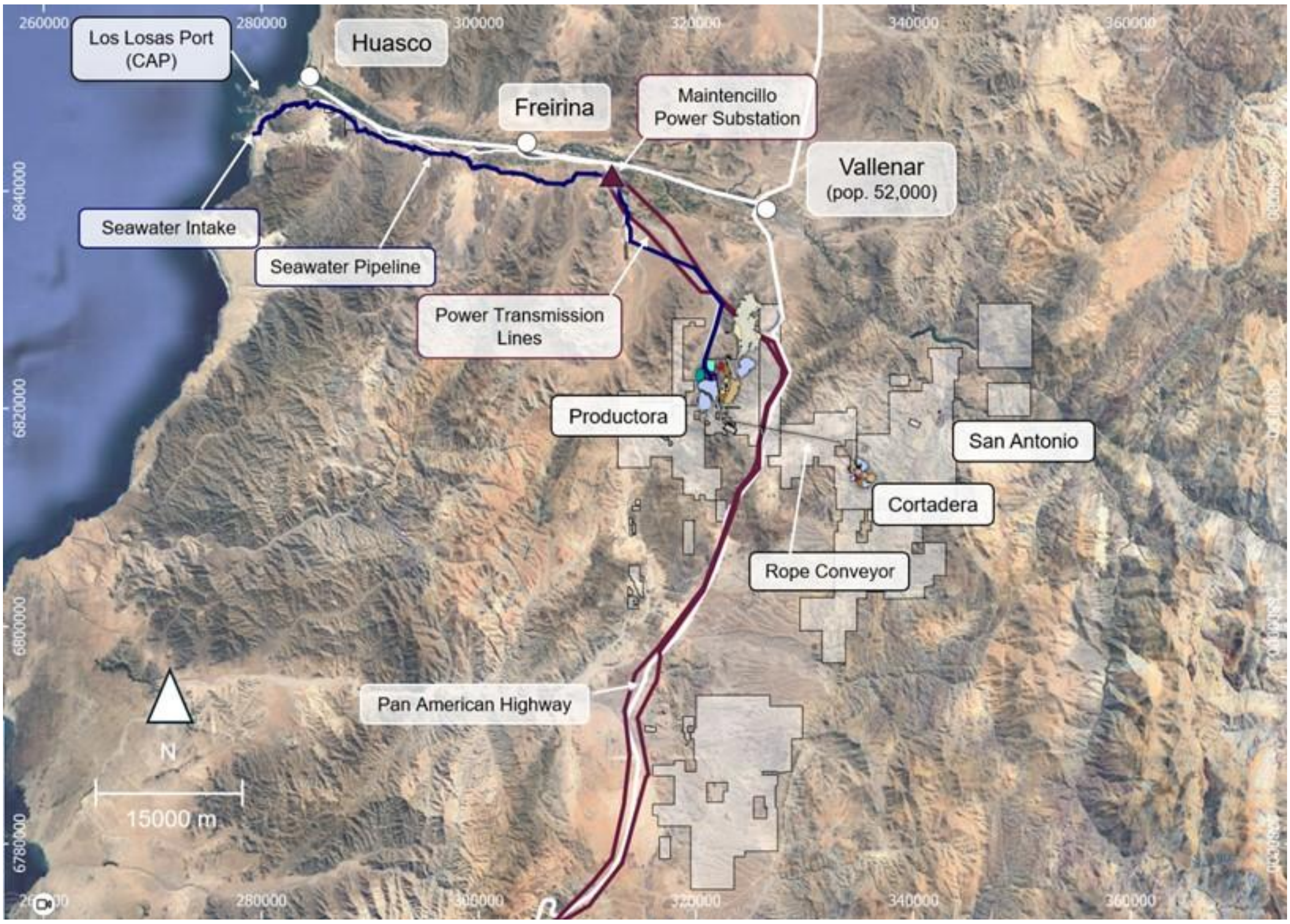
Most site buildings would be located in the area adjacent to the sulphide process plant, including the main administration building, main warehouse and change rooms. Smaller support facilities would be located at the copper oxide plant. An area for establishment of mining contractor’s facilities has been provided to the north of the Alice pit and the Productora pit.
The Tailings Storage Facility (TSF) would be located to the northeast of the sulphide process plant. The TSF embankments are bounded by naturally steep valley walls and would be constructed predominantly from open pit mine-waste.

The key infrastructure required for the Project includes the following:

- Roads (area road, plant roads, ramps and accesses, connections and signage, weighbridge). A new road from the Productora process plant joins the C472 to enable transport of concentrate and copper cathode to the nominated ports, and reagents to the plant.
- Waste management
- Tailings storage facility
- Surface water management (drainage networks, sediment control, bridges, culverts, etc.)
- Seawater transfer system
- On-site buildings
- Firefighting systems (detection and control)

- Power distribution
- Communications – telephone networks, radio systems, data networks, CCTV.

Figure 18.1 : Costa Fuego Project Planned and Existing Regional Infrastructure (HCH, 2025)



18.2 Seawater

18.2.1 General

Seawater supply, as the raw water source for the Project, is transferred from the coast (south of the port of Huasco) to seawater storage ponds located at Productora. The pipeline from coast to the process plant site is approximately 62 km in length.

The seawater transfer system comprises one intake pump station, one seawater transfer pumping station and the above-ground transfer pipeline. The design volumetric capacity of the seawater system is 500 l/s.

Power supply for the seawater intake pump station and transfer pump station will be via a 33 kV high voltage power line from the main Productora site.
The seawater transfer system will consist of the following:

- Seawater intake pipeline
- Seawater intake pit and pump station
- Seawater transfer pump stations
- Transfer pump station emergency storage pond and tank
- Seawater transfer pipeline
- Sulphide plant and oxide plant seawater storage ponds.

A reverse osmosis (RO) water plant treats seawater to provide low chloride content water, utilised in both the mining and process plant operations. RO water is further treated through a demineralisation plant, for use within the SX-EW facility of the Oxide process plant. The RO plant will also supply a treatment plant to produce potable quality water for use across the Project.

A separate water treatment facility is included in the design for the treatment of leach raffinate, TSF seepage, and mine contact water.

18.2.2 Switchrooms and Motor Control Centres (MCC)

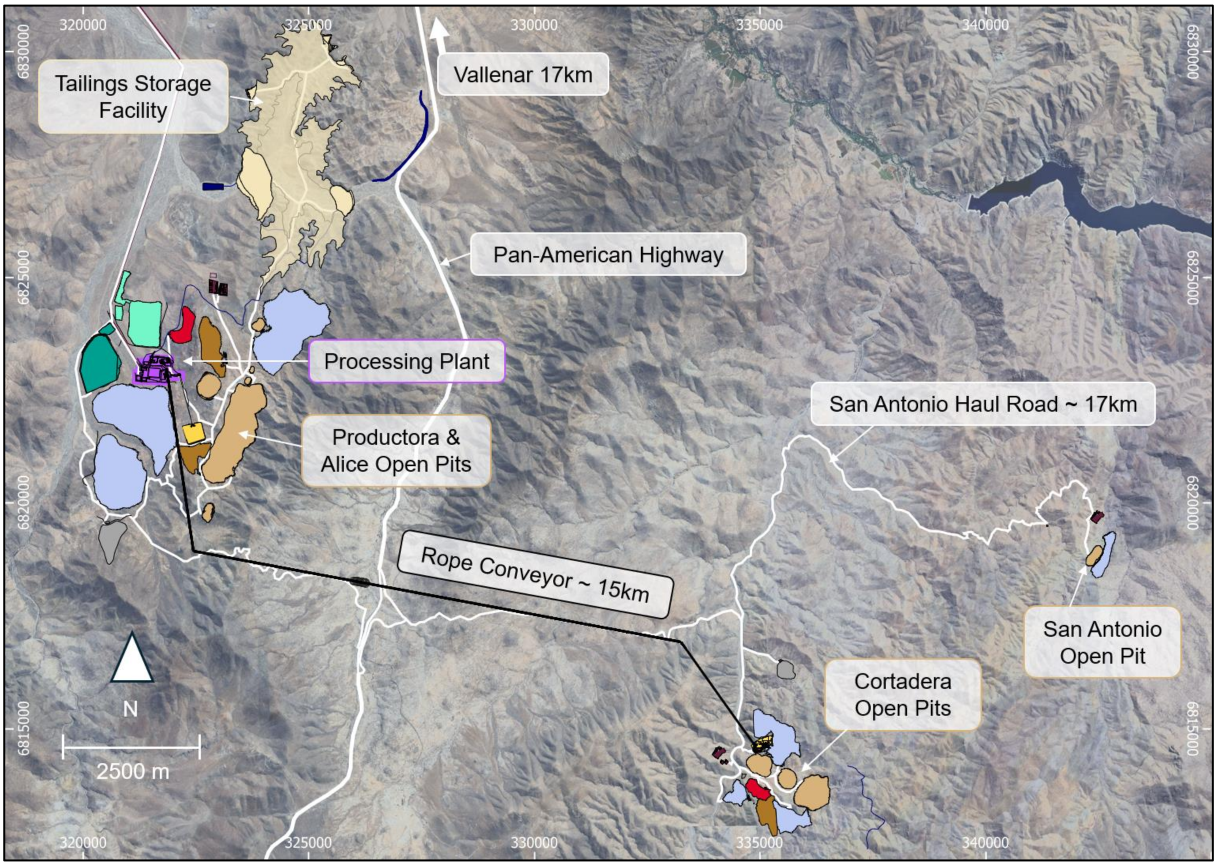
There would be low voltage (LV) electrical switch rooms installed in the following locations:

- Seawater supply
- Seawater pumping station 1 MCC
- Seawater intake pumping station MCC.

18.3 Productora Infrastructure

Figure 18.2 confirms the general layout of the Costa Fuego project to support the process plant location at Productora.

Figure 18.2 Costa Fuego Project Infrastructure (HCH, 2025)



18.3.1 Power to Productora and Electrical Design

18.3.1.1 General

Electrical supply for the Project will originate at the Maitencillo substation, feeding the main Project substation at Productora, sited close to the sulphide plant grinding and classification facilities (main Project electrical load).

The electrical substation for the seawater transfer system would be located adjacent to the seawater pumping station where the main loads are located.

The electricity transmission and distribution design approach exclude the provision of redundancy or standby equipment. However, the main substation at the sulphide processing plant has space for a redundant transformer.

18.3.1.2 Point of Supply for the Productora Substation

Power supply from Maitencillo to Productora will be via a 220kV overhead powerline. The ± 25 km route for the 220 kV overhead line is nominal and will be detailed in further stages of Project development.

The maximum demand is estimated at 149 MVA for the Project, and the Maitencillo substation represents the nearest location with sufficient capacity to provide power to the Project. The substation has numerous 220 kV power lines which connect to the country's transmission and generation network and has space for available for a non-redundant 220 kV point of supply.

A design assumption for the Project is that the Maitencillo substation would not require significant changes to provide the point of supply and that a spare gas insulated switchgear (GIS) circuit breaker will be available for connection at Maitencillo.

At the Productora substation and switchyard, the 220 kV supply is stepped down to 23 kV via an 80 MVA transformer for Cortadera and a 140 MVA transformer for Productora. The Productora substation design includes the main transformers, outdoor 220 kV SF6 switchgear and 23 kV air insulated switchgear. The 23 kV supply is connected to the 23 kV Substation via a 400 mm^2 cable; power from the 23 kV substation will be distributed to each of the load centres via switchboard and cabling.

18.3.1.3 Power Distribution

The Project would utilise the following voltages:

- 220 kV feed to the main substations at Maitencillo
- 23 kV would be used for electricity distribution as well as the power line to the tailings booster pumps, the mining infrastructure, sulphide and oxide crushing facilities, the refining facilities and transformers that supply the 380 V loads, noting that power distribution within the processing plants would be via cables with copper conductors and armouring
- 12 kV would be used for the sulphide plant SAG and ball mills
- 4.16 kV would be used for supplying motors which are greater than 200 kW and less than 2000 kW. This voltage would also be used for the transformers that supply 380 V at the crushing and reclaim areas
- 380 V for process equipment
- 220/380 V for general light and power.

The transformers would be located in close proximity to their major loads. The transformers would be oil filled and the terminals air insulated and enclosed in a termination box.

The switchgear would include circuit breakers for the 23 kV and 12 kV switchboards. The 4.16 kV switchboards would include fused contactors for motor applications. The 23 kV power lines would include pole top isolators and reclosers.

18.3.1.4 Emergency Power

The emergency diesel driven generators will deliver rated power at 23 kV and operate in parallel to meet the emergency power requirements. A load shedding scheme would be implemented when the main grid connection is offline due to a fault or maintenance. The scheme would progressively shut down non-critical areas and connect power to critical drives and systems.

18.3.1.5 Power Factor Correction

The power factor at maximum load is approximately 0.87 and the typical requirement for a 220 kV supply is 0.95 to unity. Power factor correction would be required and operate at 23 kV. To manage the step change in voltage when switching capacitors, multiple stages would be required.

18.3.2 Project Road Access and Design

The Project is located 29 km southeast of the regional mining centre of Vallenar where HCH staff and contractors would be accommodated in houses and hotels. Personnel would commute by utility vehicle or company bus to the Project for work each day.

The Project would be accessed by following the main sealed Pan-American Highway connecting Vallenar to Coquimbo in the south for 21 km. The Projects haul road connecting Productora and Cortadera intersects the Pan-American, with a further 8km to Productora or 10km to Cortadera travelled before reaching the respective sites.

Permanent roads to the mine site will have to accommodate large traffic volumes to and from Vallenar each day and heavy vehicles carrying copper concentrate to the Huasco port facility. Concentrate will be transported on a route that requires partial construction. The selected option has the cheapest net present cost, with and added community benefit of avoiding Vallenar, Freirina and Huasco, despite having the largest length of new road to construct.

The main Project access road terminates at the entrance to the sulphide process plant, with separate road access to the mining contractor's area, the oxide plant area, and the TSF.

Sulphide process plant internal roads provide access to all buildings and facilities including deliveries to the plant warehouse, fuel farm, reagent storage building and laydown area, and for concentrate transport trucks. Periphery road access to the high voltage switch yard and seawater storage pond is also provided.

The access road to the oxide plant terminates at the oxide plant back end (SX/EW plant). Internal front and back-end roads provide access to all buildings and facilities including deliveries to reagent storage and plant warehouse and for copper cathode transport trucks. Access is also provided to the crushing, agglomerator and heap leach areas

The access road to the TSF will be unsealed. A separate unsealed track will service the decant return water pump station and return water pipeline to the plant.

18.3.3 Rerouting of Existing Overhead Powerlines

A conceptual design has been proposed for rerouting of overhead power lines currently traversing a portion of planned TSF footprint. The power line re-routing will be developed with the power authorities as part of the Project development.

18.3.4 Plant Buildings

18.3.4.1 General Plant Buildings

The Sulphide and Oxide process plants will include administrative and auxiliary buildings that have been sized based on proposed operational requirements. A transportable building design has been proposed for the following:

- 3140-BD-006 Productora Safety & ERT Building
- 3140-BD-001 Productora Administration & Training Building
- 3140-BD-005 Productora Process & Maintenance Office
- 3140-BD-012 Productora Laboratory Building
- 3140-BD-013 Productora Plant Main Control Room
- 3140-BD-014 Productora Sulphide Crushing Control Room
- 3140-BD-009 Productora Canteen
- 3140-BD-004 Productora Change Room Building
- 3140-BD-002 Productora Plant Maintenance Workshop & Warehouse
- 3140-BD-003 Productora Light & Service Vehicle Workshop
- 3140-BD-017 Productora Oxide Crushing Control Room
- 3140-BD-018 Productora SX/EW Control Room
- 3140-BD-015 Productora Oxide Office + Workshop

18.3.5 Laboratory and Sample Preparation

The laboratory and sample preparation buildings would be provided by the laboratory services provider (yet to be appointed) and would be located in Vallenar. The laboratory facility would be approximately 500 m^2, including the outside fenced storage area.

The laboratory and sample preparation facility would include the following:

- Wet chemical room
- Balance room
- AAS room
- Metallurgical laboratory
- Sample preparation area

- Office and stores.

18.3.6 Switchrooms and Motor Control Centres (MCC's)

There would be low voltage (LV) electrical switch rooms installed in the following locations:

- Sulphide Plant
  - Crushing area MCC
  - Reclaim area MCC
  - Grinding and flotation
  - Concentrate storage, offices and molybdenum MCC
  - Oxide Plant
  - Crushing MCC
  - Oxide solvent extraction MCC
  - Cortadera Infrastructure
- Mining
  - Mining workshop / office MCC

18.3.7 Concentrate Storage and Loading Area

Concentrate storage and loading shall be within a storage shed building which would include a wheel wash prior to the shed entrance, weighbridge inside the shed and truck wash at the exit end of the shed.

18.3.8 Weighbridge

A weighbridge would be constructed near the entrance to the sulphide process plant. The weighbridge would be used for incoming deliveries and for outgoing product shipments.

18.3.9 Fuel Storage

The sulphide plant area fuel farm would have storage capacity based on twenty days of site storage and annual consumption for administration and maintenance vehicles.

18.3.10 Washdown Area

A wash down area and a waste oil facility would be provided for maintenance support vehicles and light vehicles.

The wash down slab would incorporate a silt and oil trap and an oil separator to remove any contaminant oil. The mining contractor would construct their own facility to manage the safe removal of waste oil (using approved contractors as required by the law).

18.3.11 Sewage

Three independent packaged sewage treatment plants (STP) would be installed to service the sulphide plant area buildings and the mining contractor's area. STPs would be sized according to personnel requirements for each area.

The STPs would produce effluent to Class A standards. The treated water can be used for dust suppression around the mine site.

The STPs would be constructed, operated and maintained in accordance with local government health regulations and as per permitting requirements.

18.3.12 Tailings Deposition

Process tailings are planned to be deposited in the Tailings Storage Facility (TSF) via pipeline. Details of the TSF are contained in Section 18.4.

18.4 Tailing Storage Facilities

18.4.1 Introduction

The Tailings Storage Facility (TSF) is located approximately 5.4 km northeast of the Project plant site (centroid to centroid) at Productora (Figure 18.3). The main embankment traverses the area of lowest elevation within the footprint of the TSF at the west of the facility. The topography of the proposed TSF area is steep and hilly and the perimeter is characterised by steep and undulating ridges. The basin is in a valley that is bounded by ridges to the north and south. A second embankment will be constructed on the eastern side of the valley, and four saddle embankments will be constructed to the North (3 No.) and West (1 No.) of the facility. The valley falls in the westerly direction towards the main embankment, with the minimum ground surface levels at the proposed main embankment approximately RL845 m above mean sea level. A general arrangement of the final TSF is provided below in Figure 18.4 and Figure 18.5.

In Pit tailings deposition will occur within the Main Productora Open Pit following the completion of mining and the main TSF has reached capacity. A general arrangement of the Productora Open Pit at the end of deposition is provided below in Figure 18.6.

Figure 18.3 : Location of Proposed TSF (HCH, 2025)

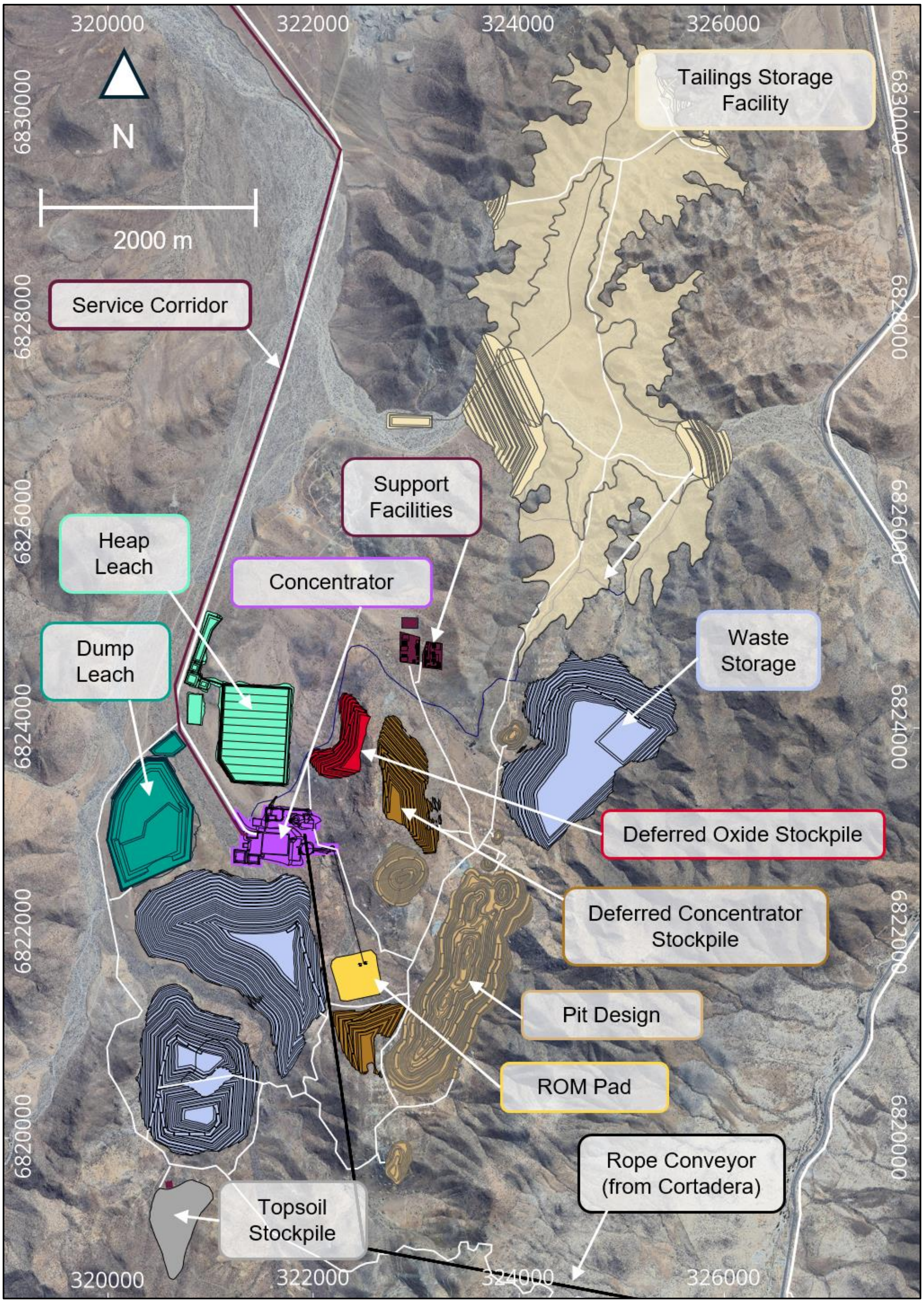
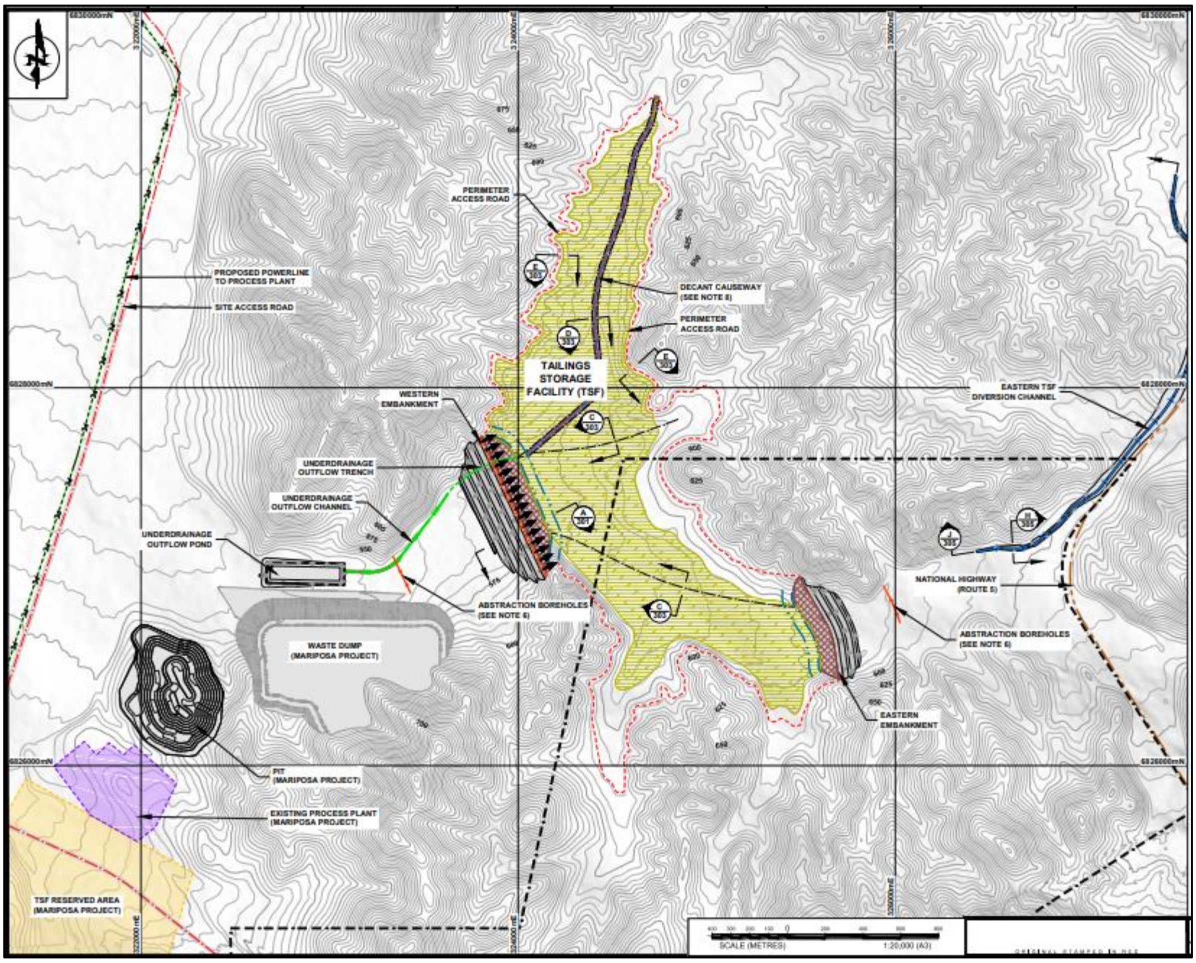


Figure 18.4: General Arrangement of Stage 1 TSF (31.1 Mt) (Knight Piesold, 2025)



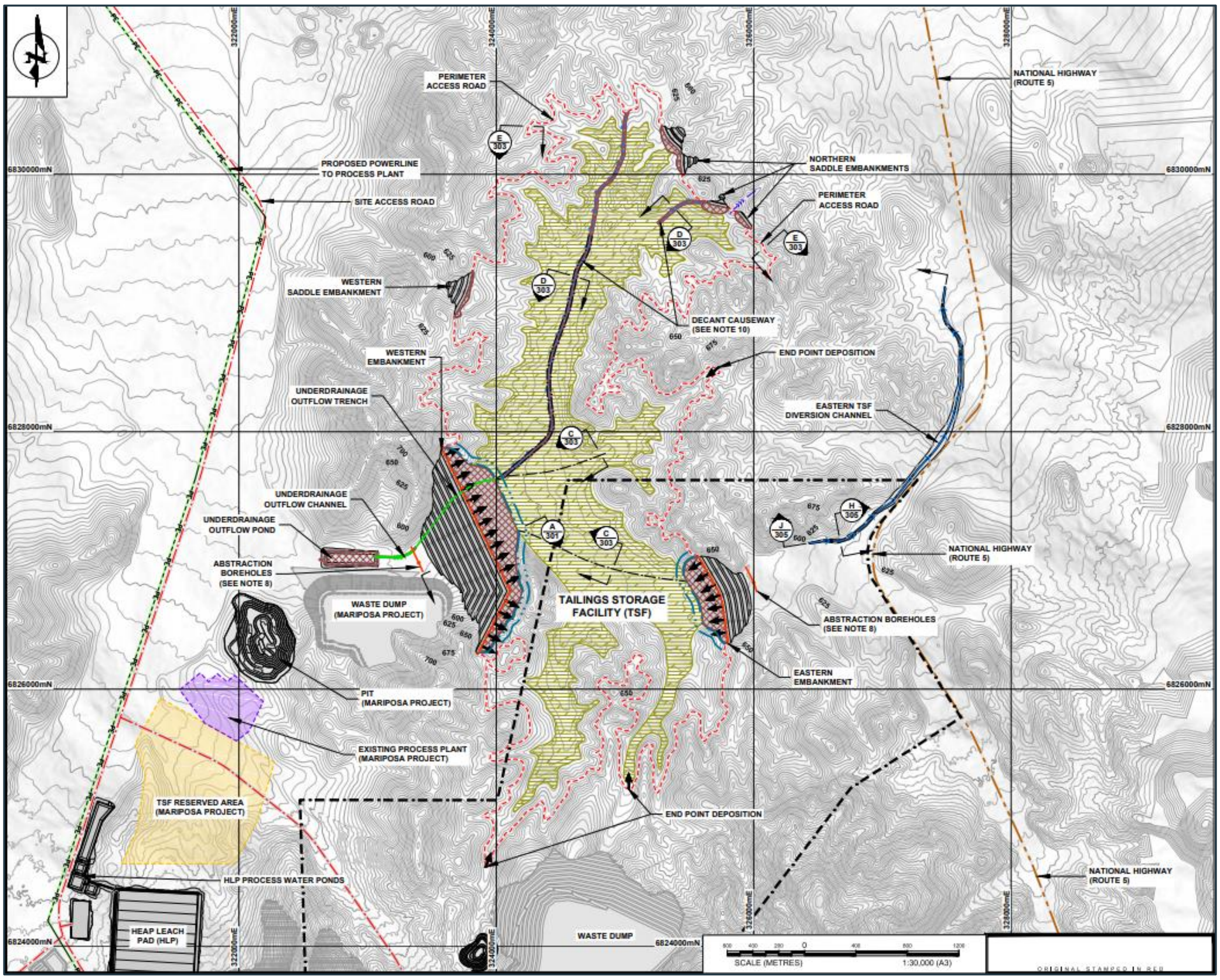
LEGEND:

- MINE LEASE BOUNDARY
- TAILING DEPOSITION PIPELINE
- DECANT TURRET CAUSEWAY
- SPIGOT LOCATION
- TEXTURED HDPE LINER
- SMOOTH HDPE LINER
- PRIMARY TOE DRAIN
- COLLECTOR DRAIN
- UNDERDRAINAGE OUTFLOW TRENCH
- UNDERDRAINAGE OUTFLOW CHANNEL
- ABSTRACTION BOREHOLES TRANSECT (SEE NOTE 6)
- PERIMETER ACCESS ROAD
- BUTTRESS FOOTPRINT

NOTES:

1. 5m CONTOUR INTERVAL SHOWN. TOPOGRAPHY BASED ON THE AMALGAMATION ON DATA RECEIVED FROM HOT CHILI LIMITED FROM MAY, 2024 UNTIL SEPTEMBER, 2024.
2. ALL COORDINATES SHOWN IN GRID PROJECTION UTM(WGS84) ZONE 19 SOUTH.
3. SITE INFRASTRUCTURE LAYOUT SHOWN RECEIVED DECEMBER, 2024.
4. PITS AND WASTE DUMPS LAYOUT RECEIVED, AUGUST 2024.
5. TSF STAGE 1 DESIGN SHOWN.
6. ABSTRACTION BOREHOLES TO BE INSTALLED 20m DOWNSTREAM FROM THE FINAL BUTTRESS TOE LINE. SIX BORES TO BE INSTALLED DOWNSTREAM OF BOTH EASTERN AND WESTERN EMBANKMENTS. DEPTHS AND LOCATIONS TO BE SPECIFIED BY OTHERS.
7. FINGER DRAINS TO BE INSTALLED THROUGHOUT STAGE 1 BASIN AT 25m SPACINGS. THE FINGER DRAIN SECTION IS PROVIDED ON DRAWING 201-478-A3000-302.
8. DECANT TURRET (DESIGNED BY OTHERS) TO BE SLED OR TRAILER MOUNTED TO ALLOW FOR RETREAT AS THE TAILINGS BEACH DEVELOPS. ABSTRACTION INFRASTRUCTURE AND PIPELINES DESIGNED BY OTHERS.

Figure 18.5: General Arrangement of Final TSF (386 Mt) (Knight Piesold, 2025)



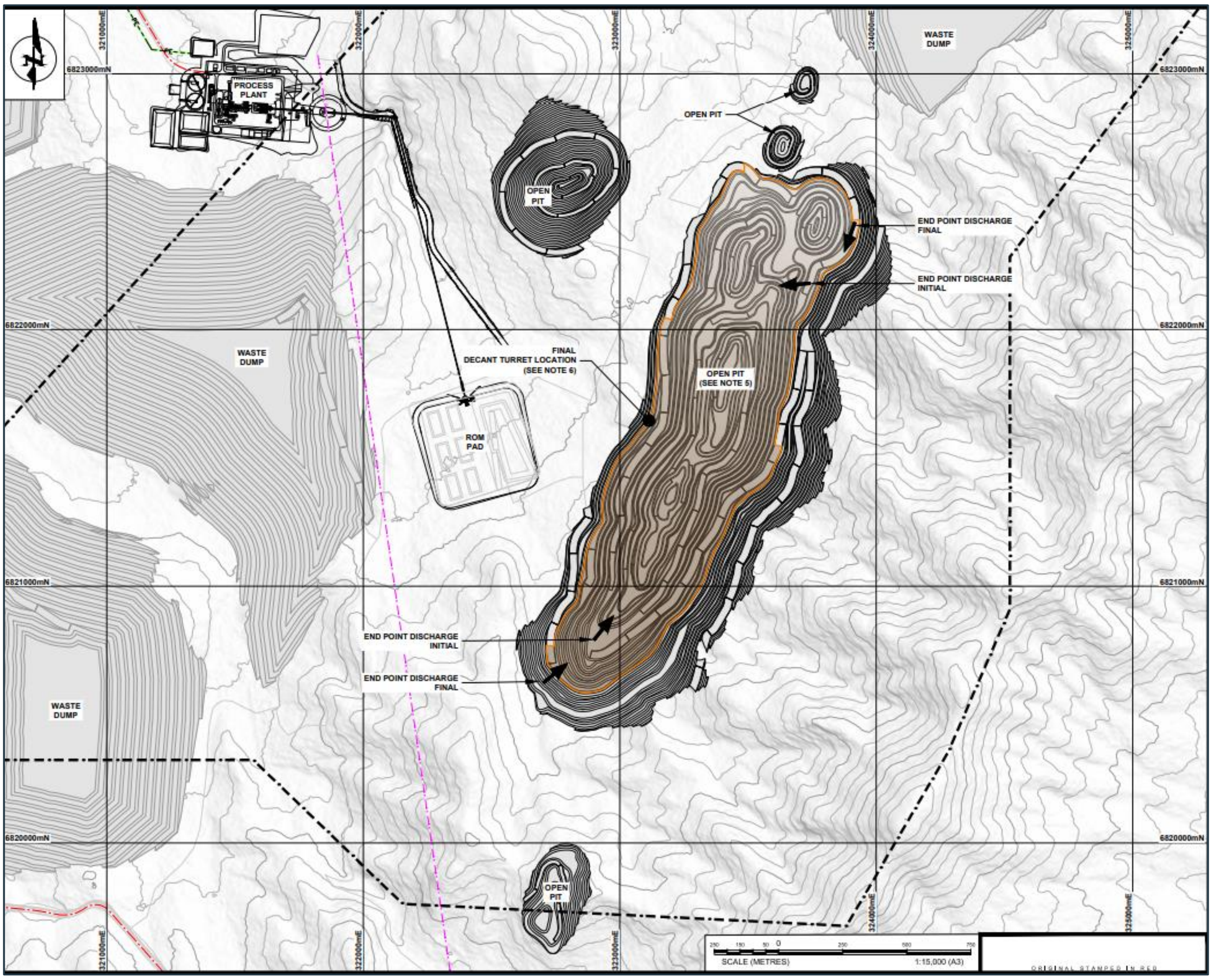
LEGEND:

- MINE LEASE BOUNDARY
- TAILING DEPOSITION PIPELINE
- DECANT TURRET CAUSEWAY
- SPIGOT LOCATION
- TEXTURED HDPE LINER
- SMOOTH HDPE LINER
- PRIMARY TOE DRAIN
- COLLECTOR DRAIN
- UNDERDRAINAGE OUTFLOW TRENCH
- UNDERDRAINAGE OUTFLOW CHANNEL
- ABSTRACTION BOREHOLES TRANSECT (SEE NOTE 8)
- CLOSURE SPILLWAY (SEE NOTE 7)
- PERIMETER ACCESS ROAD
- BUTTRESS FOOTPRINT

NOTES:

1. 5m CONTOUR INTERVAL SHOWN. TOPOGRAPHY BASED ON THE AMALGAMATION ON DATA RECEIVED FROM HOT CHILI LIMITED FROM MAY, 2024 UNTIL SEPTEMBER, 2024.
2. ALL COORDINATES SHOWN IN GRID PROJECTION UTM(WGS84) ZONE 19 SOUTH.
3. SITE INFRASTRUCTURE LAYOUT SHOWN RECEIVED DECEMBER, 2024.
4. PITS AND WASTE DUMPS LAYOUT RECEIVED, AUGUST 2024.
5. TSF FINAL STAGE DESIGN SHOWN.
7. NORTHERN SADDLE EMBANKMENT TO BE REMOVED AT CLOSURE SO CLOSURE SPILLWAY CAN BE CONSTRUCTED.
8. ABSTRACTION BOREHOLES TO BE INSTALLED 20M DOWNSTREAM FROM THE FINAL BUTTRESS TOE LINE. SIX BORES TO BE INSTALLED DOWNSTREAM OF BOTH EASTERN AND WESTERN EMBANKMENTS. DEPTHS AND LOCATIONS TO BE SPECIFIED BY OTHERS.
9. FINGER DRAINS TO BE INSTALLED THROUGHOUT STAGE 1 BASIN AT 25m SPACINGS. THE FINGER DRAIN SECTION IS PROVIDED ON DRAWING 201-478-A3000-302.
10. DECANT TURRET (DESIGNED BY OTHERS) TO BE SLED OR TRAILER MOUNTED TO ALLOW FOR RETREAT AS THE TAILINGS BEACH DEVELOPS. ABSTRACTION INFRASTRUCTURE AND PIPELINES DESIGNED BY OTHERS.

Figure 18.6: General Arrangement of In Pit Tailings (114 Mt) (Knight Piesold, 2025)



LEGEND:

- MINE LEASE BOUNDARY
- INDICATIVE TAILINGS BEACH
- ROPE CONCENTRATED CONVEYOR
- SITE ACCESS ROAD CENTRELINE
- PROCESS PLANT POWERLINE
- DISCHARGE LOCATION

NOTES:

1. ALL COORDINATES SHOWN IN GRID PROJECTION UTM(WGS84) ZONE 19 SOUTH.
2. 5m CONTOUR INTERVAL SHOWN. TOPOGRAPHY BASED ON THE AMALGAMATION ON DATA RECEIVED FROM HOT CHILI LIMITED FROM MAY, 2024 UNTIL SEPTEMBER, 2024.
3. SITE INFRASTRUCTURE LAYOUT SHOWN RECEIVED DECEMBER, 2024.
4. PITS AND WASTE DUMPS LAYOUT RECEIVED, AUGUST 2024.
5. 114Mt IN PIT STORAGE SHOWN.
6. DECANT TURRET (DESIGNED BY OTHERS) TO BE SLED OR TRAILER MOUNTED TO ALLOW FOR RETREAT AS THE TAILINGS BEACH DEVELOPS. ABSTRACTION INFRASTRUCTURE AND PIPELINES DESIGNED BY OTHERS.

The site is dry, with the average yearly rainfall for the Project area approximately 50 mm and an estimated annual Pan evaporation of 2,400 mm/year (1,702 mm/year lake evaporation).

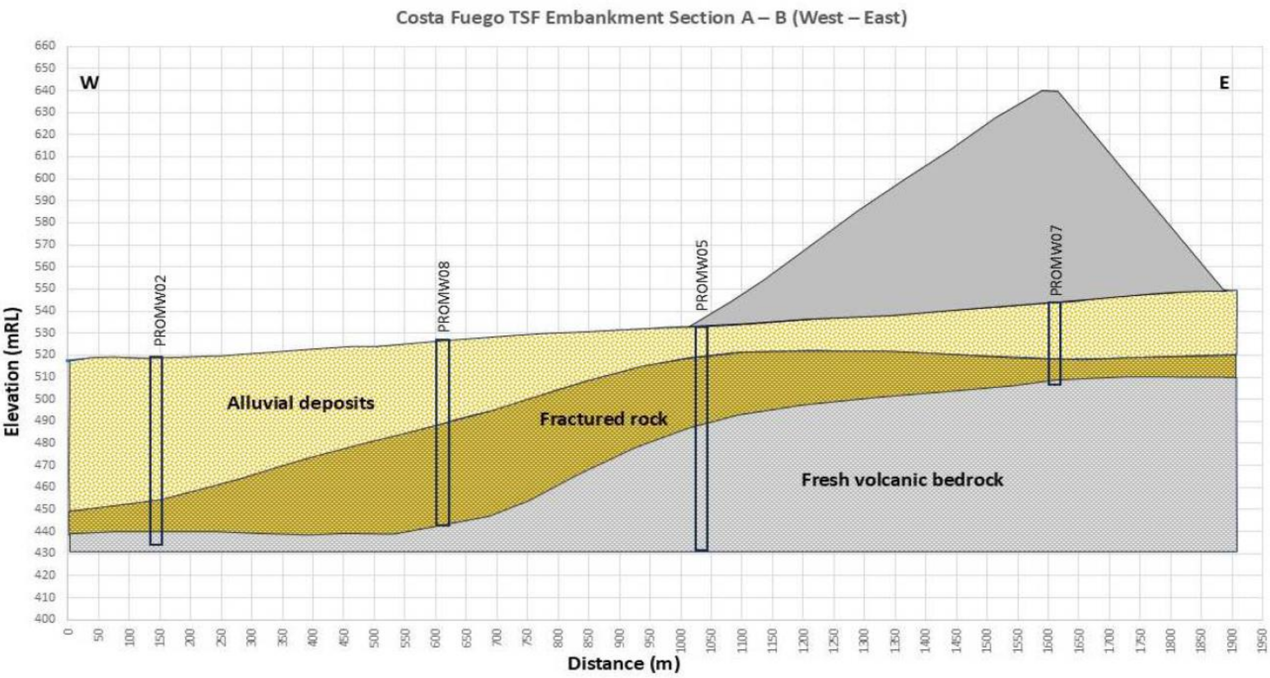
A preliminary seismic hazard assessment has been completed for this phase of design, which confirmed that the Project area is located in an area of high seismic activity. An in-country seismic specialist has been engaged to conduct a site-specific seismic hazard assessment, which is in progress.

A geotechnical investigation of the TSF was conducted, which comprised diamond drilling, test pitting, and a MASW survey. The site typically comprises a compact granular soil layer of variable thickness, underlain by residual soil/extremely weathered rock (granular soil), transitioning to rock. Towards the valley base, Alluvium was encountered to depths of between approximately 15 m and 30 m in some locations, whilst away from the valley base, test pits refused at an average depth of 1.6 m, typically on or close to the surface of highly weathered rock. Further geotechnical investigations are planned for the next design phase to expand on these findings.

A groundwater investigation within the TSF footprint was conducted, which comprised Reverse Circulation (RC) Drilling, installation of groundwater bores, and testing of these bores. The depth to bedrock ranges from 12 m to 80 m, and the depth to groundwater (where intersected) ranges from 18 m to 66 m. The depth of rock within the eastern embankment footprint is between 12 m and 26 m, and the depth to groundwater is between 28 m to 42 m based on historical drilling. In-situ water quality parameters were collected during the investigation.

Based on the interpretation of drilling data and MASW survey the western embankment was moved ~400 m to the east to avoid the deeper alluvial deposits. The interpreted ground profile showing the proposed embankment location is provided below in Figure 18.7.

Figure 18.7: Interpreted Ground Profile Below Western TSF Embankment



Historical groundwater depth measurements indicate that the stabilised water depth in the Productora Pit is around 20 to 80m below the surface (RL 705 m to RL 780 m). Three geological units were identified for the

Project: shallow sediments (HU1), fractured rock (HU2), and fresh bedrock (HU3). The shallow sediments are insignificant for the Productora Open Pit. It is characterised by fractured rock and fresh bedrock. These units have moderate ($1x10^{-7}$ to $1x10^{-8}$ /s) to low permeabilities ($1x10^{-8}$ m/s) and low groundwater capacity. The structural model indicates that many faults intercept the Productora pit, and these will control the hydrogeological conditions around the pit. Additional hydrogeological studies regarding the Productora pit in relation to tailings deposition are recommended to quantify seepage and any preparation or closure requirements for the pit upon the transition from production to tailings deposition.

18.4.2 TSF Type and Design

18.4.2.1 Design Data

The proposed facility is for valley storage with three multi-zoned embankments and, once mined out, in pit disposal within the Main Productora Pit. The data used for design is summarised in Table 18.1.

Table 18.1 : TSF Design Data		
Item	**Value**	**Source**
Total tonnage (Mt)	500	HCH Ltd
-Tailings Storage Facility (above ground)	386	
- In pit Storage	114	
Throughput (Mt/a)	20.7	HCH Ltd
Discharge % solids	68	Wood
Estimated beach slope (70% solids)	1V:150H	Knight Piésold
Tailings properties (tested at 63% solids)		
P_{80} (µm)	150	Knight Piésold
Density (t/m^3)		
- Undrained	1.37	Knight Piésold
- Drained	1.43	Knight Piésold
- Air-drying	1.67	Knight Piésold
Supernatant production (%)		
- Undrained	33	Knight Piésold
- Drained	18	Knight Piésold
Freeboard		
- Tailings (m)	1.0	Decree 50
- Storm (m)	0.8 to 0.9	Knight Piésold
Stormwater Containment (ANCOLD)		
- Wet Season	1 in 1,000 yr ARI Wet Season	ANCOLD 2019
- Extreme Storage	1 in 10,000 yr ARI, 72hr Flood	ANCOLD 2019
Flood Design (GISTM)		
- Active Care	1 in 10,000 yr ARI	GISTM
- Passive Care	1 in 10,000 yr ARI	GISTM
Flood Design (Decree 50)	1 in 10,000 yr ARI or PMF, whichever is greater	Decree 50
Dam Classification		
- Decree 50	Category C	Decree 50
- ANCOLD	Extreme	ANCOLD 2019
- GISTM	Extreme	GISTM
Earthquake Loading		
- Operating Basis Earthquake	1 in 1,000 yr ARI Earthquake	ANCOLD
- Safety Evaluation Earthquake	1 in 10,000 yr ARI Earthquake or Maximum Credible Earthquake (MCE)	ANCOLD
- Active Care	1 in 10,000 yr ARI Earthquake	GISTM
- Passive Care	1 in 10,000 yr ARI Earthquake	GISTM
- Seismic Design	Maximum Credible Earthquake (MCE) or Magnitude 8.5 earthquake, whichever is greater	Decree 50
Stability Design (ANCOLD)		
- Drained	1.5	ANCOLD
- Undrained (no release)	1.3	ANCOLD

Table 18.1 : TSF Design Data		
Item	**Value**	**Source**
- Undrained (release)	1.5	ANCOLD
- Post Seismic	1.0 to 1.2	ANCOLD
Stability Design (Decree 50)		
- Static	1.4	Decree 50
- Pseudo Static	1.2	Decree 50
- Post Seismic	1.0	Decree 50
Embankment Design		
- Crest width (without/with buttress) (m)	24 m / 27.5	Knight Piésold
- Upstream Slope	3H:1V	Knight Piésold
- Downstream Slope (interim)	4H:1V (effective)	Knight Piésold
- Downstream Slope (final, overall)	5H:1V (effective)	Knight Piésold

Knight Piésold previously tested a sample of representative tailings in 2015. Samples for tailings test work were composited from the rougher tailings from flowsheet development flotation tests. Two composites were created, one at a P_{80} grind size of 106 µm and the other at 150 µm. Additional physical tailing testwork is currently underway on samples generated for each deposit. These will be incorporated into the next design phase.

A suite of static geochemical testing was conducted on the 2015 tailings samples, which indicated that the tailings were potentially acid forming (PAF), had a moderate number of elemental enrichments and that the supernatant water quality was poor (partially due to seawater used in processing). Seepage control measures such as a clay liner in parts and an above-liner underdrainage system were incorporated based on the testing. Due to the elevated enrichments, a mine waste cover was incorporated to limit access to the tailings and manage dust generation at closure. Furthermore, due to the PAF nature of the tailings and supernatant water quality, an engineered low permeability layer to reduce oxygen diffusion and water ingress was incorporated. Additional measures, such as designing the process flow sheet to remove sulphide-bearing materials, will also be considered as part of the next design phase.

In 2024/2025 a further 14 tailings samples underwent geochemical characterisation. Results were received at the time of PFS finalisation, and these will be reviewed as part of the next design phase, which will inform the tailings management requirements and whether further geochemical testing is required. Geochemical testing on operational samples will also be continued throughout operations and rehabilitation trials conducted to inform the final closure requirements. The 2015 sample was adopted for the basis of the design within the PFS.

Based on the tailings test work, design parameters provided in Table 18.2 were estimated by Knight Piésold. At 68% solids, the tailings are expected to exhibit highly thickened behaviour.

Table 18.2 : TSF Estimated Design Densities and Maximum Water Return	
Item Design Densities (t/m^3)	**Value**
Stage 1	1.54
Stage 2	1.57
Stage 3	1.59

Table 18.2 : TSF Estimated Design Densities and Maximum Water Return	
Item Design Densities (t/m^3)	**Value**
Stage 4+ In Pit Disposal	1.59 – 1.65 1.49
Expected max. decant pump capacity	47% Return (228 L/s)

18.4.2.2 General

The TSF will comprise a cross-valley storage facility formed by multi-zoned earth fill embankments, comprising a total footprint area (including the basin area) of approximately 276 ha for the Stage 1 TSF, increasing to 889 ha for the final TSF and 88.5 ha within the Productora Open Pit. The TSF is designed to accommodate 386 Mt of tailings within a conventional TSF and 114 Mt within the Productora Open Pit, for a total of 500 Mt. Further expansion of the TSF is possible; however, it should be evaluated if as and when required. General arrangements for the TSF Stage 1 and final stage are shown in Figure 18.4 and Figure 18.5.

A total of 18 raises of the TSF will be required over the life of mine. Stage 1 will provide storage for 18 months (31.1 Mt of dry tailings) and have a maximum embankment height of 38.3 m (RL 582.3 m). Subsequent stages will be raised annually to suit storage requirements. Stage Final will provide storage for 386 Mt of dry tailings and have a height of 71.4 m (RL 632.4 m). An additional 114 Mt of dry tailings will be stored within the Productora pit commencing once mining is completed; no embankment will be required as the tailings are expected to be below the pit mouth, and the facility will have sufficient capacity to contain the design storm without overflow. The final tailings level within the Productora pit will be RL 770.6 m, which is generally found in fresh bedrock, although some fractured rock zones exist. Significant excess stormwater capacity is available within the pit (refer to Section 18.1.2.3).

<u>Conventional TSF</u>

The Stage 1 TSF is designed for 18 months storage capacity. Subsequently, the TSF will be constructed in annual raises to suit storage requirements. However, this may be adjusted to biennial raises to suit mine scheduling during the operation. Downstream raise construction methods will be utilised for all TSF embankment raises. A downstream seepage collection system will be installed downstream of the TSF embankment to capture and return seepage to the plant site from the TSF. The abstraction system will comprise a series of abstraction bores located downstream of the embankments to capture seepage. Typical embankment cross sections and details are shown in Figure 18.8 and Figure 18.9.

Figure 18.8 : Typical Embankment Cross Section, tailings located on left side

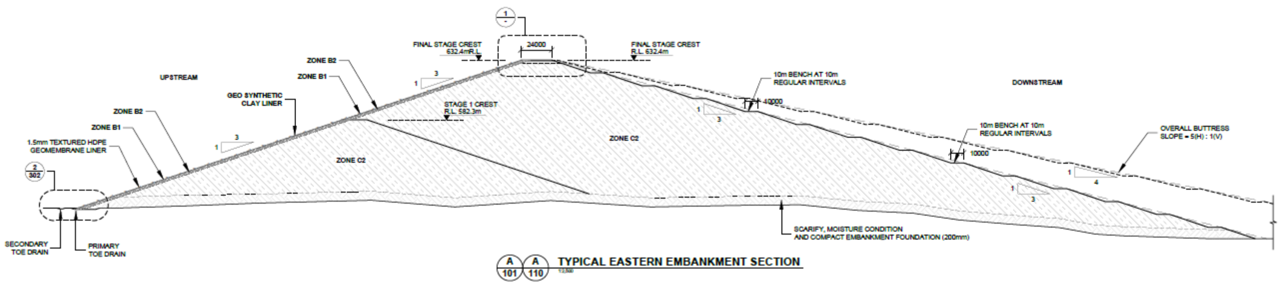
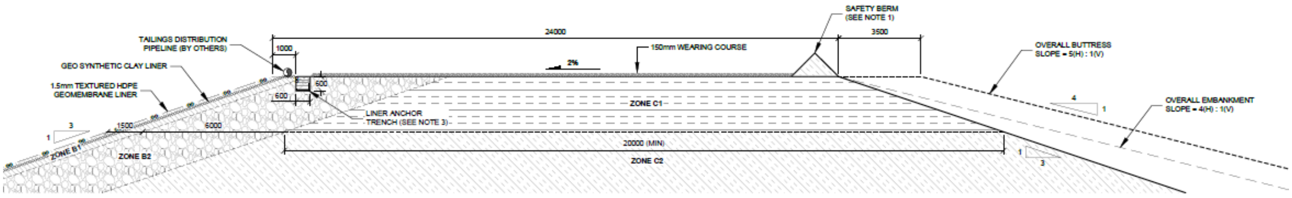


Figure 18.9 : Typical Crest Detail, tailings located on left side



The TSF basin area will be cleared, grubbed and topsoil stripped. A basin liner comprising a geosynthetic clay liner (GCL) is proposed across the higher permeability alluvium and within the embankments. After additional tailings geochemistry testing is completed, the basin lining requirements will be reviewed.

The TSF design incorporates an underdrainage system to reduce head pressure near the embankments, reduce seepage, increase tailings densities, and improve the geotechnical stability of the embankments. The underdrainage system comprises a network of finger and collector drains. The underdrainage system drains by gravity to a collection sump located at the lowest point in the TSF basin. The solution recovered from the underdrainage system will flow to an HDPE-lined outflow pond via an HDPE-lined outflow trench. The TSF underdrainage system layout is shown in Figure 18.5, and typical underdrainage system sections and details are shown in Figure 18.13 to Figure 18.15.

Supernatant water will be removed from the TSF via a floating turret system located on a decant causeway constructed at start-up with the low points of the valley and extended up the valley during operation as the supernatant pond migrates. The supernatant pond will be maintained in the northern valley within the TSF basin. Solution recovered from the decant system will be pumped back to the plant for re-use in the process circuit. The TSF decant system layout is shown in Figure 18.11, and typical decant system sections and details are shown on Figure 18.18.

An operational emergency spillway will not be provided during operation as the facility will be designed to attenuate the probable maximum flood (PMF), which is in addition to the capacity of the target operational pond. The facility's excess stormwater capacity is significant, and the containment of the PMF is considered practicable with minimal additional freeboard being applied.

The closure spillway will be excavated through the northern saddle adjacent to the facility, at the low point on the final tailings beach, running north. It will safely discharge into the existing watercourse after operation ceases, thus ensuring the TSF becomes a fully water-shedding structure on closure. Typical closure spillway sections and details are shown in Figure 18.10 and Figure 18.11 below.

Figure 18.10 : Typical Closure Spillway Long Section

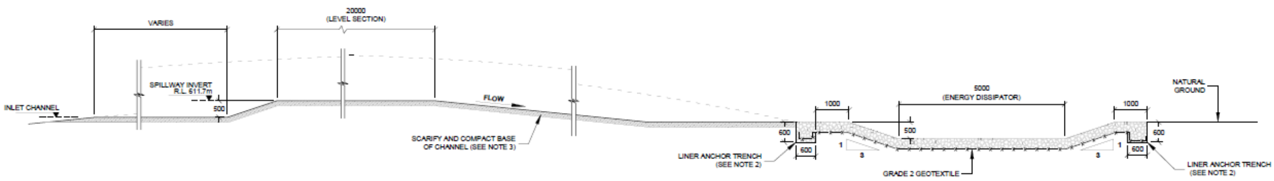
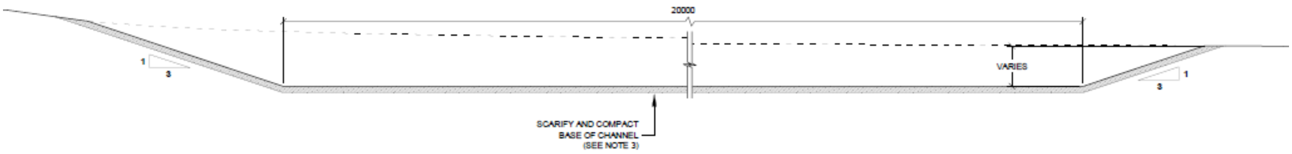


Figure 18.11 : Typical Closure Spillway Transverse Section



Tailings will be discharged into the TSF by sub-aerial deposition methods, using a combination of spigots at regularly spaced intervals around the TSF embankment and end point discharge locations within select valleys. Deposition extents are shown on Figure 18.4 and Figure 18.5. During the early stages of operation, when the risk of water shortages is highest, the deposition plan will be managed to optimise water recovery. The southern endpoint discharge will operate 8 m above the spigot deposition at the embankments, to ensure that the supernatant pond is located in the northern valley of the facility.

Tailings will be delivered to the TSF via the tailing's delivery pipeline. The pipeline will report from the process plant to the TSF. The pipeline will be contained in a HDPE liner trench, with an access road constructed adjacent to allow for daily inspection by process personnel. The delivery pipeline trench will contain tailings and liquor outflow in the event of pipeline rupture. The pipeline will be fitted with telemetry and automatic shut offs to prevent continued discharge in the event of pipeline failure. The pipeline shall drain via gravity to catch basins located at the low points along the pipeline alignment where tailings discharge can be contained in the event of a rupture.

The tailings delivery pipeline will report to the tailings distribution pipeline located on the crest of the embankment and perimeter of the facility. The distribution pipeline will be positioned on the access road around the western perimeter of the TSF. Due to the region's high seismicity and steep topography, the access road has been founded in cut. Fill sections have been avoided to reduce risks associated with a slip failure into the facility. The access road and embankment crest will be graded to promote runoff into the facility in the event of pipeline failure to prevent uncontrolled release of tailings to the environment.

Due to the limited access to the embankment area and the requirement to raise all the embankment zones simultaneously, it is anticipated that the embankment construction schedule might not align with annual lifts. Therefore, the staging may be modified to suit the mine waste production schedule.

<u>In Pit Storage of Tailings</u>

No embankments will be required around the Productora Open pit, as tailings deposition will be kept below the pit mouth (~ RL 780 m). Sufficient stormwater capacity will be provided within the pit during operation to handle a PMF. Monitoring bores will be provided around the pit to monitor water quality. The bores will be large enough in diameter to allow the installation of a pump to return seepage if water quality is deemed unsuitable. No basin treatment is proposed within the Pit.

Supernatant water will be removed from the Pit via a floating turret system located on the pit access ramp and relocated as the tailings beach and supernatant pond level rise. Solution recovered from the decant system will be pumped back to the plant for re-use in the process circuit. The TSF decant system layout is shown in Figure 18.12, and typical decant system sections and details are shown in Figure 18.12.

An operational emergency spillway will not be provided during operation as the facility will be designed to attenuate the probable maximum flood (PMF) in addition to the target operational pond. The facility's excess stormwater capacity is significant, and the containment of the PMF is considered practicable with minimal additional freeboard being applied.

At closure, the facility is expected to be water-negative and will be able to attenuate the probable maximum flood (PMF) without discharge. Additional work is required to understand the pit inflows during operation and closure and possible pit lake development. If the subsequent design determines a spillway is needed, the closure spillway will be excavated through the pit mouth to the north. It will safely discharge into the existing watercourse before reaching the TSF after the operation ceases. At closure, the facility will be a store and release system.

Tailings will be discharged into the In Pit TSF by end-point discharge. Deposition extents are shown on Figure 18.6. During the early stages of operation, when the risk of water shortages is highest, the deposition plan will be managed to optimise water recovery.

A pipeline containment trench (TDRT) will be constructed before commissioning to contain both the tailings delivery pipeline and decant return pipeline between the Pit and Plant Site to reduce environmental impact if the pipeline bursts.

18.4.2.3 Stormwater Storage Capacity

A deposition model was run for the TSF, assuming deposition off the eastern and western embankments and endpoint discharge at two locations along the southern ridge of the facility and one location along the eastern ridge. The nominated minimum tailings freeboard for tailings was set at 1 m in accordance with Chilean regulations (Decree 248 and 50, Refs. 1 and 2). An additional 0.8 to 0.9 m freeboard was applied, above the supernatant pond generated by the design storm event (Probable Maximum Flood 72 hr), superimposed over the operational pond. Table 18.7 below summarises the available stormwater capacity within the facility following a number of rainfall events.

Table 18.7 : Stormwater Capacity

Rainfall Event (Probable Maximum Flood)	PMF Inflow plus average pond (Mm^3)	Excess Stormwater Capacity above PMF inflow (Mm^3)
Tailings Storage Facility		
Stage 1	13.97	2.84
Stage 4	13.97	3.09
Stage 18	13.97	41.75
In Pit Tailings		
Stage 19	3.56	79.2
Final	3.56	4.40

The climate at the site is arid, with an average annual rainfall of about 50 mm, so water storage capacity will be a function of short-term storm events (providing that the facility is operated in accordance with the design intent). The 24-hour and 72-hour storm events were estimated based on meteorological data from Santa Juana. The quoted storm events are provided in Table 18.

Table 18.8 : Maximum Storm Events		
Return Period (yr)	**24-Hour Rainfall (mm)**	**72-Hour Rainfall (mm)**
5	38	40.1
10	54	63.8
20	69.1	92
50	87.2	136.6
100	99.4	176
1,000	161.5	286
PMP	410	725.9

The data sequence to determine these events was relatively short (51.5 years of valid data). For the study, it was decided that the design requirement will be that the TSF will need to hold a probable maximum flood 72-hour storm event plus a freeboard allowance of 0.8 to 0.9 m (depending on the stage). The calculation of the storage volume required is summarised in Table 18.3. A hydrologic model was developed to assess the upstream catchment and the performance of the eastern TSF diversion channel and the eastern TSF embankment.

Table 18.3 : TSF Storm Capacity	
Item	**Value**
TSF catchment area (ha)	1,897
Probable Maximum Precipitation 72-hour (mm)	726
Catchment run-off coefficient[1]	1.0
Design storm volume (Mm^3)	13.7
Upstream catchment area (ha)	6,040
Probable Maximum Precipitation 72-hour (mm)	726
Catchment run-off coefficient[1]	1.0
Design storm volume (Mm^3)	43.8
Peak Attenuation (m^3)	4,240,000
Maximum Inflow (m^3/s)	1,013
Maximum Outflow (m^3/s)	739.7

1 – Probable Maximum Flood (PMF) conditions assume 100% runoff.

As the facility will be designed without an operational spillway, it was designed to contain the probable maximum flood; therefore, a runoff coefficient of 1.0 was adopted. At closure, a spillway will be incorporated to ensure the facility is water-shedding. It is noted that the embankment elevations during the initial 4 years of operation will be governed by the design storm event (PMF); tailings will govern the stages for Year 5 onwards and there will be significant excess stormwater capacity above the extreme stormwater containment requirement for all stages (refer Table 18.7).

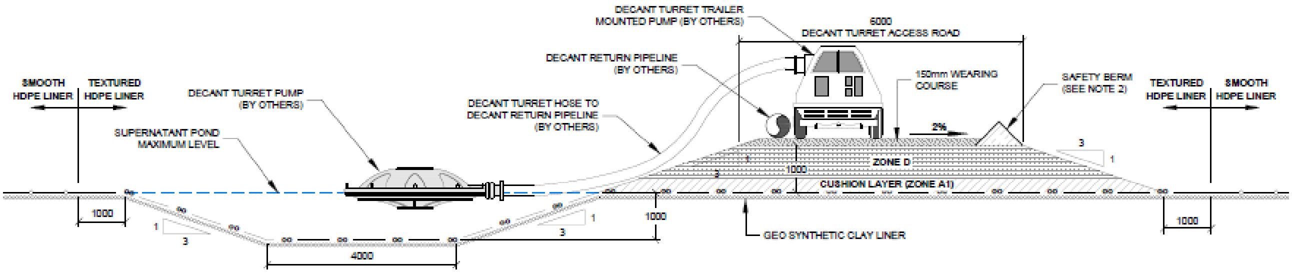
It is noted that the water freeboard available is very large under normal operating conditions (much greater than one metre). However, Knight Piésold reviewed the water freeboard for the typical operating pond plus the extreme storm events and applied an additional minimum freeboard of 0.8 to 0.9 m (depending on the stage) for this case.

18.4.2.4 TSF Decant Water System

In the initial years of operation, the tailings move up the valley by 3.1 km. Decant recovery in these initial stages will require a mobile decant pump system, either a trailer or skid-mounted system. In 10 years, the location of the supernatant pond will be near the north end of the basin, and a permanent decant causeway will be established.

The decant system proposed is to construct a decant trench along the base of the valley and use the material to construct a 1 m high decant causeway. The trailer or skid-mounted decant pump will be placed on the causeway, with the decant turret placed in the trench. The pump will be relocated regularly along the causeway as the pond advances up the valley. The general arrangement of the decant system is provided in Figure 18.5 and the section is provided below in Figure 18.12.

Figure 18.12 : Typical Decant System Section



Decant water will be recovered from the TSF at a maximum value of 902 m^3/hr (based on 8000 hrs of operation). A third party will design the pump system and pipework.

18.4.2.5 Closure Concept

For closure, it is proposed that the final tailings profile be shaped to direct runoff to the north of the facility, where a closure spillway will be excavated in the northern ridge so that any rainfall runoff will run over the tailings surface to a sediment control area before discharge downstream. The tailings surface will be covered by a suitable cover of local borrow and waste rock to prevent erosion, especially wind erosion. For this study, the following was assumed:

- Low permeability cover (200 mm)
- Store and Release layer (500 mm)
- Topsoil cover (100 mm)

For this study, an allowance for installing a geosynthetic clay liner (10% of the tailings beach) has been included. This was assumed to be installed at the low point of the tailing beach to reduce rainfall infiltration into the tailings. Once more tailings geochemistry data is available, this allowance will be assessed in the next design phase.

A closure spillway will be constructed through a saddle at the north of the facility to ensure that the facility will be water-shedding at closure. A small sediment control area will be provided at the spillway invert before discharge.

18.4.2.6 Seepage Management

The seepage management for the TSF will focus on containment and recovery of seepage at the main embankment location or downstream of the embankment. The main TSF embankment is located in the narrow section of the valley where the alluvium depths are ~15 to 30 m, which is underlain by fractured bedrock and then bedrock. Investigations have determined that the intact bedrock permeability is generally low. However, the fractured bedrock has a reasonably high permeability (similar to the alluvium). The system design includes the following components:

- A cut-off trench under the embankments will be constructed and keyed into suitable low-permeability materials where practical. It is anticipated that due to the significant depths of alluvium within the centre of the valley, this may not be practical along its entire length. Therefore, a nominal depth cut-off will be specified for these locations. The cut-off trenches will be keyed into the underlying bedrock on the embankment abutments, where rock is anticipated to be shallower. A typical section of the cut-off trench is shown in Figure 18.8.
- The TSF basin and upstream embankment faces will be lined with a geosynthetic clay liner (GCL) anchored in a trench around the facility perimeter and along the embankment crests. The liner will extend to the cut-off trench integrated with the upstream toe drain. The extents of the liner are shown on Figure 18.4 and Figure 18.5.
- An underdrainage system comprising finger and collector drains will be installed throughout the Stage 1 TSF basin to reduce the pressure head acting on the liner and assist with drainage of the tailings mass. In addition, an upstream toe drain will be installed along the upstream toe of both the western and eastern TSF embankments. The underdrains will report to an underdrainage sump, where the solution will report downstream in a lined drainage channel excavated below the embankment to an outflow pond, where the solution will be pumped back to either the TSF or the plant site. The underdrains will comprise of the following:
    - Finger drains—80 mm draincoil surrounded by a sand drainage medium (Zone F1), wrapped in geotextile, and overlain by erosion protection material (Zone E). These drains will be installed at a 25 m spacing throughout the TSF basin. They will be the primary collectors of the solution and will report to either the collector drains or toe drains. A typical section of the finger drain is provided below in Figure 18.13.
    - Collector drains— two 160 mm draincoil surrounded by a sand drainage medium (Zone F1), wrapped in geotextile, and overlain by erosion protection material (Zone E). These drains will be installed in the spine of the valley and report to the underdrainage sump. A typical section of the collector drain is provided below in Figure 18.14.
    - Toe drains— three 160 mm draincoil surrounded by a sand drainage medium (Zone F1), wrapped in geotextile, and overlain by erosion protection material (Zone E). These drains will be installed at the upstream toe of the eastern and western embankments. They will report to the underdrainage sump. A primary toe drain will be installed adjacent to the upstream embankment toe and a secondary toe drain

offset 50 m into the facility. Bridge drains will interconnect these drains. A typical section of the toe drain is provided below in Figure 18.15.

- Downstream of the TSF embankment will be a fence line of abstraction bores. The bores will comprise a row of shallow bores installed into the top of fractured bedrock to collect seepage from the alluvial sediments and a row of deeper bores installed into bedrock to collect seepage from the fractured bedrock. Several monitoring bores will be installed downstream of the abstraction system to assess water quality and performance. Additional abstraction bores will be installed downstream of the existing bores, if needed. The proposed location of the abstraction bores is shown on Figure 18.5 and a typical section is provided below in Figure 18.16.

Figure 18.13 : Typical Finger Drain Section

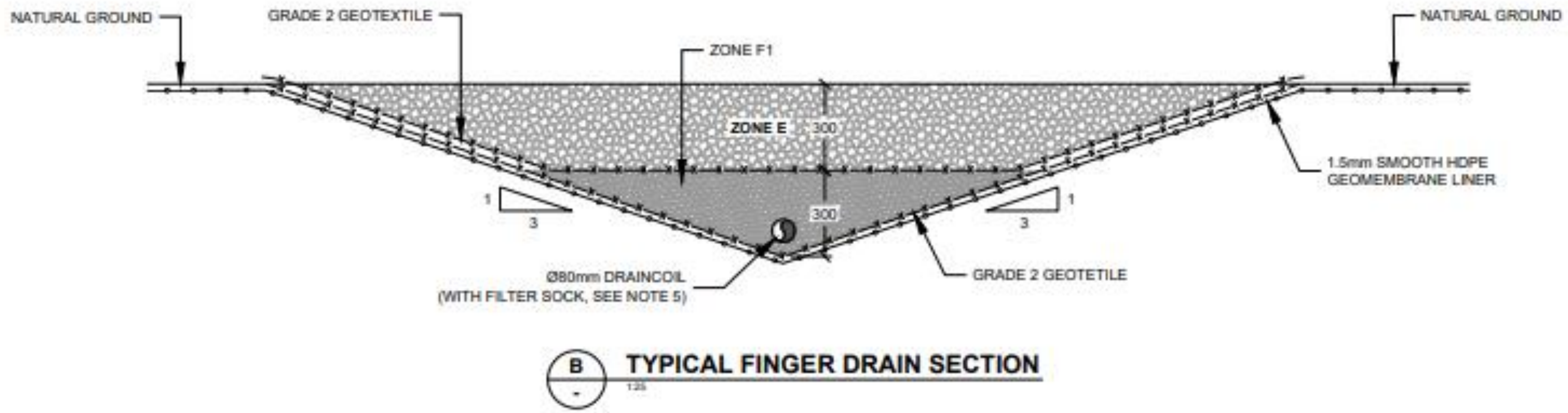


Figure 18.14 : Typical Collector Drain Section

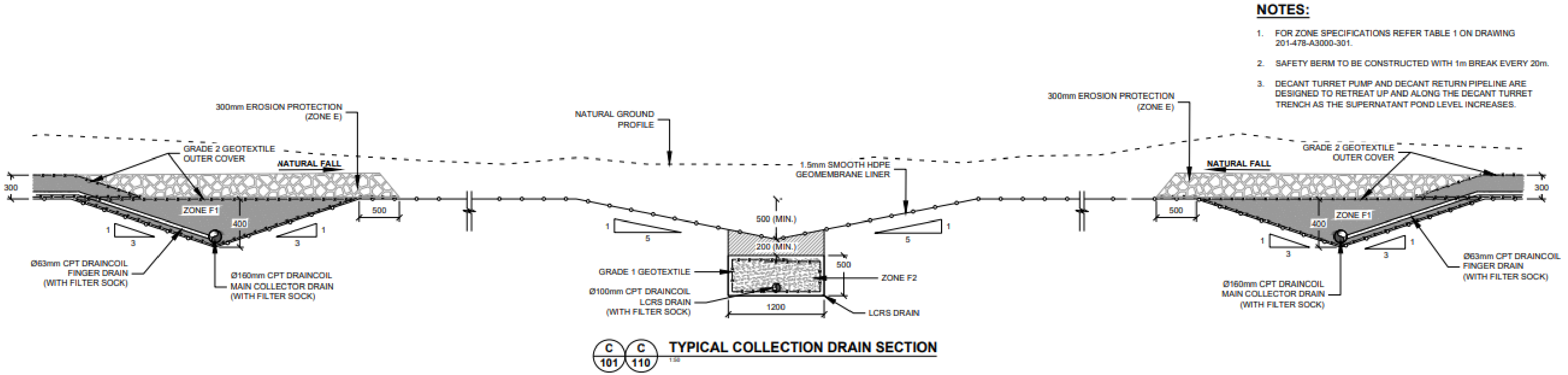


Figure 18.15 : Typical Embankment Toe Drain Section

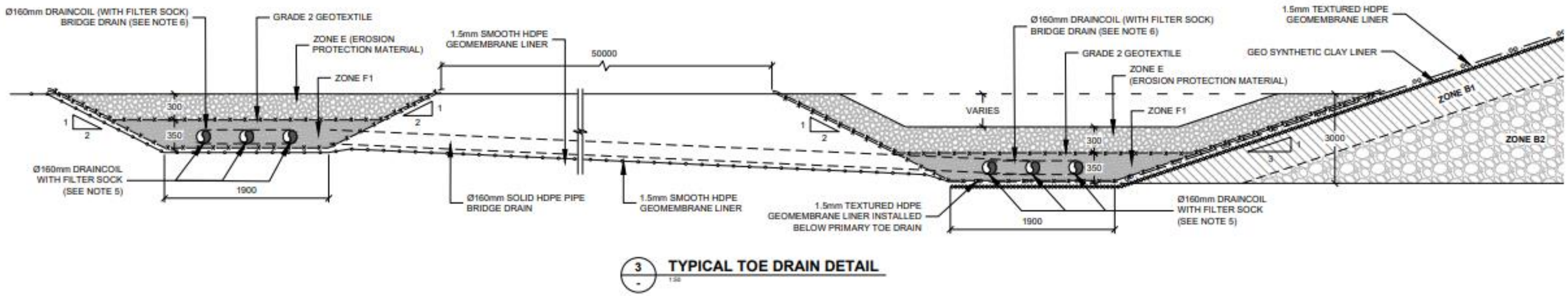
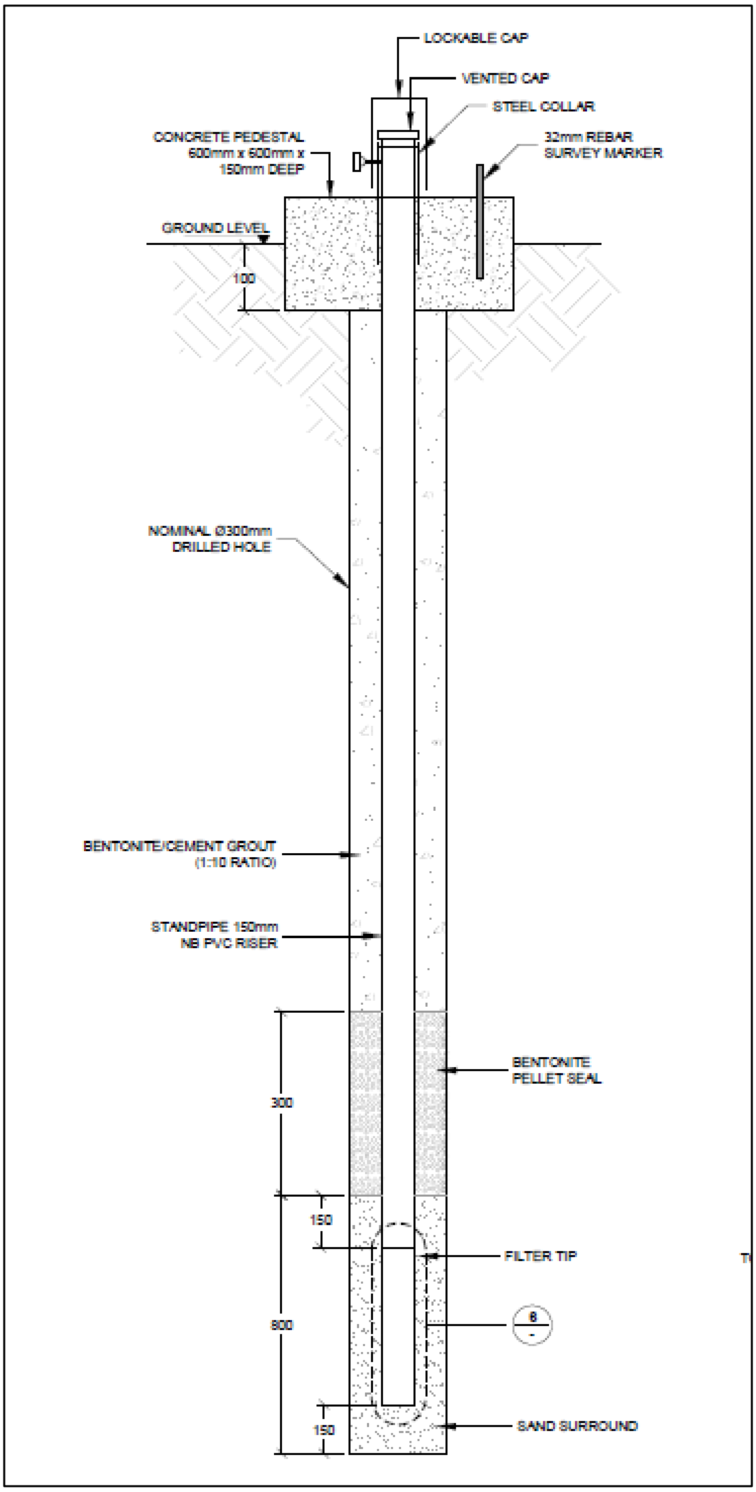


Figure 18.16 : Typical Embankment Abstraction Bore Detail



Beyond the drainage control system, several monitoring bore stations (with one shallow and one deep monitoring bore) will be installed to monitor any potential seepage. These bores will be constructed with sufficient diameter to be equipped with a pump if the seepage rates are high.

18.4.2.7 Quantities

The embankment will have the following zones:

- The upstream face of the embankment will be lined with a GCL liner.
- Zone B1 material shall be screened alluvium to form the bedding material for the GCL. After Stage 1, the use of tailings should be evaluated. The use of tailings as construction material will be subject to further technical evaluation to determine whether they are suitable. The material will be moisture-conditioned (before placement), spread, and compacted along the embankment face with a dozer.
- Zone B2 material (if required) shall be won from borrow to provide sufficient transition between the finer Zone B2 material and coarser Zone C1/C2 material after conditioning and compaction. Zone B2 is assumed to be won from the alluvium within the TSF basin and screened to produce a suitable filter. Material processing of mine waste via crushing and screening may also be required to make enough of suitable filter material. However, this will be subject to the grading of natural materials across the site.
- Zone C2 material shall be delivered to the embankment by the mining operation, levelled with a dozer and traffic compacted by loaded haul trucks on an ongoing basis during the operation. The layer thickness will vary depending on the material types, with an indicative guide as follows:
  - Oxide waste material – 0.5 m to 1 m layers (uncompacted thickness), paddock dumped.
  - Transitional waste material – 1 m to 2 m layers (uncompacted thickness), paddock dumped.
  - Fresh waste material (coarse rockfill) – 2 m to 5 m layers (uncompacted thickness), paddock dumped and spread by a dozer.
- When material from the Open Pit is not available or for areas not easily accessible by the mining fleet, Zone C1 material shall be won from borrow. The material will be moisture-conditioned, spread, and compacted in 500 mm (uncompacted) layers.

18.4.2.8 Monitoring

A monitoring programme for the TSF will be developed to monitor for any potential problems which may arise during operations. The monitoring will include:

- Monitoring bores and surface water sampling stations downstream of the TSF.
- Vibrating wire piezometers in the TSF embankment and foundations to monitor the phreatic surface.
- Survey pins to check embankment movement.

The proposed locations are provided in Figure 18.17. The typical details of the instrumentation are shown in Figure 18.18 and Figure 18.19.

If the monitoring programme indicates that potential problems are developing, monitoring frequency will be increased, and a response plan will be developed.

Abstraction bores around the Productora Pit will be specified. The locations will be specified to intercept structures to allow groundwater quality to be monitored and pumped, if required.

Figure 18.17 : Monitoring and Instrumentation Layout (Knight Piesold, 2025)

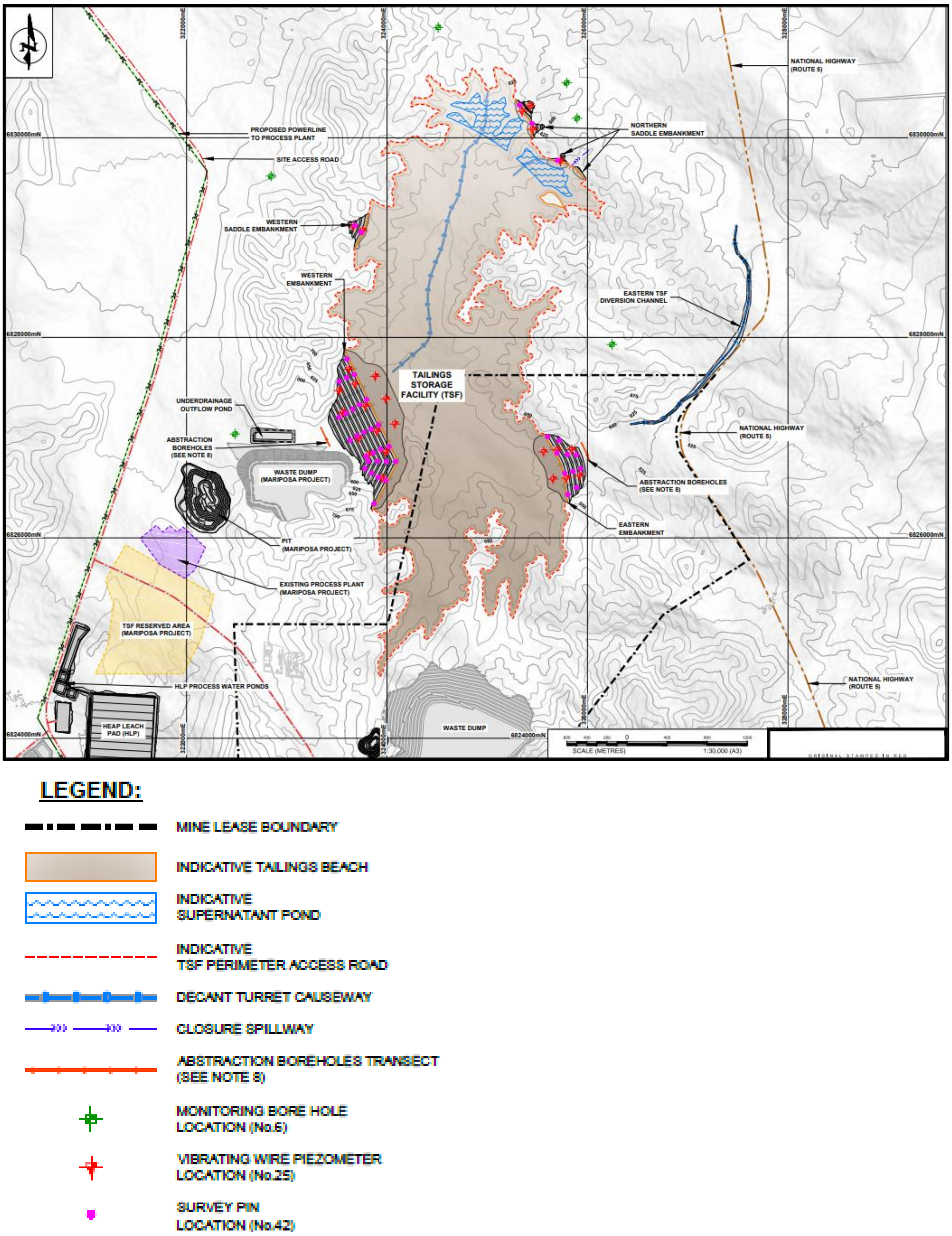


Figure 18.18: Survey Pin Detail

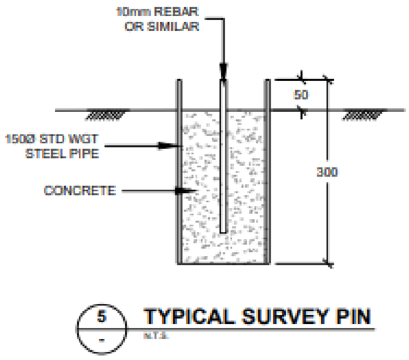
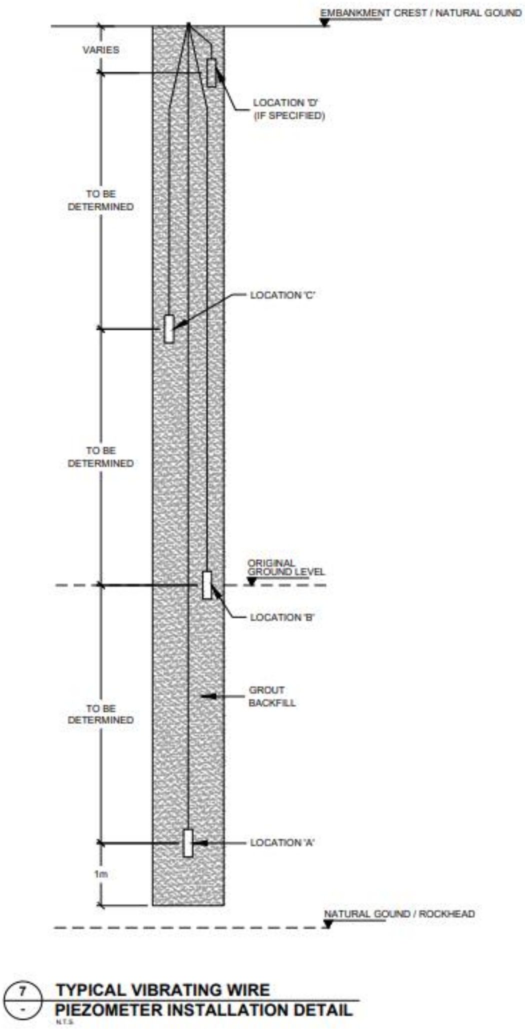


Figure 18.19: Vibrating Wire Piezometer Detail



18.4.2.9 TSF seepage monitoring and collection

Water infiltrating from the TSF is expected to saturate the porous and permeable alluvial deposits of the Quebrada Las Arenas. Most seepage will likely occur under the West embankment, where a greater thickness of the alluvial deposits is detected (35 to 80 m, currently unsaturated), and will migrate downstream through the same unit with an estimated flow of approximately 20 to 40 L/s. Additionally, TSF solutions may migrate through the bedrock through the geological structures, depending on the extension and connectivity of any open fractures.

Compared to the naturally occurring groundwater, any seepage from the TSF will have higher concentrations of chloride (from sea water used during processing), and sulphates, and potentially other soluble ions (mining processes). Standard groundwater monitoring (e.g., groundwater depth, electric conductivity, pH, TDS, Salinity) of each hydrogeological unit should be sufficient to detect any filtration. Although piezometers exist, additional monitoring locations should be evaluated for the EIA and feasibility level studies.

Partially saturated alluvial deposits are found 700 meters downstream of the West embankment (the confluence of Quebradas Las Arenas with Quebrada La Higuera) and groundwater discharges at the Agua Verde wetland, about 14 km downstream, near the Maitencillo area. This constitutes the main environmental receptor identified downstream of the TSF.

To reduce the risk of affecting the Quebrada La Higuera alluvial deposits and the Agua Verde wetland, a combination of measures will likely be required to control infiltration including e.g. liners, cut-off walls, grout injections, while a seepage collection system (SCS), based on a trench and/or pumping wells downstream of the West embankment. Options should be evaluated and designed in post PFS studies as they will be required in the future EIA submission.

Below the East wall the alluvial sediments are thinner, at about 10 to 20 m. Limited seepage through these sediments can likely be managed with shallow pumping wells and/or evaporation ponds. Any contact seepage water should be kept separated from non-contact runoff.

The North wall will be constructed over bedrock, therefore, any seepage is expected to be limited to the open fractures in the weathered bedrock. Shallow piezometers and trenches may be sufficient to detect and control minor seepage.

18.4.2.10 Risks and Opportunities

18.4.2.10.1 Beach Slope

The design is based on an average tailings beach slope of 0.67% (150H:1V). However, the beach slope is heavily dependent on the grind size and the ore blend. Thus, small changes in plant performance or design, ore type, or the ore blend have the potential to change the tailings beach slope.

There are a number of approaches which can be used in response to measured beach slopes that are consistently different to the beach slope used for design. One advantage of staging construction on an annual basis is the ability to modify the design each year based on measured data obtained from the TSF. In these cases, the timing and height of the subsequent embankment raises can be modified to bring the schedule back into line with the design, and the subsequent lifts will be on an annual basis essentially as per the design raised heights.

Steeper Beach Slope

If the measured beach slope is steeper than the design slope, the tailings rate of rise against the TSF embankment will be faster than expected, and the Stage 1 TSF will reach its tailings storage capacity earlier than the design. If this were to become an issue, the response will be to move Stage 2 construction of the TSF forward. Commencing the construction one or two months earlier will not have a significant impact as the construction will still be predominantly in the dry season. It should be noted Stage 1 capacity is 18 months, which provides a high level of flexibility for the construction schedule, if required. In addition, the deposition line could be extended to the eastern valley to provide additional tailings storage capacity without impacting the operation significantly.

It should be noted that for steeper beach slopes the potential tailings storage will be reduced, but the storm water storage capacity will be increased accordingly.

Flatter Beach Slope

If the measured tailings beach slope is flatter than the design slope, the capacity of the Stage 1 TSF to store tailings will be increased. The overall TSF stormwater storage capacity will not be affected, unless Stage 2 construction is deferred beyond the original construction schedule.

18.4.2.10.2 Achieved Densities

The staged TSF embankment crest elevations are based on the ore blend and throughput used for the water balance modelling. Changes in these characteristics and/or throughput will result in changes in the achieved densities in the TSF. Similar to the variations in tailings beach slope, this may result in an adjusted construction schedule for the first raise, either earlier or later than the design timing. It is recommended that monitoring of throughput, ore blend, rate of rise and achieved densities be undertaken so that suitable planning and staging of the future embankment construction can occur.

The densities presented in this Report are based on a single tailings sample from the Productora open pit, therefore a change in density is possible. The density modelling will be reviewed at the completion of the physical tailings testing to determine the implications on the TSF design.

18.4.2.10.3 Life of Mine Planning

Any changes to the life of mine plan or throughput may impact the tailings management requirements for the site. Any significant increases in throughput may result in lower tailing densities being achieved within the TSF, thus increasing construction costs. Any decrease to the total tonnage may require reconsideration of the proposed closure plan, as the closure spillway may become prohibitively deep.

In addition to the impacts on the TSF design, any changes to the operating throughput and percent solids of the tailings may impact water demands.

18.4.2.10.4 Engineered Soil Cover

After decommissioning, the final soil cover for the tailings surface will be confirmed during operation based on ongoing operational tailings geochemistry testing results. The closure capping profile has been assumed for this Study and will be reviewed after the tailings geochemistry testing is completed. Based on the arid Project climate, it is anticipated that a store-and-release cover will be the most appropriate cover solution.

18.4.2.10.5 Operating Embankment Downstream Profile

If the TSF embankment Zone C material comprises coarse, clean rockfill of high strength sourced from the open pit, the stability of the TSF embankment downstream face using a steeper slope may be considered, subject to confirmation by a stability assessment once the rock fill properties are known. The final profile of 4H:1V (overall) is required for rehabilitation purposes.

Due to the liquefaction potential of the embankment foundation, this is not considered to be likely.

18.4.2.10.6 Availability Of Mine Waste

Design of the TSF is based on structural fill material being sourced from the open pit mining operations for Stage 1 and construction of future raises. If waste is not readily available during the Stage 1 construction, additional borrow areas will be required in proximity to the TSF. Although this is possible, the capital cost will increase significantly. Utilising a civil earthworks fleet to win material from the Open Pit footprints may prove to be uneconomical due to the long haul distances. In this scenario, material may be sourced from within the TSF basin area, which may offset some of the increased costs by providing additional capacity within the TSF (thus reducing the embankment fill volumes). Geochemical testwork, specifically ARD, of the TSF basin materials is recommended for this opportunity. Based on the current mining schedule there will be sufficient benign mine waste for all stages of TSF construction. Material placement should be carefully planned as part of mining operations to ensure waste placement is sufficiently advanced to allow stage construction of the upstream embankment zones and avoid double handling of mine waste.

Likewise, suitable low permeability fill material may be stockpiled by the mining operation at locations in close proximity to the TSF embankment, for use by civil contractors in future stages. This may reduce civil earthworks rates during future raise construction. The availability of low permeability material sourced from the open pit is not expected, however it will be reviewed when data are available.

18.4.2.10.7 Tailings Geochemistry

Static geochemical testing was carried out on a single tailings sample in 2015. Further static geochemical testing has been completed on tailings samples to assess the differences in the tailings generated from different mining areas and confirm the historical results. Following review of the updated geochemical testing and review of the results, further kinetic testing should be considered.

Further geochemical testing of the tailings should be conducted at points throughout the life of the facility (nominally within the first year of operation and then every 2 years thereafter) to ensure that initial testing remains valid. Measurements will need to continue as part of ongoing operations to ensure information is available on the geochemical and physical behaviour of the tailings.

If tailings are confirmed to be potentially acid generating (PAG) and/or metal leaching (ML), then the proposed seepage control measures should be reviewed. It may be possible to separate the PAG +/- ML material and non-acid generating (NAG) non-metal leaching (NML) materials into a separate tailing stream that can be stored in separate cells. Alternatively, the PAG material could be stored on the heap leach pad, which will be lined with HDPE and could provide incremental metal recovery. The necessity of feasibility of separating the tailings into a PAG/ML and NAG/NML stream, if required, should be reviewed by HCH in the next phase of design.

18.4.2.10.8 Survey Data

Inaccurate base survey is a common cause of variations between expected and actual quantities, particularly in reference to bulk fill earthworks volumes. Topographical contours at times can be generated with small amounts of survey pickup, and as such there is significant interpolation by computer programs.

Accurate basin pickup is required as this will have a significant impact on both bulk fill volumes and the storage capacity of the facility. If there is less storage capacity than currently designed, an earlier start to Stage 2 construction may be required in order to continue to provide the required stormwater storage capacity. It is recommended that a stripped ground survey be completed during construction of the TSF to reconcile the required embankment heights prior to embankment construction. This may result in some cost reductions (or increases).

18.4.2.10.9 Low Permeability Fill (Zone A) Availability

Limited naturally occurring low permeability materials have been identified across the Project area. Further geotechnical investigations will be conducted to identify potential sources of low permeability materials. The current base case assumption is that no materials will be available, however if a borrow location in close proximity to the Project can be identified it may be possible to reduce costs.

If low permeability materials are unavailable, screened alluvium won from the TSF basin could be considered as an alternative. A desktop evaluation of the laboratory results will be conducted to determine if screening of the basin alluvium achieves the target material specification. Screening trials will be required to determine if screening can achieve the target material specification. If the desktop evaluation demonstrates that this is possible, then this will be adopted as the base case for the Feasibility study.

After Stage 1, the use of tailings as a construction material should be considered. Further technical evaluation will be needed to confirm that the material will be suitable as a construction material.

It is noted that the requirement to line the TSF with low-permeability material will significantly increase TSF construction costs. The use of geomembranes as the basis to reduce seepage from TSFs is not permitted in Chile, so they should not be considered as the primary seepage control measure even if adopted.

The current base case is that a composite liner system of HDPE liner and a GCL will be used due to the lack of naturally occurring low-permeability materials.

18.4.2.10.10 Seepage

The TSF was designed to capture seepage downstream of the TSF via a series of abstraction bores; this is the primary seepage control measure for the facility. In addition, a partial basin liner comprising a GCL is proposed as a secondary seepage measure, which will work in conjunction with a cut-off trench, toe drain and underdrainage to reduce seepage. Seepage is still expected from the facility and will be managed by the abstraction bores downstream.

A groundwater model will need to be developed as part of subsequent design to specify the bore locations and installation depths to ensure an adequate level of performance is achieved. The impact to downstream users of groundwater in contact with TSF seepage needs to be evaluated and understood, to ensure adequate contingency is incorporated into the design. At this stage, analysis by HCH baseline studies confirmed that there are currently no dwellings present downstream of the TSF, with the closest being 14km to the northwest. Other receptors in the region include the Mariposa operation and other abandoned mining sites. As of the date of this Report, HCH was unaware of any users of the groundwater downstream, however, this will be further investigated in future study analysis.

Water quality from the TSF must be monitored closely during operation and closure to ensure that performance criteria are being met. Should performance criteria not be met, a response plan should be developed and implemented to address the issue.

18.4.2.10.11 Seismicity

A preliminary seismic hazard assessment has been completed for this phase of design. An in-country seismic specialist has been engaged to conduct a site-specific seismic hazard assessment, which is in progress. It is anticipated that further characterisation and site investigations will be required to refine the seismic loadings determined for the TSF as part of this initial assessment. If the seismic loadings are determined to be larger than initially estimated, the embankment slopes may need to be flattened, and additional buttressing could be required, which will increase costs.

18.4.2.10.12 HDPE Liner and GCL

Based on discussions with HCH, the installation of a geosynthetic clay liner overlain by an HDPE liner is proposed as an alternative basin treatment within the TSF basin (3H:1V and flatter). It is noted that the incorporation of these will increase costs significantly; however, while it is not a primary seepage control measure, it will help lower seepage and improve water recoveries. Therefore, a cost-benefit analysis and risk assessment should be conducted to evaluate the advantages and disadvantages of the proposed basin treatments.

It is noted that the installation of just an HDPE liner or GCL may be appropriate, which will reduce costs, whilst achieving a similar design intent.

The requirement for HDPE liner and GCL should be reviewed as the updated tailings geochemical testing and groundwater modelling are completed.

18.4.2.10.13 Buttressing

A minimum crest width of 24 m and 4H:1V profile is required to achieve the target factors of safety. A final crest width of 27.5 metres and 5H:1V overall buttressing profile are proposed to improve the factor of safety of the facility.

If sufficient mine waste is available, further buttressing could be considered. The haul distance from the Productora open pit is 5 to 6 km. Therefore, the additional overhaul costs are deemed reasonable. HCH should consider the additional haul costs and an additional factor of safety to determine whether further buttressing is needed or desirable to manage the facility's risks.

If the width and slope of the buttress were to increase, then the downstream abstraction bores will need to be relocated further downstream or alternatively raised vertically. The embankment was adjusted to allow the positioning of the abstraction bores based on the better underlying foundation conditions; therefore, a change in the buttressing will impact this design (or the boreholes can be located within the buttress footprint).

18.4.2.10.14 Foundation Liquefaction

Due to the region's seismicity and nature of the alluvium and fractured bedrock, the liquefaction of the embankment foundation was considered possible under post-seismic loading conditions. This was assumed to occur where the alluvium below the embankment was saturated. Based on preliminary steady-state seepage modelling with no HDPE liner or GCL, the alluvium was shown to be partially saturated, and adequate factors of safety can be achieved with the design profiles discussed above. If HDPE liner and a GCL are incorporated into the seepage model the factors of safety improve. The seepage modelling will be reviewed following the second phase of site investigation to confirm the findings and ensure that adequate factors of safety are achieved. The methodologies adopted are considered conservative based on available information. However, further verification of the foundation conditions and site-specific seismicity is needed as part of the subsequent design. If slopes need to be flattened further or buttressing increased, to achieve compliant factors of safety, then costs will increase.

The removal of the liquefiable material within the embankment foundation could be considered. However, it is envisaged that this may not be practical due to the depth of alluvium, resulting in increased costs. An additional geophysical survey of the revised embankment alignments in conjunction with additional boreholes are planned to estimate the volume of alluvium and determine if this approach is practical. This is proposed as part of the Phase 2 investigation.

18.4.2.10.15 In Pit Deposition

If Pit deposition were not practicable, additional capacity within the conventional TSF will be needed to accommodate the entire 500 Mt of capacity, which will increase costs. A preliminary assessment indicated sufficient capacity is available within the current TSF location for the entire 500 Mt. However, further technical evaluation will be needed to assess the viability of this expansion. It is noted that expansion to 500 Mt will increase the height of the northern saddle dams, so the deposition plan will need to be amended to push the supernatant pond away from these saddles to reduce the dam breach risk to Vallenar.

A groundwater model will be developed in the next stage to assess the impacts of tailings deposition into the Productora Pit. The water balance model will be reviewed and updated as additional information is available to assess the implications of in-pit tailings management.

18.5 Security

Access to the site would be controlled by boom gates. Boom gates would secure the Project from tourists traveling in the vicinity as well as unauthorised people attempting to access the restricted areas.

Security would be further strengthened with a manned security gatehouse at the plant entrance. Security inside the mining contractor's area of activities would be the responsibility of the mining contractor.

18.6 Infrastructure Connecting Productora to Cortadera

Sulphide and oxide processing material is planned to be transported 15km, from Cortadera to the Productora site via a rope conveyor.

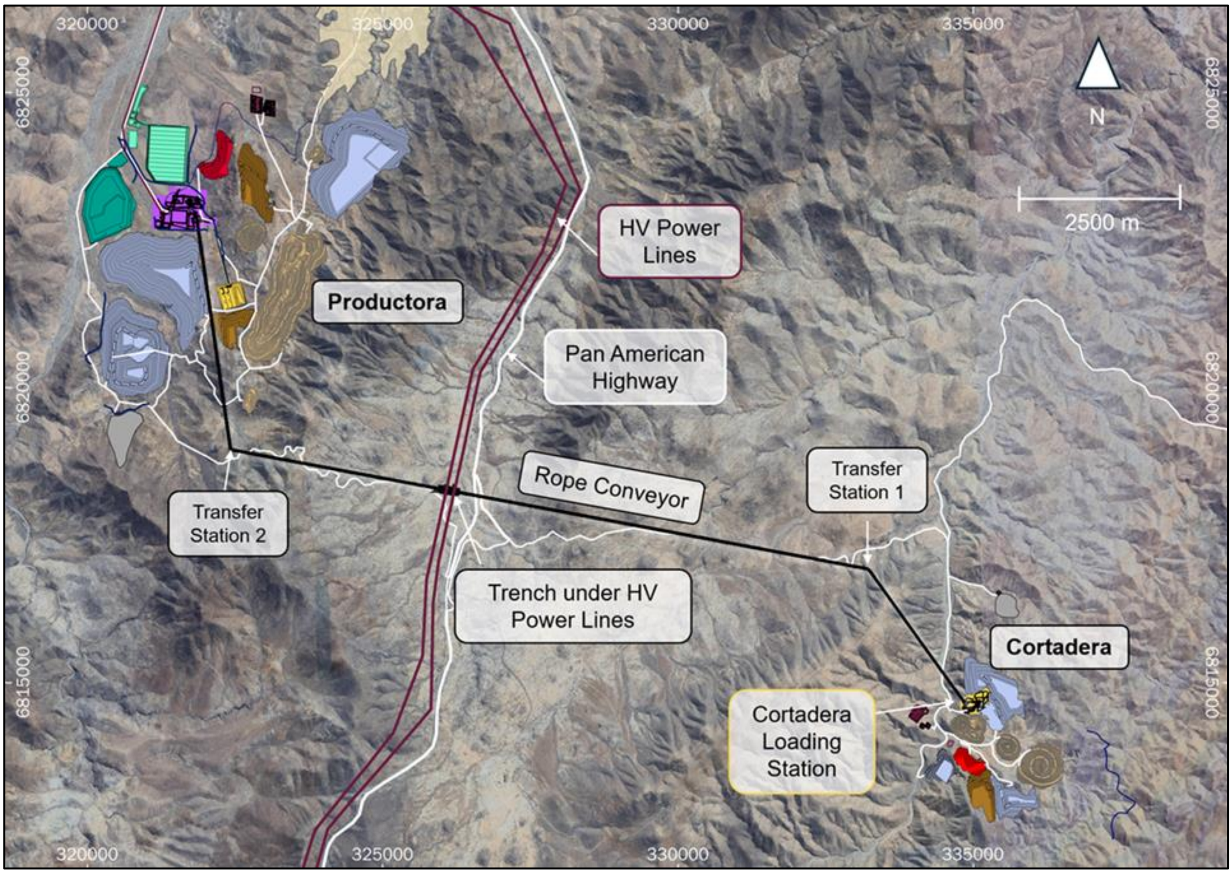
Dopplemayr completed a PFS level Engineering Study to support the design and construction of the rope conveyor (RopeCon). The RopeCon has been designed with a nominal capacity of 25 Mtpa.

The PFS technical deliverables included:

- Technical elaboration of the route, general arrangement of stations and towers, and specifications for the belt, track rope frame and motor rating,
- Foundation loads, and
- Capital and operating cost estimates.

Figure 18.20 shows the infrastructure corridor between Productora and Cortadera to support the following sections.

Figure 18.20 : Rope Conveyor Location, HCH (2025)



18.6.1 Cortadera Materials Handling – Rope Conveyor

Sulphide and Oxide processing material is planned to be transported 15km, from Cortadera to the Productora site via a rope conveyor.

Dopplemayr completed a PFS level Engineering Study to support the design and construction of the rope conveyor (RopeCon). The RopeCon has been designed with a nominal capacity of 25 Mtpa.

This section summarises the key aspects of the documentation provided by Dopplemayr in support of the Engineering Study. The key documents are summarised in Table 18.4.

Table 18.4: Dopplemayr PFS Report

Report Name	Filename
Engineering Study Report	FCA0910PD002_D02_Costa Fuego PFS_Hot Chili_EngineeringStudyReport_20241009
Design Criteria	Design Criteria Costa Fuego RopeCon - PFS (D_P_TM_2030 - FCA0910TE001 - EN - D04) - 1
Electrical Equipment	Main electrical equipment (D_P_TM_2030 - FCA0910TE400 - EN - D02) - 1
Capital Cost Estimate	GRRA-24-0086-01_FCA0910PD003_Budget Quotation_PFS
Operating Cost Estimate	FCA0910PD004_CostaFuegoPFS_Hot Chili_RopeCon_GRRA-24-0086-01_Estimated_OPEX_20240927
Supporting Documents	Included in Table 18.7

18.6.1.1 Description

RopeCon offers the advantages of a ropeway and combines them with the properties of a belt conveyor. The system therefore opens up new perspectives in material transport. The line structure with track ropes allows for long rope spans to cross obstacles of all kinds. For some areas of the RopeCon the track ropes are guided across lower tower structures.

The detailed description of the RopeCon provided in Section 5.2 of the Engineering Study Report is summarized below.

18.6.1.2 Line Structure

Six fully locked steel wire track ropes form the line structure of the RopeCon system. Track rope frames are mounted at regular intervals to keep the six track ropes in alignment. The track ropes which form the line structure on which the running wheels of the conveyor belt travel are tightly tensioned. The ropes are anchored via a separate concrete block or the track ropes can be anchored in a station building designed for the purpose.

To ensure sufficient clearance the track ropes are guided over towers. Depending on the terrain, different types of towers could be used.

The RopeCon transports the material on a continuous cross-reinforced flat textile or steel cord belt with corrugated side walls. The corrugated side walls are either bonded or vulcanized to the belt to a height determined by the material to be transported. The conveyor belt performs a haulage function and is driven by a drive and gearbox arrangement and is equipped with two independent mechanical braking systems.

At the return drum the material is unloaded (discharged) from the conveyor belt. When the conveyor belt has passed the unloading point, the belt turning unit turns the belt by 180° to bring the soiled side of the belt upwards again. This prevents soiling of the track. The conveyor belt is turned once more before it reaches the loading point.

18.6.1.3 Belt Guidance

The RopeCon conveyor belt is screwed onto axles arranged at regular intervals which support the conveyor belt. A polyamide running wheel is fitted to each end of an axle. Together, the axle and its running wheels form a wheel set. These moving parts are attached to the conveyor belt and not to the support structure. Therefore, they pass through the stations at regular intervals.

The running wheels travel on the track ropes guiding the conveyor belt. The running wheels of the top belt travel on the central track rope pair and the running wheels of the bottom belt travel on the bottommost track rope pair. The uppermost track rope pair provides additional stability and forms the track for the inspection vehicle and RopeCon roofing is also clamped to them.

18.6.1.4 Roof cover for conveyor belt

The top belt and is fitted with a roof cover to protect the material on the belt from the effects of the weather. The roof cover is fixed to the line structure and also allows the High Voltage Cables (HCH to supply) to be laid on the valleys of the corrugated sheet roofing.

18.6.1.5 Material containment

The rope conveyor transports material on a flat belt with corrugated sides (an example photo in Figure 18.21). Material remains stationery once loaded onto the belt for the duration of its transit.

18.6.1.6 Protection plates

Shielding plates (an example photo in Figure 18.22) are able to be fitted where additional protection below the rope conveyor path, such as over road crossings, is required.

18.6.1.7 Safe maintenance environment

The majority of maintenance works can be carried out in a safe environment in the stations. The towers are fitted with platforms and handrails. The system can also be fitted with an inspection vehicle in which any point along the line can be reached for inspection purposes.

With the inspection vehicle it is possible to travel to any point along the track to carry out inspections and/or perform maintenance work. The inspection vehicle travels on the uppermost track rope pair. It offers room for maintenance personnel and tools. Its drive operates independently of the RopeCon drive.

18.6.1.8 Operation

RopeCon installations have been operating in a number of countries across the world for several years (see Engineering Study Report). Safety is of the utmost importance to Doppelmayr. Doppelmayr is certified according to ISO 9001:2015, ISO 14001:2015 and ISO 45001:2018.

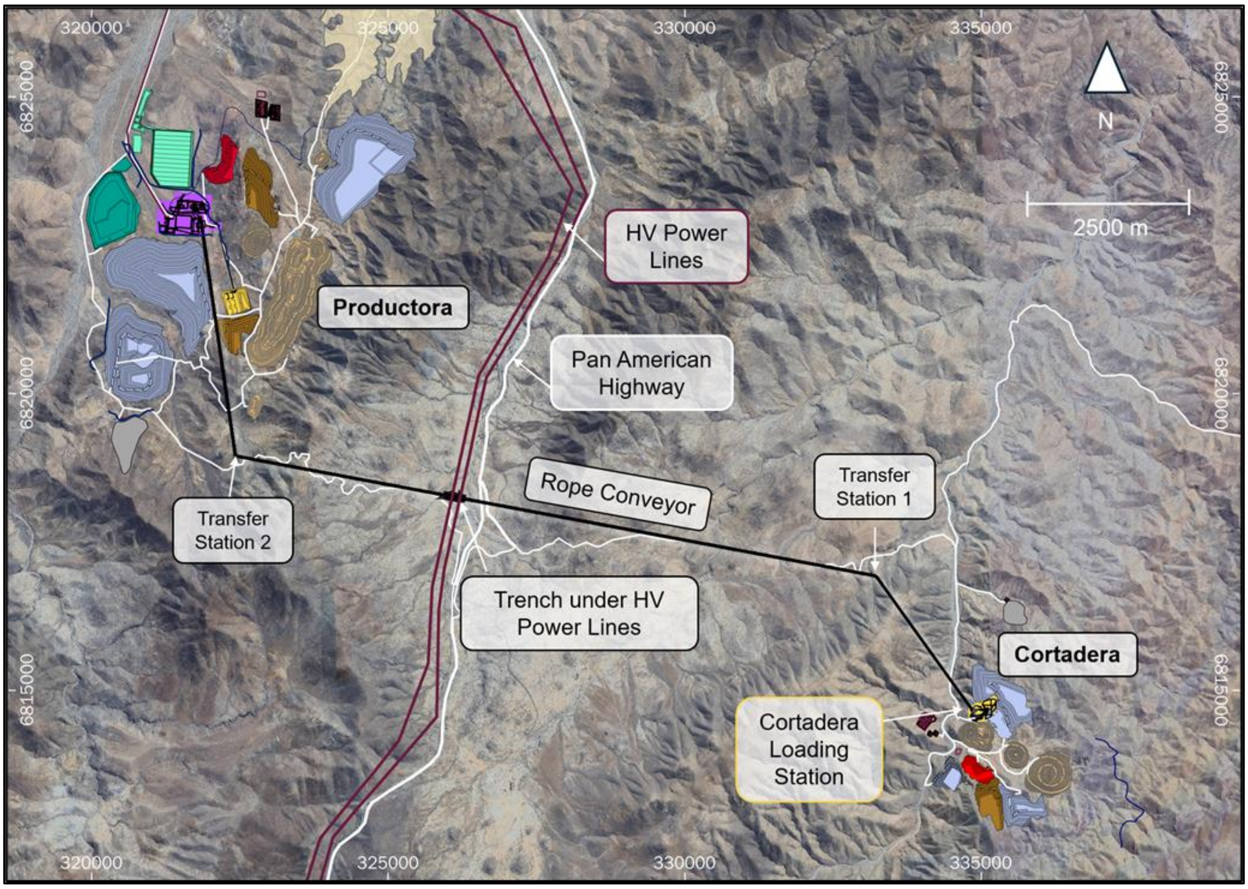
Figure 18.21 Close up view of Rope Conveyor belt with corrugated side walls and track system

Figure 18.22 Rope Conveyor installation example including shielding plates above roadway

18.6.1.9 Route

The proposed route for the RopeCon presented in Figure 18.23.

Figure 18.23 Rope Conveyor Location



The final arrangement includes four sections, shown in and Figure 18.25 to Figure 18.27.

- Section 1 – Cortadera Loading Station to Transfer Station 1 is 2.4 km long and includes an elevation difference of 74m.
- Section 2 – Transfer Station 1 to Transfer Station 2 is 11 km and includes an elevation difference of 66m. This section is designed as a hybrid RopeCon Low-Line Structure so it can cross highways, secondary roads, and high voltage powerlines.
- Section 3 – Transfer Station 2 to Distribution Conveyor at Productora is 3.8 km long and includes and elevation change of 265m.
- Section 4 - Distribution Conveyor to separate Sulphides and Oxides Stockpiles at Productora is 12m long and includes an elevation change of 13m. The Section 4 conveyor is planned to be supported on the track ropes of RopeCon Section 3, allowing discharge onto both ore stockpiles.

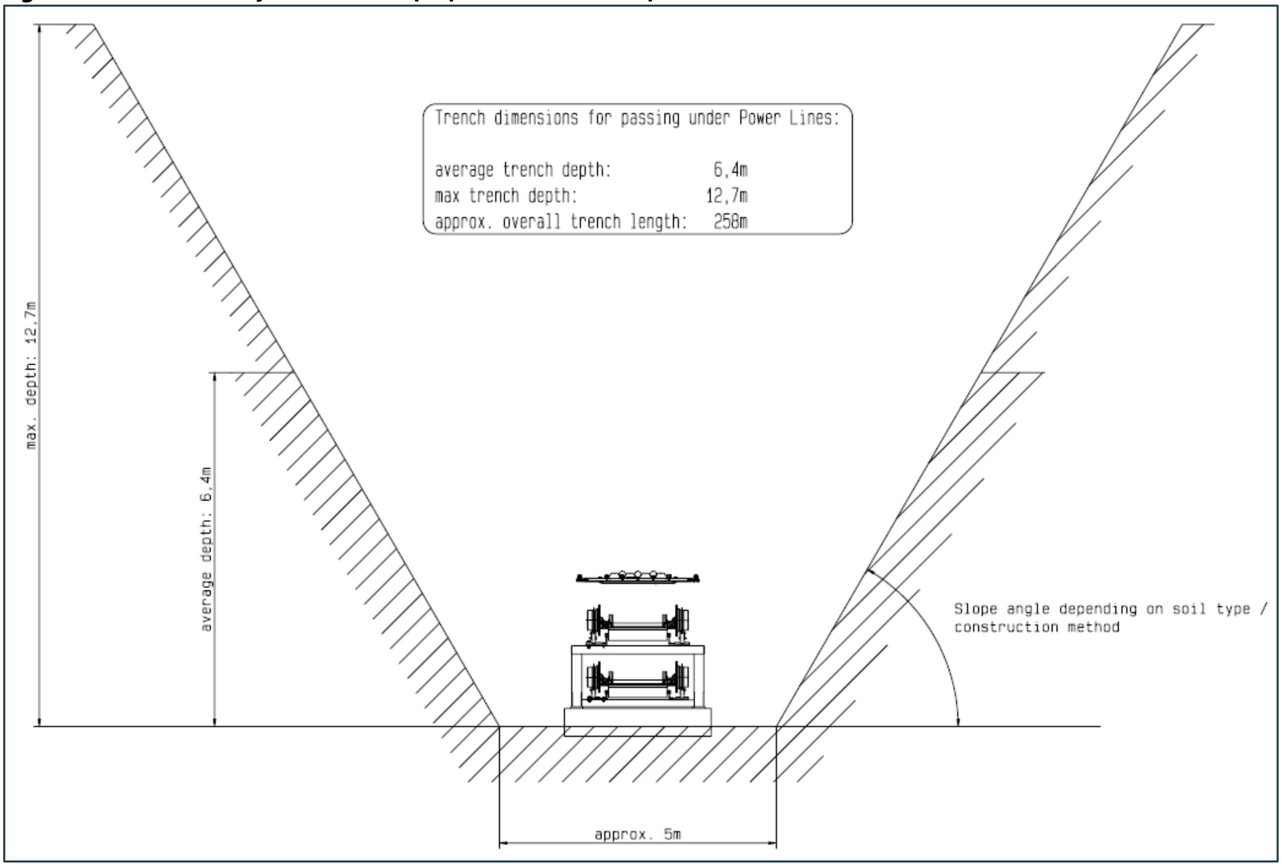
The intended route underwent multiple iterations, taking into consideration environmental sensitivities identified during the winter and spring environmental baseline surveys, interactions with high voltage powerlines and the Pan American highway, topography and anticipated stockpile dimensions.

18.6.1.10 Pan American Highway / Ruta 5 Highway

The design includes the following when considering a RopeCon design to cross the Pan American Highway:

- Ground clearance with Ruta 5 highway of 10 m.
- Clearance between RopeCon and the auxiliary road running in parallel to Ruta 5 of 7 m
- To achieve the safety clearance between the high voltage power cables and the RopeCon: 9.5 m for the 550 KV line and 7.8 m for the 220 KV line, the RopeCon is designed to be placed on an approx. 260-m-long trench, Figure 18.24.
- New power line close to the Ruta 5 highway: HCH plans to bury this new powerline, with the RopeCon designed to be approximately 10 m above the buried powerline (same ground clearance as for Ruta 5 Highway)

Figure 18.24: Preliminary Details of the proposed trench for RopeCon Section 2



The PFS included a trade-off study for an alternate underpass method to the Pan American Highway, discussed in Section 24, and a final design will be reviewed prior to the commencement of a definitive feasibility study.

Figure 18.25: Preliminary Longitudinal profile for RopeCon Section 1

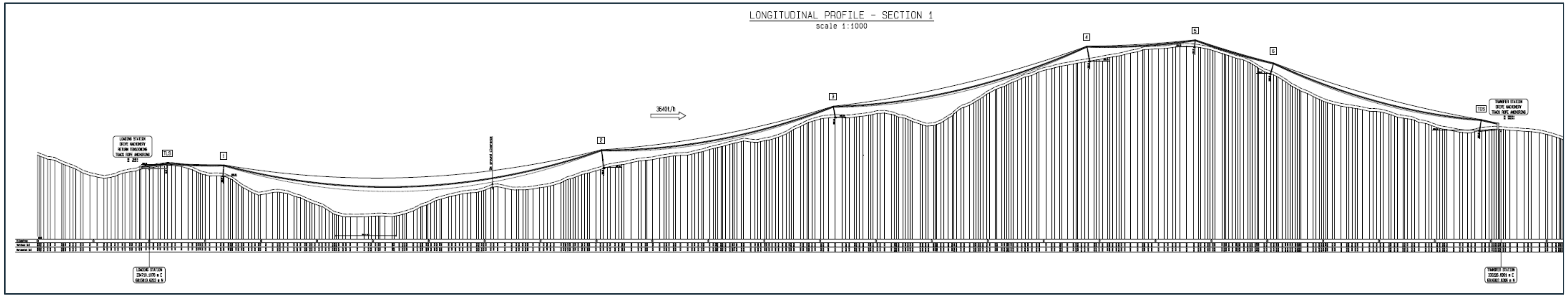


Figure 18.26: Preliminary Longitudinal profile for RopeCon Section 2

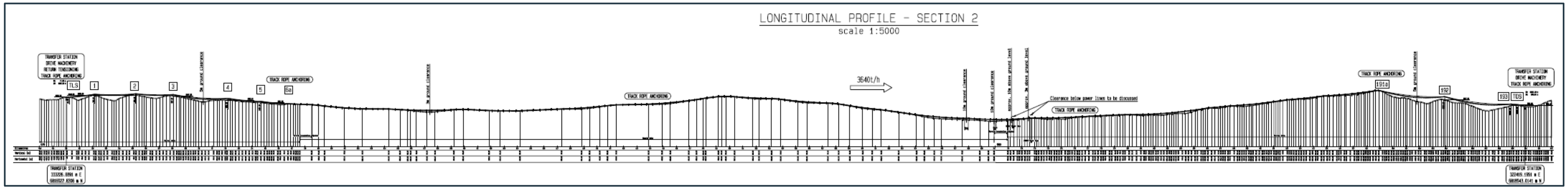
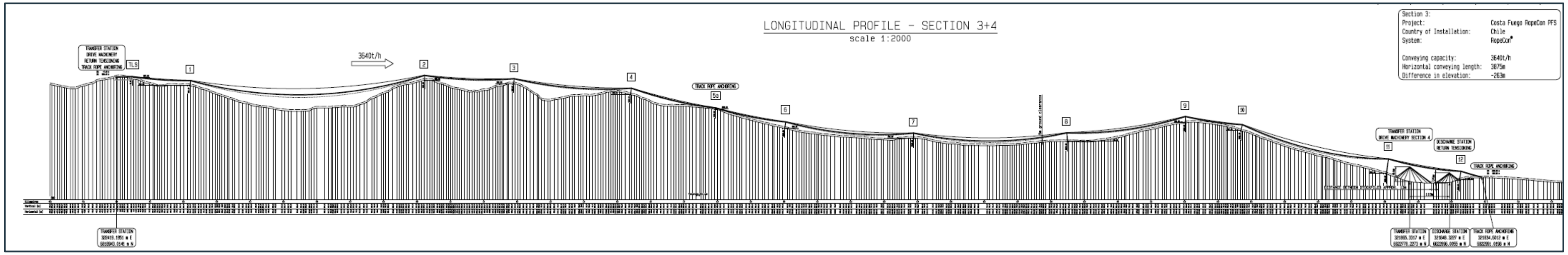


Figure 18.27: Preliminary Longitudinal profile for RopeCon Section 3 & 4



18.6.1.11 Specifications

The material specifications for each section of the rope conveyor design are provided in Table 1.14.

Table 18.5: Technical Data Design Criteria.

Criteria	Units	Section 1	Section 2	Section 3	Section 4
Horizontal conveying length	m	2,411	10,995	3,875	119
Difference in elevation	m	74	-66	-265	-13
Position of drive		Both sides	Both sides	Loading	Loading
Nominal conveying capacity	t/h	3,140	3,140	3,140	3,140
Maximum conveying capacity	t/h	3,640	3,640	3,640	3,640
Belt speed	m/s	5.2	5.2	5.2	2.8
Maximum grain size	mm	300	300	300	300
Maximum bulk density	t/m^3	2.08	2.08	2.08	2.08
Minimum bulk density	t/m^3	1.5	1.5	1.5	1.5
Belt width	mm	1,050	1,050	1,050	1,200
Height of corrugated side walls	mm	240	240	240	400
Motor rating continuous	kW	1,351	1,694	-1709[1]	-102[1]

[1] This system generates power from braking actions during operation.

18.6.1.12 Design Criteria and Considerations

This section summarises the information presented in the Design Criteria and Electrical Equipment report.

18.6.1.13 Mechanical and Electrical Data

The technical data for the design is summarised on the cross-section drawings in the Dopplemayr report and summarised in Table 18.6.

Table 18.6: Technical Data and Electrical Design

		Section1	Section 2	Section 3	Section 4
Mechanical Power Data					
Start-up	kW	2,556	1,706	-92	-110
Continuous operation	kW	1,358	6,211	-1,690	-160
Braking	kW	-338	-4477	-3,945	-215
Max. Power	kW	4,228	9,201	3,079	176
Min. Power (reg)	kW	~-2,728	-7,392	-4,943	-305
Belt Data					
Belt Type	ST	4850 11T/BT-Y	6550 13T/10T-Y	5420 12T/9T-Y	EP 1250 6T/3T-Y
Belt Width	mm	1,050	1,050	1,050	1,200
Sidewall height	mm	240	240	240	400
Belt utilisation width	mm	894	894	894	1,000
Endless belt length	mm	~4,690	~22,264	~7,982	~242
Track Rope Data					

		Section1	Section 2	Section 3	Section 4
Track rope diameter	mm	2 x 67	2 x 67 / 42	2 x 67	NA
	mm	2 x 67	2 x 67	2 x 67	NA
	mm	2 x 57	2 x 57	2 x 57	NA
Pretension force	kN	~10,100	~10,100	~10,100	NA
Rope Length	m	6 x ~2,487	6 x ~11,055	6 x ~4,150	NA

18.6.1.14 Foundations

Doppelmayr investigated within the PFS-Level Engineering Activities, analysing the loads transmitted by the RopeCon system to foundations under all operational scenarios and environmental loads. The investigation's outcome provides a summary of foundation loads, enabling HCH to preliminarily design the required civil works for the RopeCon system.

18.6.1.15 Dust

The design includes specific features to manage to dust generation. RopeCon chutes at transfer points (loading, transfer and discharge stations) have a hood-and-spoon design to ensure the material speed on the chutes is the same, or close to, the belt speed. This approach minimizing dust generation on the transfer stations.

For the Costa Fuego Project, considering the material provided, no additional dust control system on the stations has been considered. If needed during the commissioning or later operation of the RopeCon system, water spraying systems could be installed in case the material handled would generate dust issues in the stations.

Once material is on the RopeCon belt, it is transported similar to a “carpet”, with no bumping as material passes over the rollers. As there are no rollers along the line no dust is anticipated to be generated. Regarding cross wind and possible dust generation due to it, RopeCon belt has vulcanized sidewalls for transporting the material, which also protects transported material against crosswind. In addition to this, depending on the size and type of the material handled, as well as on wind speed, different types of roofing/cover can be installed along the RopeCon systems. Considering the material size, wind conditions and precipitation, it is proposed that a trapezoidal sheet roofing is used, which has been used in similar operations with similar conditions.

Table 18.7: Doppelmayr Supporting files

Description	**Doppelmayr Ref. No.**
Design Criteria	FCA0910TE001
Route evaluation Costa Fuego	FCA0910TE002
Longitudinal Profile Costa Fuego PFS - RopeCon Section 1	20049738T206001
Longitudinal Profile Costa Fuego PFS - RopeCon Section 1	20049739T206001
Longitudinal Profile Costa Fuego PFS - RopeCon Section 3 + 4	20049740T206001
Layout A-Frame Tower Section 1	20049763T206001
Layout A-Frame Tower Section 2	20049764T206001
Layout A-Frame Tower Section 3 + 4	20049764T206001
Layout Ground Based RopeCon System Section 2	20049766T206001
Layout Loading Station Section 1	20049787T206001
Layout Loading Discharge Section 1	20049788T206001

Description	Doppelmayr Ref. No.
Layout Loading Station Section 2	20049789T206001
Layout Loading Discharge Section 2	20049790T206001
Layout Loading Station Section 3	20049791T206001
Layout Transfer Station Section 3 + 4	20049792T206001
Layout Loading Discharge Section 4	20049793T206001
General Arrangement	20049817T206001
Example Layout Track Rope Divide	20049836T206001
20240917_FCP910_CostaFuegoRopeCon-PFS.kmz	--
Coordinates of Tower Foundations	FCA0910TE600
Foundation Loads Section 1	FCA0910TE003
Foundation Loads Section 2	FCA0910TE004
Foundation Loads Section 3 + 4	FCA0910TE005
Main Electrical Equipment	FCA0910TE400
Budget Quotation (+/-25% CapEx Estimate)	FCA0910PD003
Estimate of Operational Expenditures (OpEx Estimate)	FCA0910PD004
Final Engineering Study Report	FCA0910PD002
DOPPELMAYR Main Standards for Design	T230AD007E

18.6.1.16 Cost Estimate

The capital cost estimate based on the design described above is provided by Doppelmayr in the Engineering Study, GRRA-24-0086-01_FCA0910PD003_Budget Quotation_PFS, is Euro 126M. The includes supply, installation and delivery as defined in the Scope of Supplies and Services.

18.6.2 Productora to Cortadera Roads

An access road has been aligned to follow the rope conveyor, the road is to be constructed following the route as shown in Figure 1.19.

The road is planned to be non-paved road.

18.6.3 Power to Cortadera

The PFS has allowed for power to be supplied from Productora to Cortadera using a 3x 1C 220 kV cable installed above the rope conveyor roofing structure at 220 kV.

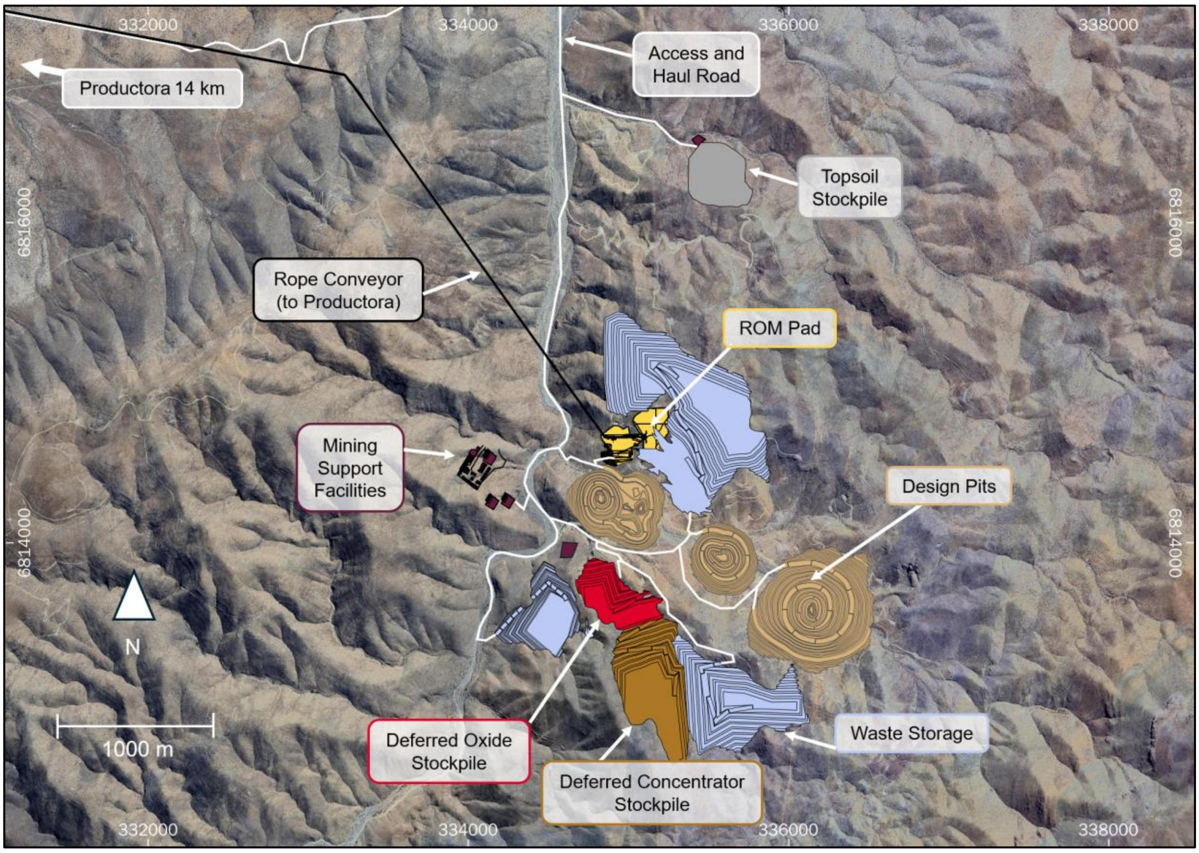
18.6.4 Water Supply to Cortadera

The PFS has allowed for a water distribution of seawater from Productora seawater storage ponds to a storage pond set water, to be supplied from Productora to Cortadera. The seawater consumption is calculated at 488 m^3/h from the Concentrator Sea Water Pond via a steel HDPE lined pipeline and a multi-stage pumping system (duty/standby) rated at 750 m^3/h to the Cortadera Sea Water Pond which has a capacity of 20,000 m^3. The Cortadera Sea Water Pond is located behind the open pit.

18.7 Cortadera Infrastructure

Figure 18.28 shows the planned infrastructure at Cortadera.

Figure 18.28 Cortadera Surface Infrastructure (HCH, 2025)



18.7.1 Mining – Open Pit

The mine surface infrastructure area, located near the mine as part of the mine surface facilities for the area would be provided by the Mining Contractor. The Project has included costs for:

- Explosive stores
- Fuel farm (heavy vehicles, mobile trucks for refuelling and a separate light vehicle fuel farm would be installed to serve non-mining vehicles)
- Utilities (water, sewage/waste management)
- Cortadera Materials Handling
- Block Cave Underground Crushing.

18.7.2 Mining – Block Cave

Refer to section 16.4.15 for Block Cave infrastructure.

18.7.3 Cortadera Open Pit Crushing

Refer to section 16.4.17 for Material Handling Design – Crushers.

18.7.4 Buildings

The Project would include administrative and auxiliary buildings located in the proximity of the Cortadera Crushing Plant. These buildings have been specified as transportable buildings to be installed for the Project, for the following offices:

- 5100-BD-001 Cortadera Crushing Control Room
- 5100-BD-002 Cortadera Office + Crib + Changeroom
- 5100-BD-003 Cortadera Workshop + Warehouse

The buildings have been sized based on planned operational requirements.

18.7.5 Water

Water from the Cortadera Sea Water Pond is then distributed as follows:

- Sea water for dust suppression 203 m^3/h
- Cortadera RO Plant 285 m^3/h

From the RO plant water is distributed to:

- Primary Crusher Sprays
- Mine Wash
- Underground Mining

The brine is stored in the Cortadera RO Brine Pond and then used for Dust Suppression.

A separate Potable Water treatment package for the offices and workshops at Cortadera has been allowed for, as well as a fire water tank and fire water pump package.

18.7.6 Power

Power is received at Cortadera via the 220 kV cable which is installed on the rope conveyor. The cable is then routed to the switchyard which is sited on the hill above the loading station. The switchyard then distributes power transformed from 220 kV to 33 kV to the various load centres via the 33kV switchboard. The load centres are noted as follows:

- Cortadera Primary Crusher
- Cortadera Office, Workshop and change area
- Seawater pond
- Mining Offices and Warehouse

- Underground Portal
- Open Pit Perimeter

The PFS assumes that the magazine will be powered by a diesel generator given the minimal load and remote location.

18.7.7 Air Services

A compressed air package with a receiver, dryer and filters for distribution for the site services has been allowed as part of the PFS. In addition, an instrument air receiver has also been allowed for any instruments.

18.8 Port Concentrate Storage and Loadout

18.8.1 Introduction

An existing port facility would be utilised for receival, storage, reclaim and ship loading of copper concentrate. The facility would require upgrading to handle the volume of concentrate to be stored and shipped.

New facilities required to be constructed for the Project are described in further detail in the following sections, but encompass the following:

- Access roads
- Concentrate storage yards
- Conveyors from concentrate storage yards via wharf to ship loader facility
- Ship loader facility.

18.8.2 Port Facilities

The Port would include facilities for unloading, storage and loading of the concentrate. Trucks would deliver the concentrate from the process plant to the concentrate storage shed via a weighbridge at the security gate at the entrance of the port area.

The truck would tip the concentrate onto the floor of the shed to reduce dust, where the concentrate is then moved around by Front End Loader (FEL) to be either directly loaded to a ship or stacked onto the stockpile waiting for the next ship. The concentrate would be loaded via the stockpile the material reclaimed via FEL into the reclaim hoppers to loaded onto the port reclaimer conveyors and then onto the ship loader. If the concentrate is loaded directly from the truck the concentrate would be conveyed via the bypass conveyor and then onto the reclaim conveyors to the ship-loader. As part of the loading the concentrate would be weighed by the weightometer located on the first reclaim conveyor as part of the loading process.

In addition, the following infrastructure is included as part of the Port facilities:

- Concentrate Unloading Area Sump
- Concentrate Storage Shed Sump Pump 1
- Concentrate Storage Shed Sump 1

- Concentrate Storage Shed Sump Pump 2
- Concentrate Storage Shed Sump 2
- Port Wastewater Tank
- Port Wastewater Tank Agitator
- Port Wastewater Pump
- Port Water Tank
- Port Fire Water Package
- Port Water Pump A
- Port Water Pump B
- Port Air Compressor
- Port Air Dryer Filter 1
- Port Air Dryer
- Port Air Dryer Filter 2
- Port Air Receiver

18.8.3 Buildings

To support the operations and maintenance personnel at the port the PFS has included the following buildings:

- 5140-BD-001 Port Office + Crib + Changeroom
- 5140-BD-002 Port Workshop + Warehouse

The buildings have been specified as transportable buildings and sizing is based off the expected number of people occupying the building.

18.9 General Infrastructure

18.9.1 Plant Control Systems (PCS) and Instrumentation

The PCS would be configured as a networked fully integrated system incorporating the sulphide and oxide process plants. The lowest level would comprise field instrumentation and control equipment. The middle level would comprise the process control system hardware. The top level would comprise the operator interface hardware and network interfacing for remote monitoring and administrative reporting.

Field instrumentation would interface to the PCS through remote Input/Output (IO) panels across a device level communications network. Switchboard devices for monitoring, such as protection relays and variable speed control panels, would also interface to the PCS by means of a device level communications network.

A Supervisory Control and Data Acquisition (SCADA) system allowing plant operation and calibration across all areas is proposed. The SCADA system would be designed with multiple servers for redundancy of operation and distribution of tasks. The system would include a historian for data logging purposes and change management system for security over programs and configurations.

The control philosophy would be to provide a comprehensive automated start up and shut down of all plant areas. Automatic interlocking, sequence control and analogue control would be implemented by the PCS equipment. Safety interlocks would be hard-wired.

The PCS would provide detailed information including:

- Plant status monitoring
- Fault annunciation and logging
- Management reporting.

The PCS hardware would be powered by uninterruptable power supply (UPS) equipment, providing smooth, fully synchronised power for 30 minutes after total power failure to allow for controlled shutdown of PCS equipment prior to complete loss of power.

The plant would normally be controlled from a main operator control room via the PCS.

18.9.2 Communications and Data Systems

As of the date of this Report, there was no telecommunication infrastructure in the immediate mine site area. It is expected that telecommunications would be established by connecting into available facilities in the Vallenar town vicinity.

The communications infrastructure would be built on a fibre optic backbone including some redundancy. This would ensure reliable communications to all parts of the operations. Another loop would interconnect the main sulphide plant area buildings (including the administration building), ensuring a fault tolerant ring that connects all buildings. The oxide plant buildings would be connected to the sulphide plant area buildings via another fibre optic link. Multi fibre optic cables would be specified to separate the dedicated fibre systems and ensure speed and reliability of data from each system.

The general data and telephone network would have dedicated fibre to allow all facilities to be integrated and to be able to use IP telephony. The data and telephone network would include wireless routers in the administration areas in order to provide a wireless data network (Wi-Fi).

The mining contractor would be responsible for the provision of communications infrastructure for its facilities. A conventional very-high frequency (VHF) radio system with handheld radios and chargers would be provided for site coverage. Radio communications would be via separate channels for mining and process plant. There would be a separate, dedicated emergency channel.

18.9.3 Closed-Circuit Television (CCTV)

The process plant, TSF, port and mine would utilise a CCTV system to provide coverage of the operating areas of the site. A separate CCTV surveillance system would also be installed for use by the security personnel to monitor the periphery of the mine site.

The CCTV system would consist of movable Pan-Tilt-Zoom (PTZ) cameras equipped with protective housings for outdoor industrial environments, and the dome type for indoor use.

All cameras would include native Internet Protocol (IP) technology and high-definition resolution transmitting over optical fibre links in a video traffic network, with security protocols allowing access to this information only to authorised personnel.

18.9.4 Site Water Management

The water management plan includes PFS designs of infrastructure required to manage the water collected inside mining areas (mainly open pits and underground mine inflow and precipitations run-off) and the surface deviation channels required around projected mining areas.

In the case of the Waste Rock Dumps (WRD), the semi-arid climatic conditions will limit the potential seepage from the WRD. Based on existing mining projects in the region, there's generally no requirement for emergency ponds, however a specific study required by the Chilean regulator (DGA and Sernageomin), will be completed in the next stage of the Project.

The water in the Project area is classified as:

- Non-contact water: surface water and/or groundwater that doesn't contact mining material. This category generally includes surface water of the catchment areas required to be control through deviation channels.
- Contact water: surface water and/or groundwater in contact with mining material which can affect its chemical quality.

The water management plan is designed to minimize contact water and separate from non-contact water based on:

- Surface catchment areas differentiated on a micro-basin scale
- Estimation of maximum daily precipitation in each micro-basin.

18.9.4.1 Pumping requirements for Costa Fuego pits

During operation, contact water accumulated in the pit floors will be required to be continually evacuated from the pit, facilitating the mining operations and limiting the hydrogeological recharge after rainfall events (Table 18.8).

- A sump and pumping system recommended to manage in-pit groundwater inflow.
- Additionally, standby pumping equipment have been considered for sporadic precipitation events, to manage water during these rare events (2 days a year).

Table 18.8: Pumping requirements across the Costa Fuego Project

Open Pit	Sump and Pumping System requirement	Standby pumping requirement
Productora	Up to 25 l/s	100 l/s
Alice	Up to 5 l/s	40 l/s
Cortadera	Up to 15 l/s	100 l/s

At San Antonio the drilling has largely been dry (ground water not encountered in the drillholes). Hot Chili Limited will be expanding its hydrogeological network to San Antonio within its EIA submission.

18.9.4.2 Emergency contact water ponds for WRD

The semi-arid climatic conditions of the site will limit the potential seepage from WRD infrastructure, which has been confirmed with a benchmarking review from other mining projects in the Coquimbo and Atacama regions.

It is therefore expected that most of the water from precipitation will evaporate or retained in WRD voids (due to its high drainable porosity). In case of extreme rainfall events repetition, water could potentially percolate and produce negligible fluxes at the toe of WRD, or potential for no water discharge at all. Additional analysis is required in future stages to determine whether if trenches and/or emergency ponds are necessary for Productora and Alice's WRDs.

At Cortadera it is not anticipated there is a specific need for emergency contact water ponds, considering the proximity of the WRD to the planned open pits, which will collect the majority of the potential WRD seepage during extreme precipitation events. As open pits are closed and, in some cases, backfilled with waste, emergency contact water ponds will become a requirement.

18.9.4.3 Productora & Alice areas

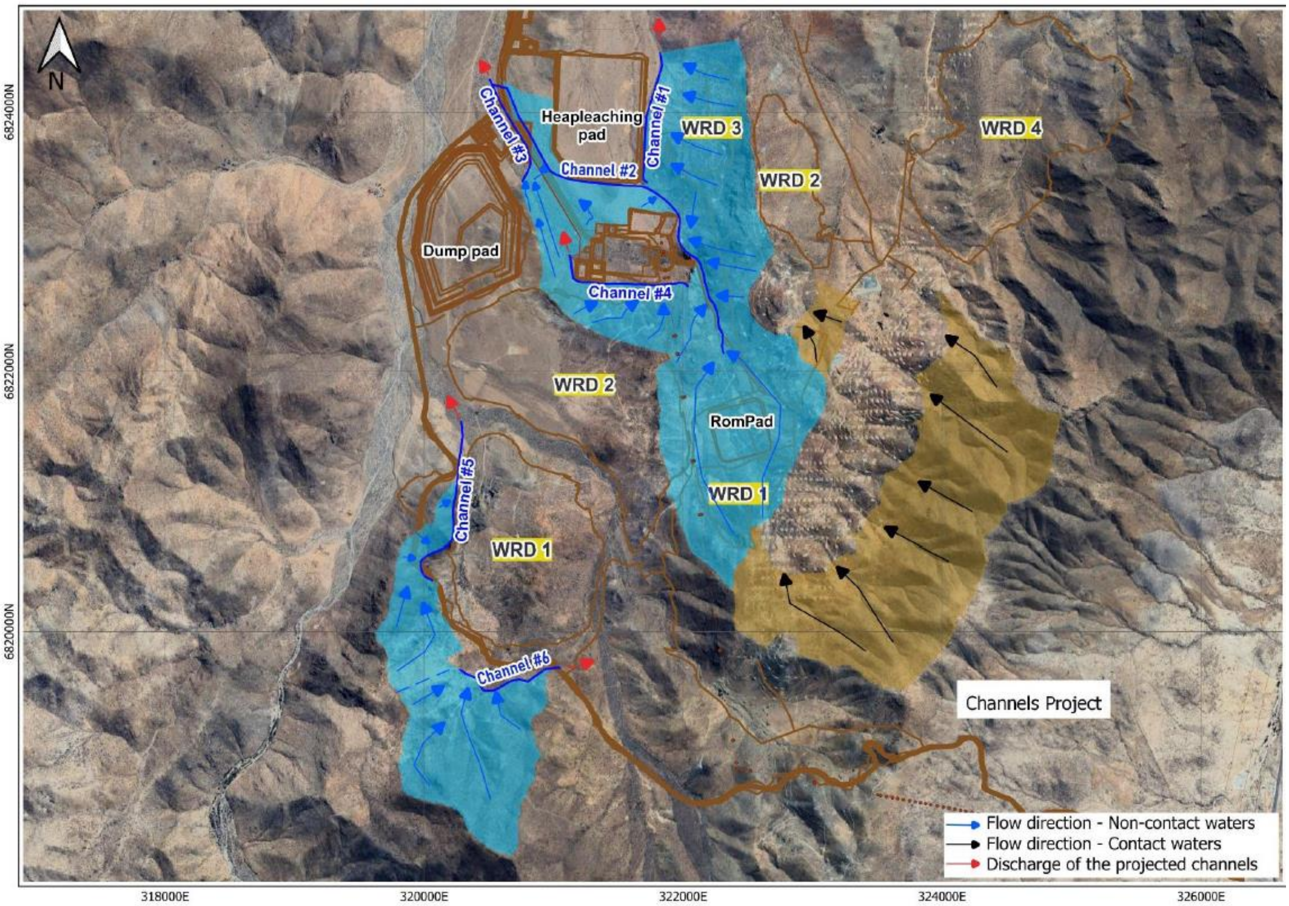
18.9.4.3.1 Layout and conceptual water management plan

The main mining infrastructure of Productora and Alice areas include, as shown in Figure 18.29:

- Productora open pit: final pit area is 1.44 km2 and the pit bottom elevation is 490 masl
- Alice open pit: final pit area is 0.15 km2 and the pit bottom elevation is 585 masl
- Two waste rock dumps (WRD): North and South, with respective areas around 0.70 km2 and 2.02 km2, located on the SW of the area.
- A dump leach pad with an area around 1.45 km2, located on the W side of the area in La Higuera valley.
- Run of Mine (ROM) and heap leaching pads, located on the NW side of the area.

The micro-basins associated with each infrastructure layout are presented below (Figure 18.29), highlighting the areas that contribute to non-contact water (to diversion channels) and contact water (to mining infrastructure).

Figure 18.29. Conceptual water management plan in the Productora and Alice areas (HCH, 2025)



18.9.4.3.2 Non-contact water diversion channels

A series of micro-basin with mining infrastructures are delimited, in which six diversion channels are required to route the non-contact water. The peak runoff flows are estimated for different return periods, based on the three methods proposed the Chilean Regulator DGA (Rational method; Modified Verni-King (MVK) method and DGA-AC method).

18.9.4.3.3 Contact water contribution in Productora and Alice open pits

The maximum contact water contribution to each open pit is the sum of the continuous groundwater inflow and sporadic surface runoff inflow for precipitation events, limited to few days per year (e.g. 2 days for the major rainfalls recorded in 1997) usually occurring from June to August:

- The groundwater contribution to the pit inflow is estimated in 10 to 25 l/s range for Productora pit and in 1 to 5 l/s range for Alice pit.
- Sump pumping to evacuate groundwater seepage and surface water drainage was considered; based on the geotechnical analysis, no specific groundwater drainage or depressurization is anticipated for the open pits.

- Pit floor sumps should be sized to manage operational flows of about 25 l/s. However, it is likely that the pit floors will become completely flooded following extreme precipitation events. Additional pumping capacity should be available to evacuate water following sporadic extreme precipitation events.
- Water extracted from the pits into contact water ponds would be available for other uses such as irrigation of mine haul roads for dust suppression.

18.9.4.4 Cortadera

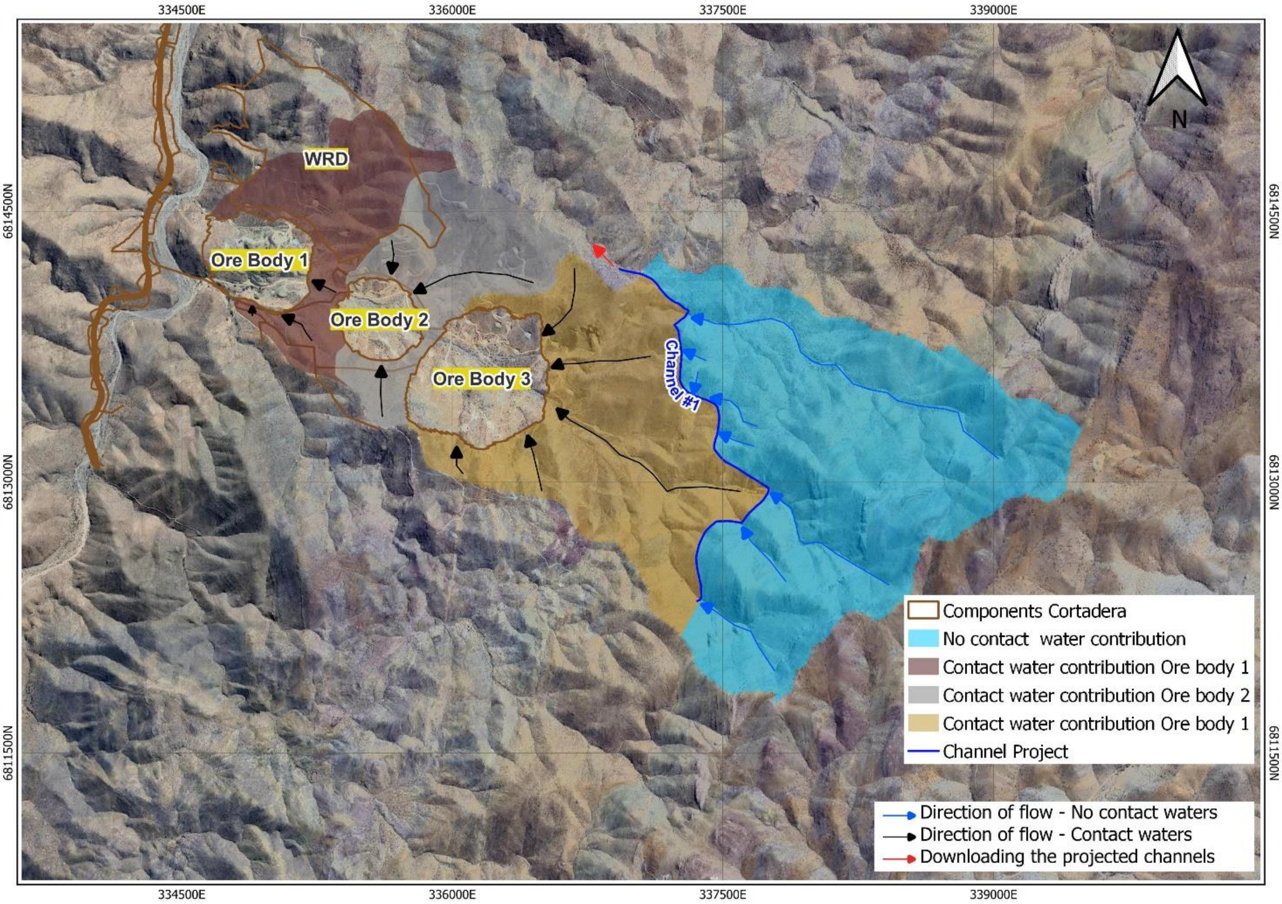
18.9.4.4.1 Layout and conceptual water management plan

The conceptual surface water management plan (Figure 18.30) was designed from an infrastructure layout that included:

- Cortadera Cuerpo 1 open pit: final pit area of 0.26 km2 and pit bottom elevation at 725 masl
- Cortadera Cuerpo 2 open pit, with a final pit area of 0.17 km2 and pit bottom elevation at 810 masl
- Cortadera Cuerpo 3 open pit, with a final pit area of 0.46 km2 and pit bottom elevation at 765 masl
- One waste rock dump (WRD) located in the North side of the area.

Following this design, the layout was adjusted to include in-pit placement of waste rock in Cuerpo 1 and 2 open pits. This change impacted the surface water management plan where drainage into these pits as shown in Figure 18.30 would be compromised, and resulted in the inclusion of a emergency contact water pond between Cuerpo 1 and the quebrada. The micro-basins associated with each infrastructure layout are presented in Figure 18.30, highlighting the areas that contribute to non-contact water (to diversion channels) and contact water (to mining infrastructure).

Figure 18.30 Conceptual Surface Water Management at Cortadera (HCH, 2025)



18.9.4.4.2 Contact water contribution in Cortadera open pits

The contact water contribution to each of the three Cuerpo open pits is the sum of the continuous groundwater inflow and sporadic surface runoff inflow for precipitation events, limited to few days per year usually occurring from June to August:

- The groundwater contribution to the pit inflow is estimated to be 4 to 15 L/s cross the Cortadera open pits.
- Sump pumping to evacuate groundwater seepage and surface water drainage was considered; based on the geotechnical analysis, no specific groundwater drainage or depressurization is anticipated for the open pits.
- Water extracted from the pits would be available for other uses such as irrigation of mine haul roads for dust suppression.

18.9.4.5 San Antonio area

San Antonio is not located in a catchment, as it is at a higher elevation. Surface water will be limited to direct precipitation during extreme events, and deviation channels are not required. Considering the estimated groundwater depth, the inpit inflow is estimated to be low and managed directly with inpit evaporation, without requiring a water extraction system.

18.10 Camps and Accommodation

During the PFS, three options were assessed in terms of accommodation of temporary personnel during the construction phase, and permanent mine site personnel during the operations phase.

The three options considered were:

1. Build a new camp adjacent to the unused agricultural facility near Productora
2. Refurbish the existing buildings and upgrade to camp facilities at the unused agricultural facility near Productora
3. Accommodation of personnel in Vallenar and the surrounding towns

For the PFS , the selected option was that personnel would be accommodated in Vallenar and surrounding towns. Construction rates etc have been updated to reflect this.

Personnel would be transported to the mine site by company bus, excluding where personnel have been allocated a company vehicle.

18.11 Risks and Opportunities

The following risks and opportunities are noted:

Risks

- The TSF powerline re-routing requires consultation with power authorities to progress this work in future stages of the study. Currently a budget allowance has been included for the rerouting to the powerlines.
- Permitting is required for the RopeCon with the local authorities to span the Pan American highway, this permitting activity is planned as a future activity

Opportunities

- Opportunities to use the currently identified brine at Productora are underway, this includes the use of brine for heap leaching (replacing salt as the reagent)
- Opportunity during further detailed studies to optimise the ROM wall height at Cortadera following the completion of the material flow test work and further optimisation of the ROM wall height.

18.12 Comments on Section 18

The QPs note:

- The infrastructure design is in line with the planned mining and processing rates, and is appropriate for a development project within the region

- Borrow pits would be required to provide the materials needed for construction through to mine closure
- Several opportunities for improved tailings management outcomes, including combinations of liners, alternative locations, and reducing the amount of potentially acid forming tailings formed through pyrite floatation, are currently under review for the TSF design
- Water for processing demands would be sourced from the seawater pumping and storage pond facility throughout the life of operations
- Port facilities for the Project are planned to utilise an existing port area. The PFS has considered the construction of new storage and loading facilities with discussions ongoing to secure port access and services.
- No camp is planned for the accommodation of personnel.

19 Marketing

19.1 Executive Summary

The Costa Fuego Project will produce a copper concentrate suitable for smelting and refining at all of the world's major copper smelters which feed primary concentrate.

The copper concentrate produced at Costa Fuego will be relatively high in copper content and low in deleterious elements. It will be sought after by smelters globally due to its quality.

Copper cathodes and molybdenum concentrate will also be produced as by-products at Costa Fuego. Sulphuric acid will be purchased to facilitate ore-leaching and the production of copper cathodes.

It is estimated that the copper concentrate (containing copper, gold and silver) will account for ~ 86 % of gross revenue, with copper cathodes and molybdenum concentrate accounting for ~8 % and ~6 % respectively.

Glencore International AG has committed to purchase 60% of the copper concentrate production for a period of 8 years. The commercial terms governing this sale will reflect prevailing global market terms at the time of shipment (Copper Concentrate Benchmark - annual basis).

Additional sales for the balance of the concentrate production will be made to copper smelters or traders. Sales to traders are expected to attract a material discount to the copper concentrate benchmark, favourable to the Costa Fuego Project.

The treatment and refining charges applied to copper concentrates are at historical lows in 2025. Sales to traders are attracting further discounts favourable to miners, to the extent that the charges have been negative, meaning that traders are paying miners in order to procure the concentrates. This situation is unprecedented in the market and reflects the structural changes which have become evident with the growth in global smelting capacity relative to a paucity of growth in aggregate mine production.

19.2 Key Risks

Risks associated with copper concentrate sales will pertain to the saleability and deliverability of the concentrate. These risks will be both endogenous and exogenous.

Concentrate Quality and Specifications:

- Variability in the concentrate's grade (e.g., copper grades or levels of impurities such as arsenic) can impact its marketability. If the concentrate fails to meet the contractual specification, buyers may reject the concentrate or seek to renegotiate commercial terms.

Offtake Risks: Financial and Performance

- Buyers may not pay for concentrate as agreed
- Buyers may not accept shipments as agreed.

Operational Changes

- Disruptions in mining or processing can impact production and delay deliveries, necessitating changes to agreed delivery schedules
- Storage and handling at the mine site or port of loading may lead to contamination or metal losses.

Contractual Obligations

- Inflexible or poorly negotiated contracts may lead to unfavourable renegotiation of terms if Costa Fuego Project is unable to meet all its obligations.

Market Demand and Pricing

- Copper concentrate prices are influenced by global market conditions, which can fluctuate based on supply and demand dynamics, economic factors, or geopolitical events
- Changes in demand from major importing countries, such as China, can significantly affect sales.

Logistics and Transportation

- Delays or disruptions in shipping due to port congestion, adverse weather, or logistical bottlenecks can hinder timely deliveries.

Regulatory and Environmental Factors

- Stricter environmental regulations, especially concerning the management of byproducts such as arsenic, can complicate the sale of the concentrate.
- Export/import restrictions or changes in tariffs can also affect deliverability.

Geopolitical and Economic Factors

- Political instability, trade disputes, or sanctions in exporting or importing countries could disrupt sales
- Currency fluctuations impact the profitability of international transactions.

19.3 Key Assumptions

19.3.1 Economic & Physical Parameters

The Costa Fuego Project will have a 21-year Mine Life with metal volumes as follows. The production schedule is shown in Section 16.

Table 19.1 : Life of Mine Production

Metal Product	Unit	Life of Mine
Copper in Concentrate	kt	1,446
Copper in Oxide Cathode	kt	93
Copper in Sulphide Cathode	kt	11
Gold in Copper Concentrate	koz	784
Molybdenum in Concentrate	kt	33

The key elements of the Costa Fuego Project concentrate are shown in the table below. Deleterious elements are discussed and shown in Section 19.5.7.

Assumed prices are shown, with consensus price ranges taken from an average of 25 leading banks is shown for payable metals.

Table 19.2 : Commodity Price and Copper Concentrate Quality Assumption					
Element	**Unit**	**Price Consensus**	**Price Used**	**Unit**	**Grade**
Copper	US $/lb	4.29	4.30	%	25.0
Gold	US $/oz	2,278	2,280	g/t	4.2
Silver	US $/oz	27.76	28.00	g/t	14.5
Molybdenum	US $/lb	18.28	20.00	%	50.0
As				ppm	44
F				ppm	Not Detected
Cl				ppm	238
Hg				ppm	1
Al2O3 + MgO				%	2.35

Table 19.3 : Copper Concentrate Sales - Key Commercial Assumptions		
Item	**Unit**	**Value**
Treatment Charge	US $/dmt	54.50
Refining Charge Cu	US c/lb	5.45
Refining Charge Au	US $/oz	5.00
Refining Charge Ag	US $/oz	0.45
Payable Cu	%	96
Payable Au	%	92
Payable Ag	%	90

Provisional payment 95% shipment date plus 5 days.
Final payment 5% when all final details (weights, assays and prices) are known – estimate shipment plus 120 days.

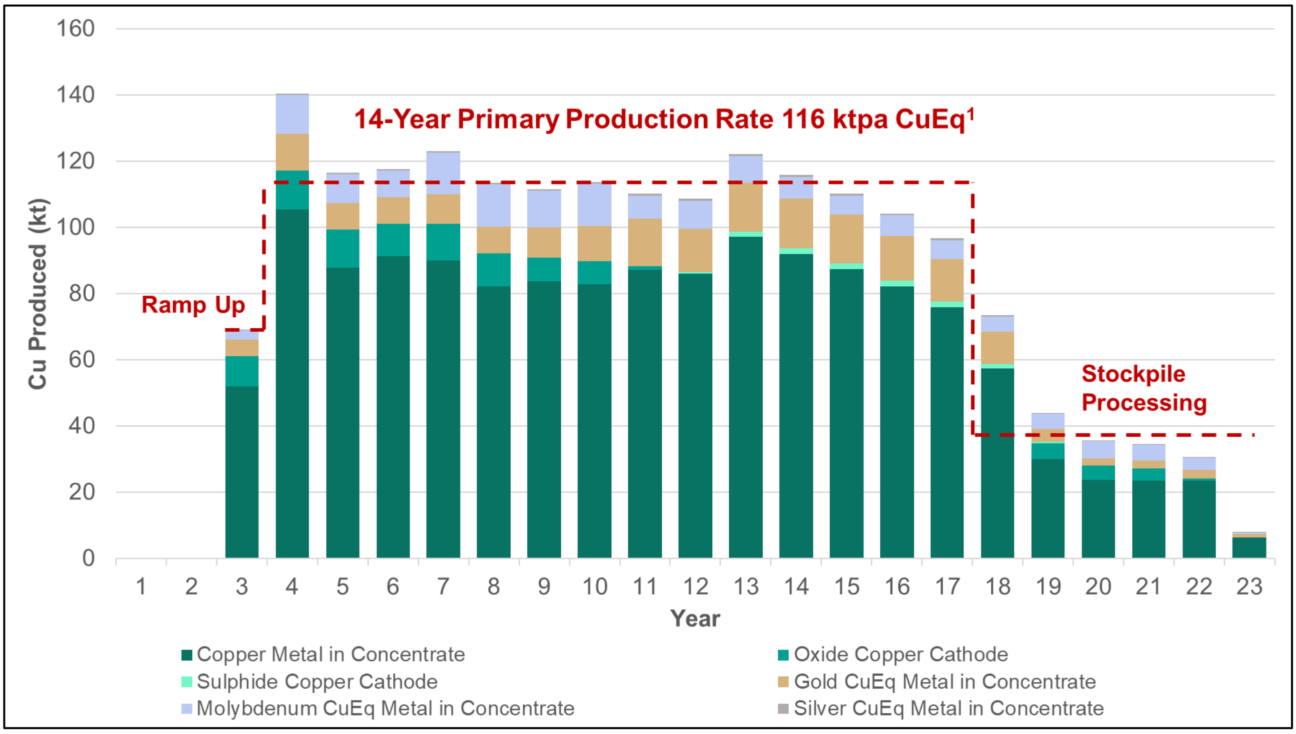
19.4 Overview

19.4.1 Concentrate Volumes

The Costa Fuego Project will produce significant quantities of copper-in-concentrate, as well as molybdenum concentrate and copper cathodes.

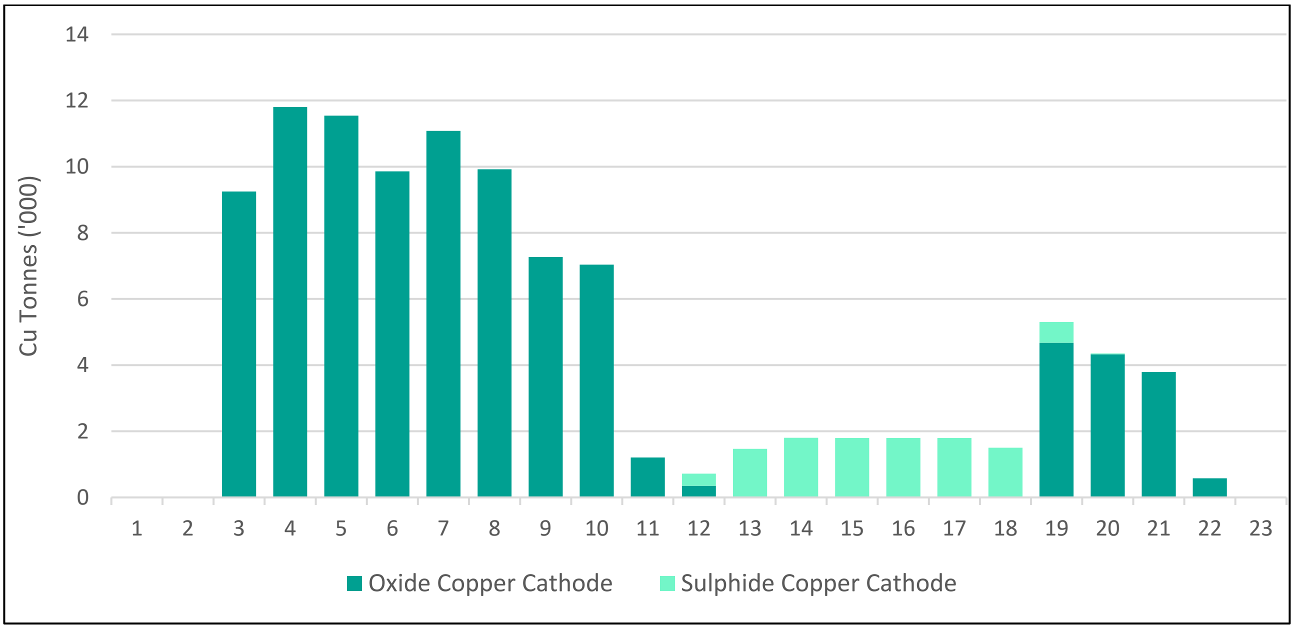
Copper concentrate volumes will range from ~330 to 420 kwmt per annum during full production years in the Life of Mine [LOM].

Figure 19.1 : Total Copper Metal Produced



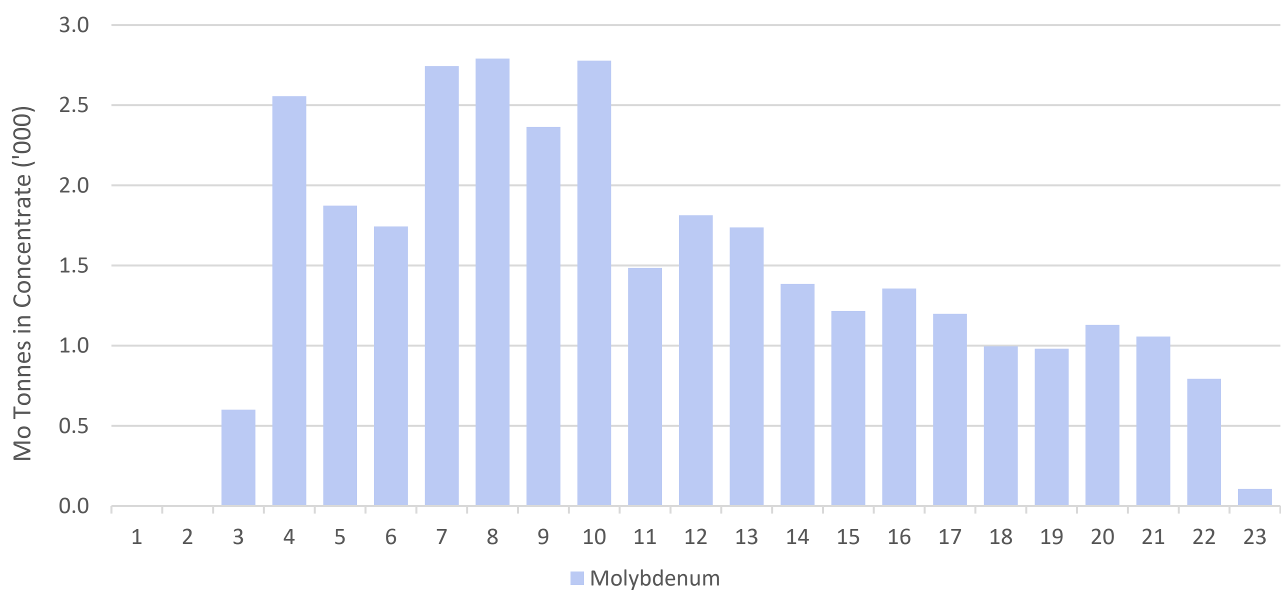
Cathode volumes will range from 1,000 ~ 12,000 with initial production from an oxide leach circuit, transitioning to a sulphide circuit in the middle years of the LOM.

Figure 19.2 : Copper Tonnes in Cathode



The high levels of molybdenum in concentrate will facilitate its removal and sale as a separate by-product stream.

Figure 19.3 : Molybdenum Tonnes in Concentrate



19.4.2 Concentrate Quality

19.4.2.1 Copper Concentrate Quality

The concentrate specification contains copper at relatively high levels and impurities at materially low levels. As discussed in Section 19.5.7 certain deleterious (or "penalty") elements not only attract financial penalties from copper smelters but may also render the concentrate impractical or problematic to process. The Costa Fuego Project concentrate will contain none of these elements at meaningful levels.

Table 19.4 : Copper Concentrate Specification1,2,3

Element	Unit	Final Cu Concentrate Costa Fuego Project
Cu	%	25.6
Mo	ppm	586
Au	ppm	3.82
Ag	ppm	23.1
Al2O3	%	2.66
As	ppm	18.9
Ba	ppm	96
Bi	ppm	2.6
CaO	%	0.59
Cd	ppm	2.0
Cl	ppm	200
Co	ppm	323
F	ppm	238
Fe	%	28.1
Hg	ppm	0.78
K	ppm	4568
MgO	ppm	3599
Mn	ppm	122
Na	ppm	2,611
Ni	ppm	178
P	ppm	134
Pb	ppm	45
S	%	32.6
Sb	ppm	9
Se	ppm	69
SiO2	%	9.5
Sn	ppm	6
Sr	ppm	33
Te	ppm	3.0
Th	ppm	3.9
Ti	ppm	563
Zn	ppm	301
Zr	ppm	125

[1] Weighted average by copper metal produced by deposit on a LOM basis
[2] Final Cu concentrate stream includes the two streams reporting to the Final Cu concentrate (Molybdenum rougher flotation tail and Molybdenum cleaner flotation tail, as per Figure 17.6).
[3] Refer to 13.3.2.8

A copper feed grade of 25% approximates the average feed grade of many smelters. Consequently, the Costa Fuego Project concentrate will not require significant blending with other concentrates prior to feeding to a smelter.

Molybdenum will be floated from the initial concentrate in the specification to produce a separate molybdenum concentrate and removing 85 to 95% of the molybdenum, reducing the level in concentrate below 1,112 ppm.

19.5 Market Analysis

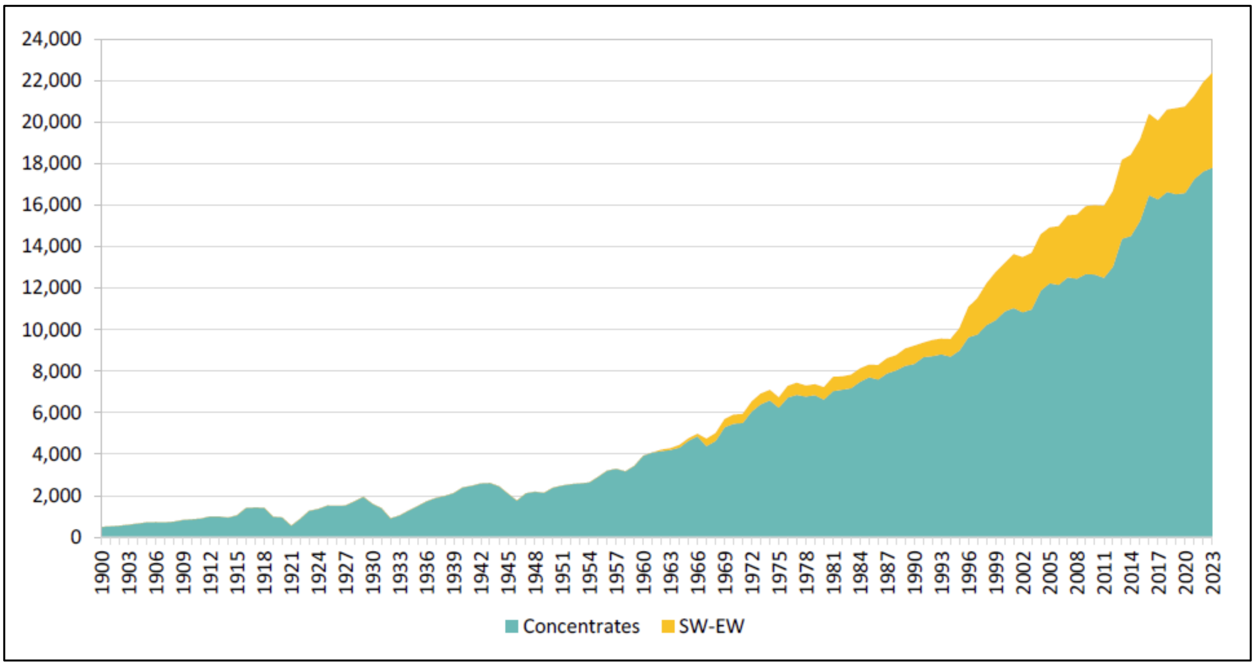
19.5.1 Copper Concentrate – Mining, Smelting & Refining

The majority of the world's copper concentrate production is processed through pyro-metallurgical processes in copper smelters and refineries throughout the world. Primary smelting technologies may be further broken own to Outokumpu, Mitsubishi, Teniente, Noranda, Isasmelt and Vanyukov processes. Recent technological advances have seen the introduction of "double-flash" and "bottom-blown" furnaces, with both technologies being advanced significantly in China. The Bottom-Blown furnaces are said to be able to treat lower concentrate grades with higher impurities whilst maintaining high metal recoveries.

19.5.1.1 Global Copper Mine Production

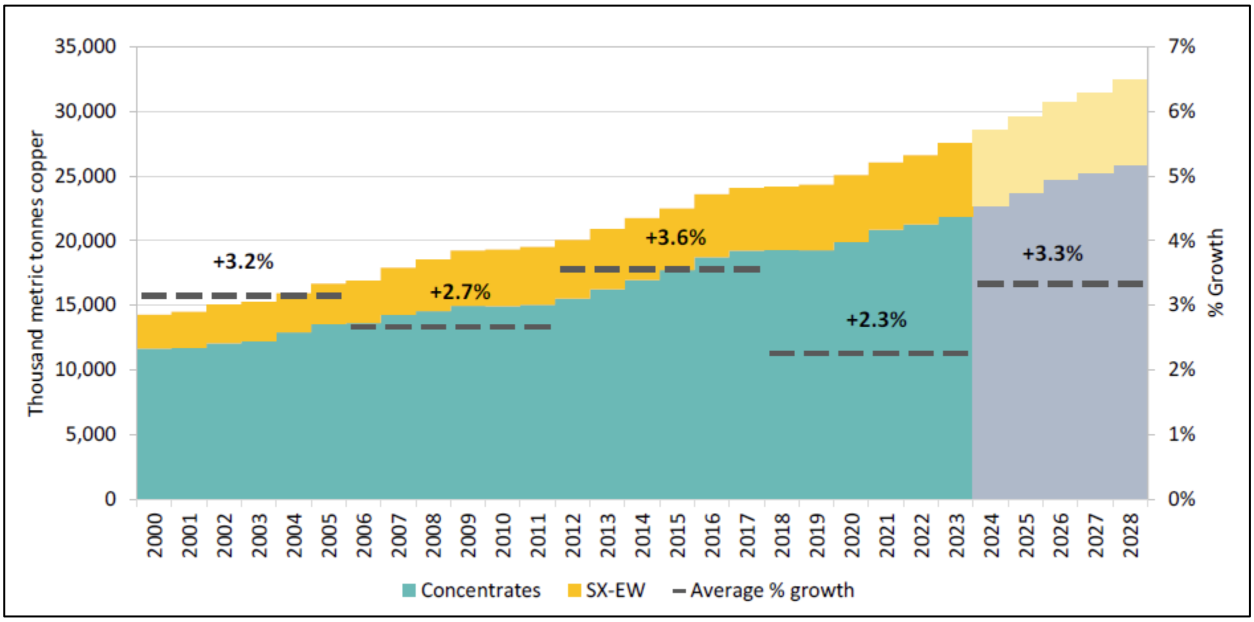
Global copper mine production (on a copper-contained basis) is estimated to be ~22.2 Mt in 2024. The addition of committed and probable projects, less attrition, will see production at ~20.8 Mt in 2029[1]. Of these quantities, ~18.0 Mt in 2024 and 20.6 Mt in 2029 is attributable to concentrate[2].

Figure 19.4 : Global Copper Mine Production 1900 ~ 2023 (kt copper contained)[3]



[1] CRU Data 2024
[2] CRU Data 2024 – *Copper Concentrates Market Outlook, September 2024*, pp8
[3] *The World Copper Factbook 2024*, ICSG, pp11

Figure 19.5 : Copper Mine Capacity 2000 ~ 2028(f)[1]



Mines producing concentrate and smelters smelting and refining concentrate can be categorised as either *Integrated* or *Custom*. Integrated mines/smelters produce concentrate from their own mines for feed to their own smelters. Custom miners and smelters buy or sell concentrate on the open market. Some integrated producers cross the arbitrary definition by buying or selling concentrate on the market from time to time to supplement smelter feed, or to off-load excess mine production.

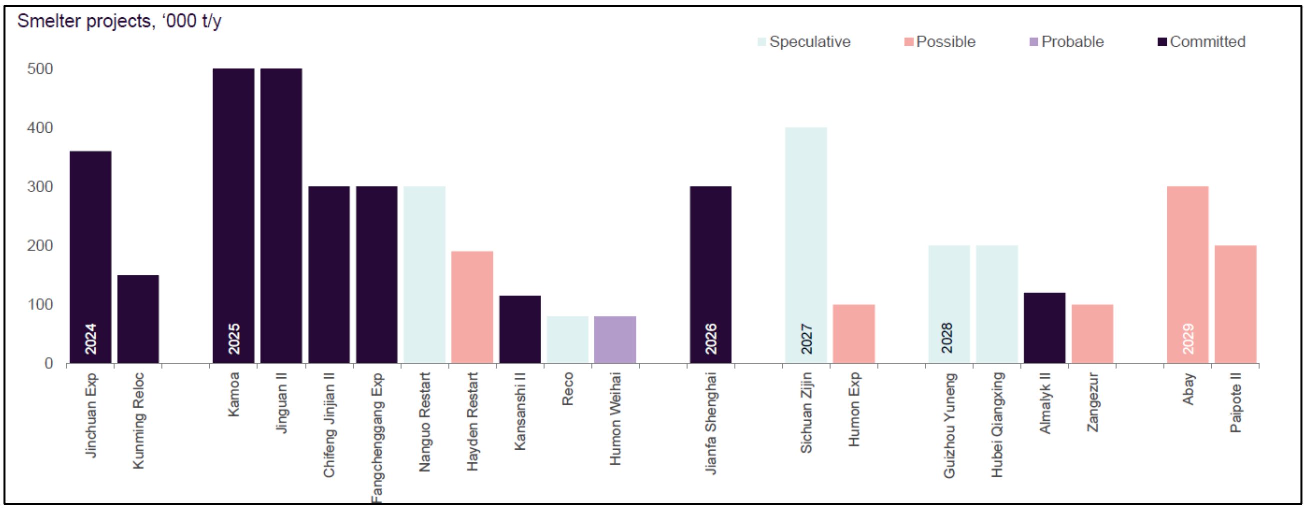
In CY2023, the Custom market for copper concentrate is likely to have accounted for 55~65% of global copper concentrate production. This is a market which has grown markedly over the last 20 years. The overall market size is estimated as ~50mDMT per annum.[2] In contrast, the integrated share of the market has remained static.

It is estimated that approximately 75% of Custom concentrate was traded internationally (exported) in CY2023 and that traders handled some 30~35% of this portion. Due to the increase in blending operations carried out by traders since 2014, it can be reasonably expected that their share of global Custom concentrate will increase.

CY2024 saw new smelter projects added in Indonesia, India and China.[3] Looking forward to 2029, additional smelter projects and expansions can be identified.

[1] *The World Copper Factbook 2024, ICSG,* pp14
[2] Industry sources
[3] It is noted that at the time of writing, delays are impacting some of these developments.

Figure 19.6 : Smelter projects (>100,000 t/y)[1]



It is notable that arsenic has been identified as a key deleterious element in copper concentrates:

> *According to an estimate by the International Copper Study Group (ICSG), less than half of the world's copper concentrate has an arsenic content equivalent to or lower than 0.5% [5000 ppm].*[2]

The ICSG also noted that a large volume of additional concentrates to be produced in Chile over the foreseeable future contained elevated levels of arsenic.

Smelter feed requirements on metal grades and impurity levels, as well as import limitations in some jurisdictions (notably China) have facilitated the development of concentrate blending operations globally. The extent of these facilities is discussed in Section 19.5.10.

19.5.1.2 Global Copper Smelter Production

The following table shows the top 20 copper smelters by capacity. It is noted that thirteen of these smelters are located in the Asian region.

[1] CRU Data, Copper Concentrates Market Outlook, September 2024, pp11
[2] *China to tighten import thresholds for impurities in metal concentrates, sources say,* Fastmarkets, 31 October, 2022

Table 19.5 : Major Copper Smelters 2024 [1]

Thousand metric tonnes of copper

Source: ICSG Directory of Copper Mines, Smelters, and Refineries – August 2024 Edition /1

Rank	Smelter	Country	Operator/Owner(s)	Process	Capacity
1	Nanko Copper (smelter)	China	Guangxi Nanko Copper Co.	Side-Blown	675
2	Guixi (smelter)	China	Jiangxi Copper Group 43.72%,Hong Kong Securities Clearing Company Ltd. 31.03%, Other companies and private 25.25%	Flash Smelter	520
3	Adani (smelter)	India	Adani Entreprises	Flash Smelter	500
4	Jinguan (Flash Smelter)	China	Tongling Nonferrous Metals 100%	Flash Smelter	480
5	Chuquicamata (smelter)	Chile	Codelco	Outokumpu/ Teniente Converter	450
5	Hamburg	Germany	Aurubis	Outokumpu, Contimelt, Electric	450
5	Saganoseki (smelter)	Japan	JX Nippon Mining & Metals Co., Ltd.	Outokumpu Flash	450
5	Toyo (smelter)	Japan	Sumitomo Metal Mining Co. Ltd.	Outokumpu Flash	450
9	Birla Copper (Dahej)	India	Birla Group (Hidalco)	Outokumpu Flash, Ausmelt, Mitsubishi Continuous	420
10	Chifeng Yunnan (smelter)	China	Chifeng Yunnan Copper (Yunnan Copper 45%, Chifeng state-owned capital operation Co.,Ltd. 45%, Jinfeng Copper 10%)	Side-Blown	400
10	Chinalco Southeast Copper (smelter)	China	Chinalco (Yunan Copper 60%, Fujian Investment & Development Group Co.,Ltd. 40%)	Flash Smelter	400
10	El Teniente (Caletones)	Chile	Codelco	Reverberatory/ Teniente Conv.	400
10	Guangxi Xingyue	China	Guangxi Xingyue Material Technology	Side-Blown	400
10	Hongsheng Copper	China	Yangxin Hongsheng Copper Industry Company Limited (Daye Nonferrous 52%,China NO.15 Metallurgical Construction Group 24%, Huangshi Xingang Development Co., Ltd 16%, Huangshi State Asset Management Co., Ltd 8%)	Flash Smelter	400
10	Jinchuan (Fangchenggang smelter)	China	Jinchuan Group 70%,Trafigura Pte Ltd. 20%, Trafigura Investment (China) Co., Ltd.10%	Flash smelter	400
10	Manyar (smelter)	Indonesia	PT-FI (Freeport-McMoRan Inc. 48.76%)	Outokumpu Flash	400
10	Norilsk (Nikelevy, Medny)	Russia	Norilsk Nickel	Reverb, Electric, Vanyukov	400
10	Shandong Fangyuan (smelter)	China	Dongying Development Zone Fangyuan nonferrous metal industry and Trade Co., Ltd 71.39%, Singapore Meijin jeweler 28.61%	Bottom-Blown	400
10	Sterlite Smelter (Tuticorin)	India	Vedanta	Isasmelt Process	400
10	Yanggu C&D (smelter)	China	Xiamen C&D 51%	Outokumpu Flash	400

The dominant technology for copper smelting is the flash furnace for the processing of sulphide concentrates. It is said to be energy efficient, as the process is autogenous. Sulphur dioxide off-gas is captured and transformed into sulphuric acid, adding further economic value.

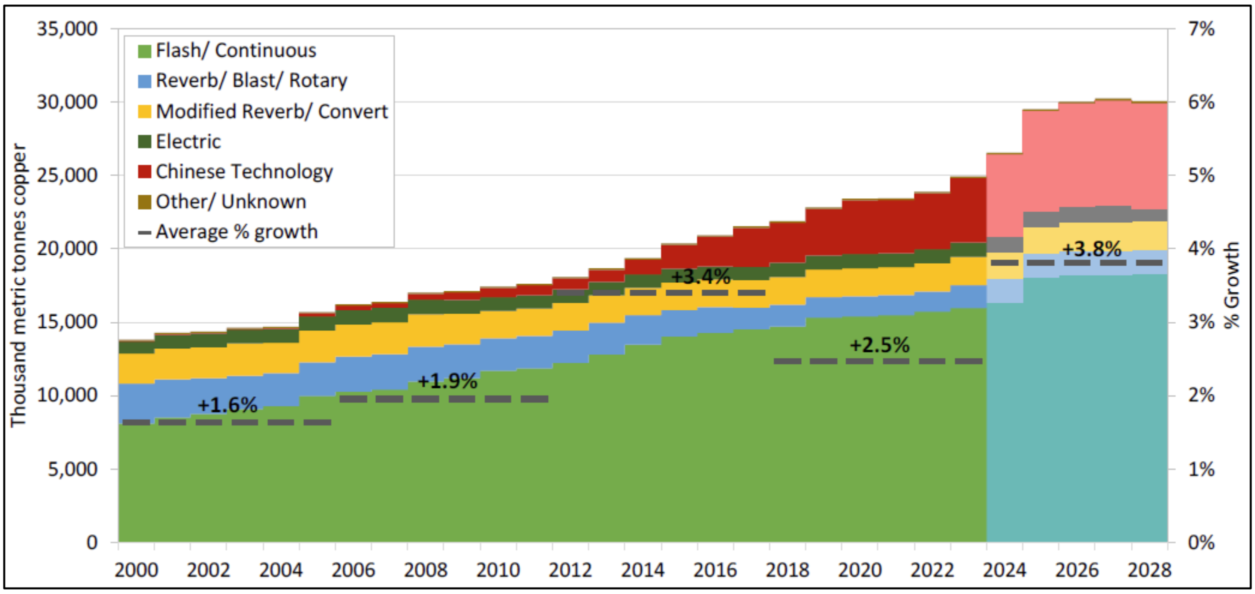
In recent years, the application of "Chinese Technology" has grown. Chinese corporations have modified established flash technology to optimise the treatment of concentrates. Two Chinese corporations have developed proprietary technology:

- **Jiangxi Copper's Oxygen-Enriched Smelting**: A modified flash smelting process that integrates oxygen-enriched air to improve energy efficiency and reduce emissions.
- **Dongying Fangyuan Process**: Developed by Shandong Fangyuan Nonferrous Metals, this process is a variation of flash smelting with enhanced efficiency and scalability for large-scale operations.

The ICSG provides an estimate of additional smelter capacity to 2027. The expansion of "Chinese Technology" is very clear.

[1] *The World Copper Factbook 2024,* ICSG, pp21

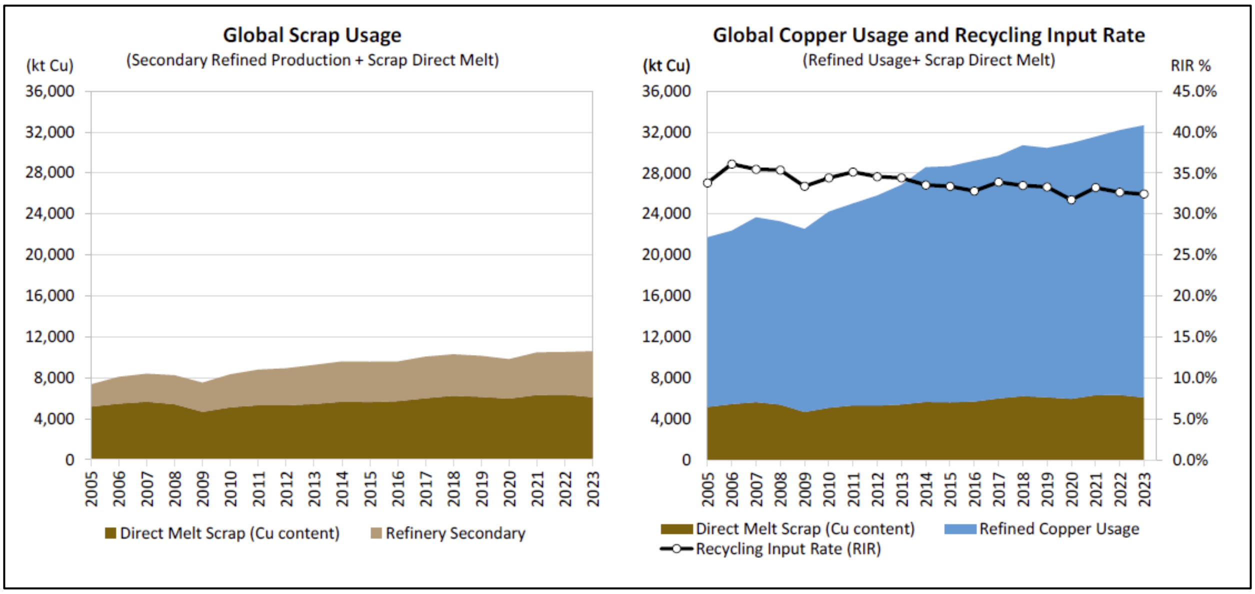
Figure 19.7 : Copper Smelting Capacity Trends 2000 ~ 2028[1]



Copper scrap has come to play an important role as a feedstock to the copper smelting industry. Some smelters have come to rely heavily on copper scrap as a feed source, particularly at time of limited, and expensive, copper concentrate supply.

[1] *The World Copper Factbook 2024,* ICSG, pp18

Figure 19.8 : Global copper scrap use 2005 ~ 2023[1]



19.5.1.3 Refined Copper Consumption

As with copper smelting and refining, copper consumption is now also Asia-centric.

In 2000, China accounted for 11% of Global consumption. By 2010, this share had increased to ~ 36% and by 2021 it was recorded as 47%.[2]

Growth in other Asian nations (excluding China and Japan) is also substantial. Refined copper consumption in Japan is expected to remain stagnant in the near term and decrease in the longer-term. The fall in demand will necessitate increased refined metal exports by Japanese smelters if all are to remain operating.

The key risk to this demand forecast is Chinese consumption. Growth in marginal demand for global refined copper is largely dependent on Chinese demand. This growth is largely reflected in and correlated to Chinese GDP and will be strongly influenced by macroeconomic factors like potential Yuan devaluation, industrial production and the speed of the transition of the Chinese economy into a higher proportion of services, as well as sustained increases in Chinese living standards.

However, potential opportunities for increases in refined copper consumption do exist in developing world economies. These have low absolute consumption levels at present but are forecast to grow strongly out to CY2040. Given the large populations in these regions and low copper intensity-of-use, even a relatively small increase in GDP per capita and copper intensity-of use in these economies can drive a large increase in total metal consumption across many commodities, including copper.

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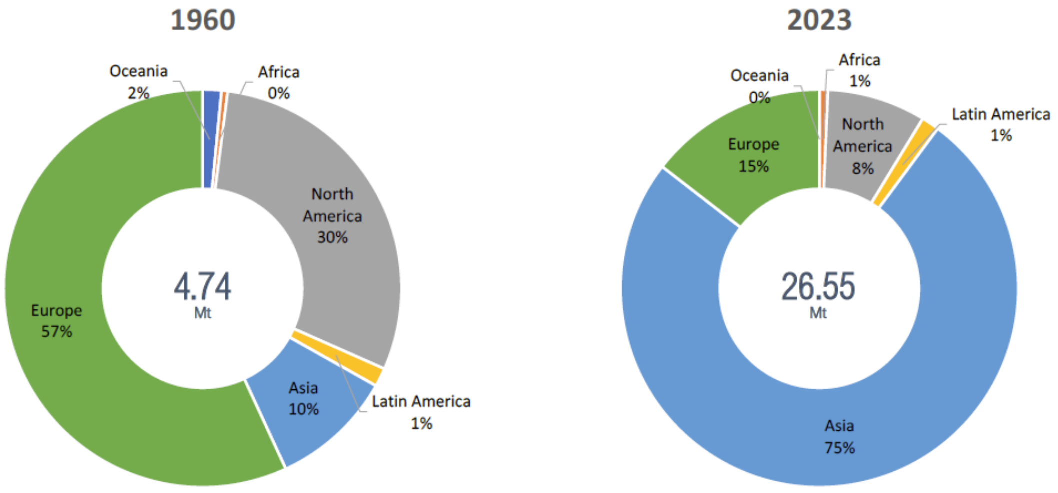
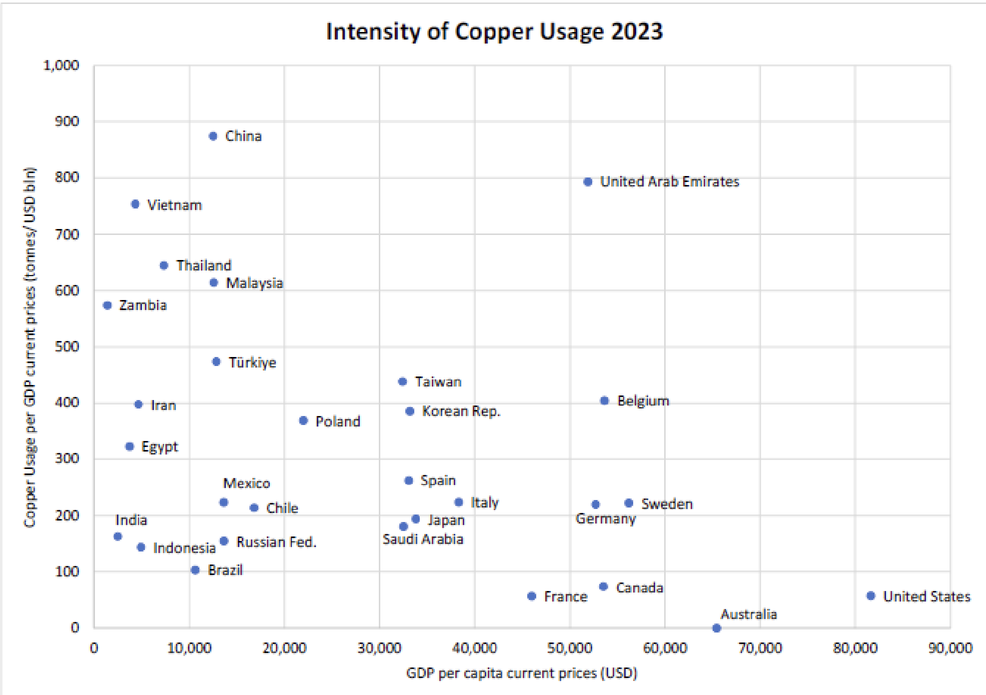
[1] *The World Copper Factbook 2024,* ICSG, pp55
[2] *Global copper strategic planning outlook - Q1 2022.* Wood Mackenzie March 2022 Data Tables

Figure 19.9 : Regional Refined Copper Consumption 1960 v 2023[1]



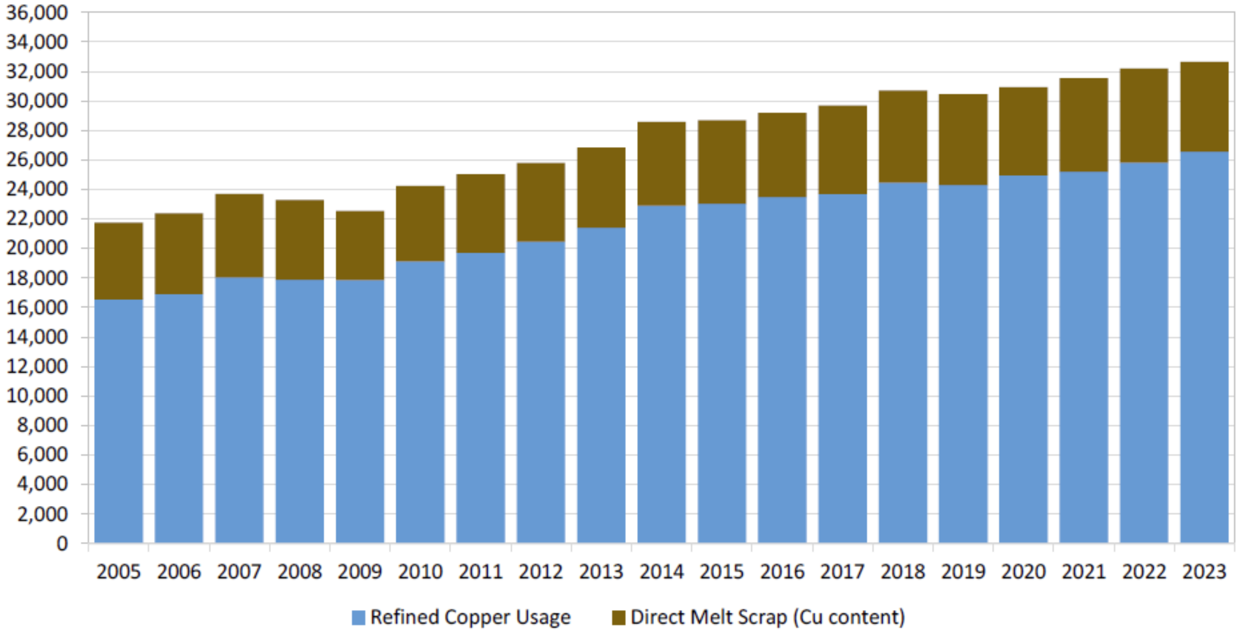
The following chart depicts the current relative intensity-of-use positions of selected countries.[2]

Figure 19.10 : Copper – Intensity of Use 2023[3]



[1] The World Copper Factbook 2024, ICSG, pp37
[2] ICSG, World Copper Fact Book, 2021, pp 39
[3] *The World Copper Factbook 2024*, ICSG, pp39

Figure 19.11 : Total Global Annual Copper Use 2005 ~ 2023 [1]



19.5.1.4 The Chinese Smelter Purchasing Team

The Chinese copper smelters which are State-Owned Enterprises formed an association during the early 2000s called the Chinese Smelter Purchasing Team [CSPT]. The CSPT is led by one of two smelters on an alternating basis, Jiangxi Copper and Tongling Nonferrous. The CSPT will negotiate the annual Benchmark on behalf of all CSPT members. A quarterly floor-price in terms of TC/RC is also set by the CSPT. Member companies are said to be bound to not purchase concentrate below the floor price.

Membership has expanded since the CSPT was established in the early 2000s.

The CSPT smelters comprise:

1. Tongling Nonferrous
2. Jiangxi Copper
3. Daye Nonferrous
4. China Gold
5. Baiyin Nonferrous
6. Gansu Jinchuan
7. China Copper (formerly Yunnan Copper)
8. Zhongtiaoshan
9. Yantai Guorun
10. Zijin Mining

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[1] *The World Copper Factbook 2024,* ICSG, pp40

11. Fuye Heding
12. Huludao Zinc's subsidiary copper smelter
13. China Gold Lignan

The CSPT nominates which smelter will lead annual benchmark discussions with miners and also sets a quarterly floor price (TC/RC) for the purchase of spot copper concentrates by CSPT members.

Until the late 1990s, a loose coalition of smelters also existed in Japan called The Japanese Smelter Pool [JSP]. This group was not a representative body, but rather a contractual entity to jointly purchase concentrates. The JSP effectively ceased to exist in the late 1990s.

19.5.1.5 The Copper Concentrate Market Balance

When the availability of copper concentrate exceeds smelter demand, the market is said to be in surplus. The opposite is true when concentrate availability is insufficient to meet smelter demand (deficit).

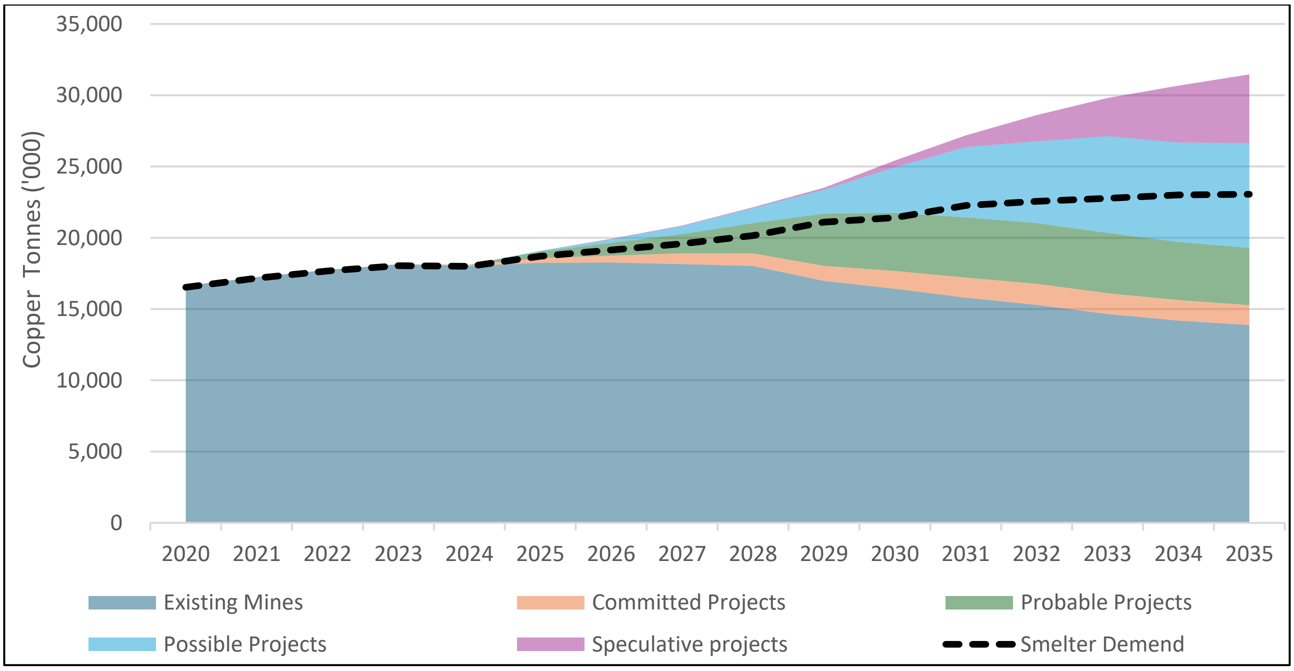
A smelter's appetite for concentrate may also be impacted by the availability of copper scrap as a feed source as well as by smelter maintenance plans. Therefore, a theoretical feed capacity (usually expressed in units of contained copper), will be reduced. Actual feed of material, as a percentage of the nameplate capacity, is known as the utilisation rate.

As shown in Figure 19.12 : , demand for copper is expected to increase in future years. CRU estimates that the CAGR from 2025 ~ 2035 will be ~1.9%[1].

Furthermore, CRU expects that supply from committed projects will be insufficient to meet this demand, and that a number of probable projects will need to be realised.

[1] CRU Data 2024 Long Term Outlook

Figure 19.12 : Concentrate Supply ~ Smelter Demand 2020 ~ 2035[1]



19.5.2 Global Copper Refined Demand

There are three key end-use sectors for copper. These are:

- Wire rod
- Copper sheet
- Copper tube.

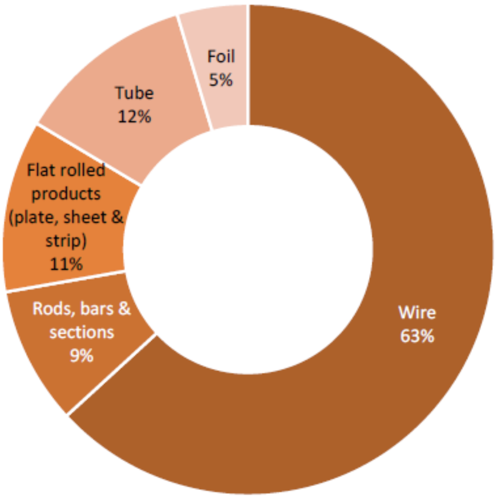
Wire rod accounts for more than 60% of total copper consumption[2] (when scrap/secondary copper is included) and more than 70% of primary copper consumption. Most wire rod is transformed to cables and wiring and mostly applied to the following product sectors:

- Low-voltage energy cable
- Power cable
- Telecom cable
- Internal telecom and data cable
- Winding/magnet wire.

[1] CRU Data 2024
[2] CRU Data set

These cabling products are then used across several sectors: construction, electrical network infrastructure, industrial machinery and equipment, transportation equipment and consumer & general products.

Figure 19.13 : Major First Uses of Copper 2023[1]



The acceleration of EV demand has led to increased demand for certain copper products. Apart from wiring, there is also materially greater demand for copper foil, which is utilised in lithium-ion battery applications.

It is worth noting that ICE passenger vehicles contain about 23 kgs of copper, whereas hybrid vehicles contain 40 kgs and full EVs contain more than 83 kgs, on average. Hybrid electric buses contain ~ 89 kgs and Battery powered electric buses can contain up to ~370 kgs[2].

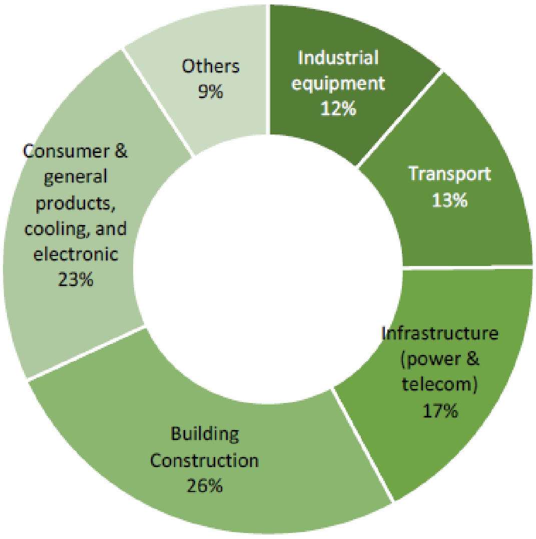
The ICSG also lists seven sectors where it envisages growing demand for copper in future years[3]:

- **Antimicrobial** – copper is gaining popularity as an alternative to plastic in medical applications, such as sterile table-tops and medical cart handles
- **Aquaculture** – marine aquaculture nets and pens made with copper-alloy mesh are emerging as an effective solution to important problems facing the near-shore fish farming industry
- **Electrical Propulsion** – powering EVs requires changes to the electrical infrastructure that will benefit from copper
- **Renewable Energy** – copper plays important roles in clean energy systems from wind to solar thermal plants

[1] *ICSG World Copper Factbook 2024* pp48
[2] *ICSG World Copper Factbook 2024* pp46
[3] *ICSG World Copper Factbook 2023* pp47

- **Seismic Energy Dissipation** – earthquake damage can be controlled through the use of copper-based devices that absorb energy to limit building movements
- **Ultra-conductive Copper Components** – progress is being made in methods of incorporating nanocarbon materials into copper in a way that promises to deliver substantial efficiency improvements in electrical energy transmission and distribution networks
- **Electrical Vehicles** – to reduce carbon emissions. The rising number of EVs is expected to result in increased copper usage.

Figure 19.14 : Copper Consumption by Industry Sector[1]

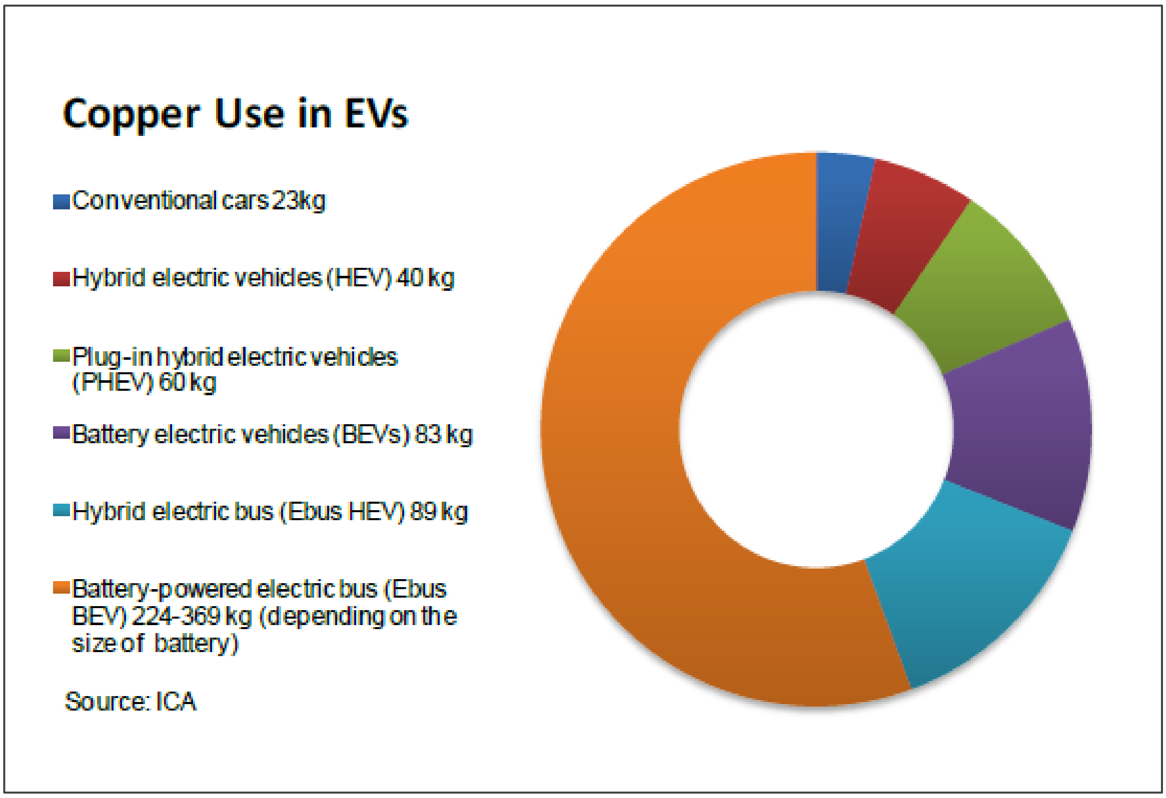


19.5.3 Copper as a Green Metal

The acceleration of EV demand has led to increased demand for certain copper products. Apart from wiring, there is also materially greater demand for copper foil, which is utilised in lithium-ion battery applications.

[1] *The World Copper Factbook 2024,* ICSG, pp48

Figure 19.15 : Copper use in EVs[1]



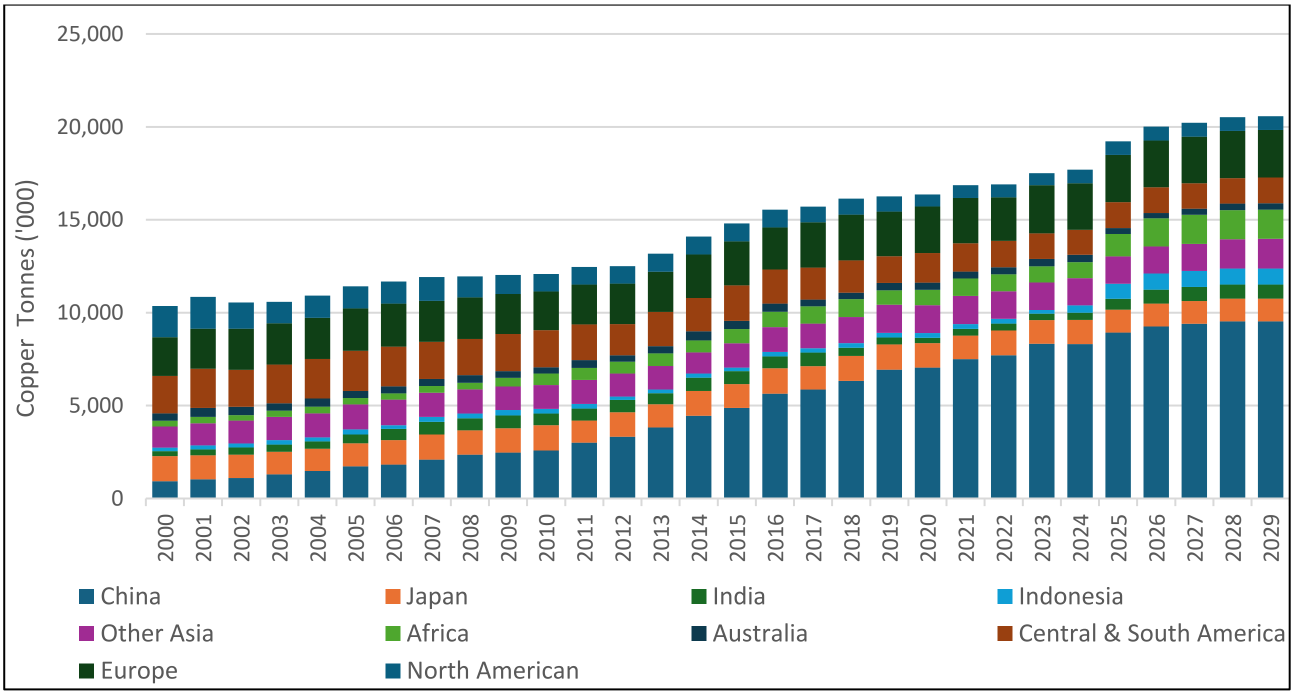
What is estimated to have been 0.5MT of copper use in 2021 may increase to ~3.0Mt by the end of the decade[2].

19.5.4 Overview of key international demand centres for copper concentrates

Asia has eclipsed all other regions in terms of smelter demand (and consequently copper concentrate demand). In 2023, global primary smelter production was about 19.6Mt[3]. China's share was about 9.2Mt, or 47%. Planned smelter expansions will take the total capacity in China to ~13Mt within a few years.[4] Due to limited mine supply – less than 2Mt – China remains a key consumer of custom copper concentrates.

[1] *The World Copper Factbook 2024* & ICA, ICSG, pp46, CRU data set
[2] Industry Data
[3] CRU data set
[4] *Industry sources*

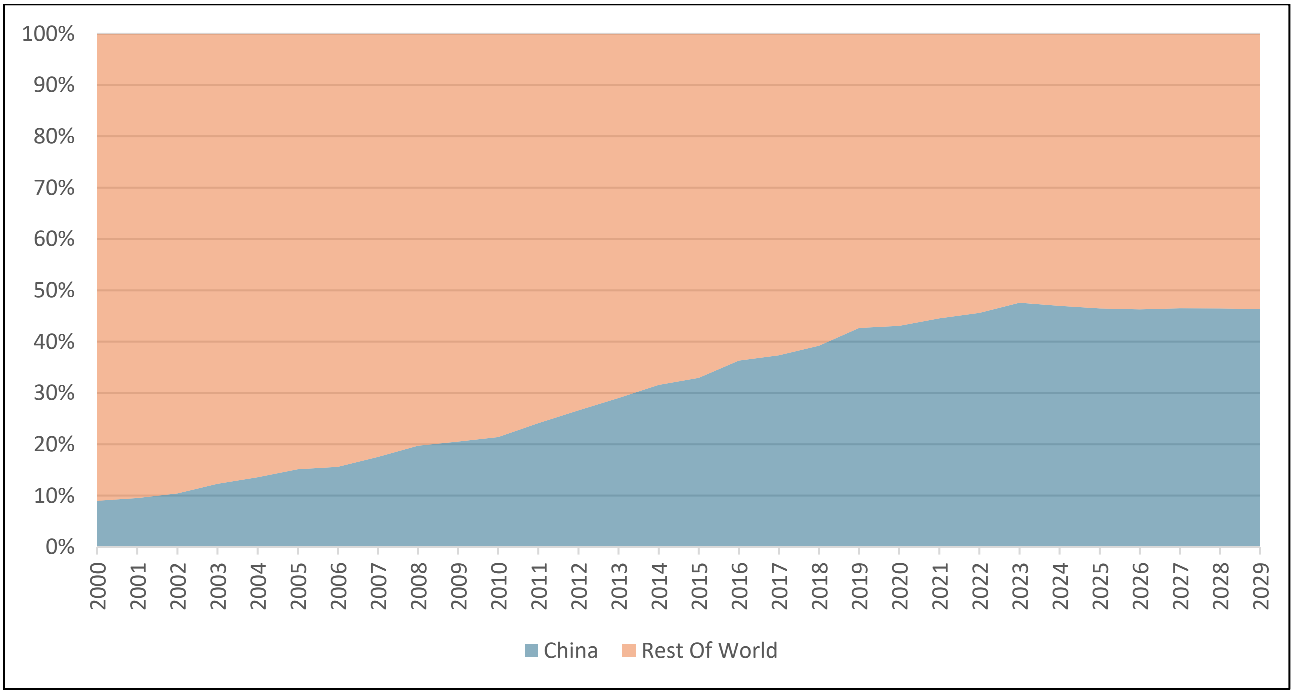
Figure 19.16 : Copper Smelter production by region[1]



As shown in Figure 19.16 : , China's share of production has increased substantially since 2000.

[1] CRU Data 2024

Figure 19.17 : China v the Rest of the World 2000 ~ 2029 [1]



19.5.5 Review of International Copper Concentrate Contractual Structures & Commercial Terms

Sales contracts for copper concentrates can be either direct mine to smelter, mine to trader or trader to smelter contracts.

The structure of direct contracts between mines and smelters can be either "block" contracts or "brick" contracts. In a block contract, 100% of the quantity for each contractual period is priced at one time (usually one-year periods). Brick contracts typically price 50% of each year's quantity over a two-year period and are typically only used when large quantities of concentrate are concerned. In brick contracts, the volatility of commercial terms is decreased, as commercial terms applied in any one-year period will be the average of agreed terms over two years. Brick contracts have become less prevalent over recent years.

The duration of a direct Mine ~ Smelter "long-term" contract will be at least three years and may be as long as 10 years. Contracts may be rolled over automatically for an agreed period of time if neither party objects. These contracts are also known as frame contracts.

Mine to trader contracts long-term generally will not exceed 1 ~ 3 years (other than spot contracts).
In the open market, Life of Mine contracts are rare. Generally, they will only exist where the buyer has provided financing to the mine, the miner has acquired the asset from the trader or a related entity, or the trader has another kind of inter-company relationship with the mine.

[1] *CRU Data 2024*

Spot contracts may cover direct business between mines and smelters however they almost always involve contracts between mines and traders, or traders and smelters. Also, spot contracts may cover a single shipment, or multiple shipments over an extended period of time.

It has been more common for miners to sell a specific quantity of concentrate to traders over an extended period of time and not aligning to any particular calendar year. For example, a sale of ~30,000 DMT made up of three 10,000 DMT shipments may be sold over a forward period of, say 6 months.

Some contracts may also reference the key commercial terms (Treatment Charge and Refining Charge) to the spot market through a market index. Wood Mackenzie, CRU and Fast Markets, amongst other publications, state such terms in monthly publications. Although not pervasive, some concentrate sales contracts now reference these terms instead of applying fixed terms or referencing the annual benchmark.

19.5.6 Copper Payment Terms

19.5.6.1 Copper Concentrate - Overview of Typical NSR Deductions

The Net Smelter Return [NSR] for concentrate sales is calculated by deducting the smelter deductions from the gross metal value.

The deductions are typically the treatment and refining charges, payable metal deductions and penalties.
Some methodologies will also take into account freight (domestic and export), handling costs and royalties.

19.5.6.1.1 The Copper Concentrate Benchmark

A Treatment and Refining Charge Benchmark framework currently exists for copper concentrate. The Benchmark is negotiated by major producers and major smelters on an annual (Calendar Year) basis. From about 1990 until the early 2000s, the Benchmark was set by one of Escondida, Freeport Indonesia, Highland Valley or Ok Tedi with, almost always, the Japanese smelters. In recent years, the benchmark has been dominated by Freeport and Antofagasta Minerals, setting the benchmark with one of the Chinese smelters.

The Benchmark is reported in a number of industry journals, such as Wood Mackenzie and CRU.

Therefore, it can be said that the Benchmark reflects the major commercial terms (treatment and copper refining charges) applying to offtake contracts between mines and smelters in a given contract/calendar year.

Many other mines and smelters will follow the Benchmark in their own offtake contracts (for direct mine to smelter transactions).

Notwithstanding the applicability of the Benchmark in mine to smelter sales, peripheral terms are usually not reported. Such terms may include changes to gold and silver refining charges, metal payment rates, quotational periods, delivery terms and contractual tonnages.

Historically, a number of major contracts were also agreed between mines and smelters on a July ~ June basis. Therefore, there was also a "mid-year" benchmark in addition to the annual "calendar year" benchmark. As the number of contracts linked to mid-year commercial terms were relatively small, this benchmark lacked depth and was more susceptible to short-term market perceptions. In recent years, Chile's Antofagasta has sold significant quantities of concentrate on a July ~ June basis, but these sales are not considered to be a "benchmark" for third-party transactions.

The Price Participation [PP] mechanism, where the copper refining charge was linked to the prevailing copper price, was removed from most market contracts in ~2007. The traditional PP arrangement allowed the copper refining charge to vary by an amount calculated by applying a percentage of the prevailing price, over or under a specified trigger price. Historically this percentage was 10% of the price above or below $0.90/lb. Consequently, at a price of $3.00/lb, the delta of $2.10/lb above the trigger price would generate an additional charge of $0.21/lb to be added to the contractual refining charge.

Although generally eliminated from the market at present (it is likely to still exist in some contracts), PP may return to the market at some point in the future. However, if this is the case, it is also likely that the trigger price that will be applied to PP will be re-aligned with current long-term price expectations for copper.

Table 19.6 shows the calendar year copper concentrate benchmark since 2006.

Table 19.6 : Historical Copper Concentrate Treatment & Refining Charges [1]							
Year	**TC US $/dmt**	**Annual BM RC US c/lb**	**Spot TC Avg (CRU)**	**Spot RC Avg**	**Cu Price US $/t**	**Cu Price US $/lb**	**Spot Δ to BM**
2006	95.00	0.095	70.00	0.070	6,736	3.06	-26%
2007	60.00	0.060	26.00	0.026	7,124	3.23	-57%
2008	45.00	0.045	29.58	0.030	6,966	3.16	-34%
2009	75.00	0.075	31.50	0.032	5,170	2.35	-58%
2010	46.50	0.047	23.25	0.023	7,536	3.42	-50%
2011	56.00	0.056	56.25	0.056	8,810	4.00	0%
2012	63.50	0.064	41.08	0.041	7,949	3.61	-35%
2013	70.00	0.070	77.58	0.078	7,322	3.32	11%
2014	92.00	0.092	93.83	0.094	6,862	3.11	2%
2015	107.00	0.107	84.50	0.085	5,494	2.49	-21%
2016	97.35	0.098	86.58	0.087	4,862	2.21	-11%
2017	92.50	0.093	72.83	0.073	6,166	2.80	-21%
2018	82.50	0.083	75.75	0.076	6,523	2.96	-8%
2019	80.80	0.081	51.75	0.052	5,999	2.72	-35%
2020	62.00	0.062	44.08	0.044	6,005	2.72	-29%
2021	59.50	0.060	37.58	0.038	9,315	4.23	-37%
2022	65.00	0.065	67.42	0.067	8,815	4.00	4%
2023	88.00	0.088	60.00	0.060	8,483	3.85	-32%
2024	80.00	0.080	-2.00	-0.002	8,444	3.83	-103%
2025 YTD	21.25	0.021	-20.00	-0.020	9,153	4.15	-194%
Average	**76.23**	**0.076**	**61.68**	**0.062**	**4,979**	**2.26**	**-37%**

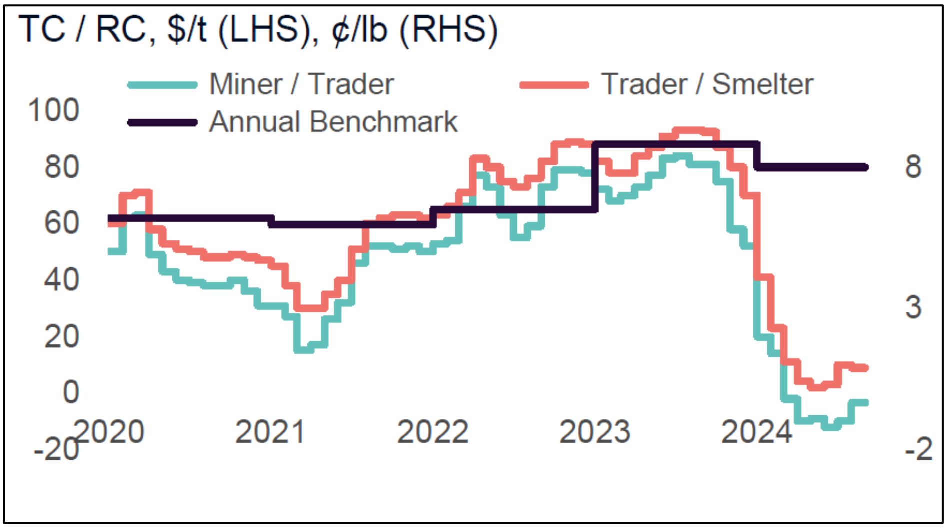
Over the period shown in Table 19.6 spot TCs/RCs have averaged ~28% below annual Benchmark charges.

Standard grade copper concentrates direct mine-smelter treatment charges since 2006 have ranged from \$21.25/DMT to \$107.00/DMT, and refining charges from 4.5¢/lb to 10.7¢ per lb payable copper. These terms are for long-term contracts between major producers and major Asian smelters agreed on an annual calendar year basis. The following chart shows the trend in treatment charges and refining charges for copper concentrate and copper metal (excluding price participation) for spot and frame contracts over the period 2020 to 2024.

In calendar year 2025, the Benchmark settlement was first made by Antofagasta and Jiangxi Copper at terms of \$21.25/DMT and 2.125c/lb. This level is unprecedented and reflects the growth in copper smelting capacity which has not been matched by growth in copper concentrate production.

[1] CRU, Industry data; AFX Commodities

Figure 19.18 : Historical Treatment and Refining Charges 2020 ~ 2024 [1]



Given the favourable terms which a miner can capture by selling only into the spot market, the question arises of why a miner would seek terms always at Benchmark in any sales contract. The answer will be in the miner's risk assessment of spot sales versus long-term offtake contracts. Very large miners producing significant quantities of concentrate have typically sought to allocate a significant portion of production to long-term contracts.

19.5.6.1.2 Standard Payables

The provisions shown below are *typical* in a copper concentrate sales agreement. No two concentrates are identical in terms of payable metals and deleterious elements. Some concentrates sit at the extreme end of the spectrum for some elements.

As an example, Prominent Hill concentrate is known to be very high in copper (~40%). At times of a pronounced concentrate deficit, the high copper grade may lead to more favourable outcomes in the assessment of deleterious elements or other contractual commercial provisions such as payment terms or freight considerations.

- **Copper:** Payable at 96.5% subject to a 1.0 unit deduction for a copper content of 20-30%. At levels exceeding 30-33%, certain smelters may agree to a higher copper payable rate of up to 96.75%. In excess of 40%, a payable of 97.0% or more could be achieved, subject to other considerations. It should be noted that some smelters may seek a higher unit deduction for concentrate where the copper grade is below 20 ~ 24%.
- **Silver:** In Asian markets, silver is paid at 90% of the analytical silver content subject to such content being higher than 30 g/t. No payment is made below 30 g/t. European smelters typically exact a deduction on the silver content. This deduction may be as high as 30 g/t.

[1] CRU Data 2024, *Copper Concentrate Market Outlook,* September, 2024, pp7.

- **Gold:** The gold payable scale in a sales contract may vary from smelter to smelter. A typical scale for an Asian smelter is shown below in Table 19.7. European smelters will seek a minimum deduction of 1 g/t.

Table 19.7 : Indicative Gold Payable Schedule	
<=1g/t	0%
>1g/t, <=3g/t	90.0%
>3g/t, <=5g/t	92.0%
>5g/t, <=7g/t	94.0%
>7g/t, <=10g/t	95.0%
>10g/t, <=15g/t	96.0%
>=15g/t, <20g/t	96.5%
>=20g/t, <30g/t	97.0%
>=30g/t, <40g/t	97.5%
>=40g/t, <50g/t	97.75%
>50g/t	98.00%

Given certain market conditions, traders and/or smelters may pay for gold at 98.25 ~ 98.5% from time to time.

This gold payability scale is well-established and the information shown above is generally accepted throughout the market for copper concentrates sold without any encumbrances on the sale (such as Buyer project equity or debt). However, the highest point of the scale, 98% > 50 g/t is usually subject to intense negotiation between the contracting parties. Chinese smelters are reluctant to pay for gold at this level, and traders on-selling to Chinese smelters will also resist payment at this level. As with all commercial terms, the final outcome will be linked to the prevailing market at the time of contract negotiation.

19.5.6.1.3 Deductions

Apart from Treatment and Refining charges applied to copper, refining charges will also be applied to payable precious metals.

Refining Charges for gold and silver are typically paid on a dollar per payable ounce basis. These items are also a matter of negotiation between buyers and sellers.

A typical gold refining charge will range from $4.00 ~ $7.00/oz. Escalation should not typically apply to the gold refining charge (although some buyers will request a gold price escalation clause).

A typical silver refining charge will range from $0.40 ~ $0.60/oz. Escalation will not typically apply to the silver refining charge.
It should be noted that these charges are applied on payable metal-in-concentrate (ie after the smelter metal deductions) rather than analytical metal-in-concentrate.

19.5.7 Deleterious Elements & Chinese Import Limits

19.5.7.1 Common Deleterious Elements

Excessive levels of some elements may preclude acceptance by some smelters or destination countries. Limitations on mercury have become stricter in recent years, most probably linked to the ratification of the Minamata Convention on mercury, placing certain controls on the production, storage and transportation of this element.

Arsenic has also faced scrutiny, partly due to the large number of mines producing concentrates with high levels of arsenic. Elevated levels of carbon and lead may also cause issues with some smelters, as will reduced levels of sulphur or elevated levels of insolubles.

Fluorine has also become an element which is avoided by smelters. Due to a significant increase of fluorine-in-concentrate from Freeport's Grasberg mine in Indonesia, high levels of fluorine in other concentrates negatively impact on their marketability.

Each smelter will differ in its own tolerance levels for certain elements. For example, if a certain smelter is purchasing a large quantity of material containing elevated levels of arsenic, its ability to purchase additional material with high arsenic will be limited.

Although levels of some elements may exceed national import limits or smelter technical limits, the advent of blending facilities by both smelters and traders has meant that some concentrate previously considered untreatable due to the chemical specification, could be purchased and blended with another material to provide a blended concentrate of acceptable quality.

As some smelters will be more sensitive to certain elements, those smelters may demand more punitive charges or will set a firm ceiling for those elements in procurement contracts. From a commercial perspective, smelters will sometimes seek an "income stream" from penalties, even when those elements may not contribute to an unacceptable deleterious impact on their processes. Also, from time to time, smelters may attempt to introduce a new penalty element or seek a favourable variation to the penalty structure for a specific element, without a sound technical basis.

19.5.7.2 Penalties for Deleterious Elements[1]

The tables below generally describe market penalty scales for deleterious elements in copper concentrates. These penalty scales are most likely to apply in direct mine ~ smelter sales. However, they should be taken as an indication of penalty scales which will apply in mine ~ trader sales.

Some disparity exists in the market with respect to penalty scales. Some miners have been able to achieve favourable, or reduced scales, whereas others have experienced higher charges. The differences are shown in Scale A and Scale B below.

[1] Internal Memo - Deleterious Element

Table 19.8 : Indicative Copper Concentrate Penalties		
Elements	**Indicative Penalty Scale A**	**Indicative Penalty Scale B**
Al2O3 + MgO	$4.50 / 1.0% > 5.0%	$4.50 / 1.0% > 5.0%
As	$3.00 / 0.1% > 0.1% $6.00 / 0.1% > 0.5% $8.50 ~ $10.00 / 0.1% > 1.0%	$2.00 / 0.1% > 0.2%
Bi	$1.50 / 0.01% > 0.03%	$1.00 / 0.01% > 0.05%
Cd	$4.00 / 0.01% > 0.03%	$4.00 / 0.01% > 0.03%
Cl	$0.50 / 100ppm > 300ppm	$0.50 / 100ppm > 500ppm
Co + Ni	$0.30 / 0.1% > 0.5%	$0.30 / 0.1% > 0.5%
F	$1.50 / 100ppm > 300~330ppm Potential higher penalty > 700ppm	$1.00 / 100ppm > 400ppm
Hg	$0.10 / 1ppm > 5ppm (or > 10ppm)	$0.20 / 1ppm > 10ppm
Pb	$1.50 / 1.0% >1.0%	$1.50 / 1.0% >2.0%
Se	$1.50 / 100ppm > 300ppm	$1.50 / 100ppm > 300ppm
Sb	$1.00 / 0.01% > 0.03%	$0.50 / 0.01% > 0.1%
Zn	$1.50 / 1.0% > 3.0%	$1.50 / 1.0% > 3.0%

Given that China is dominant as a demand centre for global copper concentrates, it is important to note the limitations placed by the Chinese authorities on contained deleterious elements in concentrates. The limits for all base-metal concentrates are currently understood to be:

Table 19.9 : Chinese Import Limits - Deleterious Elements[1]			
Element/Concentrate Type	**Copper**	**Zinc**	**Lead**
Arsenic	<0.5%	<0.5%	<0.6%
Lead	<6.0%	-	-
Cadmium	<0.05%	<0.3%	<0.4%
Fluorine	<1,000ppm	-	-
Thallium	-	<0.02%	<0.02%
Mercury	<0.01%	<0.05%	<0.05%

The Costa Fuego Project concentrate will contain very low levels of all deleterious elements. The overall chemical specification will mean that the concentrate will be a desirable feed for most global smelters, as well as traders.

[1] National Standard of the Peoples Republic of China GB/T 20424-2025 – issue date 2025-02-28

19.5.7.3 Copper Concentrate Payment for other contained metals

Overwhelmingly, payment for metals other than copper, gold and silver cannot be achieved. Even though some elements can provide a value-in-use case to a smelter, no smelter will set a precedent by recognising the value of platinum, palladium or sulphur in copper concentrates.

Other metals, such as nickel, lead and zinc will not be payable in a copper concentrate as the smelters which smelt and refine the concentrate will not achieve recovery of these metals. Rather, they are likely to be penalised at certain levels. Similarly, excessive levels of molybdenum in concentrate may result in a penalty being incurred, although there is no consensus or market standard relating to the structure of the penalty. It is indicated that molybdenum can be deleterious to the smelting and refining process at levels in excess of 5,000 ppm.

19.5.8 Payment Terms[1]

Payment terms for concentrate range from pre-payment for concentrate stored at the port of loading to payment after a specified number of days after the arrival of the carrying vessel at the port of discharge.

In the case of pre-payment in-store at the port of loading (that is a payment made against concentrate in the storage warehouse prior to loading on an export vessel), an interest charge will be applied to the pre-payment, and payment will be subject to suitable warehouse receipts or forms of verification.

Sales to traders will generally capture payment after presentation of loading documentation (Bills of Lading, Assay & Weight Certificates etc) to the Buyer. Sales to traders and some smelter customers (typically those in China and India) may also be covered by Letters of Credit issued by the Buyer on a bank acceptable to the Seller.

The payment percentage will typically be 90 ~ 95% of the provisional invoice value (based on mine estimates or assays on samples taken upon loading and port weights). Final payment of the balance will be subject to final weights and assays determined after discharge at the port of discharge. Finalisation typically occurs 90 ~ 120 days after concentrate is discharged at the receiving port.

19.5.9 Quotational Periods and Metal Pricing[2]

Quotational Periods [QPs] refer to the pricing period used to determine the price(s) to be applied to the payable metals.

Copper is typically priced on the London Metal Exchange. Gold and Silver are referenced to the London Bullion Market Association. These prices are published for each trading day (United Kingdom) in a number of industry journals.

Other exchanges also exist, such as the Shanghai Futures Exchange and Comex, but reference to prices on these exchanges is typically limited to local trades.

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[1] Internal Memo - Concentrate Invoicing and Pricing Conventions
[2] Internal Memo - Concentrate Invoicing and Pricing Conventions

The QP(s) for provisional invoices are typically taken as the prices covering a calendar week or, say, five trading days, prior to the Bill of Lading date of the carrying vessel.

The QPs for the final invoice will usually be referenced to either the month of shipment (scheduled or actual) or the month of arrival of the carrying vessel at the port of discharge.

For example:

- The QP for copper shall be the third month following the month of arrival (3 MAMA) of the carrying vessel at the port of discharge.

In the case of sales to traders, the trader may be given a QP option. Such an option may be, for example:

- The month prior to the month of shipment (M+1) or the third month after the month of shipment (M+3), declarable by the Buyer prior to the commencement of the earlier QP.

At the time of declaring the option, prices for either of the QPs will be unknown.

Granting optionality on QPs to a buyer (typically a trader) will almost always result in improved commercial terms for the seller.

The trader will use a mismatch on the QPs between their buy and sell transactions to generate a margin by hedging the material between the two quotational periods. The QP option, in the example above, provides the trader with an opportunity to generate revenue regardless of a market contango or backwardation.

Traders may also be granted a *look-back* or retrospective QP option. In such case, at least one of the QP periods will be known. The value of the optionality will depend on the metal price and the market volatility for that metal at the time the option is granted.

In a spot sale of a single cargo, a trader will ask to be allowed to declare the QP option prior to the month of shipment. In a contract covering more than one shipment or over multiple years, the option declaration requirement could be on a ship-by-ship basis, or quarterly or annually.

The value of the option to the trader will be that a *hedge* of the metal can be made to the trader's best advantage. In a *contango* market, where the forward price is higher than the spot price, the trader will declare the shortest possible QP (say M+1) so the fixed price for the trader is lower. In a backwardation market, the trader will elect the longer-dated QP. That quantity of metal in the hedge is then matched against the physical quantity in the contract with the miner.

In order to achieve a perfect hedge, the trader may ask for a *fixed quantity* to be priced per the agreed contractual QP (say M+1) with any overage or underage repriced at a later QP (typically when all final weight and assay values are known).

Providing QP optionality in favour of a buyer (typically a trader) may render hedging for the miner problematic, as the miner will always be hedging to the least favourable QP. Also, if the buyer is granted a ship-by-ship QP option over multiple shipments, the miner will not be able to effectively hedge the metal in advance, as the final QP will not be known. In order to mitigate the complications caused by QP options, a trader might be given an *annual* option, where the declaration is made prior to each contract year and the uncertainty of the pricing period for future shipments (in that year) is removed.

19.5.10 Copper Concentrate Blending

Since the early 2000s, the blending of concentrates by third-parties has expanded significantly. All smelters buying third-party concentrate must blend material to achieve a desired feed-grade. Smelters will seek to achieve a minimum level of copper units (copper grade) and blend down impurities which are harmful or costly in their processes. They may also seek to increase gold in the blend to improve overall recoveries.

Traders have been adept at taking advantage of the need to blend concentrates. Dirty concentrates can be blended with cleaner concentrates to reduce impurity levels.
Concentrate low in gold (gold below 1 g/t in generally not payable) can be blended with higher gold-bearing material to improve payability.

The blended concentrate, with lower deleterious elements and potentially higher precious metals credits, can then be sold to smelters at more favourable commercial terms (lower treatments and refining charges, and penalties).

Blending facilities are relatively inexpensive to construct and operate. No metallurgical process equipment is required. The key costs are facility rental or purchase, stevedoring equipment and the costs associated with unloading and loading concentrates for on-sale. The blended concentrate must then be delivered to smelter customers, so additional freight costs will be incurred.

Figure 19.19 : Concentrate Blending Facilities[1]

Onshore blending also takes place in China. The locations of the facilities are shown in Figure 19.20 :

[1] CRY Data 2024

Figure 19.20 : Chinese Onshore Blending[1]

19.5.11 The Current Market

In CY2024, the global market for copper concentrates was in a pronounced deficit. The construction of new smelter capacity has eclipsed mined concentrate availability. There has also been a sudden supply disruption with the cessation of concentrate exports from the Cobre Panama mine (First Quantum) and production cuts by Anglo-American. In late 2024, exports of Grasberg and Batu Hijau concentrate from Indonesia were expected to cease, as new domestic smelters are completed, removing a significant quantity of material from the traded market. Although this did not eventuate due to operational difficulties, the new smelters are expected to come online in 2025.

The concentrate Benchmark was set at $88.00/DMT and 8.8¢/lb for calendar year 2023. This industry reference point was reduced to $80.00/DMT and 8.0¢/lb for calendar year 2024.

In early December 2024, Antofagasta agreed terms with Jiangxi Copper for their 2025 annual contract at $21.25/DMT & 2.125¢/lb.[2] It is expected that this agreement will form the Benchmark for calendar year 2025 contracts. Previously, the lowest point for the annual Benchmark was $42.75/DMT and 4.275/¢/lb set for

[1] CRU – Copper Concentrates – Market Outlook, May 2023
[2] Industry sources.

CY2004. It is worth noting that at that time, price participation for copper still applied in the Benchmark settlement, providing some relief to the smelters.[1]

At the time of writing, spot TCs/RCs are being reported at less than negative $15/DMT & negative 1.5¢/lb for clean, standard grade material. This level is materially below the recent Benchmark settlement.
In January 2024, the CSPT smelters (as discussed in Section 19.5.1.4) announced production cuts in an effort to stabilise TC/RCs[2]. However, no further detail was provided, and it is uncertain if any such cuts will take place.

Long-term TC/RC assumptions range from $50.00/DMT & 5.0¢/lb to $70.00/DMT and 7.0¢/lb.

Therefore, a long-term Benchmark Treatment and Refining Charge assumption of $54.50/DMT and5.45¢/lb is suggested for the purposes of this study.

It is noted that the quantity of the Costa Fuego Project concentrate which will be allocated to Glencore International AG will attract Benchmark terms.

Sales to traders under long-term contract can be expected to attract a deduction from the Benchmark ranging from 10 ~ 40% (subject to the overall quality of the concentrate and prevailing market conditions).

Consequently, a Treatment and Refining Charge of $45.00/DMT and 4.50¢/lb ($60.00 & 6.0¢ less 24%) is suggested for open market tonnage.

On a combined basis, the weighted average Treatment and Refining Charge is calculated to be:

Table 19.10 : TC/RC Estimate - All Costa Fuego Project DMT

TC/RCs Years 1~10	Unit	GIAG	Open	Average
Treatment Charge	US $/dmt	65.00	45.00	54.50
Refining Charge	US c/lb	6.50	4.50	5.45
Refining Charge (Au)	US $/oz	5.00	4.50	4.75
Refining Charge (Ag)	US $/oz	0.50	0.45	0.48

In estimating forward-looking TCs/RCs, consideration must be given to the supply ~ demand balance for concentrate. There are several factors which will impact that balance, including:

- Unexpected smelter closures (e.g. Vedanta's Tuticorin smelter in India)
- Unexpected mine production variances (eg. higher or lower head-grades than forecast, open-pit or underground mining problems)

[1] Industry sources.
[2] *China's top copper smelters call for production cut to stabilize market,* S&P Global, 26 January 2024

- Changes in government regulations (e.g. the Indonesian curtailment of export permits for Freeport Indonesia from 2025)
- The copper price (encouraging or discouraging additional production).

For higher grade copper concentrates, smelters may also seek higher TCs from producers when the market is in favour of the smelters (concentrate surplus) to partially compensate the smelters for lower revenue received from the higher-grade producers in terms of ¢/lb Cu. This scenario has not eventuated since the 1990s.
It is noted that, as of the date of this Report, Glencore has recently announced that the PASAR smelter in The Philippines will be placed on care and maintenance. The fact that this announcement has not caused spot market TCs and RCS to rise is testimony to the structural concentrate deficit which exists in the market.

19.6 Export Logistics

Concentrate pricing is overwhelmingly on a CIF or CFR basis, where the Seller arranges and pays for freight (and insurance if CIF).

Some purchasers of concentrate, notably traders, will be prepared to purchase concentrate on an FOB basis, where the seller's obligations for delivery cease once the concentrate is loaded upon an export vessel. The trader will then charge the seller for the costs associated with delivering the cargo. This arrangement can be problematic for a seller, for although it removes the onus of the shipping and administration of export operations, it also causes the seller to lose control of export timing (and therefore cashflows) and costs. The trader may also attempt to impose higher than actual achieved costs on the Seller.

As with most markets, price levels in the market for ocean freight are determined by supply and demand factors. Other factors exogenous to the supply ~ demand balance also come into play from time to time. In recent times there have been disruptions to global trade due to global events such as the war in the Ukraine (less grain exports), terrorist activity (Houthi attacks in the Red Sea causing diversions) and diversions from the Suez and Panama Canals[1].

In 2021 and 2022 freight rates traded at high levels due to a confluence of factors. Those were principally: lack of vessel supply due to a hiatus in vessel construction because COVID; increased demand driven by increased exports of certain commodities (driven by higher commodity prices); a hang-up of vessels awaiting discharge at (mostly) Chinese ports due to COVID-related issues and high fuel prices, somewhat impacted by an IMO requirement for low-sulphur fuels in cargo vessels.

[1] Suez diversion around the Cape of Good Hope adds 7 ~ 10 days in vessel transit times. Panama diversions due to drought impacted the trade between the US and Asia.

Figure 19.21 : Baltic Handysize 7TC Average[1]

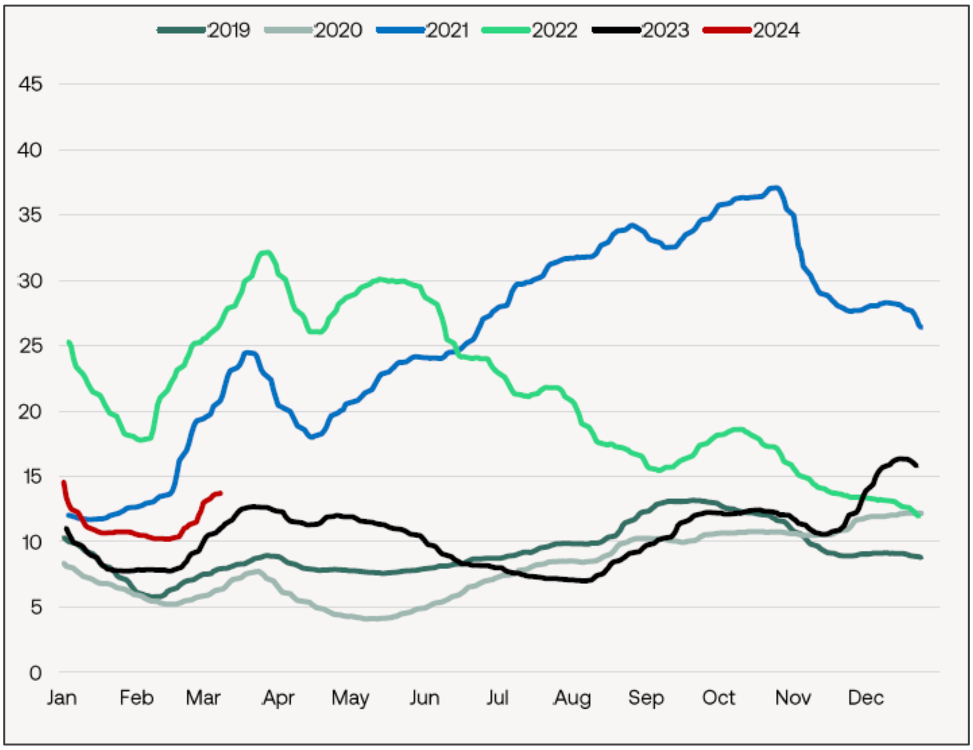
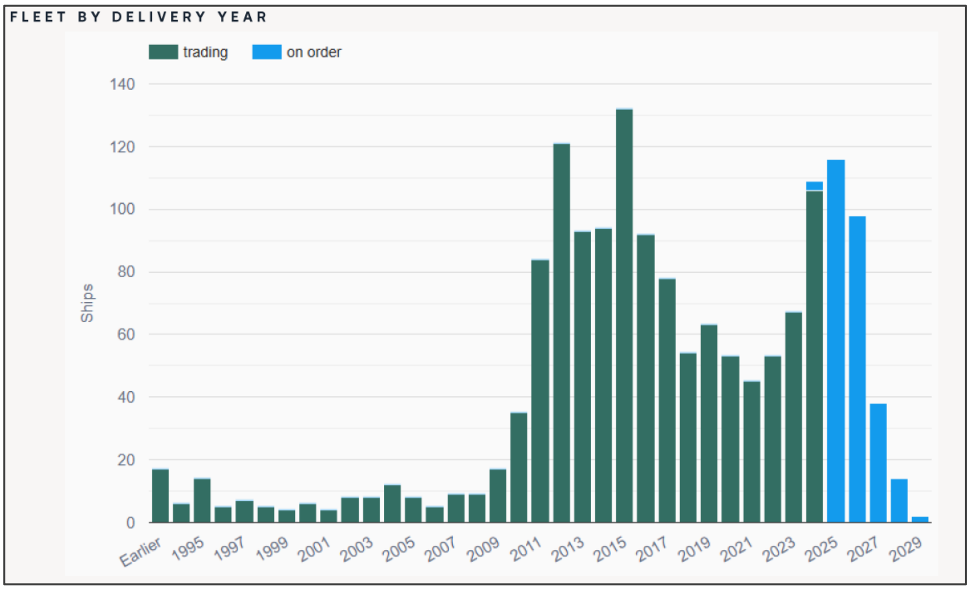


Figure 19.22 : Large Handysize Vessels - Fleet by Delivery Year[2]



[1] Dry Bulk Market Update. Braemar Shipbrokers December 2024
[2] Dry Bulk Market Update. Braemar Shipbrokers December 2024

19.6.1 Inland Logistics, Port Storage and Ship-loading

There are two logical export points for the Costa Fuego Project concentrates:

- Puerto Las Losas

  Puerto las Losas is the closest (~60 kms) export point. However, this facility does not yet have the infrastructure in place to handle copper concentrates. A Memorandum of Understanding has been executed with the Costa Fuego Project for the development of a bulk port for exporting concentrates.

- Puerto Coquimbo – TPC

  Puerto Coquimbo or TPC, is the next closest port to the Project (some 187 km south). Copper concentrates are the most important cargo handled by the port in terms of volume (5.3 million tons over the 11 years of operation) representing 66% of total cargo exported. Copper exporters such as Caserones, Codelco, Trafigura, Glencore, Ocean partners and IXM are regular users of the port.

19.6.2 Export Ocean Freight

Miners in West Coast South America [WCSA] typically work through one or more shipbrokers to procure freight contracts for concentrate export. The freight is fixed on a spot or single-shipment basis, or a multi-vessel Contract of Affreightment [CoA]. These CoAs may range from 1 to 3 years.
The standard shipping parcel size from WCSA is 10,500 ~ 11,000 WMT for copper concentrates. On the supply side, ship owners have started to shift towards bigger vessels, mostly ultramaxe classes (60,000 mt capacity) which allow them to lift 5 distinct parcels at one time.

Most vessel owners are targeting regular (almost monthly) orders from large shippers, such as BHP (Spence/Escondida) and MMG (Las Bambas) of 3 x 11,000 WMT or 4 x 11,000 WMT each used as base cargo, Completion cargos are then sought from other miners.

Main destination for copper concentrates for the region is the Far East, and mainly to ports in the North China/Shanghai River regions.

Table 19.11 : Freight Rates

Ocean Freight Matrix	11k WMT 1:1	22k WMT 1:1	22k WMT 1:2
Japan	$53.00	$50.00	$54.00
North Chine / South Korea	$53.00	$50.00	$54.00
South China	$58.00	N/A	N/A
Philippines	$65.00	$58.00	N/A
Europe / Brunsbuttel	$55.00	N/A	N/A
Europe / Finalnd	$75.00	N/A	N/A

Shipments of 22,000 WMT, instead of 11,000 WMT will enable the Costa Fuego Project to capture a lower freight rate for shipments to Asia. The saving is estimated as $3.00 ~ $7.00/WMT.

19.6.2.1.1 Ocean Swell Occurrence Risks

Over the past few years, ocean swell has become increasingly an issue for ports located in WCSA, mainly Peruvian and Chilean Ports.

Shipowners on the coast have experienced heavy delays due to ocean swell mainly in concentrate ports, as these ports are normally built in bays without a breakwater. most of them are finger pier type with dolphins for mooring.

Downtime due to ocean swell can range from 120 days in a year (las Ventanas during 2022) or 8 days such as TPC (2023).

By fixing cargoes with owners with a large number of vessels on the coast, shippers can have some protection as these owners would have the ability to substitute vessels without the need to look for vessels in the market to replace fixtures.

19.7 Ancillary costs and charges

Other costs associated with concentrate sales are:

- Assay charges for assays performed on samples taken at both the ports of loading and discharge for the Seller's party assays (for assay exchanges) and potential Umpire analyses.

These charges are estimated at $2.00 ~ $3.50 per WMT of concentrate.

- Supervision at discharge port:
  - If required, these charges are estimated as $0.30 ~ $2.00 per WMT of concentrate depending upon destination
  - If specialist attendance is required, out-of-country fees (such as airfares and accommodation) may be charged
  - In the case of sales to China, the local authorities (CCIC) may charge a "trespass" fee for access to be granted to a supervisor. Even if a fee is paid, access to all aspects of WSMD may not be permitted
  - Supervision charges may be shared on an equal basis between the Buyer and the Seller where the Buyer is a trader, on-selling the concentrate to a smelter
  - In the case of direct sales to a smelter, the Seller typically pays for the full cost of supervision.

Table 19.12 : Discharge Supervision Fees (USD)								
Country	**Port**	**5,000 ~5,500 wmt**	**10,000 ~11,000 wmt**	**Minimum Fee**	**Draft Survey**	**CCIC/ Customs Fees (per wmt)**	**Fee Basis 11k wmt**	**Fee Basis 11k wmt**
China	All	0.48	0.45	$2,250	$700	$1.45	$21,600	$1.96
India	Dahel	0.55	0.31	N/A	$385		$3,795	$0.35
Jaoan	All	Min Fee	0.278	$2,560	$760		$3,818	$0.35
Philippines	Isabel	0.80	0.50	$4,000	$500		$6,000	$0.55
S Korea	Onsan	0.27	0.27	$1,300	$500		$3,470	$0.32

19.8 Management of Weights and Assays

Best Practice management of shipment weights and assays is one of the most important aspects of sales execution. If not done correctly and to the highest standards, the Costa Fuego Project may face significant losses.

It is imperative that best practice procedures are put in place for final weight and assay determination. These procedures will assist the Costa Fuego Project in mitigating metal losses in shipment finalisations.

The key protocols are:

- Contractual provisions relating to the seller's right of representation at the Ports of Discharge
- Correct determination of the seller's assays (internal laboratory or selection of an external laboratory)
- Protocols for assay exchanges
- Selection of Umpires
- Formulae for determining final assays after Umpire results.

There are several well-known laboratories in South America, Europe and Australia that can undertake seller's analyses of samples for assay exchange. In contrast, most Umpires are located in the United Kingdom or Europe.

It is essential that well-respected and experienced Umpires are utilised in sales contracts.

19.9 Assessment of Costa Fuego Project Options

19.9.1 Marketing Execution

In order to achieve the most competitive terms for Costa Fuego Project concentrates, a broad range of market participants should be approached through a controlled tender process. The tender will call for key terms such as Treatment and Refining Charges, metal payabilities, QPs and payment terms, as well as other "minor" terms such as assaying and umpire protocols and shipping terms.

The high quality of the Costa Fuego Project concentrate will mean that it is an attractive feed in the current market environment for most smelters. Consequently, trader interest in any Costa Fuego Project concentrate sale can be expected to be very high.

Potential buyers of Costa Fuego Project concentrate are listed in the sections below.

19.9.2 Smelter customers

Regional Copper smelters are considered to be those smelters located in:

- Australia
- China – north and south China
- India
- Japan
- Korea
- The Philippines
- Europe.

19.9.3 Trader Customers

There are numerous traders now active in the market for copper concentrates. The traded copper concentrate market is the largest of the base-metals markets and over time numerous traders have entered or left the sector.

The following is a list of traders (and their trading entity domiciles) who are known to be active in the purchase of copper concentrates:

- Arrow (Switzerland)
- Concord (Switzerland)
- Freepoint Commodities (USA)
- Glencore (Switzerland)
- Greenwich Metals (USA)
- Hartree Partners (USA/Switzerland)
- International Material Group (USA)
- Mercuria (Switzerland)
- MRI Trading (Switzerland)
- Ocean Partners (USA)
- Open Mineral (Switzerland)
- Trafigura (Singapore)

- Transamine (Switzerland).

The Japanese Trading Houses (*Shōsha*)

- Marubeni (Japan)
- Mitsubishi RtM (Singapore/Japan)
- Mitsui & Co (Japan)
- Sumitomo Corporation (Japan)

There are numerous other traders who have become active in the market for copper concentrates, albeit to a lesser degree. Some of these traders are the commercial trading arms of both miners and smelters.

Three large mining houses contain active trading sections. These are:

- Anglo-American (Singapore)
- BHP (Singapore)
- Rio Tinto (Singapore).

19.9.4 Marketing Execution

Once sales contracts are in place, shipments must be scheduled, vessels loaded and documents created correctly and despatched in accordance with contractual requirements. There will also be a series of intermediary steps between a vessel's arrival at the port of discharge and the issuance of final documentation and subsequent settlement of the shipment.

Sales execution tasks will include:

- Management of agreed shipping schedules with customers
- Manipulation of available concentrate tonnages (using shipment optionality) and grades to ensure optimal economic outcomes
- Chartering of export vessels
- Monitoring of loading
- Compliance with export regulatory requirements and destination country import requirements
- Compliance with International Maritime Codes
- Monitoring of Quotation Periods for payable metals including any optionality granted to Buyers (typically traders)
- Verification of provisional (loading) weights and assays
- Verification of "final" weights and assays where shipment is made to smelters in China.
- Verification of shipping documents (Bills of Lading, Certificates of Origin, etc.)
- Issuance of provisional invoices and collection of funds
- Monitoring of vessel transit and arrival at the ports of discharge

- Appointment of discharge superintendence representatives
- Monitoring of superintendence of vessel discharge and WSMD
- Review of discharge weights
- Review of detailed party analyses (Seller) and execution of the assay exchange
- Review of exchanged assays and selection of Umpire and sample lots to be despatched to Umpire
- Review of final assay calculations
- Issuance of final invoices and collection of funds
- Settlement of outstanding freight and laytime calculations
- Finalisation of export records with the relevant government bodies (if required).

19.10 NSR Analysis

19.10.1 NSR Summary

Costs are delineated as either Cost Insurance Freight [CIF] or Free on Board [FOB]. The delta between the two classifications reflects the freight costs for delivering the concentrate to the Buyer.

The three offtake scenarios examined are:

1. The Glencore International AG sale
2. Open market sales
3. The average of the two above classifications.

Based upon the parameters outlined in Table 19.3, the following key outputs are generated:

Table 19.13 : NSR Summary[1]

NSR Item	GIAG Offtake	Open Tonnage	All Conc Sales
Gross Revenue (US$/ dmt)	$2,844	$2,844	$2,844
Net Revenue – CIF (US$ / dmt)	$2,457	$2,481	$2,473
NSR Percentage	86%	87%	87%

[1] Source: Internal Memo - Net Smelter Returns

19.11 Molybdenum

Molybdenum production is generally divided into three steps:

- Molybdenum concentrate (MoS_2)
- Roasting to convert MoS_2 to molybdenum oxide (MoO_3)
- Smelting to convert MoO_3 to ferromolybdenum or other chemicals.

The major user of molybdenum is the steel industry. Consequently, global molybdenum demand is closely linked to global steel demand. Some 20% of annual molybdenum production is used in stainless steel, with a further 60% used in specialist steel products.

Primary molybdenum production can be classified as coming from one of two sources:

1. Primary Mines
2. By-Product Mines

The typical commercial structure for a molybdenum concentrate sale is for the buyer to pay for 99% of the analytical molybdenum content (as determined through an agreed sampling programme), less a percentage of loss based on the quality of the input, at reference price averaged over a quotational period (QP), less a deduction for processing, and less deductions for penalty elements, if any.

The principal charge ("Roasting Fee/Charge"), reflecting the transformation charge for the conversion of sulphide to oxide has varied in structure over time. It is either expressed as cents/lb or as a percentage of the quoted Oxide price.

19.11.1 Treatment Charges

The treatment charge is typically a deduction of $1.00 ~ 1.20/lb for clean concentrate greater than 48~50% in molybdenum content. As a mid-point, $1.10 for molybdenum concentrate with Mo > 48% can be assumed. The Treatment Charge may be increased if payable molybdenum is raised to 100% (from 99%).

19.11.2 Penalties

Copper penalty: not actually manifested as a copper penalty, but a larger deduction (from the molybdenum payability rate) would be incurred. This "penalty" can be manifested as an increased treatment charge. Generally, < 0.5% Cu is considered clean; > 0.5% must be blended or, generally, if >1.0%, the concentrate must go through a ferric leach process or autoclave.

If the Mo grade is 50~51% the concentrate will allow for direct roasting; add extra 5~10¢/lb to Roasting Charge/1% < 48~50% (it is recommended to add 10c/lb per 1 % < 48%).

Other elements which may incur penalties are lead, arsenic and zinc.

19.11.3 Payable Metals other than Mo in molybdenum concentrate

Overwhelmingly, rhenium in molybdenum concentrate is not paid. However, if Rhenium is not payable, the occurrence of rhenium may make the concentrate more attractive.
No one known pays for gold in molybdenum concentrate as it is not said to be recovered in the treatment process.

Table 19.14 : Historical Molybdenum Prices			
USD/lb	**2004~2013**	**2014~2023**	**2024 Jan~Sep**
Range US $/lb	10.37 ~ 32.15	6.55 ~ 24.41	$20.77

Molybdenum long-term pricing is assumed as $20.00/lb. As of the date of this Report, the current price (November 2024 Molybdenum Oxide 60% Ex-Works delivery [EXW]) is ~ $22.00/lb.

19.12 Sulphuric Acid

19.12.1 Sulphuric Acid Price in Chile and Peru

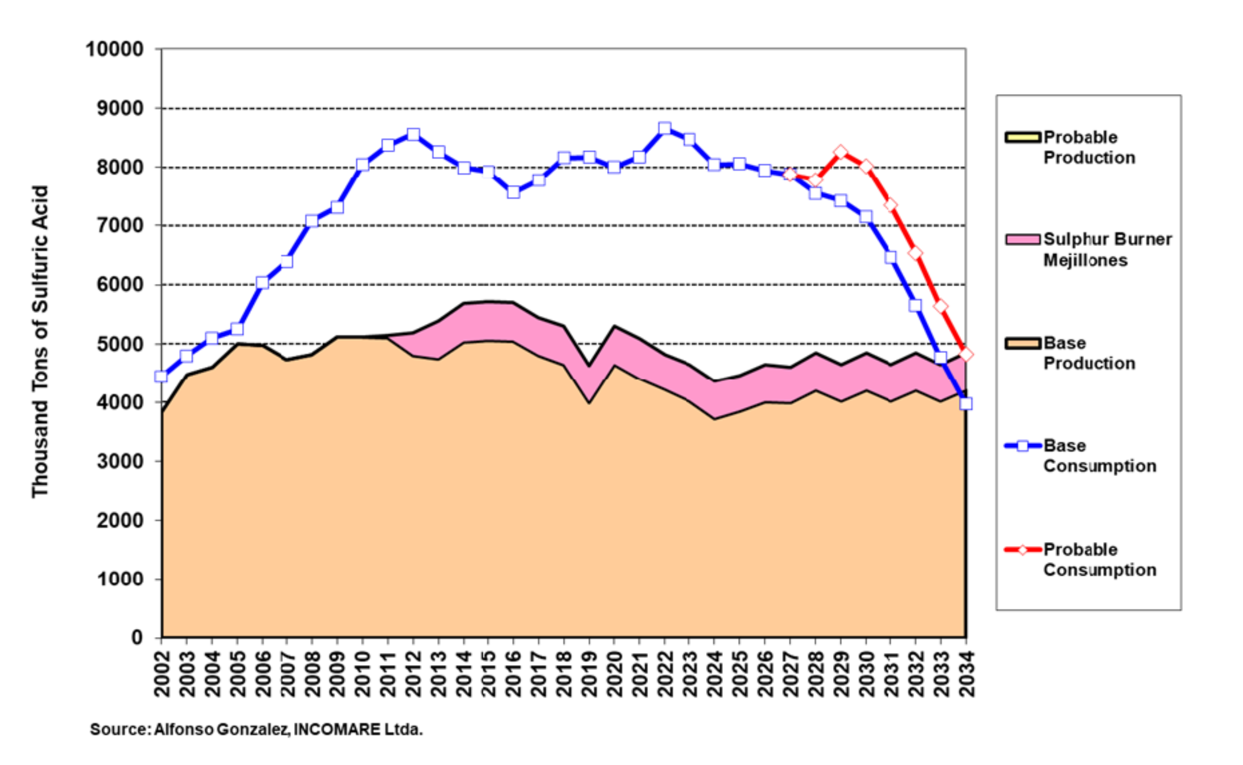
Chile is a net importer of sulphuric acid. The key import ports for acid are Antofagasta and Mejillones. It will also be possible to import acid through the ports of Barquito and Las Ventanas.

In 2024, sulphuric acid production is estimated to have been ~4.36Mt, Domestic sulphuric acid production in Chile is expected to be 4.65-4.85Mtpa for the period 2027 ~ 2034 as a base case.[1]

Consumption in Chile has ranged from ~7.5Mt to 8.5Mt in the period 2016 ~ 2023. In 2024, consumption is estimated to have been ~8.0Mts. Consumption is expected to decline through to 2034, reaching ~ 4.0Mts.[2]

[1] Future Trends for Sulphuric Acid Market in Chile and Peru, Incomare, November 2024, pp4
[2] Future Trends for Sulphuric Acid Market in Chile and Peru, Incomare, November 2024, pp5

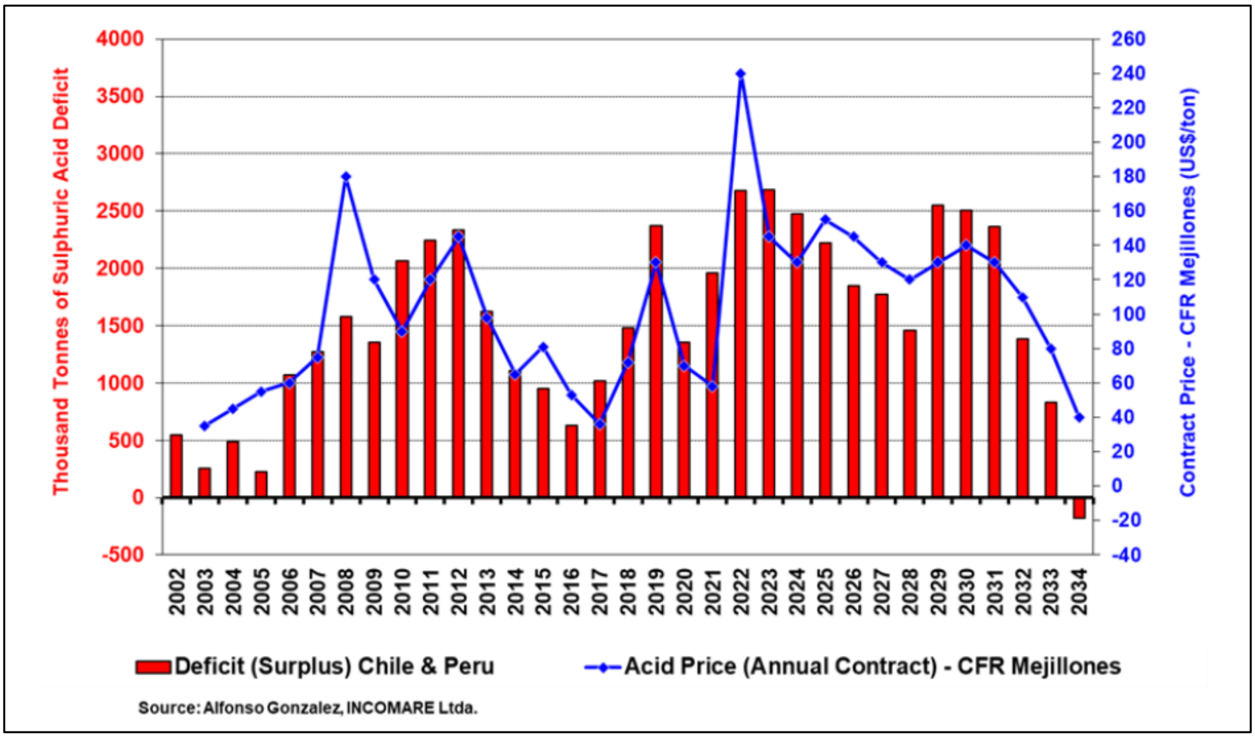
Figure 19.23 : Sulphuric Acid Balance - Chile



Sulphuric acid prices are also forecast to decline from 2029~30, reaching $40/t by 2025.

Although new leaching projects (Marimaca, Chile and Toa Maria, Peru) will increase acid consumption, depletion at other projects (Centinela, El Abra, Zaldivar, Gabriela Mistral and Codelco Norte) will materially reduce the domestic deficit and potentially lead to a surplus by early 2025.

Figure 19.24 : Sulphuric Acid Balance- Chile/Peru & Annual Contract Price



These price forecasts (CFR, Mejillones) suggest a project price of $60/t with an additional $120/t for transportation from the port to the Costa Fuego Project. Alternatively. Sulphuric acid may be transported from the port of Ventanas ($95/t) or the port Barquito ($50/t).

19.13 Salt

The Costa Fuego Project will have an annual requirement for ~ 110ktpa of salt for leaching operations. Salt can be procured within Chile with shipments from Iquique port. However, the annual requirement for salt is expected to be satisfied from the reduction of brine residues from the desalination process undertaken by Huasco Water.

19.14 Copper Cathode sales

The sale price for Costa Fuego Project copper cathodes is assumed to be the LME price, without any premium or deduction. Cathode will be delivered CIF basis to a destination in either the USA or China. The local premia achieved for cathode sales is assumed to defray the cost of export. The transportation of the cathode from the Costa Fuego Project to the port of San Antonio (being the closest port) is $50/t.

19.15 Trucking Market in Chile

Trucking costs are associated with the following products:

- Concentrate to port
- Acid to Site

- Cathode to market / port
- Moly smelter 650kms.

It is expected that the Costa Fuego Project will enter into long-term commercial arrangements with one or more trucking companies to procure favourable rates.

The trucking rate assumed is $52.00/t for moving concentrate to the port.

20 Environmental Studies, Permitting and Social or Community Impact

This section provides details of the following aspects of the Project:

- A summary of environmental studies
- Project permitting requirements, the status of any permit applications, and any known requirements to post performance or reclamation bonds
- Social or community related requirements and plans for the Project
- Plans for waste and tailings disposal, site monitoring, and water management both during operations and post mine closure
- Mine closure (remediation and reclamation) requirements and costs.

20.1 Environmental Studies

The Costa Fuego Project is located in the arid Atacama Region of northern Chile, a region characterised by its extreme environmental sensitivity.

The Project area, being a part of one of the driest places on Earth, presents unique environmental challenges, including water scarcity, biodiversity protection, and ecosystem preservation. Therefore, the environmental studies conducted for the Costa Fuego Project will be designed to provide a detailed understanding of these challenges and to establish robust measures to mitigate potential impacts throughout the mine's lifecycle, from construction to operation and ultimately closure.

These environmental studies are aligned with both Chilean environmental laws, such as Ley 19.300 (Environmental Framework Law), and international standards, including NI 43-101 and CIM ESG guidelines. This comprehensive approach ensures that the Project adheres to both national regulatory frameworks and global best practices, fostering sustainable development and responsible mining.

The Environmental Impact Assessment (EIA) of the Project will be submitted using the EIA System that is currently being applied in Chile.

20.1.1 Baseline Environmental Studies

The baseline environmental studies for the Costa Fuego Project have been conducted to establish a comprehensive environmental profile of the Project area prior to the commencement of any major construction activities. These studies provide the foundation for all subsequent environmental impact assessments (EIA) and mitigation strategies, serving as a reference point for evaluating changes in environmental conditions over time.

Since 2012, multiple baseline campaigns have been conducted over approximately 11,000 hectares, including climate/meteorology, air quality, noise, surface water and groundwater, flora and fauna, landscape, and cultural heritage. Studies on socio-economic, local communities, included indigenous communities were also undertaken.

Environmental baseline studies have been initiated for all four deposits—Productora, Alice, Cortadera, and San Antonio—as well as for the proposed tailings storage facility (TSF), and the Project's infrastructure corridors, including the seawater intake, and seawater supply pipeline, access roads, and RopeCon route. A comprehensive map of the baseline coverage areas and findings will be included in the EIA.

20.1.1.1 Biodiversity and Ecosystem Health

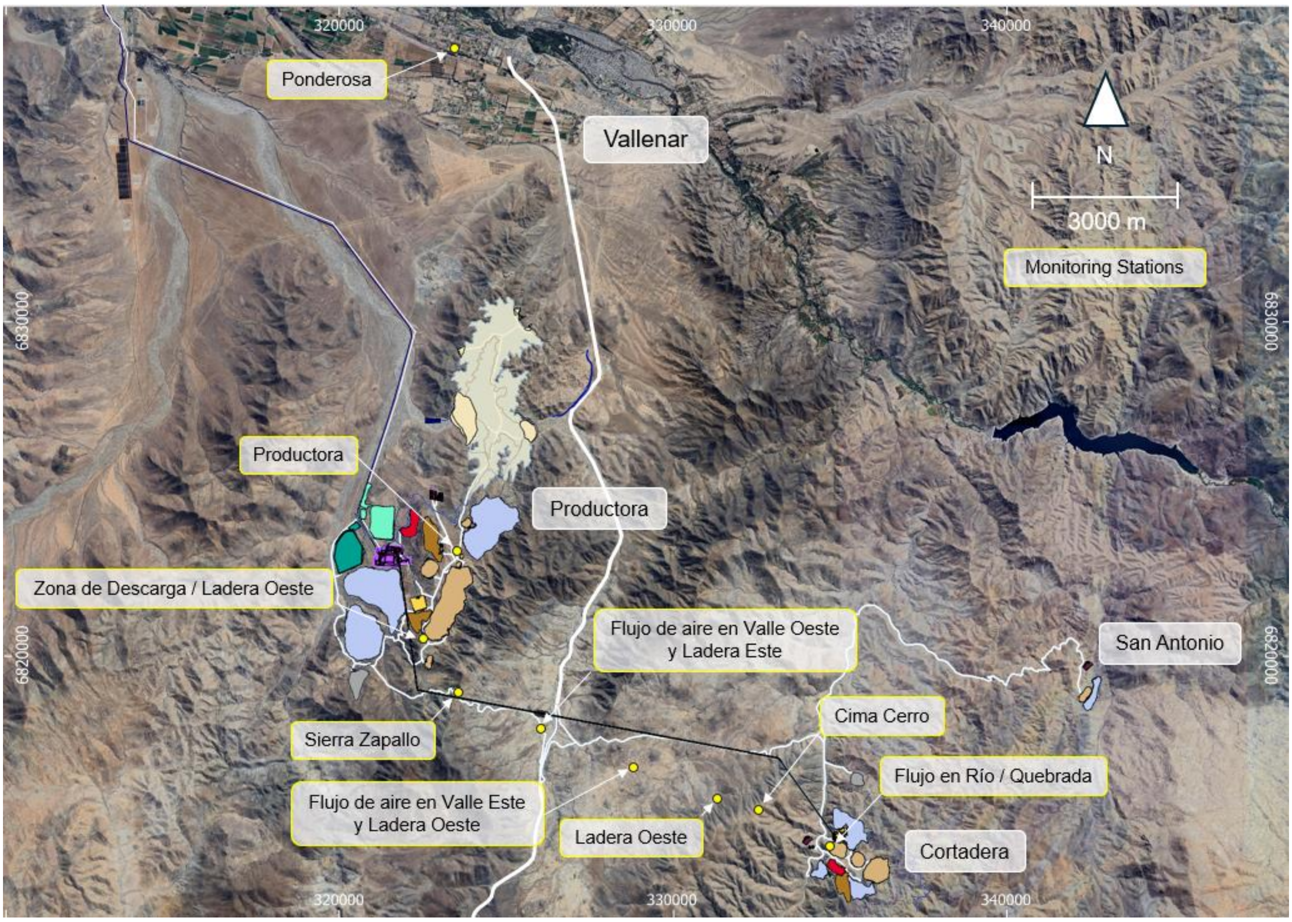
The Atacama Region is home to a variety of unique plant and animal species that have adapted to survive in this extreme environment. The baseline biodiversity studies focused on identifying the presence of species classified as endangered, vulnerable, or endemic to the region. Field surveys conducted in collaboration with local ecological experts documented the presence of xerophytic flora (shrubs and cacti) and relevant fauna, such as viscacha, guanaco and puma, in areas potentially affected by the mining activities of the Project.

Several other species of plants and small mammals were identified in proximity to the Project site, including species that are considered vulnerable due to habitat degradation from historical mining and agricultural activities in the region. The Costa Fuego Project planning working included the identification of these areas and consideration of mechanisms to reduce impact within the design.

20.1.1.2 Air Quality and Climate Conditions

Given the location of the Costa Fuego Project in an arid desert environment, the potential for air quality degradation due to mining activities—particularly from dust generation during drilling, blasting, and hauling operations—has been identified. Baseline air quality measurements were taken across nine monitoring stations set up within and around the Project area to establish pre-development levels of particulate matter (PM_{10} and $PM_{2.5}$) and other potential airborne pollutants (SO_x and NO_x gases). The historical database includes three years of continuous monitoring of PM and gases and is expected to continue towards EIA submission. Figure 20.1 shows the location of all Air Quality Monitoring stations that have informed the PFS. At the time of the PFS three monitoring stations remain active being Ponderosa, Productora and Agua Amarga.

Figure 20.1 Locations of Air Quality Monitoring Stations (HCH, 2025)



In response to the arid conditions and the resulting susceptibility to dust storms, the Project has integrated a dust suppression strategy that relies on the use of seawater for dust control. This approach is both environmentally responsible and efficient, as it minimises the consumption of scarce freshwater resources while controlling airborne dust emissions. This strategy is accepted practice within Chile[1].

The baseline air quality studies also evaluate the impact of prevailing wind patterns, which can influence the dispersion of dust and other emissions.

[1] Source The discharge of liquids that may interact with groundwaters is provided for in by Decree No. 46/2022, Law Chile - Decree 46 17-JAN-2003 MINISTRY OF THE GENERAL SECRETARIAT OF THE PRESIDENCY - Library of the National Congress. Dust emissions, including methods for suppression including the use of Bischofite (from sea water), will be evaluated according to the Environmental Law No 19.300, article 5, within the EIA permitting process.

20.1.1.3 Water Resources

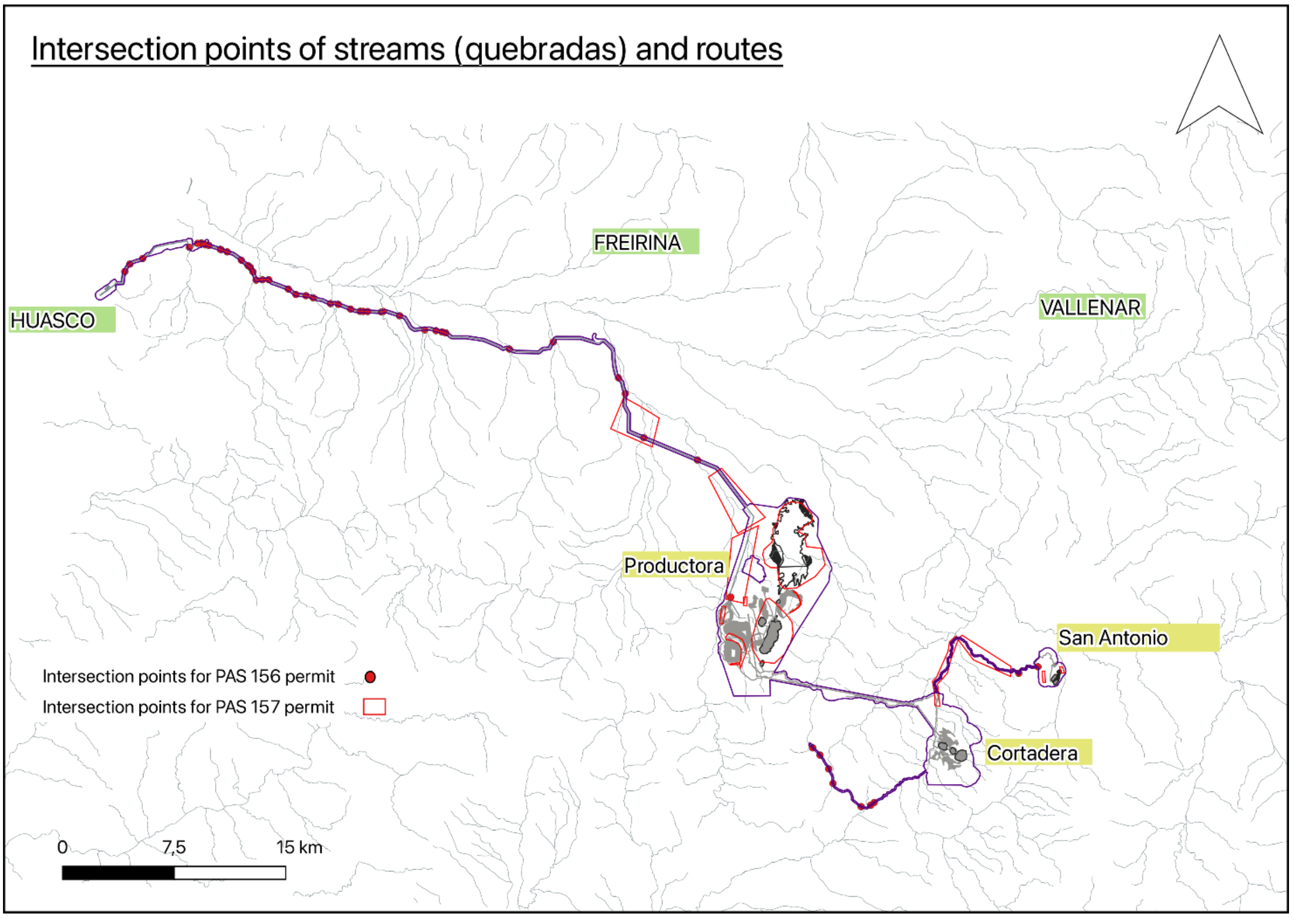
Water scarcity is a critical environmental issue in the Atacama region. As a result, the baseline hydrological studies focused on assessing the availability, quality, and potential impacts on both surface and groundwater resources. Despite the region's extreme aridity, some ephemeral watercourses and underground aquifers provide essential water sources for local agriculture and community use.

The hydrological studies[1] revealed that the surface watercourses in the vicinity of the Project are highly seasonal, flowing only after rare, intense rainfall events. Groundwater resources, while limited, are critical to the local communities and agricultural activities. To minimise the impact on these water sources, the Costa Fuego Project has implemented a water management strategy that relies primarily on seawater for processing needs and dust suppression. The PFS includes the delivery of raw seawater to the Project via a pipeline from the Pacific coast. The PFS considered the Costa Fuego Project as a customer of the Huasco Water opportunity.

The EIA will include a comprehensive assessment of the potential impacts of seawater use at the Project, specifically regarding the use as a dust suppressor. The PFS considered the delivery of seawater to Productora and Cortadera using a pipeline network and the inclusion of local small reverse osmosis plants at the sites for process and worker's use. Environmental studies related to the management of the resulting brines and waste seawater are continuing for inclusion into the EIA submission.

[1] Costa Fuego Copper Project Hydrology and Hydrogeology PFS Study. Piteau and Associates. January 2025

Figure 20.2 Hydrology network including the identification of creeks in relation to the Costa Fuego Project (HCH, 2025)



20.1.2 PFS Hydrological Studies

Hydrological studies were prepared by Piteau Associates, consulting to HCH. A summary of the PFS findings is provided below with the full report available within the appendices of this report. The hydrological study was conducted based upon the Project layout as of September 2024 and considered features relevant to that layout that have since been updated. A revision of the hydrological studies is underway to reflect the final PFS layout within the EIA submission and additional studies by Hot Chili Limited. The features to be revised are:

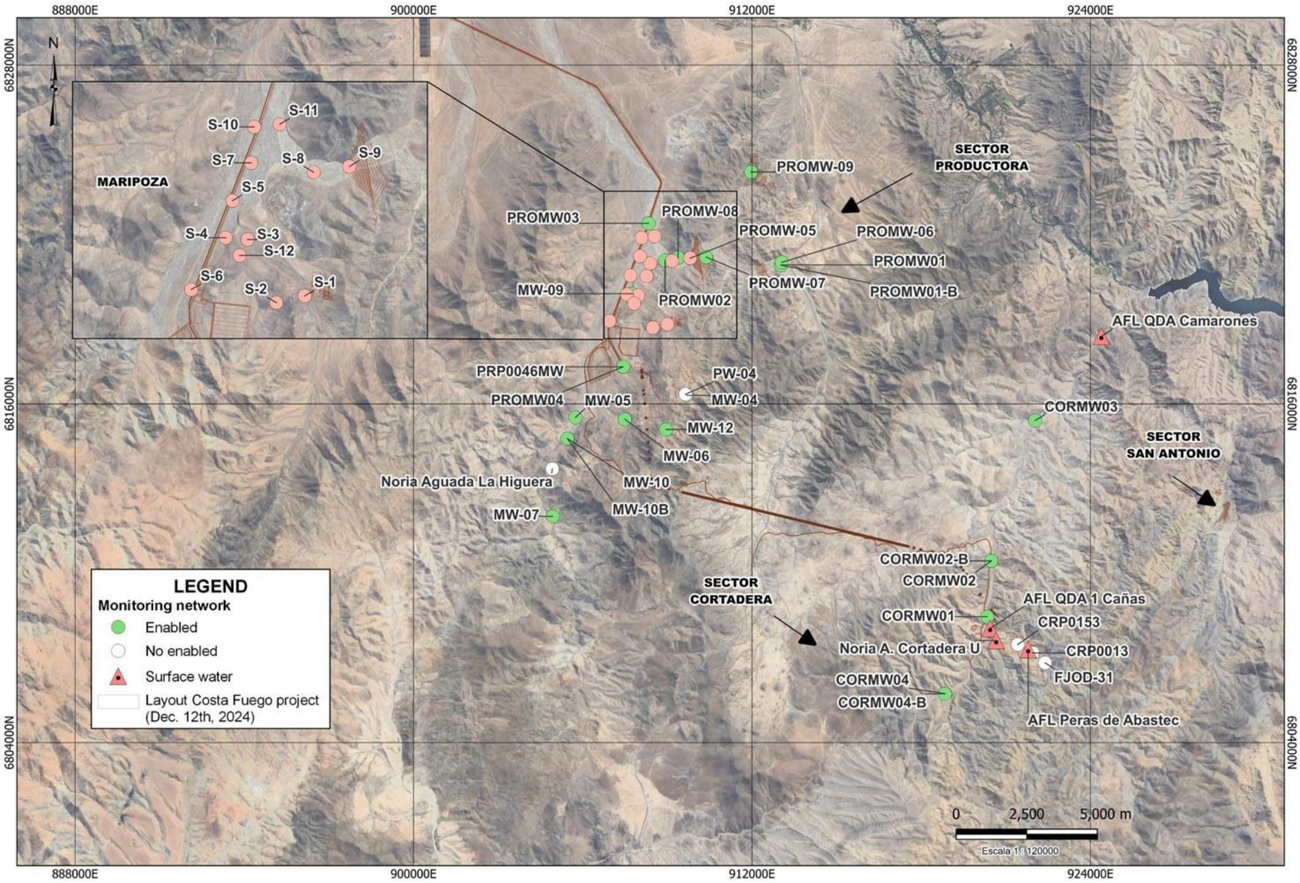
- The liner strategy within the TSF, namely the change from a dual (HDPE & GCL) liner across the embankments and basin, to a single liner (GCL) across the embankments and over alluvial sediments within the basin. A HDPE liner is also installed in the low points of the TSF basin and along the supernatant pond migration path.
- The in-pit tailings strategy at Productora, with additional studies to ascertain potential impacts on hydrology, the recommendations for liners or other preparations and for closure.
- The mining plans, including updated pit and underground designs and the impact of pit backfilling with mine waste at Cortadera on surface water management.

20.1.2.1 Hydrogeological Network

The Costa Fuego Project hydrogeological monitoring network includes standpipe piezometers, open boreholes and surface seepage or shallow wells. The locations of the network points used within the PFS hydrological studies are presented in Figure 20.3. Locations cover the Productora, Cortadera and TSF sites.

The baseline water quality studies also measured parameters such as pH, total dissolved solids (TDS), and heavy metal concentrations in both surface and groundwater sources.

Figure 20.3 Hydrogeology Network Source Costa Fuego Project Hydrology and Hydrogeology PFS Study (Piteau Associates, 2025)



20.1.2.2 Productora and Alice Sites

A previous PFS was conducted at Productora in 2016, before the consolidation of the Costa Fuego Project, by Artois. This data was reviewed and incorporated by Piteau into this current PFS hydrological study.

Table 20.1 Monitoring Network Details Productora and Alice												
Site	**Drilling Method**	**Depth (m)**	**Diameter**	**X**	**Y**	**Z**	**Pipes**	**Blind Pipes**	**Screen Pipes**	**Stick Up**	**Dip**	**Bentonite Seal**
MW-04	RC	200	6"	323478	6822169	826.7	PVC 4"	0.0 to 72.0 m	72.0 to 200.0 m	0.61	-90°	5.0 to 10.0 m
PW-04	RC	200	12"	323433	6822209	819.8				0.29	-90°	
MW-05	RC	200	6"	319584	6821191	631.3	PVC 4"	0.0 to 98.0 m	98.0 to 198.0 m	0.6	-90°	5.0 to 10.0 m
MW-06	RC	150	6"	321336	6821217	686.6	PVC 4"	0.0 to 66.0 m	66.0 to 117.0 m	0.7	-90°	5.0 to 10.0 m
MW-07	RC	150	6"	318958	6817659	710.5	PVC 4"	0.0 to 66.0 m	66.0 to 117.0 m	0.4	-90°	5.0 to 10.0 m
MW-09	RC	150	6"	321386	6825668	541	PVC 4"	0.0 to 80.0 m	80.0 to 150.0 m	0.7	-90°	5.0 to 10.0 m
MW-10	RC	150	6"	319346	6820429	645.7	PVC 4"	0.0 to 60.0 m	60.0 to 150.0 m	0.13	-90°	5.0 to 10.0 m
MW-10B	RC	45	6"	319336	6820433	645.9	PVC 4"	0.0 to 5.0 m	5.0 to 45.0 m	0.18	-90°	5.0 to 10.0 m
MW-12	RC	200	6"	322811	6820931	842.8	PVC 4"	0.0 to 50.0 m	50.0 to 200.0 m	0.4	-90°	5.0 to 10.0 m
PROMW01	RC	21	5.6"	326579	6826946	596.7	PVC 3"	0.0 to 2.74 m	2.74 to 20.08 m	1.01	-90°	0.0 to 1.0 m
PROMW01-B	RC	80	5.6"	326580	6826941	597.3	PVC 3"	0.0 to 31.49 m	31.49 to 79.13 m	1.0	-90°	0.0 to 1.0 m 18.0 to 22.0 m
PROMW02	RC	84	5.6"	322476	6826907	516.7	PVC 3"	0.0 to 53.04 m	53.04 to 82.78 m	1.0	-90°	0.0 to 1.0 m 30.0 to 34.0 m
PROMW03	RC	114	5.6"	321823	6828182	494.9	PVC 3"	0.0 to 16.94 m	16.94 to 112.2 m	1.1	-90°	8.0 to 12.5 m
PRP0046MW	RC	148	5.6"	321180	6823072	617.5	PVC 3"	0.0 to 29.95 m	29.95 to 131.2m	0.83	-90°	14.0 to 18.0 m
PROMW-05	RC / DDH	100.8	PQ: 122.6 mm	323354	6827024	531	PVC 3"	0.0 to 70.41 m	70.41 to 100.09 m	1.14	-90°	0.0 to 0.2 m 6.0 to 55.0 m
PROMW-06	RC	42	140 mm	326602.3	6827041	597.4	PVC 4"	0.0 to 35.24 m	35.24 to 41.18 m	1.12	-90°	0.0 to 2.0 m 31.0 to 37.0 m

Table 20.1 Monitoring Network Details Productora and Alice												
Site	**Drilling Method**	**Depth (m)**	**Diameter**	**X**	**Y**	**Z**	**Pipes**	**Blind Pipes**	**Screen Pipes**	**Stick Up**	**Dip**	**Bentonite Seal**
PROMW-07	RC	36	140 mm	323916.8	6827093	541.2	PVC 4"	0.0 to 22.69 m	22.69 to 34.56 m	1.37	-90°	0.0 to 1.0 m 19.3 to 21.3 m
PROMW-08	RC	84	140 mm	322926.9	6827022	523.9	PVC 3"	0.0 to 71.35 m	71.35 to 83.24 m	1.21	-90°	14.6 to 19.8 m 58.0 to 63.0 m
PROMW-09	RC	66	140 mm	325353.1	6830180	611.5	PVC 4"	0.0 to 29.35 m	29.35 to 64.98 m	1.0	-90°	0 to 5.15 m 16.5 to 19.1 m
Noria Aguada La Higuera		318858	6819335	684				Surface seepage				
AFL Poza		323505	6823783	668				Surface seepage or shallow well				
Underground mine Minera Playa Brava		323535	6822495	-				Approx.. Coordinates, flooded underground mine in La Productora deposit				

A hydrogeological characterisation and conceptual model was developed within the pit areas of Productora and Alice. This model found that the units showed low permeability and perform as an aquitard system. The phreatic surface is approximately 50 to 80 m below the surface, and groundwater can be pressurised within this system until it can be released by drilling or mining excavations.
The low permeability and arid environment result in a hydrogeological recharge assumption that is extremely low, around 1 l/s.

Future phreatic surface modelling (mine dewatering modelling) was conducted on the pit shells, with a minimum, base case, and maximum model. The inflow estimate ranged from 10 to 25 L/s. Standby pumping equipment capable of handling 100 l/s for the 1 in 100-year rainfall event was noted. Open pit pumping specifications were provided in stages with pit progression, and the recommendation noted that the areas high evaporation rate of 1,200 mm/year would accommodate the volumes of water produced.

Surface water is addressed with non-contact water diversion channels, and emergency ponds located adjacent to waste rock emplacements. These emergency ponds are to act to capture potential acid rock drainage (ARD) contaminated contact waters during a large rain event.

20.1.2.3 Cortadera Site

This PFS hydrological study is the first such study for Cortadera. Fourteen hydrogeological monitoring sites inform the hydrogeological network around Cortadera. Twenty-three hydraulic Lugeon tests using inflatable packer tools were performed to estimate the in situ hydraulic conductivity for discrete interval depths.

Table 20.2 Hydrogeological monitoring network at Cortadera

Site	Drilling Method	Depth (m)	Diam. (m)	X	Y	Z	Pipes	Blind Pipes	Screen Pipes	Stick Up	Dip	Bentonite Seal
CORMW01	RC	80	5.6"	334497	6814892	885.9	PVC 4"	0.0 to 31.38 m	31.38 to 78.27 m	0.97	-90°	0.0 to 2.75 m
CORMW02	RC	60	5.6"	334544	6816870	840.5	PVC 4"	0.0 to 12.11 m	12.11 to 59.01 m	0.99	-90°	0.0 to 2.0 m & 8.0 to 11.0 m
CORMW02-B	RC	22	5.6"	334539	6816871	840.5	PVC 4"	0.0 to 8.65 m	8.65 to 20.53 m	1.04	-90°	0.0 to 2.0 m & 5.0 to 7.5 m
CORMW03	RC	24	5.6"	335843	6821907	716.9	PVC 4"	0.0 to 6.95 m	6.95 to 18.85 m	1.02	-90°	5.0 to 7.0 m
CORMW04	RC	42	5.6"	333145	6812098	970.7	PVC 4"	0.0 to 23.25 m	23.25 to 40.88 m	0.99	-90°	5.5 to 8.0 m
CORMW04-B	RC	12	5.6"	333140	6812092	971.0	PVC 4"	0.0 to 4.73 m	4.73 to 10.71 m	0.95	-90	-
CRP0153	RC / DDH	102		335621	6813964	977.1	Borehole					
CRP0013	RC / DDH	1185.9		336163	6813692	1019.8	Borehole					
FJOD-31	RC / DDH	728.1		336621	6813365	1059.9	Borehole					
AFL QDA 1 Cañas				334619	6814488	912	Surface seepage					
Noria A. Cortadera U				334858	6814066	935	Surface seepage					
Noria A. Cortadera L				334906	6814022	922	Surface seepage					
AFL Peras de Abastec				336002	6813803	1014	Surface seepage					
AFL QDA Camarones				338029	6824997	631	Surface seepage					

A hydrogeological characterisation and conceptual model was developed within the pit and underground block cave of Cortadera. This model found that the units showed low permeability and perform as an aquitard system. The phreatic surface is approximately 17 to 85 m below the surface, and groundwater can be pressurised within this system until it can be released by drilling or mining excavations.

The low permeability and arid environment result in a hydrogeological recharge assumption that is extremely low, averaging 0.5 L/s in the pits and 1.5 L/s in the future underground depths.

Future phreatic surface modelling (mine dewatering modelling) was conducted on the mine designs, with a minimum, base case, and maximum model. The inflow estimate ranged from 3 to 22 L/s within the pits and 58 to 116 L/s in the underground workings.

Surface water is addressed with non-contact water diversion channels. No emergency ponds were recommended in the hydrology study, with contact water drainage across the planned site feeding into the pits and resulting contact water to be handled by the in-pit pumping system. The hydrological study contact water drainage design was later impacted by a change in the mining approach, where waste rock is backfilled into Cuerpo 1 and Cuerpo 2 pits. This change was mitigated by the inclusion of an emergency contact water pond at Cortadera into the PFS, and the change in mining approach will be adjusted in future hydrology studies.

20.1.2.4 San Antonio Site

Drilling at San Antonio has largely been dry (not finding water in the drillholes) and a hydrogeological network was not yet established for the PFS. Hot Chili will be expanding its hydrogeological network to San Antonio within its EIA submission.

20.1.2.5 TSF Site

The study included the hydrogeological characterisation of the Quebrada Las Arenas, which houses the planned TSF site. The PFS study focused on potential TSF seepage and an evaluation of groundwater monitoring requirements and seepage interception solutions.

The hydrogeological network includes nine monitoring boreholes and 77 in situ hydraulic tests within the TSF footprint.

Hydrogeochemical analysis of groundwater flowing within the footprint was found to be consistent within the proposed TSF site, and distinct from adjacent samples in the Quebrada La Higuera.

Hydrogeological units were conceptualised and modelled and found to show medium to extremely low permeability and extremely low groundwater storage capacity, defining it as an aquitard system where saturated.

A series of site investigations, including drill holes, test pits and geophysics, are planned within the TSF footprint in the next stage to support the environmental studies and design work in preparation for the EIA submission.

An option for in-pit tailings deposition at Productora is also being investigated, and these investigations are documented in the PFS. Investigations and analysis will conclude prior to the EIA submission for this option.

20.1.3 PFS Acid Rock Drainage Studies

Geochemical classification of Acid Rock Drainage (ARD) studies were prepared by Piteau Associates, consulting to Hot Chili Limited. A summary of the PFS findings is provided below with the full report[1] available within the appendices of this report.

The PFS was informed by 122 Acid Rock Drainage (ARD) samples from drill core and chip samples to support waste rock characterisation (Figure 20.4-Figure 20.6). During the PFS process a further 122 samples were collected for the EIA submission with results pending.
The PFS round of sampling included 85 samples at Cortadera and 37 samples at Productora, with analysis methods including acid base accounting (ABA), net acid generation (NAG) and geochemical methods.

The second round of sampling included 13 samples at Alice, 38 samples at Cortadera, 49 samples at Productora, and 22 samples at San Antonio. Samples were selected following a gap analysis of the proposed mining shapes and geological domains within these volumes. Samples were submitted for a combination of the original ABA, NAG and geochemical methods, along with compositing of samples to complete humidity cell tests (HCT) which provide validation to the original ABA and NAG suites over a longer period of time.

The PFS analysis of the original 122 ARD samples found sufficient correlation to geochemistry to allow for an interim surrogate model using S% and Ca% to approximate acid generation and acid neutralisation respectively (Equation 1). The relationship between NNP and NAG acidity to pH 4.5 is consistent across both measured and surrogate NNP. Both methods indicate that with an NNP of <0 kg $CaCO_3$/t, materials may be liable to generate acidity. This interim relationship was included within the PFS workstreams including the design and costing of base cases for waste rock dumps, the base of the heap and dump leach pads, and the TSF design including embankment construction material and capping material.

The interim relationship was coded into the mine-max mine schedule and run to categorise the net neutralising potential for waste rock from the Productora pits at the time of construction of the TSF and confirmed that adequate quantities of non-net acid generating material was available for the construction of the TSF embankments.

Waste rock dumps do not include ground preparations as an ARD mitigation, where the arid environment would not support a perpetually wet stockpile. Contact water ponds are included at Productora for collection of ARD products which are then treated by the water treatment facility. Infrastructure that is designed to be wet, being the heap and dump leach pads and parts of the TSF, feature ground preparation liners to mitigate ARD risk.

[1] Costa Fuego Copper Project Geochemical Assessment PFS Study. Piteau Associates. November 2024

Equation 1 : Surrogate Net Neutralization Potential

$$NNP_{surrogate\ (kg\ CaCO_3/\ t)} = NP_{surrogate\ (kg\ CaCO_3/\ t)} - AP_{surrogate\ (kg\ CaCO_3/\ t)}$$

Where:

$$NP_{surrogate\ (kg\ CaCO_3/\ t)} = \left\lfloor \frac{Ca\%}{40.08} \right\rfloor \cdot 100.09 \cdot 10$$

$$AP_{surrogate\ (kg\ CaCO_3/\ t)} = S\% \cdot 31.25$$

An additional stream of ARD sampling was conducted on tailings products, as part of the TSF design workstream. The results of these tailings ARD samples were forwarded to Piteau following the completion of the PFS ARD studies and will be included in the EIA and future studies for the Project.

Figure 20.4 Productora ARD Sampling Distribution (HCH, 2025)

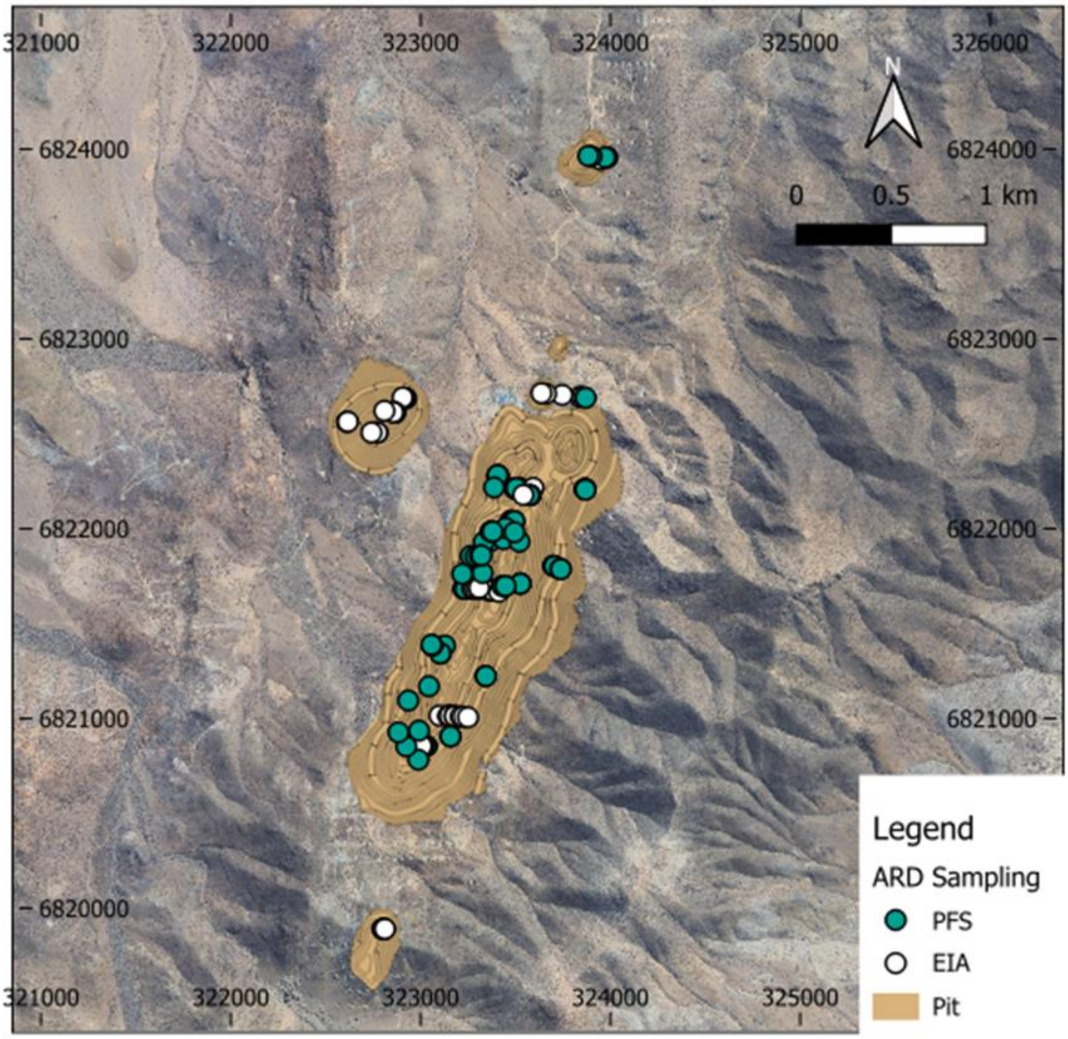


Figure 20.5 Cortadera ARD Sampling Distribution (HCH, 2025)

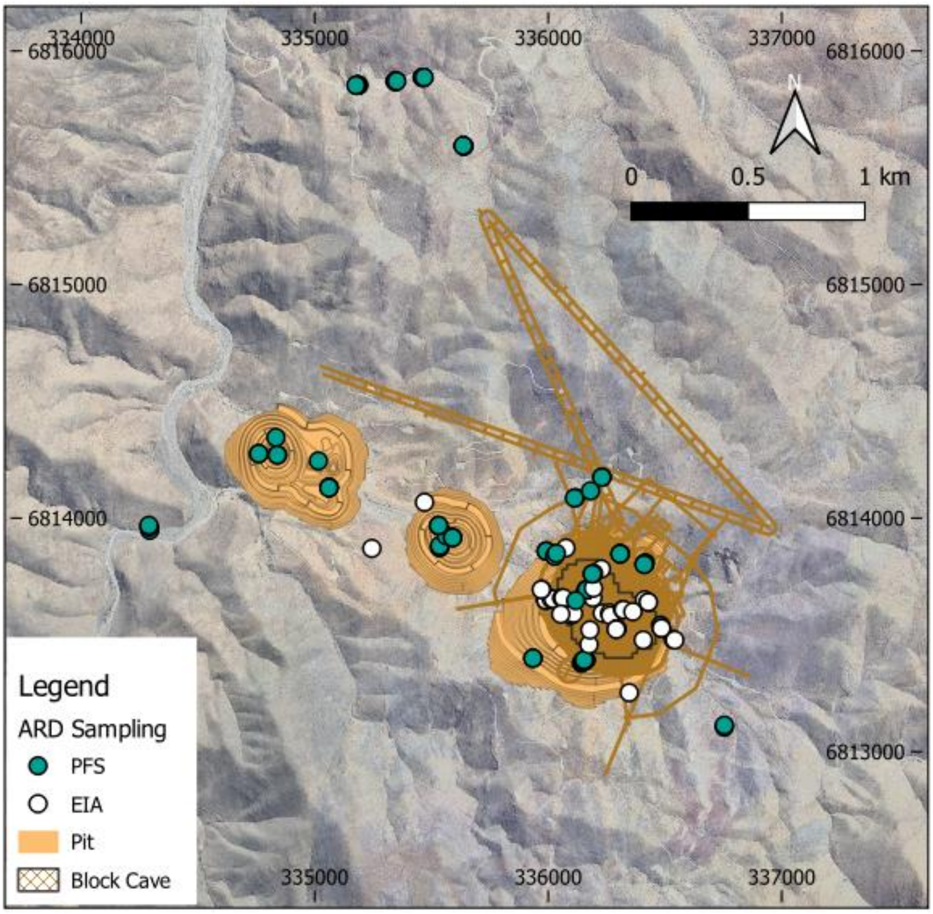
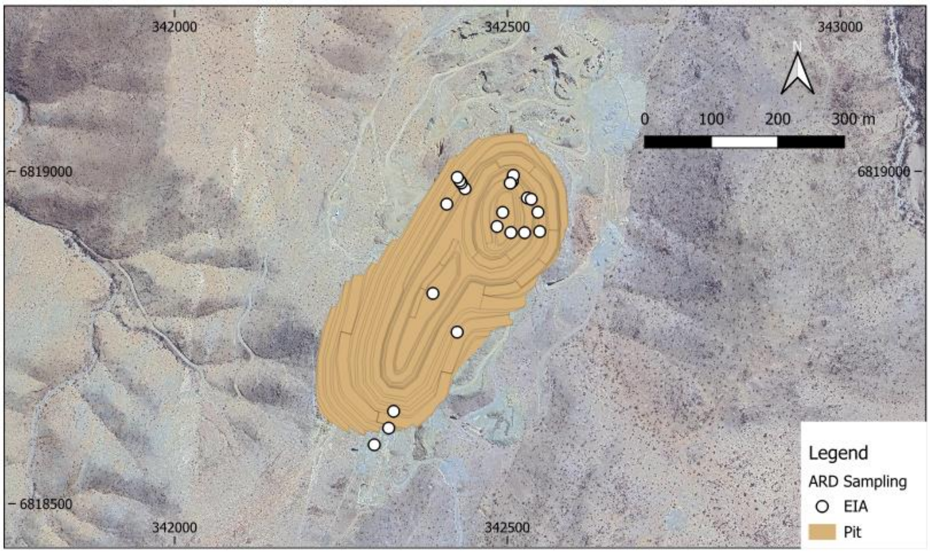


Figure 20.6 San Antonio ARD Sampling Distribution (HCH, 2025)



20.2 Permitting

The permitting process for the Costa Fuego Project is a comprehensive and multifaceted effort that ensures all aspects of the Project are aligned with Chilean legal requirements and international environmental and social standards. Permitting involves securing multiple environmental, water, land, and operational permits required at each stage of the Project's lifecycle, from exploration through to mine closure. The process is designed to guarantee that all potential impacts are thoroughly assessed and mitigated and that the rights of local communities, including indigenous groups, are respected.

Table 20.3 Permits and timing for the Costa Fuego Project

Permit Name	Applicable Authority	Status	Notes
Water Abstraction Permit	DGA (Dirección General de Aguas)	To be requested	Permit related to monitoring activities, no new water rights required
Maritime Concession	Subsecretaría para las Fuerzas Armadas	Obtained	Covers seawater intake and discharge infrastructure
Environmental Qualification Resolution (RCA)	SEA (Servicio de Evaluación Ambiental)	Not initiated	Full EIA will be submitted in next phase. The RCA is the outcome of the EIA application
Mining Concession Titles	SERNAGEOMIN	Obtained	Already granted and active
Archaeological Authorization (if applicable)	Consejo de Monumentos Nacionales	Not triggered	Based on future site clearance, to be requested if archaeological findings are encountered
Indigenous Consultation Process	CONADI / SEA	Not triggered	Current activities are identifying Indigenous Peoples under ILO 169. Formal Indigenous Consultation Process to commence following EIA submission, as prescribed by Authority.
Mining Environmental Sectorial Permits	SERNAGEOMIN	Not triggered	Permits related to stockpile configurations, TSF chemical and physical stability and closure plan. To be requested with EIA submission

Note: This list will be expanded and updated in the Feasibility Study phase and as the EIA progresses.

In Chile, mining projects must comply with a robust regulatory framework led by Ley 19.300 (Environmental Framework Law) and Law No. 20.551 on mine closure. Additionally, the Project's adherence to NI 43-101 and the CIM ESG guidelines, which governs mineral project disclosures in Canada, ensures that the Project aligns with best practices in environmental and social governance. These standards emphasise transparency, environmental stewardship, and community engagement—key principles in the Costa Fuego Project's development strategy.

20.2.1 Environmental Impact Assessment (EIA)

The Environmental Impact Assessment (EIA) is central to the Costa Fuego Project's permitting process, providing a detailed evaluation of the potential environmental impacts the Project may have throughout its lifecycle. As required by Ley 19.300, the EIA systematically addresses issues such as air and water quality, biodiversity, ecosystem health, noise, and the social impacts on surrounding communities. By conducting this rigorous assessment, the Project not only ensures regulatory compliance but also identifies mitigation measures that promote environmental sustainability and community well-being.

The EIA for the Costa Fuego Project is being prepared by a dedicated environmental and social consultancy in collaboration with Hot Chili Limited and with consultation with local stakeholders. It incorporates baseline environmental data collected during the exploration and pre-development stages, ensuring that all potential environmental impacts are thoroughly evaluated.

20.2.1.1 Social Impact Assessment

The EIA also incorporates a Social Impact Assessment (SIA), which evaluates the potential effects of the Costa Fuego Project on local communities, including indigenous groups. The SIA focuses on issues such as land use, cultural heritage, employment opportunities, infrastructure development, and community well-being. Special attention has been given to the needs and concerns of indigenous communities in the region, as required by Chilean law and international standards such as the Free, Prior, and Informed Consent (FPIC) principle outlined in ILO Convention No. 169.

The Project's social engagement efforts include public consultations, community workshops, and ongoing dialogue with local stakeholders. These activities aim to ensure that the Project's development is inclusive and that the benefits, such as job creation and infrastructure improvements, are shared with local communities. The SIA will also outline the Project's commitment to preserving cultural heritage sites in-situ or in a controlled relocation, particularly those of significance to indigenous groups, and includes mitigation measures such as the careful documentation and protection of archaeological findings.

20.2.2 Water Usage Permits

Water usage is tightly regulated in Chile, especially in arid regions like the Atacama Region, where water resources are scarce and essential for both human consumption and agriculture. The Costa Fuego Project's decision to use seawater for its operations significantly reduces its impact on local water supplies, aligning with both national regulations and international sustainability goals.

Water permits related to the dewatering activities of the proposed mining operations are required separate to the land and mining permits. Hot Chili is applying for these permits in conjunction with the Costa Fuego Project EIA submission. Also, these permits are not related to the grant of new water rights, so no issue is raised due to the Huasco valley basin is currently closed to new water rights.

Hot Chili Limited has secured permits for seawater intake, granted through its Maritime Concession permit. Seawater intake, and management of the associated infrastructure, will be addressed within the Huasco Water feasibility studies. Huasco Water is a separate entity to Hot Chili Limited, with Hot Chili Limited holding 80% ownership, and the Costa Fuego Project is anticipated to be a customer of Huasco Water.

20.2.3 Land Access and Indigenous Rights

Securing land access permits is a key requirement for the development of the Costa Fuego Project. Given the presence of local and indigenous communities in the area, the Project has taken care to engage with these groups through a transparent and inclusive process. In accordance with Chile's commitments to indigenous rights under ILO Convention No. 169, the Project has adhered to the principle of Free, Prior, and Informed

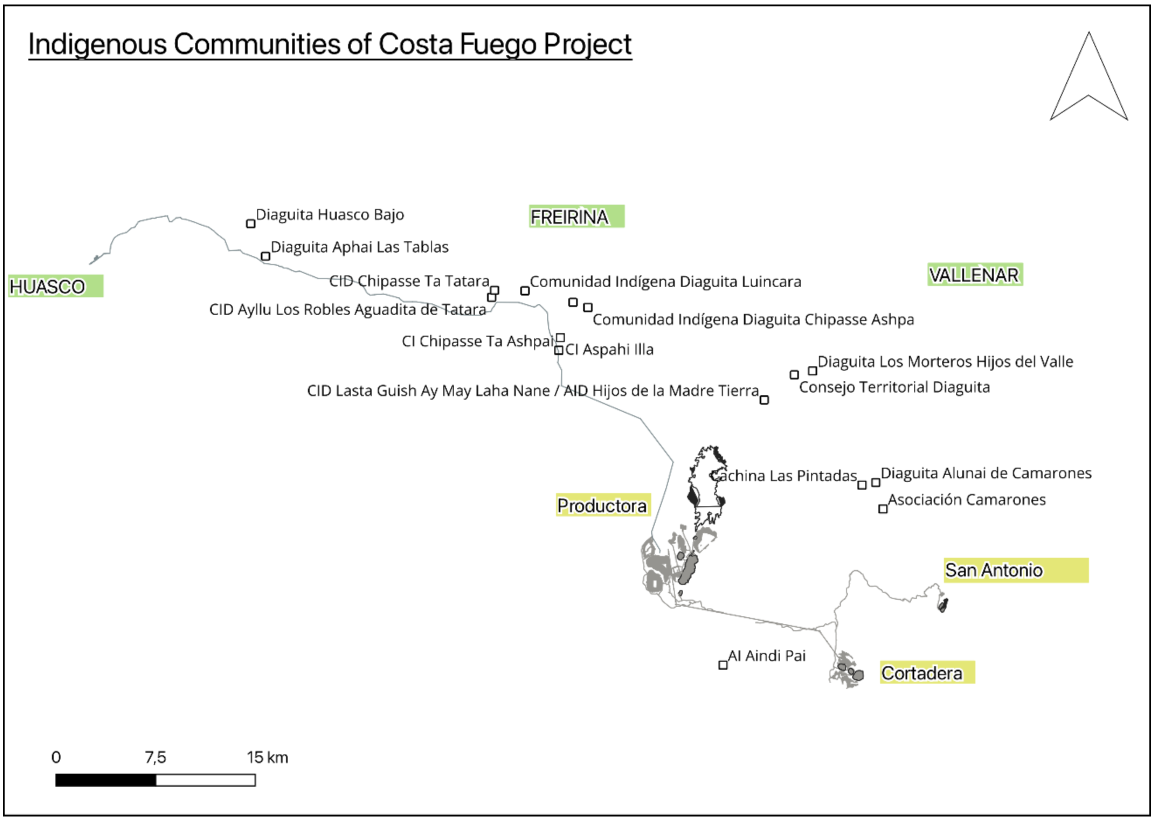
Consent (FPIC), ensuring that indigenous communities are consulted before any development occurs on their traditional lands.

The consultation process includes regular meetings with community leaders, public forums, and the distribution of information about the Project's potential impacts. These consultations are designed to address concerns related to land use, cultural heritage preservation, employment opportunities, and environmental stewardship. The Project is well advanced in reaching agreements with local landowners and indigenous groups to ensure that land access is granted in a way that respects their rights and provides tangible benefits to their communities.

In addition to the FPIC process, the Costa Fuego Project has committed to preserving and protecting archaeological and cultural heritage sites that may be impacted by mining activities. The EIA will include an archaeological survey to identify culturally significant sites, and mitigation measures will be established to ensure that these sites are documented and protected. If any new archaeological finds are discovered during the construction or operation of the mine, the Project will work closely with local authorities and indigenous groups in the preservation of these sites.

Additionally, the Project is not located in or near an area under national protection by Law, such as National Parks or Natural Monuments.

Figure 20.7 Indigenous communities identified in relation to the Costa Fuego Project (Piteau and Associates, 2025)



20.2.4 Mine Closure Permits and Financial Assurance

Chilean law mandates that all mining projects submit a comprehensive mine closure plan as part of the permitting process. The Costa Fuego Project's mine closure plan will be developed in accordance with Law No. 20.551, which outlines the requirements for environmental restoration and financial assurance in the event of mine closure and prepared for its EIA submission.

20.2.5 Declaration of National Interest

In Chile, the process to classify a mining project as of national interest involves multiple assessments and requires coordination with various governmental bodies. This permit is advantageous for projects aiming to receive expedited processing, governmental support, and enhanced credibility, particularly for large-scale investments in mining.

Preparations for a submission for the Declaration of National Interest commenced during 2024. Recent updates to the Forest Ministry legislation have resulted in a moratorium on Declaration of National Interest submissions for EIA submissions during 2025.

20.2.6 Status of Permits

As of the date of this PFS, the Costa Fuego Project has made significant progress in obtaining the necessary permits for its development. The Environmental Impact Assessment (EIA) will be submitted to the Servicio de Evaluación Ambiental (SEA).The Project will also apply for water usage permits from the Dirección General de Aguas (DGA), with approval expected following the completion of the environmental review process.

Mutual benefit agreements with local communities and indigenous groups will be in place, and the mine closure plan will be prepared following the Chilean law. The Project is on track to secure all required permits, allowing it to move forward with the construction and operational phases.

The permitting process for the Costa Fuego Project reflects Hot Chili Limited's commitment to responsible and sustainable mining. By adhering to both Chilean regulations and international best practices, the Project ensures that its environmental and social impacts are carefully managed, and that it operates with the full support of local communities and regulatory bodies.

20.3 Environmental and Social Monitoring During Operations

Monitoring during the operations phase is a crucial aspect of the Costa Fuego Project, ensuring that environmental and social impacts are consistently measured, managed, and mitigated. The monitoring framework is based on Chile's regulatory requirements outlined in Ley 19.300 and aligns with international standards such as the CIM ESG guidelines. This robust system ensures that the Project complies with legal obligations and industry best practices, with particular attention given to progressive closure steps, continuous monitoring, and adaptive management.

20.3.1 Air Quality Monitoring

Air quality monitoring is especially important due to the desert environment and the potential for significant dust generation. Dust and particulate matter (PM_{10} and $PM_{2.5}$) will be produced, particularly around excavation sites, haul roads, and processing areas. Dust suppression measures will be employed, such as the use of seawater sprays, dust suppression compounds, and windbreaks, with their effectiveness continuously monitored. Air quality monitoring stations will be placed near the Project footprint and surrounding communities to measure dust and other pollutants, ensuring compliance with regulatory limits. These will be added to the current monitoring network of the Project. Air quality will be monitored at multiple stations around the Project site to ensure that dust levels remain within acceptable limits. The EIA submission is being prepared to include continuous monitoring of particulate matter (PM_{10} and $PM_{2.5}$) and other emissions. In addition to direct monitoring, the Project intends to implement adaptive dust management practices, adjusting dust suppression techniques based on real-time data and environmental conditions.

20.3.2 Water Quality Monitoring

Water quality monitoring is another significant environmental concern for the Costa Fuego Project, especially given the use of seawater in ore processing and the potential for acid rock drainage (ARD) from waste rock and underground workings. A comprehensive water monitoring system will assess both groundwater and surface water quality, with monitoring points established around the Tailings Storage Facility (TSF), waste rock dumps, and underground workings. Water samples will be analysed for pH levels, heavy metals, sulphates, and salinity, with a particular focus on the risks of ARD and seepage from underground workings.

A water treatment facility is included in the PFS at the Productora site for the treatment of any ARD products, contact water, or bleed streams from the heap and dump leach operations.

Seawater management will also be closely monitored to prevent contamination, with real-time data collected through automated sensors to allow for early detection and rapid response to potential issues.

20.3.3 Biodiversity and Habitat Monitoring

Biodiversity and habitat monitoring will play a key role in minimising the environmental impacts on the unique ecosystems surrounding the Project site. Regular flora and fauna surveys will be conducted, particularly in flora relocation areas, to assess the health and presence of key species.

The Project will also monitor the health of local ecosystems, tracking key biodiversity indicators to assess the effectiveness of habitat protection and rehabilitation measures. This includes monitoring wildlife populations, vegetation recovery in disturbed areas, and the success of reforestation efforts with native species. The monitoring program will be adaptive, allowing for adjustments to the biodiversity management plan (BMP) as new data becomes available.

20.3.4 Waste Management Monitoring

Effective waste management is critical to preventing long-term environmental damage. Monitoring systems will be in place to ensure that all waste management activities comply with environmental regulations. The stability of the TSF will be closely monitored using geotechnical instruments, with regular inspections conducted to detect any signs of instability or seepage. Waste rock dumps will be monitored for signs of ARD, with runoff and leachate analysed regularly to prevent contamination of local water bodies. Hazardous materials, including fuels and chemicals used in ore processing, will be inspected regularly, and incidents will be documented and reported in compliance with environmental standards.

Landfill waste generated by the Costa Fuego Project will be managed at the Huasco Provincial Sanitary Landfill (Relleno Sanitario Provincial Del Huasco).

20.3.4.1 Acid Rock Drainage (ARD) Prevention

Acid rock drainage (ARD) is a significant environmental risk in many mining operations, as it can lead to the contamination of nearby water sources with harmful acids and metals. The Costa Fuego Project PFS included ARD geochemical testing at Productora and Cortadera. The PFS considered the potential for ARD in materials destined for long-term surface storage and the design of these emplacements include management for surface contact water interactions to contain any ARD products.

A further 122 rock samples were collected across the Costa Fuego Project and will be included for the EIA submission. ARD testing of tailings samples is also underway, as previously mentioned in Section 20.1.3.

A water treatment facility is included within the PFS, capable of treating ARD products during operation.

20.3.4.2 Emergency Response and Contingency Planning

In addition to preventive measures, the Costa Fuego Project will develop an emergency response plan to address any unforeseen environmental incidents, such as severe weather events or hydrocarbon spills. This plan will include detailed protocols for containment, remediation, and communication with local authorities and communities. The Project will also conduct regular environmental audits and risk assessments to identify and mitigate emerging risks, ensuring the long-term environmental integrity of the site.

20.3.5 Social and Community Monitoring

Hot Chili Limited is committed to social and community monitoring throughout the Project. This includes tracking the socio-economic impacts of the mining operations, particularly in terms of employment and income stability for local communities. Surveys will assess the effectiveness of training programs for local workers, including indigenous groups. Site impact monitoring to assess potential impacts from air and water quality will ensure the protection of local populations during both operational and closure phases. In addition, areas of cultural heritage found near the underground and surface workings will be monitored, and consultations with indigenous communities will help manage any impacts on cultural sites respectfully.

20.4 Social and Community Impact

The social and community impact of the Costa Fuego Project is a fundamental consideration in its design, development, and operation. Given the Project's location in the Atacama region, a sparsely populated area with small rural communities and indigenous populations, its success is heavily reliant on the establishment of strong relationships with local stakeholders. This subsection provides an overview of the Project’s social engagement strategy, the anticipated economic and employment benefits, cultural heritage preservation, community health and safety, and the overall approach to community development.

Hot Chili Limited has placed a significant emphasis on adopting best practices for community engagement and social responsibility, integrating international standards such as the Free, Prior, and Informed Consent (FPIC) principles from the ILO Convention No. 169, and adhering to the CIM ESG guidelines for social governance. These standards are not only necessary for regulatory compliance but are also fundamental to ensuring that the Project is socially sustainable and that its benefits are shared equitably among all affected communities.

20.4.1 Stakeholder Engagement and Free, Prior, and Informed Consent (FPIC)

From the early stages of exploration and project development, Hot Chili Limited has engaged with local communities, indigenous groups, and other stakeholders to ensure that their voices are heard, and their concerns are addressed. This stakeholder engagement process is designed to build trust, foster collaboration, and ensure that the Project's social impacts are managed in a manner that benefits all parties involved.

The FPIC principle, which is embedded in Chilean law through the country's ratification of ILO Convention No. 169, is especially important in areas where indigenous communities are present. The Project is located on the ancestral lands of several Indigenous groups with the most populous being the Diaguita people. The engagement with these communities has been conducted through a consultation process, where the Project team has shared information on the Project and will continue with consultation regarding input on how the development might affect their lands, resources, and way of life.

Indigenous leaders and community members were given the opportunity to express their concerns, particularly regarding the preservation of cultural heritage sites, access to natural resources, and the protection of local ecosystems. These consultations are not one-time events but are part of an ongoing dialogue that will continue throughout the life of the Project.

20.4.2 Employment and Economic Opportunities

The Project is expected to generate substantial employment during both the construction and operational phases. During construction, there will be a high demand for labour in areas such as infrastructure development, equipment operation, logistics, and support services. The operational phase will also require a skilled workforce to manage mining operations, processing, and maintenance activities. The PFS determined an estimate of 2,000 direct jobs during the construction phase and 800 jobs during operation will be created, providing local residents with long-term employment opportunities.

In line with its commitment to social responsibility, Hot Chili Limited has prioritised vocational training. The company has developed partnerships with local educational institutions and technical training centres to provide vocational training programs. These programs are designed to equip local residents with the skills needed to participate in the mining workforce, helping to build a legacy of skills development that will benefit the region long after the Project has ended.

The economic benefits of the Project extend beyond direct employment. Local businesses, particularly small and medium-sized enterprises (SMEs), will have the opportunity to supply goods and services to the mine. This includes contracts for the provision of construction materials, equipment, catering, transportation, and other services. Hot Chili Limited is committed to ensuring that local businesses have access to these opportunities by providing information, training, and support to help them meet the procurement requirements of the mining industry.

By fostering local economic development, the Costa Fuego Project is expected to have a lasting positive impact on the region, supporting the growth of local businesses and contributing to the overall economic resilience of the Atacama region.

20.4.3 Cultural Heritage Preservation

The preservation of cultural heritage is a core component of the social impact management strategy for the Costa Fuego Project. The Project area is rich in cultural and historical significance, particularly for the indigenous Diaguita people, whose ancestors have lived in the region for centuries. Protecting these cultural assets is not only a legal obligation under Chilean law but also a moral and ethical responsibility that Hot Chili Limited takes seriously.

As part of the environmental and social impact assessment (EIA), Hot Chili Limited will conduct an archaeological and cultural heritage survey in collaboration with local archaeologists and indigenous representatives. This survey will identify sites of historical and cultural significance, including eventually ceremonial sites and artifacts that reflect the long history of human habitation in the region. These findings will be incorporated into the Project's development plans to ensure that cultural heritage is considered throughout the life of the mine.

In addition to physical site preservation, Hot Chili Limited will partner with local communities to support the preservation of intangible cultural heritage, such as traditional languages, practices, and oral histories.

20.4.4 Community Health and Safety

Mining projects, particularly large-scale operations like the Costa Fuego Project, can have significant impacts on the health and safety of nearby communities. Hot Chili Limited recognises the importance of protecting the health and well-being of local residents and will implement a Community Health and Safety Management Plan to address potential risks associated with the Project's activities.

One of the key health risks identified during the community consultations was the potential impact of dust and emissions generated by mining operations. In response, the Project has adopted a comprehensive dust suppression strategy, which involves the use of seawater to control dust levels in areas where mining activities, transportation, and ore processing occur. This strategy helps to reduce the amount of particulate matter released into the air, thereby minimising the risk of respiratory issues for nearby residents.

Regular air quality monitoring will be conducted to ensure that dust levels remain within acceptable limits, and mitigation measures will be adjusted as necessary to respond to changing environmental conditions.

Increased traffic, particularly during the construction phase, is another safety concern for local communities. To address this, Hot Chili Limited will develop a Traffic Management Plan to outline measures to minimise the impact of construction traffic on local roads.

The health and safety management plan also addresses mental health and social well-being, recognising that large-scale industrial projects can cause stress and anxiety for local communities. Hot Chili Limited currently collaborates with local healthcare providers to offer mental health services and support programs for vulnerable residents. The company is committed to providing resources that promote mental and emotional well-being, ensuring that the community remains resilient in the face of social and economic changes.

20.4.5 Community Development and Social Investment Programs

Hot Chili Limited is committed to making a positive and lasting contribution to the development of the communities surrounding the Costa Fuego Project. The company has implemented a Community Development and Social Investment Program. These investments are designed to create long-term benefits that will continue to support the region's social and economic development even after mining operations have ceased.

20.4.6 Grievance Mechanism

Hot Chili Limited is establishing a formal grievance mechanism to provide local communities with a clear and accessible process for raising concerns or complaints related to the Costa Fuego Project. This mechanism is an important part of the company's commitment to transparency, accountability, and social responsibility.

The grievance mechanism will allow community members to submit concerns related to environmental impacts, health and safety, cultural heritage, employment, and other issues. Complaints will be logged and investigated by a dedicated team, and the company commits to resolving issues in a timely and fair manner. The grievance process is being designed to be accessible to all community members, including those from indigenous groups, and efforts are made to ensure that language barriers or cultural differences do not hinder participation.

The grievance mechanism will be regularly reviewed and audited to ensure its effectiveness, and all grievances and their resolutions will be documented and reported to relevant stakeholders. This process will help to build trust between the Project and local communities, ensuring that concerns are addressed, and that Hot Chili Limited remains responsive to the needs of the communities it impacts.

20.5 Mine Closure Plan

The mine closure plan, in compliance with Chilean regulations, will be prepared for the EIA submission.

The PFS considered closure costs for the Project, and findings of the studies in hydrology, biodiversity, acid rock drainage and the TSF design to define a closure approach for the PFS.

20.5.1 Regulatory Compliance and Closure Planning

In terms of regulatory compliance, the mine closure plan adheres to Chilean legal requirements, specifically Law No. 20.551, which dictates environmental stability and mandates financial guarantees to ensure the plan's full implementation, even if the company ceases operations prematurely.

20.5.2 Open Pit Mine Closure

Multiple options for the treatment of open pits for mine closure are under investigation and will be resolved for the EIA submission. Decommissioning of individual pits will consider bench stability requirements and will include the establishment of emergency spillway drainage channels. Allowance will be made for selective rehabilitation of benches that remain above the possible pit lake levels. Post closure maintenance activities may include repairs to earthworks, as well as care of vegetation.

Treatment options noted in the PFS documentation include:

20.5.2.1 Pit Lakes

The PFS hydrology studies found that low permeability of the pit wall materials makes the sites suitable to act as a pit lake following closure, where groundwater re-establishes and surface water pools into the pits. This option also accounts for management of surface contact water post closure.

20.5.2.2 Tailings deposition

At Productora, an option to include a component of tailings deposition (~114Mt) within the main pit is also planned, following completion of mining in Year 11. This option will be further investigated for the EIA, with additional hydrogeological modelling and analysis required. See Section 18 for more detail.

20.5.2.3 Backfilling of Pits during Operations

At Cortadera, an option includes the use of Cuerpo 1 and 2 as in pit waste rock emplacements during operations, housing waste material from Cuerpo 3 and the underground workings. Closure of these emplacements would include shaping of the profiles and management of surface contact water. Cuerpo 3 pit would not be backfilled, functioning as a pit lake during closure.

20.5.3 Underground Mine Closure

Underground operations are restricted to Cortadera and include portal and development access to a block cave underneath Cuerpo 3 pit. A conceptual closure approach was used in the PFS to calculate costs and includes sealing of the portal and any other openings to the underground workings, such as vent shafts, and construction of safety berms around the subsidence zone.

The final closure plan, inclusive of all pits, underground, and surface works, will be resolved for the EIA submission.

20.5.4 Surface Infrastructure Decommissioning

Closure costs include the decommissioning of all surface infrastructure at the Costa Fuego Project.

20.5.4.1 Surface Water Management

The PFS includes modelling and design of mitigation measures to handle surface contact water post mine closure. Chemical stability of remaining surface features, such as the waste rock dumps, leach pads, open pits and the TSF, are to be modelled prior to the EIA submission to ensure that appropriate mitigation measures are included in the design.

The PFS includes the identification of seepage control and detection systems, diversion channels to ponds and pits and a water treatment facility.

20.5.4.2 Waste Rock Dumps

Rehabilitation of waste rock dumps includes reshaping of benches for visual continuity within the landscape and earthworks for the management of surface water run-off.

As previously mentioned in this Section, the PFS included 122 ARD samples to inform waste rock management, however results of the humidity cell tests (a longer-term ARD analysis) are not yet available for the PFS. These tests will be incorporated into the EIA submission and closure plan, and will inform the requirements for encapsulation, liners, or other ARD mitigation for waste rock dumps post closure.

20.5.4.3 Heap and Dump Leach

Rehabilitation of leach pad sites will commence upon termination of rinsing, with reshaping undertaken to achieve the closure profile. An erosion protection layer of inert waste rock will be spread across the final surface and drainage chute structures will be added to control discharge from the heap.

20.5.4.4 Process Plant Area and Rope Conveyor

Decommissioning and rehabilitation of the process plant and rope conveyor infrastructure includes:

- washing of all materials that have been in contact with hydro chemicals or other chemicals
- salvage and scrapping of plant and structures
- all areas that are contaminated with hydrocarbons during the demolition operations will be disposed and sent to a licenced waste disposal facility
- steel constructions and other suitable materials will be recycled
- all concrete structures and foundations will be demolished and the area rehabilitated, to include landscaping into a natural form in alignment with the natural hydrological patterns.

20.5.4.5 Support Infrastructure

Roads, powerlines, pipelines and other utilities no longer needed will be decommissioned and removed as appropriate.

20.5.5 Tailings Storage Facility (TSF) Closure

Refer to section 20.6 for detail on the TSF.
The PFS considers two TSF sites, the conventional TSF site north of Productora and a component of in-pit tailings deposition following the conclusion of mining activities at Productora.

The PFS costings for the conventional TSF site includes and allowance for the use of inert waste material as capping upon closure. ARD testwork for tailings was not finalised at the time of the PFS conclusion, however will be studied prior to the EIA and further development studies.
The in-pit tailings component at Productora requires additional site investigation and hydrological studies to determine any ground preparations, management or closure requirements.

20.5.6 Financial Assurance and Closure Costs

The estimated total cost for closure activities at the Costa Fuego Project is $78 million, including contingency funds to cover unforeseen costs. This estimate, performed by Wood (refer to Section 21), is based on detailed closure cost assessments and benchmarked against similar projects in Chile.

Financial assurance mechanisms, such as performance bonds and trust funds, will ensure the necessary funds are available for all closure activities, with the contingency fund addressing unexpected challenges, such as unforeseen environmental issues.

Refer to Section 21 for discussion on capital and operating costs related to closure, and to Section 24 for discussion on a project execution plan that incorporates progressive closure activities.

20.6 Tailings Storage Facility (TSF) Management

20.6.1 TSF Design, Construction and Closure Concepts

Refer to Section 18.

20.6.2 PFS TSF Environmental Studies

Designs and the underpinning environmental studies for the PFS TSF design were prepared by Knight Piésold, consulting to Hot Chili Limited. A summary of the environmental studies is provided below, with the full report available within the appendices of this report.

The PFS investigated a conventional tailings facility, valley hosted north of Productora, and aa component for in-pit tailings deposition at Productora. The in-pit tailings deposition investigations are not yet complete at the PFS level, with a groundwater and water balance model to be delivered to assess this option. This assessment will be completed prior to the EIA submission.

20.6.2.1 Site Characteristics

Site characteristics works included desktop studies on the location, environment, climate, topography, surface catchments, groundwater, geology, seismicity, and social and cultural contexts for the proposed TSF site.

Additional site characterisation work for the PFS included:

- Geotechnical and hydrogeological characteristics of the facility.
- Waste rock geochemistry, including potential Acid Rock Drainage (ARD) impacts, for materials proposed for the construction of the facility.

A program of tailings physical testing and geochemistry characterisation commenced but was not yet available for the PFS. These results will be used in additional studies and reported to Hot Chili by Knight Piésold via technical memorandum once available.

20.6.2.2 TSF Dam Break Assessment

As part of the design, a dam break assessment (DBA) was prepared considering ANCOLD guidelines (2019 and 2012, Refs. 4 and 6) and the Global Industry Standard on Tailings Management (Ref. 3).

Six DBA scenarios were considered, in which catastrophic embankment failure occurred on the north, northeast, northwest, east and west embankments.

The impacts on the population at risk, business operations, social impacts, and the environment have been considered and risk assessed within the DBA. Embankment failures were considered conceptually for when the TSF is at its forecast ultimate height and capacity, when the potential volumetric outflow is the largest, and when the inundation area is the greatest. This is considered a critical case for the DBA scenarios.

20.6.2.3 Water Balance Modelling

The management of water is a critical aspect of the design for the Costa Fuego Project. In order to understand (and control) the flow of water around the site, a water balance model was developed. The primary objectives of the water balance model for the TSF centred on establishing the filling rate for tailings solids, determining the supernatant pond volumes under various climatic conditions, and determining any shortfall in water under average conditions.

The modelling found:

- The TSF is designed to hold the tailings deposited plus the design rainfall conditions and has sufficient stormwater storage capacity for all design storm events and rainfall sequences.
- The supernatant pond volume will remain at a minimum (200,000 m^3) during operation. The pond will peak following rainfall but will quickly be drawn to a minimum following decant abstraction.
- The supernatant pond balance decreases with the ceasing of the processing activities due to evaporation, and the supernatant pond is available to be removed as soon as practicable after decommissioning at which time the TSF is rehabilitated as a water shedding structure.

20.6.3 PFS TSF Environmental Controls and Design Elements

Within the broader design and construction (Section 18) for the TSF, design elements specific to delivering environmental risk mitigations and controls are discussed below.

20.6.3.1 Seepage Control

Components to manage or prevent seepage include:

- Cut-off trench
- Liners (geosynthetic clay) under embankments and alluvium bases, and HDPE within the low points of the TSF basin and along the supernatant pond migration path.
- Underdrainage system
- Embankment drains
- Downstream seepage collection systems.

20.6.3.2 Stability Assessment

The stability of the proposed TSF was assessed for static, pseudo static and post seismic loading conditions.

The stability analyses confirmed that the proposed embankment profiles will satisfy all requirements during construction, operation and closure.

20.6.3.3 Monitoring

A monitoring programme for the TSF was proposed within the design to include:

- Monitoring bores and surface water sampling stations downstream of the TSF
- Vibrating wire piezometers in the TSF embankment to monitor the phreatic surface

- Survey pins to check for embankment movement.

20.6.4 Operational Management and Risk Mitigation

The PFS considered design and construction elements, with concepts for closure. An Operational Management Plan would be developed prior to construction.

20.6.5 Community Engagement and Social Impact

As a component of Hot Chili's FPIC approach with community engagement and social impact, the TFS has been included in discussion materials for community engagement and feedback sought on the topics of TSF generally. Early feedback to Hot Chili included concern by some agricultural industry and Indigenous groups regarding TSF infrastructure generally. Engagement and discussion on the TSF continues in the advancement of the EIA submission.

20.7 Level of Detail and Additional Work

Table 20.4 provides a summary of the level of detail across multiple environmental and social topics relative to the PFS, and future work. Detailed biodiversity impact assessments, hydrological modeling, archaeological risk evaluations, and detailed closure planning are outside the scope of the PFS and will be fully addressed as part of the EIA and Feasibility Study (FS), in accordance with Chilean environmental regulations and international best practices.

Table 20.4 PFS level of detail summary and plan for additional work

Topic	Addressed in PFS	To Be Addressed in EIA/FS	Notes
Biodiversity and Species-Level Risk	Summary provided	EIA (including baseline and mitigation plan)	Per Chilean EIA Law 19.300
Cumulative Impacts	N/A	Impact Assessment	Per SEIA guidelines for EIA
Social and Indigenous Engagement	Preliminary stakeholder identification and early citizen participation	Full stakeholder analysis and consultation	
Water Balance and Aquifer Impacts	Hydrogeological models	Detailed modelling including seasonal variation	
Financial Assurance / Closure Costs	Closure approach and preliminary cost estimate included	Detailed costing and closure plan	Based on progressive rehabilitation and post-closure monitoring
Ecosystem Services Assessment	N/A	Optional for EIA	Will depend on SEA requirements

21 Capital and Operating Cost Estimates

21.1 Capital Cost Estimate

21.1.1 Introduction

This section summarises the capital cost estimates, and the basis used for building that estimate. All costs are estimated in United States dollars as at Q4 2024 and are judged to have an accuracy of ±15% to 25% and are considered to be at PFS level in accordance with Wood's Estimating Procedures and Class 4 as defined in the AACE document 18R-97.

Certain elements to complete the Project overall initial capital estimate have been provided by outside sources as listed below:

The estimate has been compiled by Wood with inputs from consultants for their responsible scope of work:

- ABGM: Mine production, mine footprint and decline development, ventilation, dewatering and underground infrastructure, underground crushing and materials handling
- Doppelmayer: Rope Conveyor
- PMC: Heap Leach and Dump Leach (above ground) MTO for irrigation and aeration piping
- Knight Piésold: Heap Leach and Dump Leach (below ground), tailings storage, surface water management
- Wood: Process plants, surface infrastructure, tailings pipelines, power supply and distribution, services and utilities, and concentrate pipelines
- HCH: Owners costs
- All: Indirect costs, EPCM
- Wood: Contingency allowance.

Wood compiled the overall Project estimate, although certain elements of the capital cost were contributed by HCH or HCH nominated specialist consultants, to achieve consistency in such things as estimate base date, exchange rate fluctuations, and rates applied.

HCH has advised that power supply, seawater pump stations and pipelines, construction/product haulage access road/s are to be constrained to tenure already established/set-aside by HCH.

The major work areas as per the WBS under Wood responsibility are as follows:

- Process Plant (Cortadera and Productora areas)
- Process Plant Infrastructure and Support Services
- Process Plant General and Miscellaneous costs
- Engineering, Procurement and Construction Management (EPCM)
- Growth Allowances
- Risk Allowances.

21.1.2 Estimate Summary

The CAPEX presented below is summarised by total (Table 21.1) and as Pre-Start and Expansion.

Table 21.1: Estimated Initial Capital Expenditure

Cost Area	Pre-Start US$000	Expansion US$000	Total US$000
Direct Cost	**1,017,975**	**1,251,147**	**2,269,122**
01 Bulk Earthworks and Drainage	9,063	-	9,063
02 Site Services	3,124	-	3,124
03 Sulphide Process	394,282	-	394,282
04 Oxide Process	172,655	17,543	190,198
05 Molybdenum Process	14,329	-	14,329
06 Power Line	31,724	19,396	51,120
07 Pipeline	-	-	-
08 Road to Port	5,688	-	5,688
09 Port	-	-	-
10 TSF	67,053	-	67,053
11 Infrastructure Other	93,793	8,477	102,270
12 Open Pit Infrastructure	74,756	60,865	135,621
12 Mining Pre-Strip	102,887	-	102,887
13 First Fills & Commissioning	48,622	24,618	73,240
18 Sulphide Leach	-	41,296	41,296
19 Processing Upgrades	-	34,254	34,254
20 Cortadera Infrastructure	-	60,565	60,565
21 Rope Conveyor	-	171,646	171,646
22 Blockcave Development	-	684,980	684,980
23 Blockcave Infrastructure	-	127,505	127,505
Indirects	**128,665**	**39,033**	**167,698**
14 Construction Facilities, Services and Equipment	8,672	3,658	12,330
15 EPCM	119,993	35,375	155,368
Total (Excluding Owners Cost)	**1,146,640**	**1,290,178**	**2,333,934**
16 Owners Costs	42,017	18,890	60,906
17 Contingency	84,033	37,780	122,813
Total Initial CAPEX	**1,272,690**	**1,346,850**	**2,619,539**
Sustaining & Closure Capital	**-**	**810,845**	**810,845**
Tailings	-	124,743	124,743
Sulphide Process	-	215,869	215,869
Molybdenum Process	-	7,845	7,845
Oxide Process	-	17,614	17,614
Sulphide Leach Process	-	2,568	2,568
Waste Stripping	-	382,370	382,370

Table 21.1: Estimated Initial Capital Expenditure			
Cost Area	**Pre-Start US$000**	**Expansion US$000**	**Total US$000**
Closure	-	78,042	78,042
Salvage	-	-18,206	-18,206

The purpose of the CAPEX estimate is to support an application for further funding to progress the Project to the definitive feasibility stage.

The estimate includes mining, process plants, Productora and Cortadera crushing areas, a rope conveyor and associated infrastructure, all located approximately 17 kilometres south of the regional township of Vallenar, in the Atacama region of Chile.

21.2 Estimate Structure

The capital cost estimate has been structured into the following major categories.

21.2.1 Direct Costs

The direct costs are those expenditures that include supply of equipment and materials, freight to site and construction labour at site.

21.2.2 Indirect Costs

Indirect costs are those expenditures covering temporary construction facilities plus engineering, procurement, and construction management (EPCM) services together with the supervision of and commissioning of the works.

21.2.3 Growth Allowances

Growth Allowances have been assigned to each discipline item based on the perceived magnitude of the risks given that the study has been executed to preliminary feasibility level. Wood has applied provisions to the estimate to make allowance for the following risks:

- minimal design input as suitable for estimates of this accuracy
- preliminary scope definition
- quantity survey errors and omissions
- rework
- gross vs. net quantities
- material and labour rate accuracy
- equipment budget costing
- incorrect "bulks" factor applications.

21.3 Estimate Cost and Scope Basis

All costs have been estimated using a cost structure developed for labour and materials as of fourth quarter 2024 and are presented in United States Dollars. The following summarises the estimation derivation and basis adopted for the study.

21.3.1 Mining Infrastructure

Mining was costed by ABGM, refer to Section 21.5.1.

21.3.2 Mining Fleet

Mining was costed by ABGM, refer to Section 21.5.1.

21.3.3 Bulk Material Rates – Process Plant

Unit rates for Bulk Materials i.e., concrete, steelwork, platework, piping etc., have been developed from in-house data from the Chile region and rates specific to this study supplied by Wood Santiago office, Client, and suppliers familiar with costs applicable to resource project developments in the Chile region plus applicable costs in the public domain.

The main all-up bulk rates used in the estimate are listed below:

Table 21.2: Bulk Material Rates – Process Plant

Description	Unit of Measure	Total unit Rate
Clear & grub shrubs	Ha	$8,218
Strip topsoil & dispose	m^3	$12.44
Bulk excavation, common dispose to soil	m^3	$10.04
Bulk fill	m^3	$2.55
Concrete (in place)	m^3	$1,490 (avg)
Steel work - general	Ton	$8,997(avg)
Steel work grating	m^2	$424
CS Rubber lined bins	Ton	$11,414 (avg)
Site erected CS Rubber lined tanks	Ton	$21,000 (avg)
Shop fabricated CS tanks	Ton	$17,900 (avg)
Shop fabricated 316 SS tanks	Ton	$20,897 (avg)

21.3.4 Equipment Costs

Equipment costs for all major equipment items are based on budget quotes received from vendors. In-house database or allowances have been used for minor equipment items. The percentage of each are tabulated in the following tables:

Table 21.3 : Equipment Costs		
Category	**Total Value (US$)**	**Percent of Total Value (%)**
Budget quotations	304,301,881	91%
In-house data and estimates	31,551,456	9%
Total	**335,853,337**	**100%**

The following vendor pricing for major Process Plant equipment is incorporated into the estimate:

Table 21.4 : Major Process Equipment	
Equipment Description	**Vendor**
Seawater pumps	KSB
Gyratory crusher	Metso
Jaw crusher	FLS
Cone crusher	FLS
Mills (sag, ball)	Mills Metso / Drives Innomotics
Mills (regrind)	Glencore
Thickeners	Above Ground Roytec / Below Ground FLS
Pressure filters	FLS
Flotation cells	Roytec / Jameson cells Glencore
Water treatment	Veolia
Apron feeders	Metso
Slurry pumps	FLS / KSB
Solution pumps	KSB
Mill servicing equipment	Metso
Vibrating screens	Derrick
Cyclones	Weir
Blowers	Daltec
SX/EW	BGRIMM
Rope conveyor	Doppelmayr

An allowance on the net price of the equipment has been made to include for fasteners, packers, wedges, grouting, guarding and signage.

21.3.5 Plant Bulk Earthworks, Drainage and Plant Roads

Bulk earthworks are based on necessary cut to fill and embankment construction suitable for plant foundation and road construction. Drainage consists of earthen side drains and concrete culverts. Water storage ponds include for HDPE liners. Plant roads are not sealed but sheeted with suitable base course. It is presumed suitable materials are available from mine waste and overburden adjacent to the Project site. Quantities for the estimate have been produced by Wood's civil engineering department from preliminary layout drawings and existing topography data.

21.3.6 Proccess Plant

The scope of the process plant(s) cost is listed below. Detailed description of the process plant can be found in other areas of this report.

Cortadera Surface Crushing:

- Rom Pad and Primary Crushing
- Services.

Overland Conveying:

- Rope Conveyor.

Productora Sulphide Concentrator:

- ROM Pad and Primary Crushing
- Stockpile and Reclaim
- Grinding
- Sulphide Flotation and Regrind
- Sulphide Concentrate Washing
- Molybdenum Flotation
- Molybdenum Concentrate Dewatering and Handling
- Copper Concentrate Dewatering and Handling
- Tailings Dewatering and Handling.

Productora Oxide Plant:

- Ore Handling and Stockpile
- Ore Preparation
- Heap Leaching
- Solvent Extraction
- Electrowinning.

Productora Dump Leaching:

- Dump Leaching.

Productora Reagents:

- Sulphide Flotation Reagents

- Molybdenum Flotation Reagents
- Flocculant
- Sulphuric Acid
- Solvent Extraction Reagents
- Electrowinning Reagents.

Productora Plant Services:

- Cooling Water
- Raw Water
- RO Water
- Process Water - High Chloride
- Process Water - Low Chloride
- Demineralised Water
- Gland Water
- Potable Water
- Air Systems
- Diesel.

21.3.7 Process Plant Buildings

Steel-clad type buildings for the Processing Plants are summarised below. All pricing has been based on the steel-clad type buildings of sufficient size to support the manning on site. All steel-clad Industrial type buildings have been costed by applying concrete, steelwork, and cladding rates etc. to calculated quantities.

- Productora Sulphide Reagents Store – 170 m^2
- Productora SX Building – 5,000 m^2
- Productora EW Building – 1,740 m^2
- Productora SX/EW Reagents Store Room – 170 m^2
- Cortadera Surface Workshop – 200 m^2
- Concentrate Storage Facility – 4,200 m^2.

The average m^2 built up rate for the Process Plant buildings used in the estimate is US$1,954.

21.3.8 Switchrooms

A preliminary scope based on the calculated load list with associated costings was developed by Wood's electrical engineering department. Sizing, scope etc. were quantified for the following:

- Switchrooms
- Transformer installations
- Motor Control Centres (MCCs)
- Variable Speed Drives (VSDs).

21.3.9 Plant Support Buildings

The cost allowances for commercial type buildings have been based on transportable type buildings and of sufficient size to provide the office area, lunchroom, change rooms, amenities etc. for the manning profile prepared for the OPEX. Offices, meeting rooms, ablutions and the like are fully fitted out with air conditioning, partitioning, furnishings, workstations, computers, appliances, and sanitary fixtures. The costing for these buildings is based on an in-house built-up rate from similar Project/Studies. Site buildings included are listed below:

- Administration Complex – 452 m^2
- Productora Security Gatehouse – 45 m^2
- Productora Safety and ERT Building – 264 m^2
- Productora Process and Maintenance Office – 296 m^2
- Productora Laboratory Building – 89 m^2
- Productora Plant Main Control Room – 80 m^2
- Productora Sulphide Crushing Control Room – 18 m^2
- Productora Canteen – 494 m^2
- Productora Change Room Building – 318 m^2
- Productora Plant Maintenance Workshop and Warehouse – 1,263 m^2
- Productora Light and Service Vehicle Workshop – 396 m^2
- Productora Oxide Crushing Control Room – 18 m^2
- Productora SX/EW Control Room – 43 m^2
- Productora Oxide Office + Workshop – 517 m^2
- Cortadera Crushing Control Room – 18 m^2
- Cortadera Office + Crib + Changeroom – 179 m^2
- Cortadera Workshop + Warehouse – 200 m^2.

The average m^2 built up rate for the Plant Support buildings used in the estimate is US$2,089.

21.3.10 Laboratory Equiment

There is a small sample laboratory onsite – the main laboratory will be offsite.

21.3.11 Fire Protection and Detection

Fire protection has been included for both Productora and Cortadera sites and includes a dedicated high pressure water supply ring main distributed around the site. This is supplemented by fire indicator panels, thermal and smoke detectors, break glass units, warning lights, fire bells and extinguishers.

This cost has been factored from a similar study done recently by Wood.

21.3.12 Water Treatment

Water treatment costs are included in the process plant.

21.3.13 Sewage Disposal and Treatment

A nominal sewage system has been included to connect to the existing network. Allowance has been made for:

- Buried gravity collection main c/w manholes
- Rising main pump station
- Buried pressure delivery main (allowance of 500 mm).

This cost has been factored from a similar study done recently by Wood.

21.3.14 Plant Support Mobile Equipment

An allowance for a basic mobile equipment fleet has been included. The cost basis is derived from budget quotations and in-house data. It is presumed requirements for certain equipment to support operations will be done by either contract or short-term hire. One example is large cranes, which will be required for certain maintenance activities.

The following vendor pricing for mobile equipment is incorporated into the estimate:

Table 21.5 : Plant Support Mobile Equipment – Vendor Pricing Source

Mobile Equipment Type	Vendor
Light vehicles	Toyota
Mobile cranes	Terex
Service truck	Plantman
Elevated work platforms	JLG
Mobile welding machines	Gentronics
Skid steer loader	Westrac
Forklifts	United Forklifts
FEL	Komatsu
54 Seat bus	BCI

Table 21.5 : Plant Support Mobile Equipment – Vendor Pricing Source	
Mobile Equipment Type	**Vendor**
Garbage truck	Hino
Fire tender	Tank Management Services
Ambulance	Toyota

21.3.15 Bulk Fuel Storage and Distribution

Included are integrated self-bunded fuel storage tanks to service plant vehicles and for emergency power generation. This cost was factored from a similar study done recently by wood.

21.3.16 Tailings Discharge Line

For the tailings disposal, a nominal 11.9 km DN600 to DN750 polyethylene (PE) pipeline to the Tailings Storage Facility (TSF) has been allowed for. This line is above ground and follows the access road to the TSF and distributes at points around the perimeter embankment.

21.3.17 Tailings Storage Facility

Knight Piésold provided the necessary engineering development quantities in the MTO build-up for the construction of the Tailings Storage Facility, which Wood applied in the CAPEX estimate. Wood has considered using the mining fleet and mining waste for constructing parts of the dam walls and causeways utilising mining rates provided by HCH.

The priced MTOs were then reviewed as a team by Knight Piésold, Wood and HCH.

21.3.18 Heap Leach

Knight Piésold provided the necessary engineering development quantities in the MTO build-up for the construction of the Heap Leach Pad, which Wood applied in the CAPEX estimate. Wood has considered using the mining fleet and mining waste for constructing parts of the leach pad base utilising mining rates provided by HCH.

The priced MTOs were then reviewed as a team by Knight Piésold, Wood and HCH.

21.3.19 Dump Leach

Knight Piésold provided the necessary engineering development quantities in the MTO build-up for the construction of the Dump Leach Pad, which Wood applied in the CAPEX estimate. Wood has considered using the mining fleet and mining waste for constructing parts of the leach pad base utilising mining rates provided by HCH.

The priced MTOs were then reviewed as a team by Knight Piésold, Wood and HCH.

21.3.20 Decant Return Line

An allowance has been included for a decant return line of 11.9 km DN200 polyethylene (PE) pipeline from the TSF.

21.3.21 Control Systems

DCS/PLC Control System for the plant has been calculated by applying a rate per kW to the installed kW of equipment installed.

21.3.22 Communications

An allowance for communications throughout the Project site is included. It covers voice/data cabling, fixed voice services and mobile phone services.

21.3.23 Emergency Power

Emergency power has been factored, based on data from a similar study.

21.3.24 Security

Security has been factored, based on data from a similar study.

21.3.25 Refuse/Waste Disposal and Storage Facility

Refuse/Waste Disposal and Storage Facility has been factored, based on data from a similar study.

21.3.26 Electrical Transmission Line

The 220 kV powerline has been developed to cover the 22 km from the Maitencillo switchyard to the site.

In the area near the TSF there are two powerlines that will need to be re-routed around the TSF footprint as part of the prestart capital, these are described below:

- 220 kV powerline that is 9.3 km long
- 500 kV powerline that is 9.7 km long which diverts around the TSF.

At the PFS level these power lines have been costed as new sections of powerline.

21.3.27 Area and Regional Roads

The engineering group developed the bulk quantities for the roads in both the Productora and Cortadera areas. MTOs have been developed from 3D models, the estimating group compiled the quantities and entered them into the estimating system:

- Productora to Port haul road, 2 sections and a total length of 22.9 km. Pavement is made of 300 mm subbase, 200 mm basecourse and 150 mm asphalt.
- Port access road including turnout. Pavement is made of 300 mm subbase, 200 mm basecourse and 150 mm asphalt.
- Productora process plant to Pan American Highway, 12.9 km unsealed road, 400 mm wearing course.

- Cortadera to Pan American Highway, 15.3 km unsealed road, 400 mm wearing course.
- Productora process plant internal access road. Pavement is made of 300 mm subbase, 200 mm basecourse and 150 mm asphalt.
- Oxide Plant internal access road. Pavement is made of 300 mm subbase, 200 mm basecourse and 150 mm asphalt.

21.4 Exchange Rates

The cost of equipment items may be subject to exchange rate fluctuation. The exchange rates used in the estimate are as follows, please note that although equipment prices may have been quoted in nominated currencies this may not be the base currency of the supply.

Table 21.6 : Exchange Rates

Currency	Converted to Other Currencies
1.00 United States Dollar (US$)	1.45 Australian Dollar (AUD)
1.00 US$	0.78 Great Britain Pound (GBP)
1.00 US$	0.9 Euro (EUR)
1.00 US$	830 Chilean Peso (CLP)

21.5 Mining Estimate Costs and Scope Basis

21.5.1 Mining Estimate Basis

The estimate for capital costs was developed using a zero-based model for mining fleet and budgetary proposals from various suppliers for the bulk of infrastructure and other equipment that would be provided.

Table 21.7 summarises elements of the mining capital estimates for the Construction, and Expansion phases of expenditure. Note the following inclusions/exclusions for each of the open pit and underground mines:

- Open Pit
  - All stripping costs have been classified as operating expenses and are excluded from the capital estimate.
  - It has been assumed that a mining contractor would be employed for the entire LOM and this contractor would provide all required production fleet and materially all ancillary fleet. The cost of this fleet would be charged as an operating expense and it is excluded from the capital estimate.
  - Elements of ancillary fleet and technology not included in the contractor bids are assumed to be provided by the Owner and have been included in the capital estimate.
  - Capital costs incurred during the initial 21 months of operation, prior to the pit providing 60% of the steady-state milling rate of 20 Mtpa, have been classified as Construction CAPEX. Capital costs incurred following this period have been classified as Expansion CAPEX.
- Underground

- All mining costs incurred during the initial 75 months of operation, prior to the initial release of cave ore, are classified as capital costs. Thereafter, only costs associated with development (lateral and vertical) and drawbell development have been classified as capital.
- It has been assumed that a contractor would be employed to install all lateral and vertical development during the initial 75-month period. All required mining fleet would be provided by the contractor and charged as an operating expense and is thus excluded from the capital estimate.
- Following the initial 75-month period, all development and production activities would be performed by the Owner, who would purchase the mining fleet. This fleet is included in the capital estimate.
- As the bulk of underground pre-production development expenses would be incurred while the Project is generating cash flows from the open pit, all underground capital costs have been classified as Expansion CAPEX.

- Open Pit and Underground
  - Costs presented exclude contingency. Typical rates at a pre-feasibility stage of project development would be 10 – 12% for fleet and 12 – 18% for all other elements.
  - Items such as growth and EPCM for the mining elements are assumed to be included within the costs presented under WBS 1000.

Table 21.7 : Summary of Mining Capital Costs

Item	Units	Construction	Expansion	Total
Open Pit Contractor Mob/DeMob	US$ 000s	9,709	9,709	19,417
Open Pit Roads	US$ 000s	20,854	17,204	38,059
Open Pit Production Fleet	US$ 000s	-	-	-
Open Pit Anciliary Fleet	US$ 000s	1,040	390	1,430
Open Pit Technology	US$ 000s	10,966	11,243	22,209
Open Pit Infrastructure	US$ 000s	14,018	22,320	36,337
Underground Development	US$ 000s	-	559,290	559,290
Underground Production Fleet	US$ 000s	-	125,691	125,691
Underground Infrastructure	US$ 000s	18,169	127,505	145,674
Mining Total	**US$ 000s**	**74,756**	**873,351**	**948,107**

Sources of the estimates presented in Table 21.5.2 are detailed in the following sections.

21.5.2 Capitalised Stripping and Contractor Mobilisation/De Mobilisation

As mentioned earlier, all open pit stripping expenses have been classified as operating costs and are thus excluded from the capital estimate. Capitalisation of waste mining is carried out during economic modelling and is discussed in section 22 of this PFS.

Open pit mining is planned to be conducted by a contractor over the entire Project life. The cost of this contractor mobilising and demobilising has been classified as a capital expense and is supported by budgetary estimates.

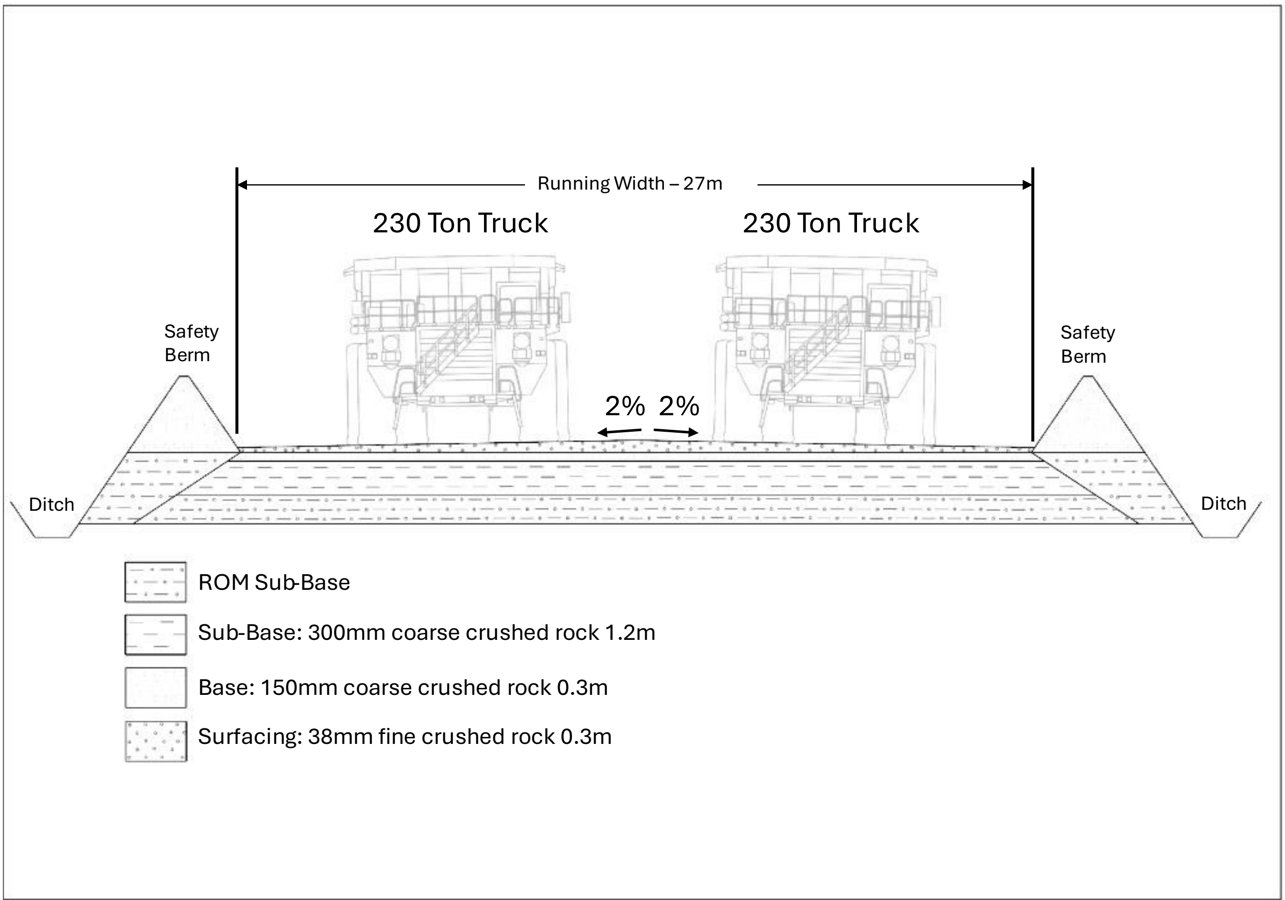
All other costs incurred by the open pit contractor have been classified as operating expenses.

21.5.3 Open Pit Roads

The estimate includes two classes of access roads:

- Roads used by Heavy Vehicles (HV). These have been sized based on two-way travel for 230 t class trucks and are depicted in Figure 21.1. Provision is made for 1 m of ROM topped with 1.2 m of material crushed to a P_{80} of 150 mm (requiring single stage crushing) while the wear surface is 300 mm of material crushed to 38 mm (requiring two-stage crushing). Berms are extended to the mid-point of the R57 tires this class of truck would be equipped with, or 2.6 m above the road surface. The running width of 27 m is equal to 3.5x the 7.6 m operating width of the truck. Mine waste would be used to construct these roads, at an estimated cost of US$1M /km.
- Roads used by Light Vehicles (LV). These would entail widening of existing roads at an estimated cost of US$0.2M /km.

Figure 21.1: HV Road Cross Section



21.5.4 Open Pit Production and Ancillary Fleet

Budgetary estimates provided by the various contractors make provision for the entire production fleet and the bulk of ancillary equipment that would be required for operation of the mine. The cost associated with these units has been classified as operating expenditures.

Units of ancillary equipment not provided in the contractor bids (primarily cranage, that would be required to assemble machines at site and subsequently for maintenance activities) have been added as capital costs.

21.5.5 Open Pit Technology

Open pit technology systems include:

- a wireless backbone and network monitoring software, to facilitate data and voice communications
- a fleet management system
- mine design and planning systems
- asset health management systems.

The estimated cost for each system was provided by a technology consultant.

21.5.6 Open Pit Infrastructure

Open pit infrastructure includes the following:

- Maintenance workshops at each of Productora/Alice, Cortadera and San Antonio. The size and number of bays for each was based on the size of largest equipment used (230 t trucks at Productora and Cortadera, 45 t articulated trucks at San Antonio) and fleet numbers, with one bay required for each increment of five trucks. Note that a lesser number of 90 t trucks will be required at both Productora (for North, South and Alice pits) and Cortadera (when ramps reduce in width and smaller trucks replace the main haul fleet). These have been included in the calculation, with a 90 t truck treated as 0.75x a 230 t truck.
- A fuel farm at each of the three pit areas. The size of each farm was a function of the calculated diesel consumption with storage of 4 days consumption provided.
- Dewatering pumps and pipeline for each pit.
- Offices at each pit area.
- A magazine at each pit area.

The costs for each of these were based on Q3 2024 quotations provided by North American based suppliers of the associated goods and services.

21.5.7 Underground Development

For the initial 30 quarters of the Project, for underground mining activities, prior to initiation of the cave, all expenditures are classified as capital expenses. During this time, mining would be performed by a contractor. Thereafter, only development (vertical and lateral) and drawbell development are classified as capital, with other items classified as operating costs.

After quarter 30, a contractor would be retained for vertical development while all other activities would be performed by the Owner.

Table 21.8 summarises the composition of development expenses.

Table 21.8 : Underground Capital Expenditure					
Item	**Units**	**Total**	**Contingency**	**$/t**	**$/m**
Development					
Vertical	metres	3,608			
Lateral	metres	65,171			
Total	metres	68,779			
Production					
Development Ore	000 tonnes	2,300			
Cave Ore	000 tonnes	143,475			
Total Ore	000 tonnes	145,776			
Capital Cost by Process					
Development	US$ 000s	453,730	15.0%	3.11	6,597
Undercut Drill & Blast	US$ 000s	8,185	15.0%	0.06	
Infrastructure Maintenance	US$ 000s	305	15.0%	0.00	
Maintenance Labour	US$ 000s	896	15.0%	0.01	
Logistics	US$ 000s	185	15.0%	0.00	
Crushing & Conveying	US$ 000s	1,840	15.0%	0.01	
Ventilation	US$ 000s	55,537	15.0%	0.38	
Management & Technical	US$ 000s	38,611	15.0%	0.26	
Sub-Total	**US$ 000s**	**559,290**	**15.0%**	**3.84**	
Fleet and Infrastructure					
Fleet	US$ 000s	125,691	10.0%	0.86	
Crushers & Conveyors & All UG Infrastructure	US$ 000s	145,674	15.0%	1.00	
Grand Total	**US$ 000s**	**830,655**	**14.3%**	**5.70**	

21.5.8 Underground Production Fleet

During the initial 30 quarters, when mining would be performed by a contractor, the cost of fleet would be charged as an operating expense. Thereafter, the Owner would take over all mining activities other than vertical development and would purchase all required mining fleet. The timing and quantum of fleet expenditures assumes that equipment would be acquired on lease-to-own terms, and the following assumptions (based on H2 2024 discussions with OEMs):

- Downpayment of 20%
- 7.5% interest rate
- 5-year repayment with $0 residual.

The majority of costs presented in Table 21.9 are based on 2025 quotations provided by Chilean based OEMs.

Table 21.9 : Underground Production Fleet			
Item	**Description**	**LOM Purchases**	**US$ 000s**
Jumbo	twin boom	4	4,718
Bolter		5	5,898

Table 21.9 : Underground Production Fleet			
Item	**Description**	**LOM Purchases**	**US$ 000s**
Cable bolter		2	2,349
Oversize rig	modified single boom jumbo	16	11,752
Long hole		8	11,636
Charger		15	11,680
Shotcreter		1	716
Scissors lift		3	2,102
Development LHD	17 tonne	2	2,709
Production LHD	21 tonne	34	51,153
Truck	63 tonne	1	1,605
Utility LHD	7 tonne	2	1,509
Grader		1	452
Water cart		1	545
Personnel carrier		6	1,495
Material car		7	1,400
Sub-Total			**111,718**
Interest charges			13,973
TOTAL			**125,691**

21.5.9 Underground Infrastructure

The infrastructure capital cost of $145.7M includes the following items:

- Mobilisation and Demobilisation of the Underground Development Contractor, at a combined cost of $4.6M. A further $11.5M is provided for the establishment of site infrastructure.
- Two underground crusher stations with an estimated cost of $18.6M each (inclusive of ground support), for an aggregate of $37.2M.
- The conveyor for transporting crushed ore and waste to surface. This will be comprised of 7 segments, each ranging from 540 – 1,000 m length and aggregating to 5,340 m. The estimated installed cost for this unit is $54.7M.
- A total of 21.2 km of underground roadways that will be concreted at a cost of $26.9M.
- An underground workshop. Note that the associated excavation is included under development costs. The cost of equipping the facility is $1.1M.
- Ventilation fans, totalling $6.3M.
- The access portal, totalling $1.5M.
- Pumps and dewatering infrastructure, totalling $0.5M.
- Equipping refuge chambers (the associated excavation included under development) at a total cost of $1.2M.
- Communications hardware and software totalling $0.4M.

21.6 Process Plant Estimation Methodology

The capital cost estimate has been assembled on an electronic spreadsheet using the following general methods of calculation.

21.6.1 "Bulk" Quantities

Preliminary global quantities for earthworks, concrete, steelwork, and platework have been determined from in-house data for similar installations, equipment lists, preliminary layout drawings and vendor data. To these quantities rates as noted above have been applied.

21.6.2 Equipment Costs

A detailed equipment list has been prepared and imported into the estimate. To each item of equipment costs have been entered as per the basis outlined above.

21.6.3 All in Labour Gang Rates

Labour rates for the estimate have been based on rates developed from the Wood Santiago office labour rate model (specific for this Project), in-house data and documents in the public domain. The rates include accommodation and travel costs, supervision, construction plant and cranes, temporary facilities, and contractor's mark-ups to give a total labour cost per hour (commonly referred to as "Gang rate"). HCH provided the contractor direct labour rates for earthwork, concrete and steel.

Main labour gang rates as listed have been used in the estimate:

Table 21.10 : Labour Gang Rates

Discipline	Rate
Earthworks	$107.05/hr varies (dependent on operation)
Concrete Works	$49/hr
Structural install	$57.18/hr
Platework	$59.58/hr
Mechanical Equipment Install	$59.58/hr
Electrical	$54.63/hr
Instrumentation & Controls	$53.61/hr
Building rate	$54.83/hr
Piping	$64.24/hr

21.6.4 Construction Hours

Site construction hours have been calculated using Australian norms as the basis. These hours allow for lost time items such as:

- Toolbox talks
- Tea breaks
- Lunch breaks

- Toilet breaks
- Mobilising to next task
- Paid breaks
- Safety briefings
- Job planning.

A productivity factor of 30% has been applied to these norms to reflect the estimated hours considered when working in the Atacama region of Chile. In other words, the norms are multiplied by 1.3 to give actual construction hours. Please note any productivity losses due to inter alia labour market, schedule requirements (fast track), boom period, or industrial climate has not been taken into consideration. Productivity takes account of such things as:

- Climate
- Greenfields/brownfields/operational plant work
- Union environment
- Working week duration.

21.6.5 Piping

Piping costs for the process plant have been calculated by applying factors to the equipment supply cost. These factors vary depending on the equipment type and size. They are based on Wood's experience of similar installations. The off-plot lines have been calculated by applying material plus installation rates to preliminary quantities.

21.6.6 Electrical

In-plant electrical costs have been calculated by applying factors to the equipment installed kW values. These factors are based on in-house data for similar installations. The Sub-Station, Transformer Installation and HV Feeder have been costed from preliminary quantities and in-house data costs.

21.6.7 Instrumentation and Controls

In-plant instrumentation and control costs have been calculated by applying a factor per kW based on equipment installed kW. These factors are based on in-house data for similar installations.

21.6.8 Plant and Other Project Buildings

Offices, meeting rooms, ablutions and the like are fully fitted out with air conditioning, verandas etc. Also included is partitioning, furnishings, workstations, computers, appliances, sanitary fixtures. Workshops, Warehouses/Stores etc. are bare shell, however an OET maintenance crane is allowed for in the Workshop. No allowance has been included for with maintenance equipment, tools and services, storage shelving etc.

21.6.9 Construction Camp

Construction camp has been excluded from the CAPEX. All workforces will be housed locally, an allowance of US$48 per day has been included in the CAPEX for this, this cost was provided to Wood by HCH.

21.6.10 Freight and Import Duty

Ocean, inland and local freight for equipment has been applied as a percentage of the equipment costs. Normally a percentage of 5% is considered an acceptable figure for in-country freight and 10% for international and ocean freight. Import Duty and Customs Duties if applicable have not been applied. Freight costs for bulk materials such as steelwork, platework and piping have been based on sourcing these commodities from the Vallenar area and local regional centres.

21.6.11 Preliminaries (Mobilisation/Demobilisation Temporary Facilities, etc.)

Preliminaries are those items that must be included to the estimate but cannot be included in specific WBS areas because they are applicable across several areas.

They consist of:

- Mobilisation and demobilisation of contractors
- Heavy lift cranes for the installation of sizers, thickeners etc., (in labour gang rate)
- Contractors site ablutions
- Construction site temporary power
- Construction site temporary communications
- Site temporary power to service contractor’s huts etc.
- Site temporary fuel storage
- Construction water standpipe and storage
- Contractors’ laydown areas
- Cost of vendor’s representatives to be present when commissioning equipment
- Assistance by contractors when carrying out commissioning the plant.

Mobilisation and demobilisation, and commissioning assistance costs have been included as a percentage allowance against the various disciplines.

Vendor’s representatives’ cost is a calculation assessing duration of personnel days on site plus travel time, fares, accommodation, expenses etc.

Contractor’s temporary facilities and services are incorporated in the labour rate as a percentage.

21.6.12 Capital Spares

Included is an amount equal to 5% of the equipment cost to cover capital and start up spares.

21.6.13 First Fills and Reagents

Included is an amount equal to 3% of the equipment cost to cover first fills and reagents.

21.6.14 Vendor Representatives

Included is an amount equal to 5% of the equipment cost to cover vendor representatives.

21.6.15 Commissioning Assistance

Included is an amount equal to 9% of the equipment cost to cover commissioning assistance.

21.6.16 Indirect Costs – Temporary Facilities and EPCM

Temporary facilities such as the establishment of construction management temporary facilities, temporary infrastructure has been included as a percentage of direct cost.

For the EPCM various percentages were (dependent on work area) applied to undertake detail design, procurement, project management, construction management and commissioning if undertaken by an engineering company familiar with this task. Also included within these EPCM amounts are such things as specialist sub-consultants, travel costs, hire vehicles, insurances etc.

The EPCM factor allowance assumes the complete design and Project execution is undertaken by an experienced and capable Engineering contractor.

21.6.17 Growth Allowances

Growth allowances have been applied at the following percentages:

- Process Plant 10%
- Solvent Extraction and Electrowinning (WBS 2540 and 2550) 15%
- General Infrastructure 20%
- Civil Infrastructure 15%.

The accuracy provision does not consider allowances outside the control of the Project such as:

- Currency exchange rate fluctuations
- Construction market forces
- Environmental considerations
- Community input considerations
- Heritage site considerations
- Unusual weather conditions
- Labour availability
- Difficult ground conditions
- Change to statutory regulations, charges, and taxes
- Scope changes.

21.7 Closure Costs

In conjunction with HCH, a closure strategy for the Project was defined. The closure capital estimate was then compiled aligned to the agreed closure strategy. For the process plant and associated internal infrastructure, closure considered the following demolition components:

- Dismantle plate steelwork
- Demolition of steel structures
- Demolition of reinforced concrete structures and foundations.

21.8 Salvage Costs

For the estimation of salvage costs the rates used against the bulk quantities did not consider any upside salvage retrieval component. Dependent on 'site' status at closure stage, there could be opportunity for 'salvage retrieval' as opposed to 'salvage disposal'. The assumption made is salvage disposal (a landfill location like the tailings storage facility would be a viable option).

However, a negotiated demolition/salvage contract would typically be put in place, with the contractors considering the following:

- Retrieval of copper from stripped out cables – the first demolition/salvage component to be agreed (providing access for demolition/salvage of mechanical/steelwork/concrete)
- Retrieval of mechanical equipment – salvage and re-purposing of existing equipment
- Retrieval of platework/steelwork - ratio of retrieval/disposal components would be lower for a hydrometallurgical operation where cleanup (due to process contamination) is required
- Retrieval of rebar from demolished concrete – labour intensive, so may only apply to major concrete structure like mill foundations.

For the financial assessment, a nominal 10% of the capital value of mechanical equipment was used as a salvage value. No further inclusion for the allowance for salvage retrieval of platework/steelwork or electrical cables. This would be a further upside potential. Salvage disposal costs on site would normally be offset against any salvage retrieval.

Table 21.11 : Salvage Values			
Item	**Total Value US$**	**Salvage Retrieval Value (10%)**	**Salvage Value US$**
Major Platework (1,194 t) - (mechanical)	182,063,031	10%	18,206,303
Plant Steelwork (6,605 t)	-		
Process Plant Buildings (Steelwork) (2,688 t)	-		
Electrical Cables	-		

21.9 Owners Costs

Owners Costs have been provided to Wood by HCH for inclusion in the CAPEX Estimate.

21.10 Owners Project Contingency

Owners Project Contingency has been provided to Wood by HCH for inclusion in the CAPEX Estimate.

21.11 Qualifications and Clarifications

- GST or any like tax has not been included.
- If Import Duty is applicable to any overseas pricing of imported equipment it has been excluded
- Estimate assumes all construction labour and plant can be sourced from within Chile
- No escalation costs have been included from base date of estimate
- No allowance has been included for dewatering, piling or specialised sub-strata improvement
- It is assumed that suitable borrow pits for building materials are near and where there is a requirement for select fill it can be produced with a minimum of screening and water conditioning
- CAR or goods in transit insurances have been included within the indirect cost
- All Mining Costs Excluded from the Process Plant Estimat
- No Construction Camp is included, all workforce will be housed locally
- Main HV Switchyard at Vallenar excluded from Estimate
- All Port Infrastructure and Seawater Pipeline is excluded in the CAPEX (Refer to Section 21.13 and 21.14).

21.12 Capital Cost Summary

The total CAPEX summary for the Costa Fuego Project is summarised in the table below:

Table 21.12 : Capital Cost Summary				
WBS Area No	WBS Area Title	Pre Start US$	Expansion US$	Total US$
Direct Cost				
1100	**Mine Pit Development**	-	-	-
1110	Exploration	-	-	-
1120	Geology	-	-	-
1130	Geotechnical	-	-	-
1200	Mine Site Preparation and Roads	-	-	-
1210	Site Preparation	544,664	421,529	966,193
1220	HV Haul Roads	19,132,500	12,236,500	31,369,000
1230	LV Service Roads	1,721,925	4,967,685	6,689,610
1240	Access Roads	-	-	-
1300	Surface Mining Productora	-	-	-
1310	Mining Contractor Mob / Demob	9,708,584	9,708,584	19,417,168
1330	Bench Development	-	-	-
1340	Drill and Blast	-	-	-
1350	Loading	-	-	-
1360	Haulage	-	-	-
1370	Mine Dewatering	820,548	-	820,548
1400	**Surface Mining Cortadera**	-	-	-
1410	Mining Contractor Mob / Demob	-	-	-
1420	Mine Pre-Strip	-	-	-
1430	Bench Development	-	-	-
1440	Drill and Blast	-	-	-
1450	Loading	-	-	-
1460	Haulage	-	-	-
1470	Mine Dewatering	-	845,157	845,157
1500	Underground Mining Cortadera	-	-	-
1510	Underground Development	-	684,980,499	684,980,499
1520	Underground Mine Infrastructure	18,168,858	127,505,454	145,674,312
1530	Underground Mining Equipment	-	-	-
1540	Ventilation & Services	-	-	-
1550	Underground Material Handling	-	-	-
1560	Mine Dewatering	-	-	-
1600	Mining Mobile Plant and Equipment	-	-	-
1610	Mine Heavy Vehicles	1,040,000	390,000	1,430,000
1620	Mine Mobile Heavy Plant	-	-	-
1630	Rescue Equipment	-	-	-
1700	Mining Facilities Productora	-	-	-
1705	Mining Offices/Facilities	495,241	-	495,241
1710	Heavy Vehicle Workshop	9,552,293	6,368,196	15,920,489
1715	Light Vehicle Workshop	-	-	-

Table 21.12 : Capital Cost Summary				
WBS Area No	**WBS Area Title**	**Pre Start US$**	**Expansion US$**	**Total US$**
1720	Plant Workshop	-	-	-
1725	Explosives (ANFO) Storage	500,000	-	500,000
1730	Fuel Farm	2,104,817	-	2,104,817
1735	Truck Refuelling	-	-	-
1740	Truck Wash Facility	-	-	-
1745	Truck Parking Area	-	-	-
1750	Waste Collection/Storage	-	-	-
1800	Mining Facilities Cortadera	-	-	-
1805	Mining Offices/Facilities	-	464,028	464,028
1810	Heavy Vehicle Workshop	-	11,466,685	11,466,685
1815	Light Vehicle Workshop	-	-	-
1820	Plant Workshop	-	-	-
1825	Explosives (ANFO) Storage	-	1,000,000	1,000,000
1830	Fuel Farm	-	1,754,014	1,754,014
1835	Truck Refuelling	-	-	-
1840	Truck Wash Facility	-	-	-
1845	Truck Parking Area	-	-	-
1850	Waste Collection/Storage	-	-	-
1900	Mine Services	-	-	-
1910	Power	-	-	-
1920	Water Treatment, Storage and Distribution	-	-	-
1930	Sewage treatment	-	-	-
1940	Comms	10,966,425	11,242,855	22,209,280
1950	Waste	-	-	-
1000	**TOTAL - Mining**	**74,755,855**	**873,351,187**	**948,107,043**
2100	**Cortadera Surface Crushing**	-	-	-
2110	ROM Pad & Primary Crushing	-	31,061,418	31,061,418
2120	Services	-	3,192,522	3,192,522
2200	Overland Conveying	-	-	-
2210	Rope Conveyer	-	171,646,333	171,646,333
2220	Rope Conveyer	-	-	-
2230	Rope Conveyer	-	-	-
2240	Rope Conveyer	-	-	-
2300	Cortadera Dump Leaching	-	-	-
2310	Dump Leaching	-	-	-
2320	Overland Piping	-	-	-
2400	**Productora Sulphide Concentrator**	-	-	-
2410	ROM Pad & Primary Crushing	45,365,230	-	45,365,230
2420	Stockpile & Reclaim	16,664,873	-	16,664,873
2430	Grinding	215,859,402	-	215,859,402

Table 21.12 : Capital Cost Summary				
WBS Area No	**WBS Area Title**	**Pre Start US$**	**Expansion US$**	**Total US$**
2440	Sulphide Flotation & Regrind	60,563,448	-	60,563,448
2450	Sulphide Concentrate Washing	4,326,186	-	4,326,186
2460	Molybdenum Flotation	7,129,698	-	7,129,698
2470	Molybdenum Conc. Dewatering & Handling	6,849,344	-	6,849,344
2480	Copper Concentrate Dewatering & Handling	16,080,802	-	16,080,802
2490	Tailings Dewatering & Handling	27,176,483	-	27,176,483
2500	Productora Oxide Plant	-	-	-
2510	Ore Handling and Stockpile	53,626,293	-	53,626,293
2520	Ore Preparation	24,879,342	-	24,879,342
2530	Heap Leaching	34,856,499	19,338,522	54,195,021
2540	Solvent Extraction	22,840,923	-	22,840,923
2550	Electrowinning	24,303,992	-	24,303,992
2560	Cathode Handling & Storage	-	-	-
2600	Productora Dump Leaching	-	-	-
2610	Dump Leaching	-	41,295,764	41,295,764
2700	Productora Reagents	-	-	-
2710	Sulphide Flotation Reagents	2,125,462	-	2,125,462
2720	Molybdenum Flotation Reagents	350,000	-	350,000
2730	Flocculant	905,512	-	905,512
2740	Sulphuric Acid	10,230,389	-	10,230,389
2750	Solvent Extraction Reagents	48,497	-	48,497
2760	Electrowinning Reagents	73,492	-	73,492
2800	Productora Plant Services - Cooling Water	1,063,185	-	1,063,185
2810	Raw Water	2,085,376	-	2,085,376
2820	RO Water	6,810,957	-	6,810,957
2830	Process Water - High Chloride	122,274	-	122,274
2840	Process Water - Low Chloride	418,380	-	418,380
2850	Demineralised Water	1,129,774	-	1,129,774
2860	Gland Water	361,349	-	361,349
2870	Potable Water	438,594	-	438,594
2880	Air Systems	2,589,316	-	2,589,316
2890	Diesel	660,000	-	660,000
2900	As Required for Process Plant	-	-	-
2000	**TOTAL - Process Plant**	**589,935,072**	**266,534,559**	**856,469,631**
3100	Infrastructure	-	-	-
3110	Site Preparation & Improvements (Earthworks)	-	-	-
3111	Site Preparation & Improvements Cortadera	-	1,960,271	1,960,271
3112	Site Preparation & Improvements Productora	6,106,002	-	6,106,002
3120	Plant Roads	-	-	-
3121	Plant Road Cortadera	-	431,603	431,603

Table 21.12 : Capital Cost Summary				
WBS Area No	**WBS Area Title**	**Pre Start US$**	**Expansion US$**	**Total US$**
3122	Plant Road Productora	996,875	-	996,875
3130	Plant Swithroom/s Electrical Distribution	34,615,892	8,045,692	42,661,584
3140	Plant Buildings	22,427,143	-	22,427,143
3141	Mine Operations Centre (MOC)	-	-	-
3142	Cafeteria/Dining	-	-	-
3143	Maintenance Workshop	3,501,184	-	3,501,184
3144	Mobile Equipment Workshop	1,709,162	-	1,709,162
3145	Stores/Warehouse Building	-	-	-
3146	Metallurgical Laboratory and Analytical	-	-	-
3147	Security/Gatehouse	-	-	-
3148	Plant Operations Buildings	-	-	-
3149	Bath House	-	-	-
3150	Fire Protection	4,200,000	-	4,200,000
3160	Water Treatment	-	-	-
3170	Sewage Disposal and Treatment	2,400,000	-	2,400,000
3180	Mobile Equipment	9,213,784	-	9,213,784
3190	Bulk Fuel Storage/Distribution HV/LV Washbay	1,200,000	-	1,200,000
3200	Electrical, Instrumentation and Controls	-	-	-
3210	Control Systems	-	-	-
3220	Communications	960,000	-	960,000
3300	Refuse/Waste Disposal & Storage Facility	600,000	-	600,000
3400	Product Loadout Concentrator	-	-	-
3410	Earthworks/Concrete Pad	-	-	-
3420	Product Reclaim	-	-	-
3500	Pipe & Cable Racks	5,622,937	-	5,622,937
3600	As Required for Infrastructure (Spare)	-	-	-
3000	**TOTAL - Plant Infrastructure**	**93,552,979**	**10,437,566**	**103,990,545**
4100	Permanent Accommodation/Village	-	-	-
4200	Water Supply & Distribution	Refer to Section 21.14		
4300	Power Supply/Transmission & Distribution	-	-	-
4310	Switchyards/Substations	-	-	-
4320	Transmission	29,804,344	19,395,600	49,199,944
4330	Distribution	Refer to Section	-	-
4340	Emergency Power	1,920,000	-	1,920,000
4400	Tailings		-	-
4410	Tailings Storage Facility	51,352,926	-	51,352,926
4420	Decant Structure/Lines	5,401,536	-	5,401,536
4430	Storage Ponds	259,618	-	259,618
4440	Tailings Slurry Pipeline	10,038,720	-	10,038,720
4500	Area Roads		-	-

Table 21.12 : Capital Cost Summary				
WBS Area No	**WBS Area Title**	**Pre Start US$**	**Expansion US$**	**Total US$**
4600	Stormwater Prevention and Diversion	1,986	-	1,986
4700	Area Communications		-	-
4800	Product Transport/Logistics		-	-
4810	Haul Road Construction Productora to Port	5,688,192	-	5,688,192
4820	Bridge/Culvert Construction		-	-
4830	Road Haulage Fleet		-	-
4840	Road Haulage Facilities		-	-
4900	Productora to Cortadera Corridor		60,565,423	60,565,423
4000	**TOTAL - Area Infrastructure**	**104,467,322**	**79,961,023**	**184,428,346**
5100	Port Storage/Shiploading	Refer to Section 21.13		
5200	Construction Camp	-	-	-
5300	Regional Roads	-	-	-
5400	Electrical Power Feeder	-	-	-
5500	Water Transmission Line	-	-	-
5600	Regional Communications	-	-	-
5700	Overland Conveying	-	-	-
5800	As required for Regional & Remote	-	-	-
5000	**TOTAL - Regional Infrastructure**	**0**	**0**	**0**
6100	First Fill Reagents & Consumables	4,705,310	2,382,411	7,087,722
6200	Ocean Freight	7,842,183	3,970,686	11,812,870
6300	Capital/Strategic/Spares	7,842,183	3,970,686	11,812,870
6400	Mobilisation and demob	6,273,748	3,176,548	9,450,296
6500	Vendor's Representatives	7,842,183	3,970,686	11,812,870
6600	Commissioning Assistance	14,115,931	7,147,234	21,263,166
6700	As required for Miscellaneous (Spare)	-	-	-
6000	**TOTAL - Miscellaneous**	**48,621,540**	**24,618,254**	**73,239,794**
	TOTAL Direct Cost	**911,332,769**	**1,254,902,589**	**2,166,235,358**
Indirect Costs				
7100	Construction Facilities, Services & Equipment	8,671,689	3,658,097	12,329,786
7200	EPCM	119,993,188	35,375,178	155,368,366
7000	**TOTAL Indirect Cost**	**128,664,878**	**39,033,275**	**167,698,152**
	TOTAL BARE COST	**1,039,997,647**	**1,293,935,864**	**2,333,933,510**
9300	Owner's Costs	42,016,620	18,889,796	60,906,416
9400	Owners Project Contingency	84,033,240	37,779,592	121,812,831
	TOTAL Initial Capital	**1,166,047,506**	**1,350,605,251**	**2,516,652,757**

21.13 Port Costs

HCH undertook a benchmarking exercise surveying costs from multiple ports, cost categories that were benchmarked were:

- Loading
- Warehousing
- Load rate.

Any port restrictions in terms of maximum draft, beam or LOA were also noted as part of the analysis. From the benchmarked data ports that utilised rotainers and port costs that were outliers were removed from the data set as shown below in Table 21.13. The average of the applicable dataset was then applied in the financial model.

Table 21.13 : Benchmark Results for Loading and Warehousing Concentrates 2025

Port	Loading (US$/MT)	Warehousing (US$/MT)	Total (US$/MT)	Load Rate (kt/day)
Salaverry	16.87	5.00	21.87	8.00
Matarani	21.10	0.00	21.10	10.00
Arica	18.35	7.77	26.12	10.00
Antofagasta- ATI	17.60	0.72	18.32	10.00
Average	**18.48**	**3.37**	**21.85**	

Source Data: 2025 Benchmark Copper Concentrates Ports.pdf

21.14 Seawater Supply Costs

The seawater costs were developed as part of the capital estimating, the results are shown below in Table 21.14. The basis of the seawater capacity is 500 L/s.

The basis of the capital costs are as follows:

- MTO and contractor pricing for Pipeline and Pumping Station
- Intake Structure quotation from HCH
- Power line based on unit rate.

Table 21.14 : Seawater Capital Costs

WBS	Description	US$ (Million)
4200	Intake Structure	19.3
-	Desalination	0
4200	Pipeline	64.1
4200	Pumping Stations	16.6
4320	Power line	14.9
Total Direct	Includes Growth	114.9
Indirects	EPCM, Owners Costs	18.4
	Project Owners Contingency	17.2
Total		**150.6**

Sustaining costs were calculated as US$1,307,655 per annum.

The operating cost was then calculated based on the PFS design and throughput rates as described above:

- Fixed Cost US$ 1,300,794
- Variable Cost, Power 50,432 MWh.

These inputs of Capex and Opex were then applied to the Huasco Water financial model for Stage 1 only. The result of the financial model was a return on investment of 19% for Stage 1, the fixed and variable costs from the Huasco financial model are:

- Fixed Cost 33,021,053 per annum
- Variable Cost $0.69/m^3$ seawater.

These fixed and variable costs from the model are then applied to the operating costs for Productora and Cortadera as noted in Section 21.16.

21.15 Mining Operating Cost

21.15.1 Operating Cost Open Pit Mining

Table 21.15 summarises elements of the open pit mining operating cost estimate. This estimate was developed using a zero-based model. As noted previously, all expenditures for the open pit have been classified as operating costs. The open pit plans to employ a contractor for the entire LOM. The contractor will have responsibility for operations, maintenance and supervision. The Owner will be responsible for technical functions, such as geology, survey and environmental management.

Table 21.15 : Summary of Open Pit Mining Operating Costs by Process

Area	Item	Units	Productora / Alice	Cortadera	San Antonio	Total
Production	Total Mined	000 t	710,604	172,691	12,391	895,687
Operating Cost by Process	Item	Units	Productora / Alice	Cortadera	San Antonio	Weighted Average
	Drilling	US$/t	0.09	0.09	0.17	0.09
	Blasting	US$/t	0.26	0.27	0.43	0.26
	Loading	US$/t	0.18	0.18	0.28	0.18
	Hauling	US$/t	0.89	0.85	0.98	0.88
	Support Equipment	US$/t	0.15	0.17	0.90	0.16
	Maintenance Labour	US$/t	0.12	0.18	0.42	0.13
	Management	US$/t	0.11	0.22	0.14	0.13
	Technical Services	US$/t	0.09	0.10	0.02	0.09
	Sub-Total	**US$/t**	**1.88**	**2.06**	**3.30**	**1.93**
	Fleet Rental	US$/t	0.20	0.18	0.32	0.19

Table 21.15 : Summary of Open Pit Mining Operating Costs by Process						
Area	**Item**	**Units**	**Productora / Alice**	**Cortadera**	**San Antonio**	**Total**
	Overheads	US$/t	0.22	0.10	0.10	0.20
	TOTAL	**US$/t**	**2.29**	**2.34**	**3.72**	**2.32**

Sources of the estimates presented in the above tables are detailed in the following sections.

21.15.1.1 Open Pit Global Assumptions

Key assumptions used in the modelling of open pit mining operating costs included the following:

- The mine would operate 24/7 for 360 days a year, allowing for 5 days of vacations and/or weather delays. Four crews would be employed, working 12-hour shifts.
- Drills and trucks would be operated manually, using on board personnel (i.e., autonomous solutions are not considered). Absenteeism of 5% was assumed, with additional personnel provided on the roster to ensure equipment did not stand down.
- A US$ exchange rate of CLP 970 has been assumed.
- A mining contractor would be employed for the entire life of mine. The additional costs associated with the contractor were determined by calibrating the zero-based model against the budgetary estimates submitted by three different contractors. This calibration assumed that energy (diesel and electricity) would be provided to the contractor at cost. Other costs (labour, consumables, maintenance spares and fleet rental) were increased to match the estimates of the various contractors. Separately, the budgetary estimates provided for overhead costs that would include their profit margin. A comparison of the bids is as follows:
  - The lowest cost bid required a calibration of 12% (i.e., the zero-based estimate was increased by 12%) while overhead costs totalled $0.20/t or approximately 9% of the pre-overhead cost.
  - The highest bid required a calibration of 22% while overhead costs totalled $0.51/t or 19% of the pre-overhead cost.
  - The average of the three bids required a calibration of 18% while the overhead costs totalled $0.48/t or 18% of the pre-overhead cost.

Note that reported costs for contract mining are based on the lowest cost bid.

21.15.1.2 Open Pit Drilling and Blasting

The operating cost estimate assumes a single size drill and blast pattern; with mining conducted on 15 m benches using 251 mm holes. The sub-drill will be 0.8 m and there will be 3.8 m of stemming.

All holes will be charged with ANFO. The powder factor will range as follows:

- Dump Leach ore will be charged at 0.36 kg/t.
- The remaining ore (Oxide Leach and Mill Feed) will be charged at 0.30 kg/t.
- Waste will be charged at 0.25 kg/t.

Manually operated drills would achieve average availability and utilisation of 85% and 80%, respectively, resulting in 5,875 engine hours annually. The assumed instantaneous penetration rate of 50 m/hr is reduced to an effective rate of 35 m/hr after non-productive time (moving between holes and patterns) is accounted.

Maintenance costs for the various drills were based on data provided by OEMs that accounts for the impact of age (hourly costs are less initially, then rise as the accumulated engine hours increase).

An ANFO cost of US$0.65/kg has been assumed, based on a budgetary estimate provided by a local supplier.

21.15.1.3 Open Pit Loading and Hauling

The operating cost estimate assumes three sizes of load and haul equipment:

- Productora Main and the bulk of Cortadera, representing approximately 91% of the total mine plan, will be loaded using face shovels equipped with 38 m^3 dippers (e.g., Komatsu PC7000) into 230 t trucks (e.g., Komatsu 830e).
- The Productora satellite pits (North and South), Alice and the deepest portions of Cortadera, representing approximately 8% of the total mine plan, will be loaded using face shovels equipped with 16 m^3 dippers (e.g., Komatsu PC3000) into 90 t trucks (e.g., Komatsu HD785).
- All material from San Antonio, representing approximately 1% of the total mine plan, will be loaded using backhoes equipped with 7 m^3 dippers (e.g., Komatsu PC1250) into 40 t articulated trucks (e.g., Komatsu HM400).

Excavators were assumed to have average availability and utilisation of 85% and 83%, respectively, resulting in 6,120 engine hours annually. Trucks were assumed to achieve approximately similar annual engine hours, with availability and utilisation of 88% and 80%, respectively.

The productivity calculation for loading units assumed 25% of total engine hours would be non-productive, either waiting for trucks or moving between muckpiles.

The productivity calculation for trucks assumed speeds based on OEM rimpull/retard curves and/or assumed property speed limits as follows:

- In-pit flat (empty and loaded) 20 km/h
- Ex-pit flat (empty and loaded) 35 km/h
- Up-hill 10% gradient loaded 11 km/h
- Up-hill 10% gradient empty 28 km/h
- Down-hill 10% gradient loaded 27 km/h
- Down-hill 10% gradient empty 40 km/h.

All equipment was assumed to be diesel powered. Consumption rates were based on data provided by OEMs.

Equipment maintenance costs were based on data provided by OEMs that accounts for the impact of age (hourly costs are less initially, then rise as the accumulated engine hours increase).

21.15.1.4 Open Pit Support Equipment

The following support equipment was provided:

- Front End Loaders (FEL) to support the various loading units at the empirical rate of 1 FEL engine hour per 10 loading unit engine hours. Two sizes of FEL were provided, with a small unit (Komatsu WA600 class, 12 t payload) provided for loading units of the 16 m^3 class (Komatsu PC 3000) or smaller, and a medium unit (Komatsu WA900 class, 25 t payload) for the larger loading units.
- Dozers to support loading units, various pit construction activities and to spot trucks tipping on dumps. The empirical rate of 1 dozer hour per loading unit hour was used. Three sizes of dozer were provided, with a small unit (Komatsu D155 class) provided for loading units of the 7 m^3 class (Komatsu PC 1250), a medium unit (Komatsu D375 class) for loading units up to 22 m^3 class (Komatsu PC 4000) and a large dozer (Komatsu D475 class) for the larger loading units.
- Graders to maintain roads at an empirical rate of 1 grader hour per 20 truck hours. Two sizes of grader were provided, with a small unit (14 ft blade, Komatsu GD 655 class) provided for 90 t trucks (Komatsu HD785 class) or smaller, and a large unit (18 ft blade, Komatsu GD 955 class) for the larger trucks.
- Water tankers to mitigate dust throughput the mine, at an empirical rate of 1 tanker hour per 20 truck hours. A single size unit was provided, with a modified 135 t truck (Komatsu HD 1500 class) that would be equipped with a 32 k gallon (121 m^3) tank.
- Utility excavators (Komatsu PC 900 class) for wall scaling and general construction. A single unit would be provided for each active pit.

Maintenance costs for the various machines were based on data provided by OEMs that accounts for the impact of age (hourly costs are less initially, then rise as the accumulated engine hours increase).

21.15.1.5 Open Pit Maintenance

Maintenance personnel were provided according to the following empirical rates:

- Drills: 0.6 maintenance artisan hours per engine hour for the size of drill assumed.
- Excavators: 1.0 artisan hours per engine hour for the smaller loading units (up to PC 3000 class) and 3.0 artisan hours for the larger PC 7000 class units.
- Trucks: 0.6 maintenance artisan hours per engine hour for units 90 t and smaller, rising to 0.8 artisan hours per engine hour for the larger 230 t class trucks.
- FELs: 0.8 artisan hours per engine hour for the range of FELs that would be employed.
- Other support equipment: 0.5 artisan hours per engine hour.

It was assumed that 90% of the artisans working on the mobile fleet would be mechanical trades and the remaining 10% higher cost electrical trades.

An allowance was made for additional personnel to maintain infrastructure such as dewatering pumps at each of the pits. An allowance was also made for lower-cost apprentices and labour, at the ratio of 1 apprentice per 10 artisans.

Both Productora and Cortadera would have separate fuel bays and tire bays, with a complement of maintenance personnel to staff each.

The workshops would be equipped with a fleet of ancillary equipment that has been discussed previously.

21.15.1.6 Open Pit Management and Technical

While the Owner would retain senior management at each of Productora and Cortadera (i.e., a mine manager), the remaining supervision of operations and maintenance personnel would be the responsibility of the Contractor. Note that San Antonio would be managed under Cortadera.

The Owner would retain responsibility for technical activities, including mining engineering, geology and survey. An average total complement of 27 technical personnel would be provided for Productora, reducing to 16 for the smaller Cortadera pit. Technical Services for San Antonio would be provided by Cortadera staff.

21.15.2 Operating Costs Underground Mining

Table 21.16 summarises elements of the underground mining operating cost estimate. This estimate was developed using a zero-based model. As noted previously it is planned to make use of a development contractor for the initial 30 quarters. During this time, all expenditures (including those incurred by the Owner, such as for ventilation) will be capitalised. Thereafter, the contractor will only be responsible for vertical development.

Table 21.16 : Summary of Underground Mining Operating Costs by Process

Item	Units	Capital	Operating	Total	US$/t
Total Ore [1]	000 t	2,773	145,693	148,466	
Development	US$ 000s	453,730	-	453,730	3.06
Undercut Drill & Blast	US$ 000s	8,185	41,528	49,713	0.33
Secondary Drill & Blast	US$ 000s	-	121,359	121,359	0.82
LHD Mucking	US$ 000s	-	170,584	170,584	1.15
Infrastructure Maintenance	US$ 000s	305	17,355	17,660	0.12
Maintenance Labour	US$ 000s	896	42,096	42,992	0.29
Logistics	US$ 000s	185	34,650	34,836	0.23
Crushing	US$ 000s	-	179,722	179,722	1.21
Conveying	US$ 000s	1,840	90,155	91,994	0.62
Ventilation	US$ 000s	55,537	130,974	186,512	1.26
Management & Technical	US$ 000s	38,611	171,940	210,552	1.42
Fleet	US$ 000s	125,691	-	125,691	0.85
Crushers & Conveyors & All UG Infra	US$ 000s	145,674	-	145,674	0.98
Total Cost	**US$ 000s**	**830,655**	**1,000,365**	**1,831,020**	**12.33**
Unit Rate	**US$/t**	**5.59**	**6.74**	**12.33**	

[1] 1.8 Mt development ore mined during initial 30 quarters.

21.15.2.1 Underground Global Assumptions

Key assumptions used in the modelling of underground mining operating costs included the following:

- As with the open pit, the underground mine would operate 24/7 for 360 days a year, allowing for 5 days of vacations and/or weather delays. Four crews would be employed, working 10-hour shifts, with 2 hours allowed for end-off shift blasting (including secondary breaking, when development and undercutting has been completed).
- All equipment would be operated manually, using on board personnel. Absenteeism of 5% was assumed, with additional personnel provided on the roster to ensure equipment did not stand.
- The exchange rate and prices for diesel and electricity are as per the open pit; at CLP 970 : 1 US$, US$0.73/litre (CLP 708/litre) and $65/MWhr (CLP 63k/MWhr), respectively. Grid power would be available from the start of the third year of the overall program, with diesel generators used prior. For the assumed price of diesel, the cost of generated power would be $249/MWhr.
- To cater for the impact of the underground environment on the productivity of production equipment, a third factor has been included in productivity calculations: productive utilisation (being the percentage of engine hours where productive work is performed).

21.15.2.2 Development and Undercutting

All lateral and vertical development is classified as a capital expense and has been discussed previously.

Undercutting includes:

- Cutting of drawbells, which is classified as a capital expense. Drawbells would be cut with a pattern of 92 holes averaging 17.5 m in length and charged with 3.1 kg explosive per metre.
- Drilling and blasting of undercut pillars, which is classified as an operating expense. Pillars would be drilled with rings of 23 holes averaging 13.8 m in length and charged with 7.0 kg explosive per metre.
- Preconditioning holes drilled into the rockmass immediately above the undercut. Precondition rings of 7 holes would average 23.7 m in length and be charged with 3.6 kg explosive per metre.

Maintenance costs for the various long hole drills were based on data provided by OEMs that accounts for the impact of age (hourly costs are less initially, then rise as the accumulated engine hours increase).

Over the life of mine, 16% of the total tonnage drilled and blasted is capitalised drawbells, with the remaining 84% of tonnage in undercut pillars and preconditioning classified as operating costs.

A key metric is water consumption by the various drill machines as well as in the workshops. Over the life of underground mine, water consumption will total 1.0 Mm^3 with peak daily consumption of approximately 450 m^3. After accounting for the effective utilisation of drill equipment, this translates to a maximum instantaneous consumption (i.e., the rate that reticulation infrastructure must be capable of delivering) of 31 L/s.

21.15.2.3 Secondary Drilling and Blasting

The tonnage of cave ore requiring secondary breaking was assumed to be as follows:

- For the initial 10% of cave draw, 20% would require secondary breaking

- Thereafter, as the cave matures, the percentage requiring secondary breaking would reduce to 10%.

The weighted average of secondary breaking over the life of mine is 10.8% of the cave.

Holes would be drilled using a modified single boom jumbo, at the empirical factor of 5.6 tonnes broken per metre drilled (approximately 1 hole per 2 m^3). Rigs were assumed to have an availability of 85% and utilisation of 75%, resulting in 4,590 engine hours annually (i.e., time spent accruing costs). Of these, the productive utilisation of 90% would result in 4,130 hours of useful work.

Maintenance costs for secondary breaking rigs were based on data provided by OEMs that accounts for the impact of age (hourly costs are less initially, then rise as the accumulated engine hours increase).

Holes would be charged at a powder factor of 2 kg/t.

21.15.2.4 LHD Mucking

Production LHDs would be 21 t class (Sandvik LH521) and would muck from drawpoints into ore passes. 1-Way tramming distances would initially be short at 120 m, as the initial drawpoints would be located close to ore passes. Thereafter the tramming distance would increase to a maximum of 300 m as the cave footprint expands. However, the longer hauls and increased number of production tunnels would have a favourable impact on congestion. Over the life of mine, the average 1-way distance would be 230 m and congestion would be 16% of total load and haul hours.

LHDs would have an availability of 88% and a utilisation of 79%, resulting in 4,942 engine hours annually. Of these, the productive utilisation of 92% would resulting in 4,547 hours of useful work.

Maintenance costs for the various sizes of LHDs were based on data provided by OEMs that accounts for the impact of age (hourly costs are less initially, then rise as the accumulated engine hours increase).

21.15.2.5 Underground Infrastructure Maintenance

Underground Infrastructure would be maintained by the following fleet:

- Graders and water cart for maintaining roadways. The empirical factor of 0.04 engine hours for each of the grader and water cart per metre of underground roadway was assumed.
- Utility LHDs for various construction activities. The empirical factor of 0.0002 engine hours per tonnage of cave ore was assumed.

All machines would have an availability of 75% and a utilisation of 60%, resulting in 3,200 engine hours annually.

21.15.2.6 Underground Logistics

Personnel would be transported underground in cars with an assumed capacity of 21 persons.

Maintenance spares and consumables would be transported in material cars. The empirical relationship of one car load per 2,500 tonnes broken was used to calculated material car requirements.

21.15.2.7 Underground Crushing and Conveying

Costs associated with crushing ore and waste include the following:

- consumable steel (liners) of $0.75/tonne crushed
- maintenance spares of $0.21/tonne crushed
- power consumption of 3 kWhr/tonne crushed
- One operator for each of the two crusher stations per crew.

Costs associated with conveying ore and waste include the following:

- replacement idlers and conveyor rubber costs totalling $2.8M per annum
- combined Installed power of 8.9 MW for the 11 conveyor segments
- a clean-up crew of 12 persons.

21.15.2.8 Underground Maintenance

Mobile equipment maintenance personnel were provided according to the following empirical rates:

- Drills and Bolters: 1.0 maintenance artisan hours per engine hour.
- LHDs: 0.6 artisan hours per engine hour for both the 17 t development LHDs and 21 t production LHDs.
- Trucks: 0.5 maintenance artisan hours per engine hour.
- Other support equipment: 0.4 artisan hours per engine hour.

It was assumed that 80% of the artisans working on the mobile fleet would be mechanical trades and the remaining 20% higher cost electrical trades.

Other maintenance personnel include:

- Lead hands: 2 total
- Light Vehicles: 4 total
- Fixed Infrastructure (pumps, fans, crushers and conveyors): 6 x mechanics and 3 x electricians.

An allowance was made for lower-cost apprentices and labour, at the ratio of 1 apprentice per 10 artisans.

21.15.2.9 Underground Ventilation

Total airflow requirements were based on the Canadian standard of 0.06 m^3/kW installed fleet and infrastructure, with the total kW derated by 8% to account for not all units operating concurrently. An additional allowance of 15% was then made for leakage.

At peak installed power of 15.4 MW, airflow of 976 m^3/s will be required while the LOM average values are 10.4 MW and 662 m^3/s, respectively.

Airflow was converted to kW installed fan capacity at a ratio of 13 kW/m^3. An allowance was made for spot coolers totalling 500 kW.

Maintenance costs for the system were estimated using the empirical ratio of $180/installed kW per annum.

21.15.2.10 Underground Management and Technical

The Owners' management and technical complement was estimated at 46 personnel for the initial phase before the commencement of caving. Thereafter, the complement will increase to 83 personnel.

21.16 Processing Operating Cost Estimate

21.16.1 Summary

Operating cost estimates have been prepared for the Cortadera surface primary crusher, Productora primary crusher and sulphide concentrator, Productora oxide plant (ore preparation, heap leach, SX/EW), Productora sulphide dump leach and port operations (copper concentrate storage and transhipment).

All costs are estimated in United States dollars as at Q4 2024 and are judged to have an accuracy of ±15% to 25% and are considered to be at PFS level in accordance with Wood's Estimating Procedures and Class 4 as defined in the AACE document 18R-97.

The average processing costs for the four treatment operations and the port facilities are summarised in Table 21.17 to Table 21.21.

Table 21.17 : Average Processing Operating Costs for Cortadera Surface Crusher			
Project Area	**Annual $/a**	**$/t Crusher Feed**	**Distribution %**
Cortadera Primary Crusher			
Labour	1,121,598	0.08	30.33
Electricity	456,088	0.03	12.33
Maintenance	1,338,232	0.10	36.19
Consumables	598,174	0.04	16.18
Miscellaneous (including water)	184,000	0.01	4.98
Total Area OPEX	**3,698,093**	**0.27**	**100.0**
Relative costs by category,			
Fixed	2,459,831	0.18	66.5
Variable	1,238,262	0.09	33.5

Table 21.18 : Average Processing Operating Costs for Productora Sulphide Primary Crusher and Concentrator				
Project Area	**Annual $/a**	**$/t Concentrator Feed**	**$/t Cu Concentrate (dry)**	**Distribution %**
Productora Primary Crusher and Concentrator				
Labour	5,423,056	0.26	14.72	3.1
Electricity	42,029,574	2.03	114.08	24.0
Maintenance	13,230,245	0.64	35.91	7.6
Reagents	12,108,597	0.58	32.87	6.92
Consumables	40,135,607	1.94	108.94	22.9
Miscellaneous	4,022,350	0.19	10.92	2.3
Sea Water Supply	38,946,647	1.88	105.71	22.25
Concentrator Total	**155,896,076**	**7.53**	**423**	**89.1**
Relative costs by category				
Fixed	56,959,231	2.75	155	32.5
Variable	98,936,845	4.78	269	56.5
Other Operating Costs				
Concentrate Transport	4,714,366	0.23	13	2.69
Operating General and Administration	9,091,257	0.44	25	5.19
Corporate General and Administration	5,303,180	0.26	14	3.03
Other Total	**19,108,804**	**0.92**	**52**	**10.9**
Total Area OPEX	**175,004,880**	**8.45**	**475**	**100**

Table 21.19 : Average Processing Operating Costs for Productora Oxide Plant				
Project Area	**Annual $/a**	**$/t Oxide Plant ROM Feed**	**$/t Cu Cathode**	**Distribution %**
Oxide Heap Leach and Project SX/EW				
Labour	4,784,095	3	398.67	13.98
Electricity	5,913,081	1.48	492.76	17.28
Maintenance	6,735,686	1.68	561.31	19.68
Reagents	13,679,683	3.42	1139.97	39.97
Consumables	1,819,638	0.45	151.64	5.32
Miscellaneous	572,906	0.14	47.74	1.67
Process Plant Sub Total	**33,505,089**	**8.38**	**2792**	**97.9**
Relative costs by category, %				
Fixed	12,092,687	3.02	1008	35.3
Variable	21,412,402	5.35	1784	62.6
Other Operating Costs				
Cathode Metal Transport	600,000	0.15	50.0	1.75
General and Administration	123,841	0.03	10.3	0.36
Sub Total	**723,841**	**0.18**	**60.3**	**2.11**
Total Area OPEX	**34,228,930**	**8.56**	**2,852**	**100**

Table 21.20 : Average Processing Operating Costs for Productora Sulphide Dump Leach				
Project Area	**Annual $/a**	**$/t Dump Leach ROM Feed**	**$/t Cu Cathode**	**Distribution %**
Productora Dump Leach				
Labour	1,916,668	0.53	1,065	21.6
Electricity	905,075	0.25	503	10.2
Maintenance	383,267	0.11	213	4.3
Reagents	5,653,066	1.57	3,141	63.8
Total Area OPEX	**8,858,076**	**2.46**	**4,921**	**100.0**
Relative costs by category				
Fixed	1,916,668	0.53	1,065	21.6
Variable	6,941,408	1.93	3,856	78.4

Table 21.21 : Average Processing Operating Costs for Port Operations				
Project Area	**Annual $/a**	**$/t Concentrator Feed**	**$/t Cu Concentrate (dry)**	**Distribution %**
Port				
Labour	1,539,200	0.07	4.18	43.34
Electricity	162,457	0.01	0.44	4.57
Maintenance	1,101,959	0.05	2.99	31.03
Miscellaneous	747,429	0.04	2.03	21.05
Total Area OPEX	**3,551,046**	**0.17**	**9.6**	**100**
Relative costs by category				
Fixed	1,752,153	0.08	4.8	49.3
Variable	1,798,892	0.09	4.9	50.7

21.16.2 Basis of Process Plant Estimates

21.16.2.1 Assumptions and Sources of Data

The operating cost estimates have been estimated by Wood with input from HCH. The operating costs are based on the following throughput criteria:

- Cortadera surface primary crusher – 13.5 Mtpa ROM feed
- Productora sulphide primary crusher and concentrator – 20.7 Mtpa ROM feed, producing 368 ktpa Copper concentrate (dry) and 4.2 ktpa Molybdenum concentrate (dry)
- Oxide plant – 4 Mtpa oxide ore, producing 12,000 ktpa copper cathode
- Sulphide dump leach – 3.6 Mtpa ROM ore producing 1,800 ktpa of copper cathode
- Port operations - 368 ktpa copper concentrate (dry).

The sources of information and the basis of assumptions used for estimating the operating costs are as follows:

- Plant throughput agreed with HCH arising from ore delivery schedule (Rev E).
- Design criteria from a combination of testwork parameters, process modelling and database projects.
- Mass balances developed using SysCAD software.
- Database pricing and vendor quotations obtained by Wood for reagents and consumables during the course of the study, supplemented with transport costs provided by HCH.
- Plant labour structures from the 2015 PFS and 2020 PEA and escalated rates benchmarked against similar projects.
- Reagent consumptions are based on the process design criteria and mass balance output.

- Electricity consumption was determined from the detailed mechanical equipment lists for the Cortadera Surface Primary Crusher, Productora Sulphide Concentrator, Productora Oxide Plant (Ore preparation, Heap leach, SX/EW), Productora Sulphide Dump Leach and the Port operations.
- Maintenance costs were factored from the direct capital cost by discipline using typical factors based on Wood experience.
- Larger equipment associated consumable costs were derived from similar projects and advice from vendors.
- Currency exchange rates have been adopted as consistent with the capital cost estimates.

21.16.2.2 Labour

The labour rates are comprised of a base rate and an overhead component. The labour costs are subdivided into administration, technical/plant management, operations and maintenance components. Labour costs were escalated from the 2015 PFS and benchmarked against similar projects. Personnel levels and the shift operating strategy to be employed was based on the 2015 PFS.

Table 21.22 summarises the plant labour structure and total costs for administration, concentrator and refinery operations.

Table 21.22 : Cortadera Primary Crusher Operation Labour Summary

Parameter	Labour Count
Administration (shared with Productora Concentrator)	-
Technical services (shared with Productora Concentrator)	-
Plant operations	7
Plant maintenance	8
Total	**15**

Table 21.23 : Productora Primary Crusher and Sulphide Concentrator Operation Labour Summary

Parameter	Labour Count
Administration	43
Technical services	6
Plant operations	31
Plant maintenance	31
Total	**111**

Table 21.24 : Productora Oxide Operation Labour Summary

Parameter	Labour Count
Administration (shared with Productora Concentrator)	-
Technical services (shared with Productora Concentrator)	-
Plant operations	44
Plant maintenance	38

Table 21.24 : Productora Oxide Operation Labour Summary	
Total	**82**

Table 21.25 : Productora Dump Leach Operation Labour Summary	
Parameter	**Labour Count**
Administration (shared with Productora Concentrator)	-
Technical services (shared with Productora Concentrator)	-
Plant operations	17
Plant maintenance	17
Total	**34**

Table 21.26 : Port Operation Labour Summary	
Parameter	**Labour Count**
Administration (shared with Productora Concentrator)	-
Technical services (shared with Productora Concentrator)	-
Plant operations	15
Plant maintenance	6
Total	**21**

21.16.2.3 Reagents

The reagent costs have been estimated using the reagent consumption indicated in the process design criteria and estimated by the mass balance (SysCAD and Excel models) and reagent prices from vendors and database sources. Reagents will be delivered to site. Salt addition to the heap leach and dump leach operations will be sourced within the operation from the seawater supply. The site RO plant produces both clean water and brine, and the brine is further treated by electrodialysis to make a highly-concentrated brine which is used in the process. The operating cost associated with producing the concentrated brine includes the operating costs (power, consumables, labour) as well as amortised capital costs associated with the electrodialysis process.

A breakdown of the reagent consumptions and costs is presented in Table 21.27.

Table 21.27 : Productora Sulphide Concentrator, Oxide and Sulphide Dump Leach Reagent Summary				
Reagent	**Main Function**	**Consumption Basis**	**Consumption**	**Delivered Cost (US$/t)**
Concentrator				
Collector (RTD2086)	Primary collector	t/a	973	6,485
Kerosene	Secondary collector	t/a	5	1,570
Frother	Flotation	t/a	1,249	3,414
NaHS	Flotation Conditioner	t/a	405	550
Flocculant	Thickening	t/a	218	5,974

Table 21.27 : Productora Sulphide Concentrator, Oxide and Sulphide Dump Leach Reagent Summary				
Reagent	**Main Function**	**Consumption Basis**	**Consumption**	**Delivered Cost (US$/t)**
Oxide Plant				
Sulphuric acid (98%)	Heap Leaching	t/a	90,640	110
Concentrated Brine (Sodium chloride)	Heap Leaching	m^3/a	302,220	7.20*
SX Diluent	Solvent Extraction	t/a	539	1,639
SX Extractant	Solvent Extraction	t/a	37	8,500
Cobalt Sulphate	Electrowinning	t/a	14	10,749
Guar	Electrowinning	t/a	3	3,734
Ferrous Sulphate	Electrowinning	t/a	30	3,948
Activated Clay (Bentonite)	Solvent Extraction	t/a	24	2,383
Sulphide Dump Leach				
Sulphuric acid (98%)	Heap Leaching	t/a	33,588	110
Concentrated Brine (Sodium chloride)	Heap Leaching	m^3/a	271,998	7.20*

* Concentrated Brine cost in $/$m^3$

21.16.2.4 Consumables

Consumable costs, which includes mill grinding balls, crusher and mill liners, etc., were calculated based on consumption rates estimated by Wood or replacement frequencies of wear items estimated by Wood based on experience or vendor recommendations. Specific consumable costs were based on previous experience or vendor quotations.

21.16.2.5 Maintenance

Maintenance costs are estimated by applying percentage factors to the direct installed capital cost estimate for each area. The maintenance factors used typically reflect the individual circumstances of the Project, including project location, the nature of the process and the ore being processed. The factors typically cover maintenance materials, hired equipment and Contractor costs where these are not specified separately. Maintenance labour is allowed for in the labour cost as part of the personnel schedule. Areas in which separate allowances are made for maintenance consumables (e.g. mill liners) typically have a reduced percentage factor.

Maintenance factors range from 1 to 5%. Average factors for the main areas of the Project are summarised in Table 21.28.

Table 21.28 : Average Maintenance Factors	
Operation	**Average Factor**
Cortadera Surface Crusher	3.9
Productora Primary Crusher and Concentrator	3.1
Productora Oxide Plant	4.0

Table 21.28 : Average Maintenance Factors	
Port Operations	3.0

21.16.2.6 Electricity

The electricity usage is based on the equipment power draw calculated from the installed capacity with utilisation factors applied to reflect actual drawn power. The electrical power supply cost of US$0.065/kWh was provided by HCH. Table 21.29 summarises the power consumption for the four operations.

Table 21.29 : Operations Electricity Demand Summary, kW

Area	Connected, kW	Operating, kW	Annual, MWh/a
Cortadera Surface Crusher	1,431	1,216	7,017
Productora Primary Crusher and Concentrator	102,750	87,274	646,609
Productora Oxide Plant	12,812	10,891	90,079
Port Operations	371	315	2,499

21.16.2.7 Process Miscellaneous Costs

The process miscellaneous costs include the laboratory and mobile equipment diesel costs with estimates for general site activities.

Allowances have been made for miscellaneous items as listed in Table 21.30.

Table 21.30 : Miscellaneous Costs Summary

Cost Item	Basis	Cost	Basis
Cortadera Surface Crusher			
Mobile equipment fuel, maintenance, and licensing	$/a	75,825	Costs derived based on selected vehicle models and associated running costs in Wood's database.
Health, Safety and Environment	$/a	3,000	Allowance for safety supplies. $200/employee
General consumables	$/a	55,000	Allowance for stationery, IT consumables and miscellaneous plant consumables. Includes 10% for freight.
Contract labour and services	$/a	80,000	Allowance for planned major shutdowns and vendor representatives.
Productora Primary Crusher and Concentrator			
Laboratory services	$/a	1,751,525	Allowance based on contract operation of laboratory.
Mobile equipment fuel, maintenance, and licensing	$/a	915,249	Costs derived based on selected vehicle models and associated

Table 21.30 : Miscellaneous Costs Summary			
Cost Item	**Basis**	**Cost**	**Basis**
			running costs in Wood's database and allowance for hire equipment.
Health, Safety and Environment	$/a	22,100	Allowance for safety supplies. $200/employee
General consumables	$/a	55,000	Allowance for stationery, IT consumables and miscellaneous plant consumables. Includes 10% for freight.
R&D (testwork and consultants)	$/a	300,000	Allowance for external consultants and external testwork.
Contract labour and services	$/a	828,476	Allowance for planned major shutdowns, vendor supervisor, contract OSA servicing.
Concentrate Loading (FEL contract)	$/a	534,476	Concentrate loading contract at $1.31/t (all in cost).
Sea Water Tariff	$/a	29,168,966	Fixed cost: $23.5m/a Variable cost: $0.66/m^3
Productora Oxide Plant			
Laboratory services	$/a	105,655	Allowance based on contract operation of laboratory.
Mobile equipment fuel, maintenance, and licensing	$/a	395,850	Costs derived based on selected vehicle models and associated running costs in Wood's database.
Health, Safety and Environment	$/a	16,400	Allowance for safety supplies. $200/employee
General consumables	$/a	55,000	Allowance for stationery, IT consumables and miscellaneous plant consumables. Includes 10% for freight.
Port Operations			
Mobile equipment fuel, maintenance, and licensing	$/a	144,753	Costs derived based on selected vehicle models and associated running costs in Wood's database.
Health, Safety and Environment	$/a	4,200	Allowance for safety supplies. $21/employee
General consumables	$/a	55,000	Allowance for stationery, IT consumables and miscellaneous plant consumables. Includes 10% for freight.
Concentrate Loading (FEL contract)	$/a	534,476	Concentrate loading contract at $1.31/t (all in cost).
Water	$/a	9,000	Allowance for potable water and dust suppression.

21.16.2.8 Raw Water Costs

The cost of raw water (sea water) is based on tariff rates provided by HCH. The tariff rates have an annual fixed cost and variable cost component and are the costs for providing water to the Productora Sea Water Pond. The fixed cost component is applied to the concentrator operating costs and the variable cost component is applied to the water requirement for each of the operations as per the water balance. An additional variable component is added for water used at Cortadera because of the additional requirements to pump the water to Cortadera. The fixed and variable water components are shown in Table 21.31.

Table 21.31 : Raw Water Tariff	
Fixed Cost	US$M 33.02/year
Variable Cost (Productora)	$0.69/m^3
Variable Cost (Cortadera)	$0.85/m^3

21.16.2.9 General and Administration Costs

Operational Administrative costs were estimated for each Project area with input from HCH and are summarised in Table 21.32. General and Administration costs for the Cortadera Surface Primary Crusher and Port operations are covered by the Concentrator costs.

Table 21.32 : Operational General and Administration Costs	
Operation	**Allowance**
Productora Primary Crusher and Concentrator	
Labour	4,444,392
Site Offices	1,288,333
Maintenance (Access Road and Buildings)	738,910
Local Transport	149,622
Consultants (Mining & Geology, Environmental, Accounting, HR, Community, Metallurgical)	350,000
Personnel Costs (Medical, Recreation, Entertainment, Recruitment, OHS, Training)	1,755,000
Contracts (Security and Janitorial)	100,000
Environmental Testing and Monitoring	265,000
Productora Oxide Plant	
Training	47,841
Systems/IT	10,000
General Consumables	60,000
General Freight	6,000

Corporate operating costs were provided by HCH and are summarised in Table 21.33.

Table 21.33 : Corporate General and Administration Costs	
	Allowance ($/a)
Corporate Costs	2,000,000
Insurances	2,048,180
Financial (Legal and Auditing Fees)	250,000
Government Charges (Mining lease, land rental, environmental license, local government)	680,000
Community (Community relations, projects and sponsorships)	325,000

21.16.2.10 Exclusions

The basis of the operating cost estimate excludes the following:

- Copper concentrate, molybdenum concentrate and copper cathode delivery costs are excluded from the process plant operating cost estimate. However, these costs have been calculated for use in the economic modelling undertaken by HCH and are quoted separately.
- All import duties, withholding taxes and other in country taxes.
- All general and administration costs (except for administrative labour and general consumables).
- Any impact of foreign exchange rate fluctuations.
- Any escalation from the date of the estimate.
- Any contingency allowance.
- Any land or crop compensation costs.
- Any rehabilitation or closure costs.
- Any licence fees or further royalties (other than those specified).
- Government monitoring, inspection or compliance costs.
- All Owners' costs and risk allowances, other than specified.
- Additional tailings storage costs (other than specified), including future lifts and rehabilitation, which are considered as sustaining capital.
- All environmental monitoring, and rehabilitation costs, other than specified.
- All mining costs.
- ROM and dump leach rehandling costs.

22 Economic Analysis

22.1 Cautionary Statement

The results of the economic analysis in this Report represent forward-looking information that is subject to a number of known and unknown risks, uncertainties, and other factors that may cause actual results to differ materially from those presented here. Forward-looking statements in this Report include, but are not limited to:

- Statements with respect to the economic and study parameters of the Project
- Forecast metal prices and currency exchange rates
- Timing and amount of future cashflows from mining operations
- Forecast production rates and amounts of product produced from the proposed Costa Fuego Project mining plan and mining method
- Estimation of the Mineral Resources and the realisation of the Mineral Resource Estimates within the PFS mine plans
- Time required to develop the Project based on the PFS mine design
- Cost required to develop the Project based on the PFS mine design
- Assumptions regarding mine dilution
- Assumptions regarding losses
- Expected grade of the material delivered to the mill
- Metallurgical recovery rates
- Initial, Expansion, and Sustaining capital costs
- NPV, IRR, payback period, LOM, production, cashflows and other financial and operational metrics
- Mine closure costs and reclamation
- Timing and conditions of permits required to initiate mine construction, maintaining mining activities, and mine closure
- Assumptions regarding geotechnical and hydrogeological factors.
- General business and economic conditions

The reader is cautioned that the actual results of mining operations may vary from what is forecast. Risks to forward-looking information include but are not limited to unexpected variations in grade or geological continuity, as well as geotechnical and hydrogeological assumptions that are used in the mine designs. There could be seismic or water management events during the construction, operations, closure, and post-closure periods that could affect:

- Predicted mine production quantities and rates
- Timing of the production
- Costs of future production

- Capital expenditures
- Future operating costs
- Permitting timelines
- Potential delays in the issuance of permits, or changes to existing permits
- Requirements for additional capital
- The plant, equipment or metallurgical or mining processes may fail to operate as anticipated.

There may be changes to government regulation of mining operations, environmental issues, permitting requirements, and social risks, or unrecognised environmental, closure costs and closure requirements, unanticipated reclamation expenses, title disputes or claims and limitations on insurance coverage.

The PFS has been undertaken to assess the viability of developing the Costa Fuego Project, it is a technical and economic study based on assessments that are sufficient to support the estimation of ore reserves and is based on the material assumptions in this document. While Hot Chili Ltd (HCH) considers all the material assumptions to be based on reasonable grounds, there is no certainty that they will prove to be correct or that the range of outcomes indicated by the PFS will be achieved.

The PFS is based on the February 2024 Mineral Resource Estimate, Probable Ore Reserves, and a PFS standard level of technical and economic assessments, which do not provide assurance of economic development or certainty that the PFS outcomes will be realised. The PFS has been completed to a level of accuracy of ±25% in line with industry standard accuracy for this stage of development.

22.2 Methodology Used

22.2.1 General

This section of the Study describes the financial model for the Project, which has been prepared by HCH. The final draft versions of the financial model have been reviewed and validated by Wood PLC for internal accuracy and consistencies and the QPs deem it acceptable for use in this PFS.

22.2.2 Financial Model

The financial analysis for the Project has been evaluated using a discounted cashflow (DCF) analysis. Cash inflows consist of quarterly revenue projections for the mine. Cash outflows such as capital, including the pre-production mining costs, operating costs, taxes, and royalties, are subtracted from the inflows to arrive at the quarterly cashflow projections. Cashflows are taken to occur at the end of each period

To reflect the time value of money, quarterly net cashflow (NCF) projections are discounted back to the start of construction using an 8% discount rate. The discount rate appropriate to a specific project depends on many factors, including the type of commodity and the level of project risks, such as market risk, technical risk, and political risk. The discounted present values of the cashflows are summed to arrive at the NPV.

The internal rate of return (IRR) is expressed as the discount rate that yields a zero NPV.

The payback period is the time calculated from the start of production until all initial capital expenditures have been recovered.

This economic analysis includes sensitivities to variation in operating costs, capital costs, grades, discount rate and metal price.

The financial model assumes 100% equity.

All pricing and costs are stated in constant (Real) fourth quarter 2024 United States dollars (US$). No inflation has been assumed in the model.

22.3 Financial Model Parameters

The financial analysis was based on:

- Royalty agreements described in Section 4
- The Mineral Resources presented in Section 14
- The mine and process plan and assumptions detailed in Sections 16 and 17, respectively
- The projected infrastructure requirements outlined in Section 18
- The metal's price, and treatment and refining (TR/RC) assumptions in Section 19
- The permitting, social and environmental regime discussions in Section 20
- The capital and operating cost estimates detailed in Section 21.

22.4 Metal Price

Long-term metal price assumptions and justification are detailed in Section 19 of this report. The metal prices utilised in the economic analysis are summarised below.

Table 22.1 : Metals Prices

Metal	Unit	Price
Copper	US$/lb	4.30
Molybdenum	US$/lb	20.00
Gold	US$/oz t	2,280.00
Silver	US$/oz t	28.00

22.5 Metal Recovery

Recoveries are discussed in Section 16 of this report. Metal recovery is variable and calculated by regression on a block-by-block basis within the model based on metal content, oxidation level, and production feed source in the mine plan.

Over the life of the modelled Project, sulphide copper recovery averages 86%, gold recovery averages 54%, silver recovery averages 37% and molybdenum averages 70%. These values are derived from the developed recoveries as described in Section 13.

Recoverable copper from leach processing is assumed to be 65% for oxide material over a 240-day leach period, and 39% for low-grade sulphide material over a two-year leach period.

22.6 Exchange Rate

For the purposes of the capital cost estimate, the operating cost estimate, and financial analysis, the assumed exchange rates for the LOM are shown in Table 22.2.

Table 22.2 : Exchange Rates

Currency	Exchange Rate
US$/AU	1.45
US$/CLP	830

22.7 Concentrate Physicals and Selling Costs

Treatment and refining costs (TC/RC) assumptions are discussed Section 19. Table 22.3 provides a summary of the concentrate physicals, product payables, and transport/selling costs used in the economic evaluation of the Project.

Table 22.3 : Concentrate Physicals and Selling Costs

	Unit	Value
Cu-Au Concentrate		
Cu-Au Concentrate Moisture	%	8
Cu Grade in Concentrate	%	25
Au Grade in Concentrate	g/t	4
Ag Grade in Concentrate	g/t	14
Mo Concentrate		
Mo Concentrate Moisture	%	5
Mo Grade in Concentrate	%	50
Payables		
Cu Payable (Cu-Au Con.)	%	96
Cu Cathode Payable	%	100
Mo Payable (Mo Con.)	%	99
Au Payable (Cu-Au Con.)	%	92
Ag Payable (Cu-Au Con.)	%	90
Selling Cost		
Sulphide Copper	US$/lb Cu	0.32
Cathode/Leach Copper	US$/lb Cu	0.02
Molybdenum	US$/lb Mo	1.21
Gold	US$/oz Au	5.00
Silver	US$/oz Ag	0.45

22.8 Taxation and Royalties

The tax model was compiled by HCH with advice from third-party taxation professionals who completed a review of the tax assumptions used in the financial model to determine if the assumptions are reasonable and in accordance with current tax rules.

The calculations are based on the tax regime in effect as of the date of this report. The following is an overview of the Chile and Costa Fuego Project tax regime applied in the financial model, as provided by HCH.

The financial model includes tax losses, at the Project start, of US$ 300 M. These encompass the existing investment into the Project (for example, drilling and study workstreams) through to the commencement of construction. The existing HCH balance sheet has been used with an additional component of projected study and finance costs.

The financial model assumes the Chilean Corporate Income Tax rate for the Project is 27% and withholding tax is 35%.

The financial model assumes that the Project is subject to the Chilean mining royalty which has two components, an Ad-Valorem component of 1% on annual copper sales and a Margin component based on Adjusted Mining Operational Taxable Income (RIOMA). The Ad-Valorem component applies when two criteria are met: CuEq sales are over 50ktpa, and more than 50% of the sales for the respective fiscal year correspond to copper. The Margin component rate applied to RIOMA is based on annual tons sold for medium-scale producers (12 to 50ktpa Cu), or on Mining Operating Margin for large scale producers (+50ktpa Cu or CuEq).

The maximum combined tax burden of mining royalty, CIT and withholding tax is 45.5% on six-year average annual production of 50-80ktpa Cu and 46.5% on average annual production over 80ktpa Cu. If the total tax burden exceeds the maximum, the mining royalty is reduced starting with the Margin component and then continuing with the Ad-Valorem component if necessary.

The Project will produce more than 50 kt Cu per year in all years, except the final two. The applicable margin component of the mining royalty rates are shown in Table 22.4, Table 22.5 and Table 22.6.

Table 22.4 : Mining Royalty, Margin Component, Large Scale Copper Production >50,000 tonnes/year

Mining Operational Margin (%) – Annualized Basis		Applicable Rate
Min	Max	
0	20	8%
20	45	12%
45	60	26%
60		26%

Table 22.5 : Mining Royalty, Margin Component, Large Scale Copper Equivalent Production >50,000 tonnes/year		
Mining Operating Margin (%) – Annualized Basis		Applicable Rate
Min	Max	
0	35	5.0%
35	40	8.0%
40	45	10.5%
45	50	13.0%
50	55	15.5%
55	60	18.0%
60	65	21.0%
65	70	24.0%
70	75	27.5%
75	80	31.0%
80	85	34.5%
85	100	14.0%

Table 22.6 : Mining Royalty, Medium Scale Copper Equivalent Production 12,000 - 50,000 tonnes/year		
Mining Operating Margin (%) – Annualized Basis		Applicable Rate
Min	Max	
-	12	0.0%
12	15	0.4%
15	20	0.9%
20	25	1.4%
25	30	1.9%
30	35	2.4%
35	40	2.9%
40	50	4.4%

Royalties in the financial model are applied according to the royalty agreements described in Section 4. There are several mining leases within the modelled Project area that are subject to the following royalty payments included in the financial model:

- Productora production from the Chilean Nuclear Energy Commission (CCHEN) mining right "Uranio 1 al 70" is subject to an NSR of 2% for all metals except gold, 4% for gold, and 5% for non-metallic products.
- Cortadera production from the mining right 'Purísima 1/8 (1/2-5/6)' ('Purisima') is subject to a 1.5% NSR on all products.
- Productora production from the mining right "Zapa 1 al 6" ('Zapa') is subject to a Gross Royalty of 1% on all products.
- Production from Productora, Cortadera and San Antonio are subject to a 1% NSR on copper and 3% NSR on gold under the royalty agreement with Osisko Resources.

22.9 Depreciation

For capital items considered in the financial model, different depreciation schemes apply. The tax depreciation scheme is based on a table provided by the local tax authority, which identifies the type of asset and timeframes over which depreciation occurs.

Based on the description of the assets provided by the company, the following figures contain the number, in years, of the straight-line depreciation and accelerated depreciation. The latter generally corresponds to 1/3 of the years assigned to the straight-line.

Depreciation begins when the company starts to generate income from its principal activities. Accelerated depreciation is applied to calculate the taxable income, while straight-line depreciation is used to determine the operating margin used to calculate the Specific Mining Tax Rate.

Table 22.7 : Depreciation Schedule

Asset description	Straight-Line Depreciation Years	Accelerated Depreciation Years
Bulk earthworks and drainage	9	3
Site services	5	1
Plant for sulphide, oxide and molybdenum processing	9	3
Tailings storage facilities (TSF)	10	3
Mining fleet and associated infrastructure	9	3
EPCM and owner's costs	5	1
Construction facilities, services and equipment	5	1
Working capital	6	1
Capitalised production waste stripping	6	1
High voltage powerline with associated substations	20	6
Water pipeline with associated pump stations and inlet	10	3
Expansion Capital	9	3
First Fills	5	1
Sustaining capital (process plants, TSF, leach pads)	5	1

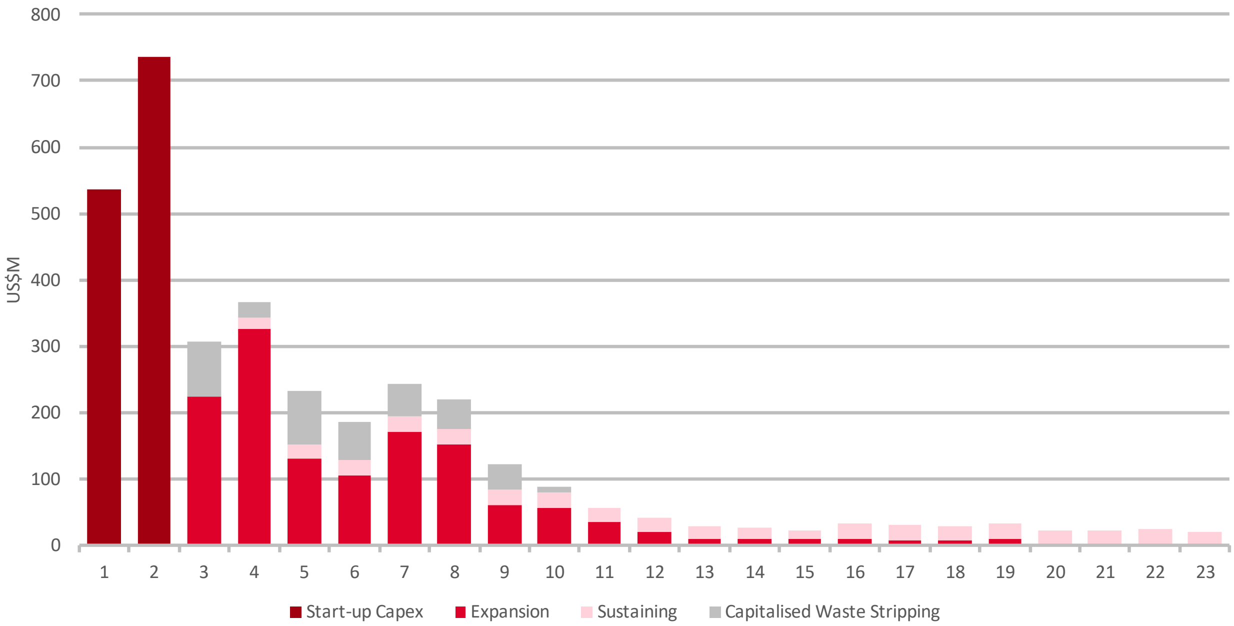
22.10 Capital Costs

Total capital expenditures over the life of the Project total US$2.77 B. A summary of capital expenditures is shown in Table 22.8.

Table 22.8 : Capital Expenditure Summary	
Capital Item	**Amount US$ M**
Initial Capex	1,273
Expansion Capital	1,347
Sustaining Capital (inc. Closure and Salvage)	811
Life of Project Capex	3,430

Closure costs in the financial model are estimated at $78 M and are based on a progressive closure schedule as mining activities at deposits are completed. Salvage value has only considered a 10% recoup of the mechanical equipment value. Detailed breakdown of capital costs can be found in Section 22.

Figure 22.1 : Annual Capital Costs



22.11 Operating Costs

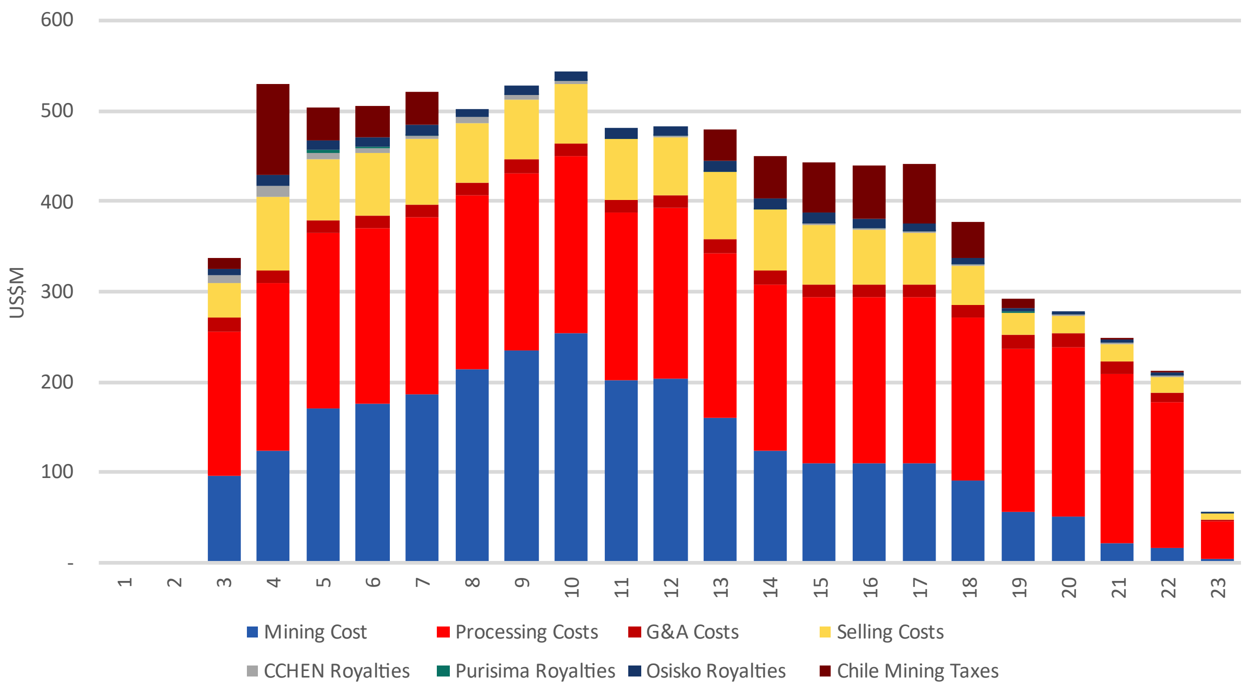
Operating costs have been applied as described in Section 21. Operating unit costs are summarised in the following table. Mining costs incurred before the start of production are capitalised. This only impacts open pit mining costs (3% of total) and is not included in the life-of-mine average below.

During operations, open pit waste mining costs in excess of the average life-of-mine strip-ratio are also capitalised (12% of total). This capitalised stripping is not included in the life-of-mine averages below.

Table 22.9 : Operating Cost Input Summary		
Operating Costs	**Unit**	**Life of Mine**
Mining Cost Average	US$/t mined	2.61
Open Pit	US$/t mined	2.46
Underground	US$/t mined	6.86
Processing Costs		
Rehandle	US$/t Process Feed	0.81
Sulphide Concentrator		
Cu/Au/Ag Concentrate	US$/t Process Feed	7.53
Mo Concentrate	US$/lb Mo in Conc.	0.23
Sulphide Leach		
Front End Processing	US$/t	2.46
Back End Processing	US$/lb Cu	0.16
Oxide Leach		
Front End Processing	US$/t	8.38
Back End Processing	US$/lb Cu	0.16
G&A	US$M/quarter	3.60

Figure 22.2 shows the profile of annual operating costs over the life of the Project.

Figure 22.2 : Annual Operating Costs



22.12 Financial Results

Based on the economic analysis, the Project generates positive before- and after-tax discounted cashflows. The after-tax NPV_8 for the Project is US$1,203 M with 19% IRR and 4.5-year payback period. Table 22.10 below presents a summary of the financial analysis results.

Table 22.10 : Summary of the Financial Analysis

Project Metric			**Units**	**Value**
Financial Measures				
Pre-Tax	Cu US$4.30/lb	NPV8%	US$M	1,712
		IRR	%	22
Post-Tax	Cu US$4.30/lb	NPV8%	US$M	1,203
		IRR	%	19
Payback period (from start of operations)			yr	4.5
Open Pit Strip Ratio			W/P	1.5
NPV/Capex			Ratio	0.94
Capital Costs				
Total Pre-production Capital Expenditure			US$M	1,273
Expansion			US$M	1,347
Sustaining			US$M	811
Total			US$M	3,430
Operating Costs				
C1 – (net of by-product revenue)			$/lb Cu	1.38
Total Cash Cost (net by-products and including royalties)			$/lb Cu	1.61
All-in-Sustaining Cost			$/lb Cu	1.85
All-In Cost LOM			$/lb Cu	2.62
Mine Life and Metal Production				
Primary Mine Production Including Ramp-up			yr	14
Mine Life (Life of Mine Processing)			yr	20
Primary Mine Production – Average Annual Copper Equivalent Metal			kt/yr	116
Primary Mine Production – Average Annual Copper Metal			kt/yr	95
Primary Mine Production – Average Annual Gold Metal			koz/yr	48
Primary Mine Production – Average Molybdenum Metal			kt/yr	2.0

Noting:

- C1 Costs consist of mining costs, processing costs, G&A, selling costs, net of by-product credits
- Total Cash Costs consist of mining costs, processing costs, G&A, selling costs, royalties and production taxes (SMT), net of by-product credits
- All-in-Sustaining Cost includes cash costs plus sustaining capital costs
- All-In Costs includes All-in-Sustaining Costs plus all other LOM capital.

The production and cashflow forecast on an annual basis using mineral resources within the PFS mine plan (Figure 22.3) are shown in Table 22.11.

Figure 22.3 : Feed Schedule Tonnes and Copper Grade

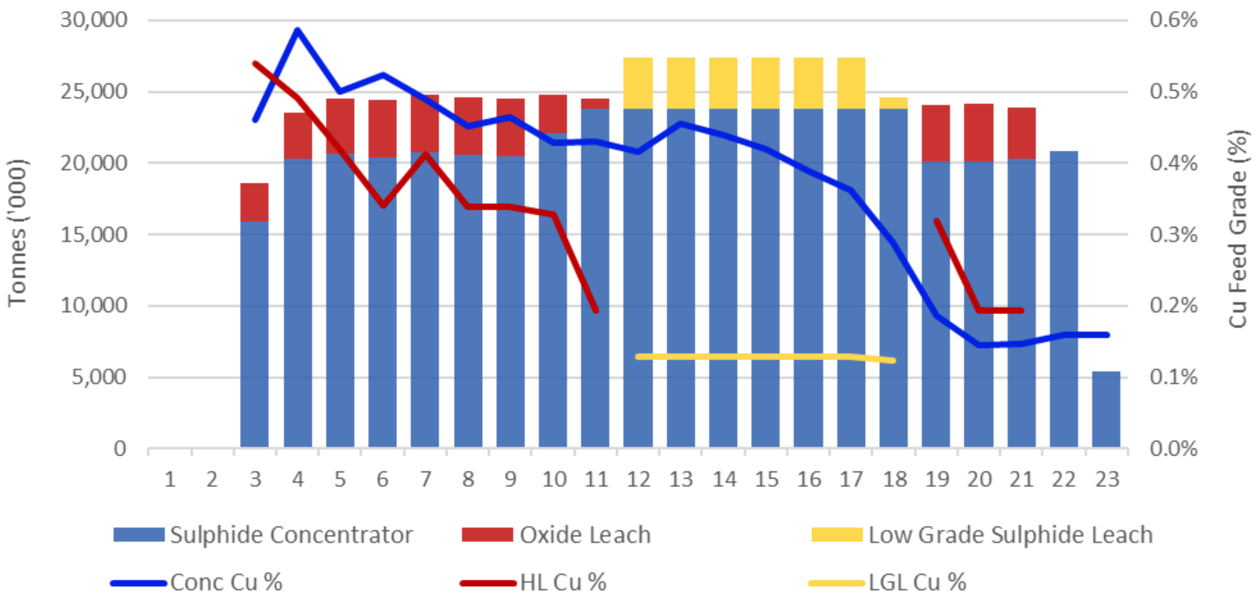


Table 22.11 : Financial Model Output

Year			1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24
		Total																								
Mining																										
Ore Tonnes Mined	kt	504,536	73	6,407	28,350	33,496	56,095	49,176	42,142	44,064	35,948	44,068	25,118	22,231	25,533	19,193	18,373	18,403	18,466	13,871	3,313	197	20	-	-	-
Waste Mined (excl capitalized)	kt	344,008	1,687	36,374	28,096	32,396	37,073	36,818	34,725	37,150	36,992	29,173	13,419	16,977	3,127	2	-	-	-	-	-	-	-	-	-	-
Rehandle	kt	150,132	-	-	1,828	963	-	670	3,966	2,104	4,287	571	2,728	6,851	480	4,651	5,425	5,399	5,338	9,944	20,799	23,954	23,857	20,875	5,443	-
Capitalised Waste	kt	195,609	-	-	49,977	13,524	44,319	28,372	21,363	18,648	16,076	3,331	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Processing																										
Sulphide Concentrator																										
Ore Processed	kt	438,594	-	-	15,866	20,308	20,704	20,387	20,770	20,607	20,502	22,107	23,802	23,817	23,800	23,800	23,798	23,802	23,803	23,815	20,112	20,150	20,326	20,875	5,443	-
Processed Grade																										
Copper	%	0.38%	-	-	0.46%	0.59%	0.50%	0.52%	0.49%	0.45%	0.46%	0.43%	0.43%	0.42%	0.45%	0.44%	0.42%	0.39%	0.36%	0.29%	0.19%	0.14%	0.15%	0.16%	0.16%	-
Gold	g/t	0.10	-	-	0.08	0.12	0.10	0.09	0.10	0.09	0.11	0.11	0.15	0.14	0.14	0.15	0.15	0.13	0.13	0.10	0.05	0.03	0.04	0.04	0.04	-
Silver	g/t	0.51	-	-	0.35	0.39	0.53	0.54	0.44	0.36	0.46	0.51	0.70	0.71	0.72	0.77	0.75	0.67	0.61	0.46	0.31	0.26	0.26	0.27	0.25	-
Molybdenum	ppm	106.35	-	-	97.24	175.53	128.05	122.08	178.54	183.30	155.92	169.72	86.11	104.17	99.99	82.39	74.29	82.65	74.16	62.71	72.33	82.40	77.05	58.74	34.89	-
Contained Metal																										
Copper	kt	1,688	-	-	73	119	104	107	102	93	95	95	102	99	108	105	100	93	86	69	37	29	30	33	9	-
Gold	koz	1,462	-	-	42	76	63	62	65	61	69	82	114	104	107	115	111	101	96	77	34	22	23	30	7	-
Silver	koz	7,221	-	-	177	256	353	353	296	238	304	364	538	540	554	592	571	513	464	351	202	166	168	179	43	-
Molybdenum	kt	47	-	-	2	4	3	2	4	4	3	4	2	2	2	2	2	2	2	1	1	2	2	1	0	-
Low Grade Sulphide Leach																										
Ore Processed	kt	22,385	-	-	-	-	-	-	-	-	-	-	-	3,600	3,600	3,600	3,600	3,600	3,600	785	-	-	-	-	-	-
Processed Cu Grade	%	0.13%	-	-	-	-	-	-	-	-	-	-	-	0.13%	0.13%	0.13%	0.13%	0.13%	0.13%	0.12%	-	-	-	-	-	-
Contained Cu	kt	29	-	-	-	-	-	-	-	-	-	-	-	5	5	5	5	5	5	1	-	-	-	-	-	-
Oxide Leach																										
Ore Processed	kt	40,807	-	-	2,744	3,231	3,851	4,000	4,000	4,000	4,000	2,716	712	-	-	-	-	-	-	-	4,000	4,000	3,552	-	-	-
Processed Cu Grade	%	0.35%	-	-	0.54%	0.49%	0.42%	0.34%	0.41%	0.34%	0.34%	0.33%	0.19%	-	-	-	-	-	-	-	0.32%	0.19%	0.19%	-	-	-
Contained Cu	kt	142	-	-	15	16	16	14	17	14	14	9	1	-	-	-	-	-	-	-	13	8	7	-	-	-
Production																										
Copper Concentrate	dmkt	5,783	-	-	207	421	351	365	360	329	334	331	348	343	389	367	349	328	303	229	120	94	94	94	25	-
Copper Concentrate	wmkt	6,286	-	-	226	458	381	397	391	357	363	360	378	373	423	399	380	357	330	249	130	103	102	102	28	-
Copper in concentrate	kt	1,446	-	-	52	105	88	91	90	82	84	83	87	86	97	92	87	82	76	57	30	24	23	24	6	-
Gold in concentrate	000 oz	784	-	-	21	46	34	33	36	34	38	44	59	56	60	63	61	56	53	40	16	10	10	11	3	-
Silver in concentrate	000 oz	2,698	-	-	62	102	127	135	127	96	120	138	181	198	210	218	211	195	177	126	74	66	64	58	13	-
Molybdenum Concentrate	dmkt	65	-	-	1	5	4	3	5	6	5	6	3	4	3	3	2	3	2	2	2	2	2	2	0	-
Molybdenum Concentrate	wmkt	69	-	-	1	5	4	4	6	6	5	6	3	4	4	3	3	3	3	2	2	2	2	2	0	-
Molybdenum in concentrate	kt	33	-	-	1	3	2	2	3	3	2	3	1	2	2	1	1	1	1	1	1	1	1	1	0	-
Sulphide Copper Cathode	kt	11	-	-	-	-	-	-	-	-	-	-	-	0	1	2	2	2	2	2	1	0	-	-	-	-
Oxide Copper Cathode	kt	93	-	-	9	12	12	10	11	10	7	7	1	0	-	-	-	-	-	-	5	4	4	1	-	-

Year			1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24
Metal Sold																										
Copper in Concentrate	000 lbs	3,187,568	-	-	111,554	234,263	189,641	200,797	200,797	178,486	184,064	184,064	195,219	184,064	217,530	200,797	195,219	178,486	167,331	128,287	66,932	50,199	50,199	50,199	19,441	-
Gold	000 oz	784	-	-	20	46	34	33	37	33	38	44	60	55	61	62	62	55	53	40	16	9	9	11	4	-
Silver	000 oz	2,698	-	-	61	103	124	135	129	95	119	138	184	193	214	216	214	193	177	128	75	64	63	56	19	-
Molybdenum	000 lbs	72,113	-	-	1,319	5,623	4,147	3,833	6,032	6,158	5,215	6,126	3,299	3,990	3,833	3,047	2,670	2,985	2,670	2,168	2,168	2,513	2,325	1,728	265	-
Copper Cathode	000 lbs	228,981	-	-	20,371	26,015	25,397	21,782	24,427	21,870	15,961	15,521	2,646	1,587	3,263	3,968	3,968	3,968	3,968	3,351	11,640	9,612	8,378	1,288	-	-

Year		Total	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24
Revenue																										
Copper in Concentrate	$'000	13,158,281	-	-	460,494	967,037	782,840	828,889	828,889	736,790	759,815	759,815	805,864	759,815	897,963	828,889	805,864	736,790	690,741	529,568	276,296	207,222	207,222	207,222	80,253	-
Gold	$'000	1,643,799	-	-	42,526	96,179	70,554	68,893	77,074	69,928	80,268	91,838	125,229	114,562	128,685	130,754	129,519	116,412	111,723	84,784	33,980	19,413	19,818	23,123	8,539	-
Silver	$'000	67,979	-	-	1,528	2,608	3,119	3,403	3,240	2,389	2,992	3,485	4,630	4,864	5,381	5,441	5,384	4,856	4,453	3,225	1,883	1,609	1,580	1,423	486	-
Molybdenum	$'000	1,427,831	-	-	26,125	111,344	82,108	75,888	119,430	121,919	103,258	121,297	65,314	78,998	75,888	60,337	52,873	59,093	52,873	42,920	42,920	49,763	46,030	34,212	5,241	-
Copper Cathode	$'000	984,619	-	-	87,594	111,862	109,208	93,661	105,037	94,040	68,634	66,738	11,376	6,826	14,030	17,064	17,064	17,064	17,064	14,409	50,054	41,332	36,023	5,538	-	-
Total Revenue	$'000	17,282,510	-	-	618,267	1,289,031	1,047,829	1,070,734	1,133,670	1,025,066	1,014,966	1,043,173	1,012,412	965,065	1,121,948	1,042,486	1,010,704	934,215	876,854	674,907	405,133	319,339	310,674	271,519	94,518	-
Expensed Operating Costs																										
Mining Cost	$'000	(2,716,700)	-	-	(96,863)	(124,439)	(171,066)	(175,693)	(186,494)	(214,711)	(235,235)	(254,199)	(201,528)	(204,532)	(159,570)	(123,524)	(109,491)	(109,239)	(109,624)	(91,081)	(57,049)	(50,422)	(21,099)	(16,604)	(4,237)	-
Processing Costs	$'000	(3,747,812)	-	-	(159,719)	(184,453)	(194,294)	(194,783)	(195,383)	(191,491)	(196,549)	(195,339)	(186,256)	(188,578)	(183,821)	(184,748)	(184,692)	(184,259)	(184,395)	(179,628)	(180,300)	(188,639)	(187,418)	(161,828)	(41,241)	-
G&A Costs	$'000	(284,290)	-	-	(14,394)	(14,394)	(14,394)	(14,394)	(14,394)	(14,394)	(14,394)	(14,394)	(14,394)	(14,394)	(14,394)	(14,394)	(14,394)	(14,394)	(14,394)	(14,394)	(14,394)	(14,394)	(14,394)	(8,997)	(1,799)	-
Selling Costs	$'000	(1,115,614)	-	-	(37,820)	(82,487)	(66,385)	(69,489)	(72,217)	(65,151)	(65,698)	(66,822)	(66,786)	(64,010)	(74,593)	(68,322)	(66,082)	(61,074)	(57,113)	(43,934)	(24,375)	(19,360)	(19,106)	(18,230)	(6,559)	-
CCHEN Royalties	$'000	(61,425)	-	-	(9,343)	(11,162)	(7,948)	(4,651)	(4,746)	(7,427)	(5,174)	(2,010)	(534)	(654)	(96)	(142)	(250)	(653)	(634)	(517)	(1,200)	(1,785)	(1,593)	(883)	(26)	-
Purisima Royalties	$'000	(8,779)	-	-	-	-	(2,391)	(1,702)	(160)	-	(240)	(1)	(36)	(490)	(56)	(543)	(518)	(18)	(35)	(671)	(378)	(91)	(307)	(900)	(241)	-
Zapa Royalties	$'000	(1,226)	-	-	-	-	-	-	-	-	-	-	-	(52)	(626)	(153)	(0)	-	-	(3)	(221)	(51)	-	-	(119)	-
Osisko Royalties	$'000	(180,390)	-	-	(6,393)	(12,913)	(10,420)	(10,641)	(10,999)	(9,826)	(10,095)	(10,423)	(11,296)	(10,507)	(12,276)	(11,730)	(11,481)	(10,451)	(9,886)	(7,567)	(4,064)	(2,904)	(2,863)	(2,659)	(996)	-
Chile Mining Taxes	$'000	(535,789)	-	-	(13,690)	(100,139)	(36,556)	(35,010)	(37,072)	-	-	-	-	-	(34,589)	(46,710)	(56,451)	(59,311)	(65,232)	(39,458)	(10,022)	-	(1,451)	(98)	-	-
Total Operating Costs	$'000	(8,652,026)	-	-	(338,222)	(529,987)	(503,455)	(506,364)	(521,465)	(503,001)	(527,384)	(543,188)	(480,831)	(483,217)	(480,022)	(450,267)	(443,360)	(439,400)	(441,313)	(377,253)	(292,004)	(277,647)	(248,231)	(210,199)	(55,218)	-
Capital																										
Construction	$'000	1,169,803	521,360	648,443	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Capitalised Expenses	$'000	102,887	15,625	87,262	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Expansion	$'000	1,346,850	-	-	224,152	327,110	130,327	105,180	171,423	152,966	60,205	56,391	34,714	20,163	10,371	8,886	9,059	9,731	8,152	7,387	10,633	-	-	-	-	-
Sustaining	$'000	810,845	-	-	83,345	38,861	101,660	80,475	71,125	68,084	62,890	31,643	20,902	20,785	18,490	18,568	14,214	22,753	22,754	22,436	22,504	22,520	22,401	24,487	20,230	(282)
Total Capital	$'000	3,430,384	536,984	735,705	307,497	365,970	231,987	185,655	242,548	221,050	123,095	88,034	55,616	40,947	28,861	27,454	23,273	32,484	30,906	29,823	33,136	22,520	22,401	24,487	20,230	(282)
EBITDA	$'000	8,630,484	-	-	280,046	759,044	544,374	564,370	612,205	522,065	487,582	499,985	531,582	481,847	641,926	592,219	567,343	494,815	435,541	297,654	113,130	41,692	62,443	61,320	39,300	-
Depreciation (accelerated)	$'000	(3,370,548)	-	-	(758,168)	(492,868)	(541,941)	(293,123)	(249,039)	(202,882)	(203,333)	(171,480)	(94,685)	(59,135)	(40,558)	(31,280)	(23,716)	(25,823)	(31,851)	(31,456)	(31,593)	(30,189)	(27,629)	(21,206)	(8,018)	(576)
Tax Loss Carry Forward	$'000	(823,150)	-	-	-	(266,176)	(39,611)	(271,247)	(238,266)	-	-	-	-	-	-	-	-	-	-	-	-	(4,464)	(3,386)	-	-	-
Taxable Income	$'000	4,965,553	-	-	-	-	-	-	124,901	319,183	284,249	328,505	436,897	422,712	601,368	560,939	543,628	468,991	403,690	266,198	81,537	14,889	31,428	40,114	36,323	-
Corporate Income Tax Rate		27%																								
Corporate Income Tax	$'000	(1,340,699)	-	-	-	-	-	-	(33,723)	(86,179)	(76,747)	(88,696)	(117,962)	(114,132)	(162,369)	(151,453)	(146,780)	(126,628)	(108,996)	(71,874)	(22,015)	(4,020)	(8,486)	(10,831)	(9,807)	-

Year			1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24
Free Cashflow																										
Revenue	$'000	17,282,510	-	-	618,267	1,289,031	1,047,829	1,070,734	1,133,670	1,025,066	1,014,966	1,043,173	1,012,412	965,065	1,121,948	1,042,486	1,010,704	934,215	876,854	674,907	405,133	319,339	310,674	271,519	94,518	-
Operating Costs	$'000	(8,652,181)	-	-	(338,222)	(530,206)	(503,434)	(506,343)	(521,468)	(503,001)	(527,384)	(543,188)	(480,831)	(483,217)	(480,022)	(450,267)	(443,360)	(439,400)	(441,313)	(377,232)	(292,000)	(277,647)	(248,231)	(210,199)	(55,218)	-
Capital	$'000	(3,430,384)	(536,984)	(735,705)	(307,497)	(365,970)	(231,987)	(185,655)	(242,548)	(221,050)	(123,095)	(88,034)	(55,616)	(40,947)	(28,861)	(27,454)	(23,273)	(32,484)	(30,906)	(29,823)	(33,136)	(22,520)	(22,401)	(24,487)	(20,230)	282
Pre-Tax FCF	$'000	5,200,100	(536,984)	(735,705)	(27,451)	393,073	312,387	378,715	369,657	301,015	364,487	411,950	475,966	440,900	613,065	564,765	544,070	462,331	404,635	267,831	79,993	19,172	40,043	36,833	19,070	282
Corporate Income Tax	$'000	(1,340,699)	-	-	-	-	-	-	(33,723)	(86,179)	(76,747)	(88,696)	(117,962)	(114,132)	(162,369)	(151,453)	(146,780)	(126,628)	(108,996)	(71,874)	(22,015)	(4,020)	(8,486)	(10,831)	(9,807)	-
Post-Tax FCF	$'000	3,859,400	(536,984)	(735,705)	(27,451)	393,073	312,387	378,715	335,934	214,836	287,740	323,254	358,004	326,768	450,696	413,312	397,291	335,703	295,639	195,957	57,978	15,152	31,557	26,002	9,262	282

Year			1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24
		Total																								
Discounted Cash Flow																										
Discount rate		8%																								
Discount Factor			0.93	0.86	0.79	0.74	0.68	0.63	0.58	0.54	0.50	0.46	0.43	0.40	0.37	0.34	0.32	0.29	0.27	0.25	0.23	0.21	0.20	0.18	0.17	0.16
Pre-Tax																										
$NPV_{8\%}$	$'000	1,662,215	(497,208)	(630,749)	(21,792)	288,760	212,620	238,668	215,689	162,629	182,334	190,813	204,134	175,088	225,423	192,281	171,514	134,950	109,360	67,030	18,536	4,113	7,955	6,775	3,248	44
IRR		22%																								
Post-Tax																										
$NPV_{8\%}$	$'000	1,167,356	(497,208)	(630,749)	(21,792)	288,760	212,620	238,668	196,041	116,069	143,942	149,729	153,542	129,764	165,720	140,717	125,243	97,988	79,902	49,042	13,435	3,251	6,269	4,783	1,578	44
IRR		19%																								

Table 22.12 and Table 22.13 summarise the before and after-tax valuation indicators for the Project. Figure 22.4 shows the annual and cumulative pre-financing project free cashflows. Figure 22.5 shows the breakdown of the annual Project cashflows.

Table 22.12 : Before Tax Financial Results		
Before-Tax Valuation Indicators	**Unit**	**Value**
Undiscounted cumulative cashflow	US$M	5,200
NPV5	US$M	2,618
NPV8	US$M	1,712
NPV10	US$M	1,269
Payback period (from start of operations)	years	4.5
IRR before tax	%	22%

Table 22.13 : After Tax Financial Results		
After-Tax Valuation Indicators	**Unit**	**Value**
Undiscounted cumulative cashflow	US$M	3,859
NPV5	US$M	1,898
NPV8	US$M	1,203
NPV10	US$M	861
Payback period (from start of operations)	years	4.5
IRR	%	19%

Figure 22.4 : Pre-Financing Project Cashflows

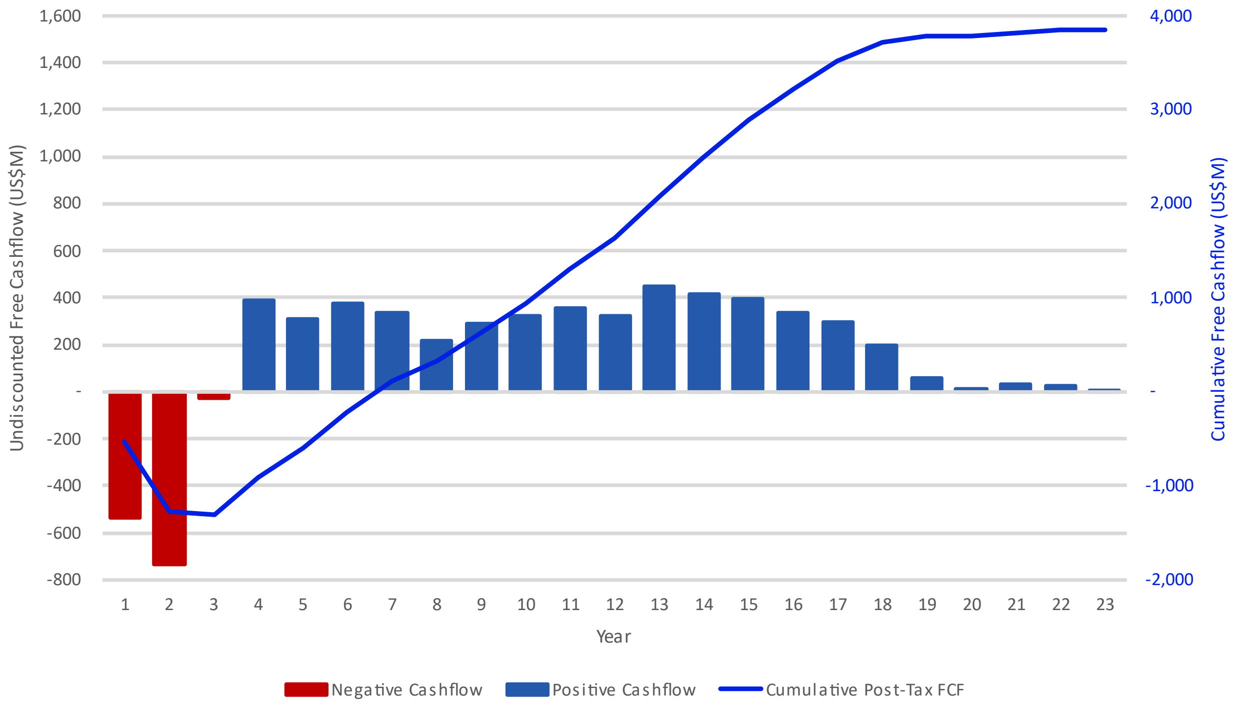
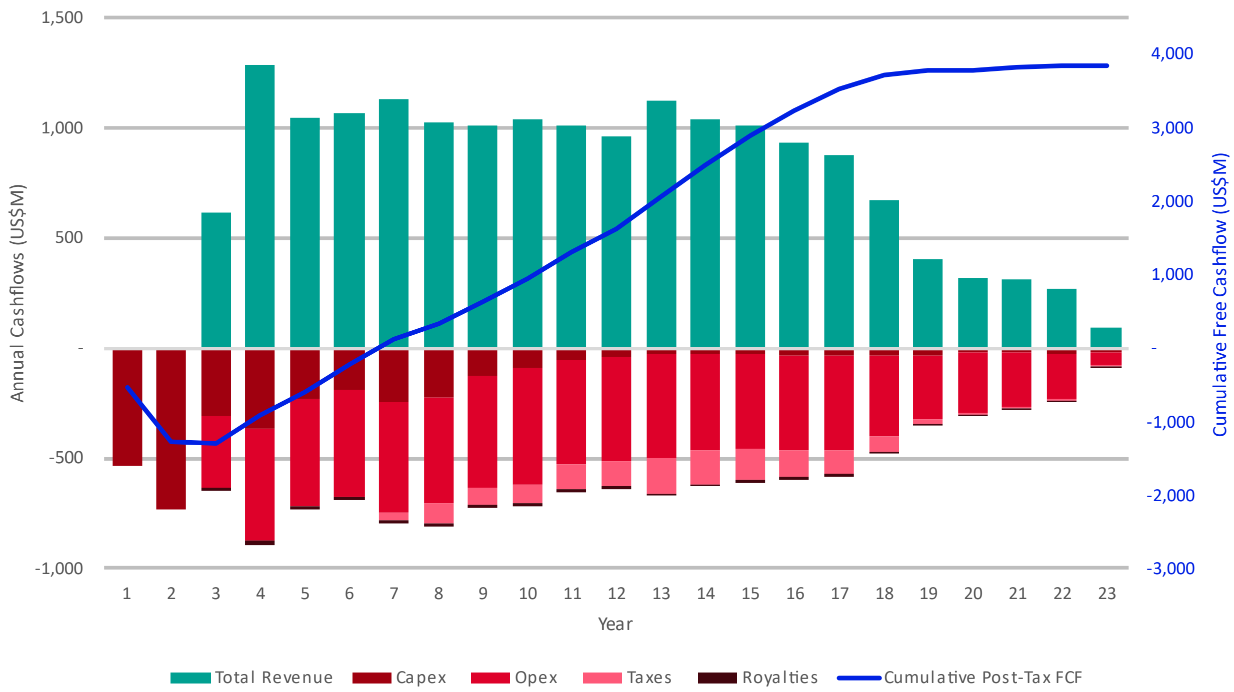


Figure 22.5 : Project Cashflows Breakdown



22.13 Sensitivity Analysis

A sensitivity analysis was conducted on the after-tax NPV and IRR of the Project, with respect to input variables including metal prices, recoveries and grades, capital, operating costs, selling costs and discount rate.

Project after-tax NPV is most sensitive to factors that affect copper revenue - copper price, grade and recovery - and discount rate. NPV is also sensitive to changes in mining cost, processing cost and construction capital.

The results of the analysis are shown in the spider graphs in Figure 22.6 to Figure 22.11.

Figure 22.6 : Project NPV Sensitivity Spider Graph – Metal Price and Discount Rate

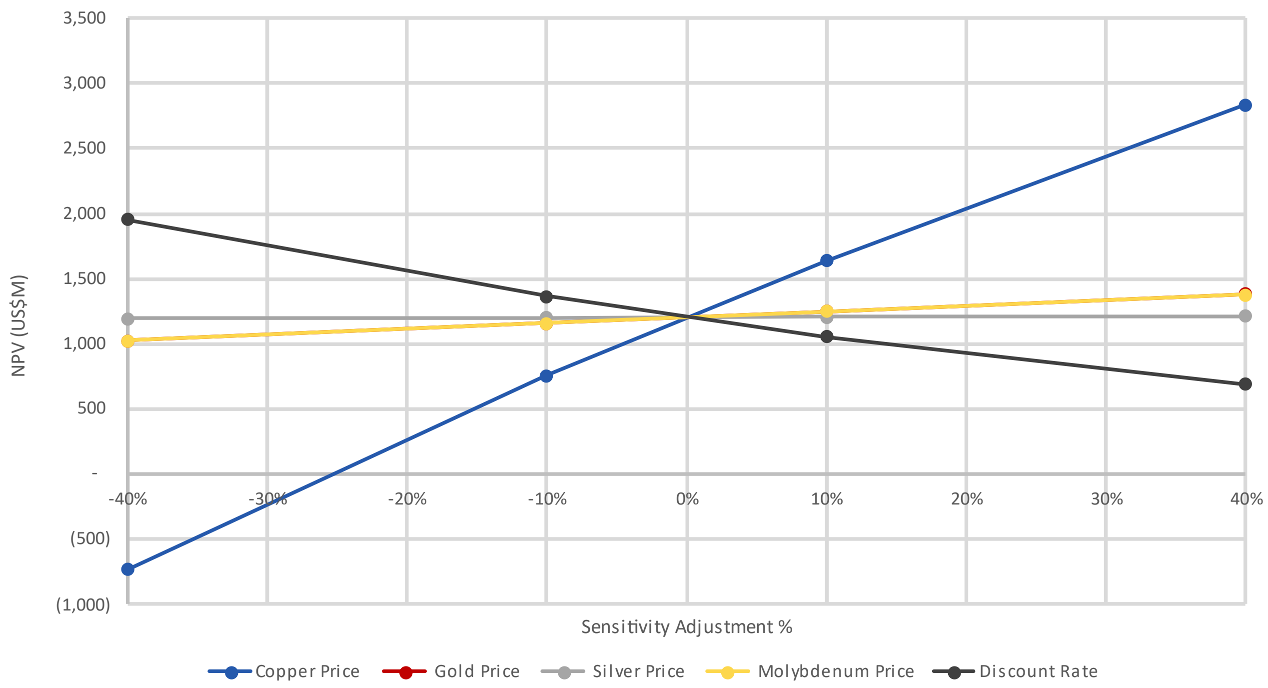


Figure 22.7 : Project NPV Sensitivity Spider Graph – Process Grade or Recovery

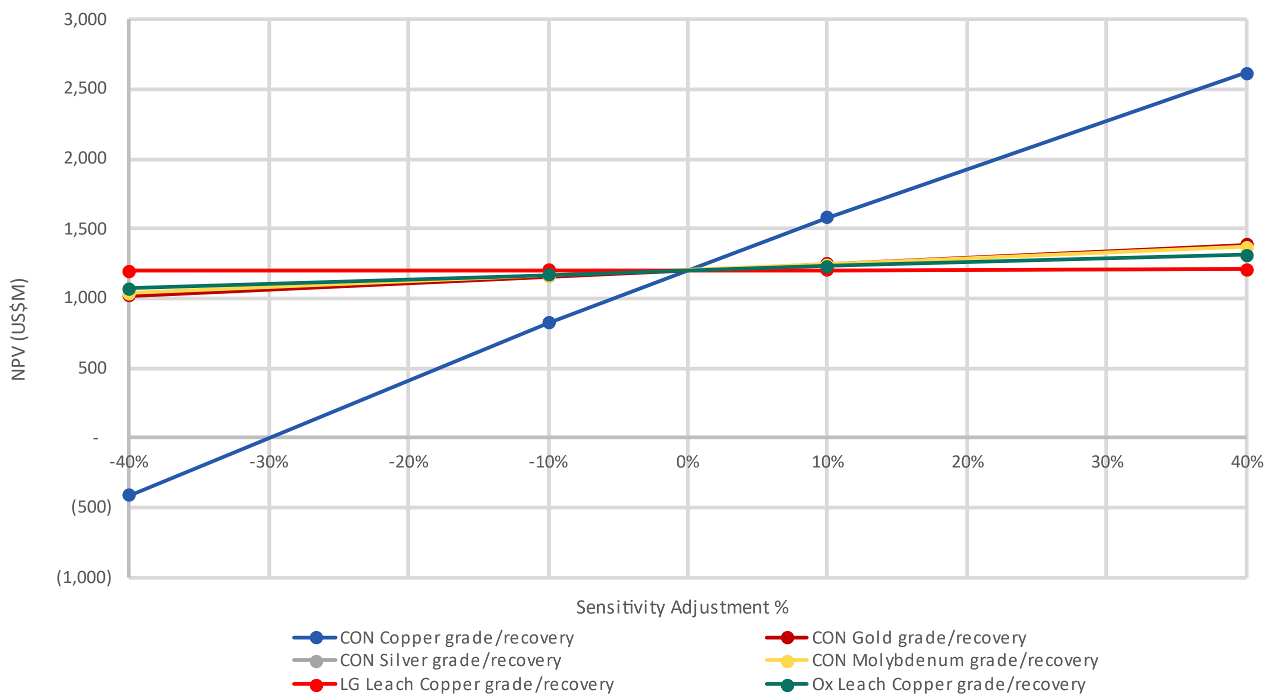


Figure 22.8 : Project NPV Sensitivity Spider Graph – Capital and Operating Costs

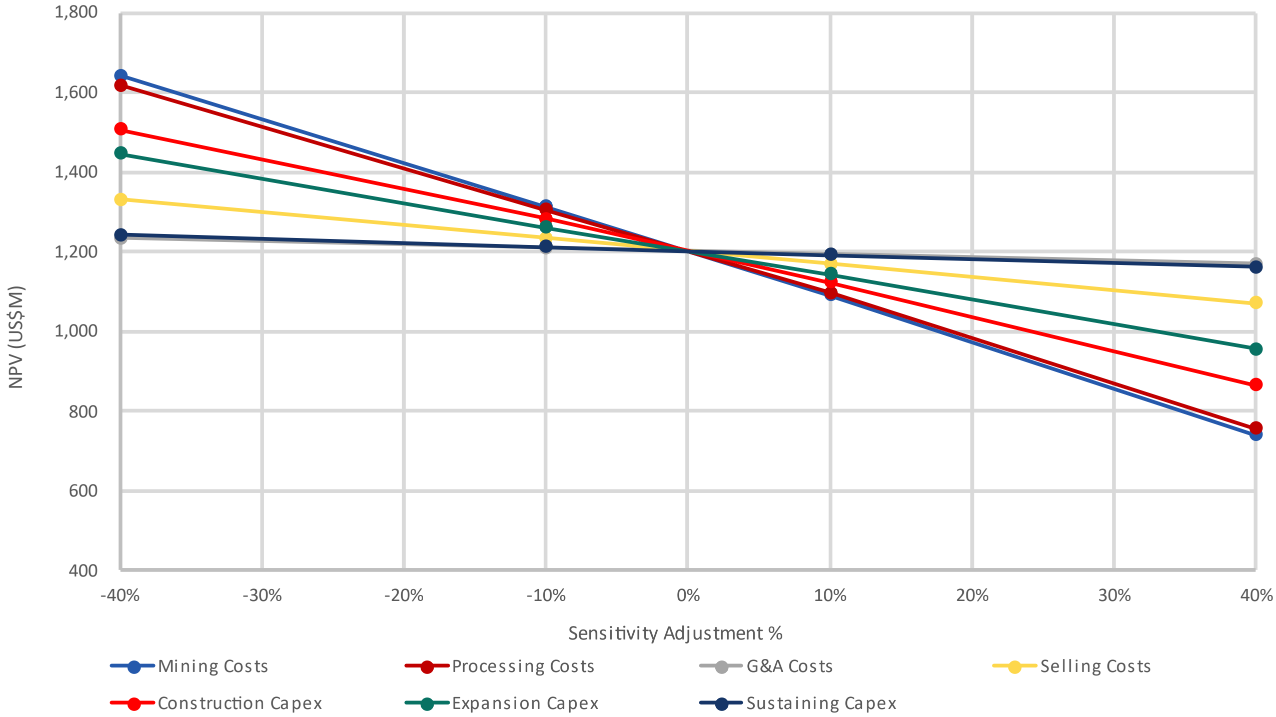


Figure 22.9 : Project IRR Sensitivity Spider Graph – Metal Prices

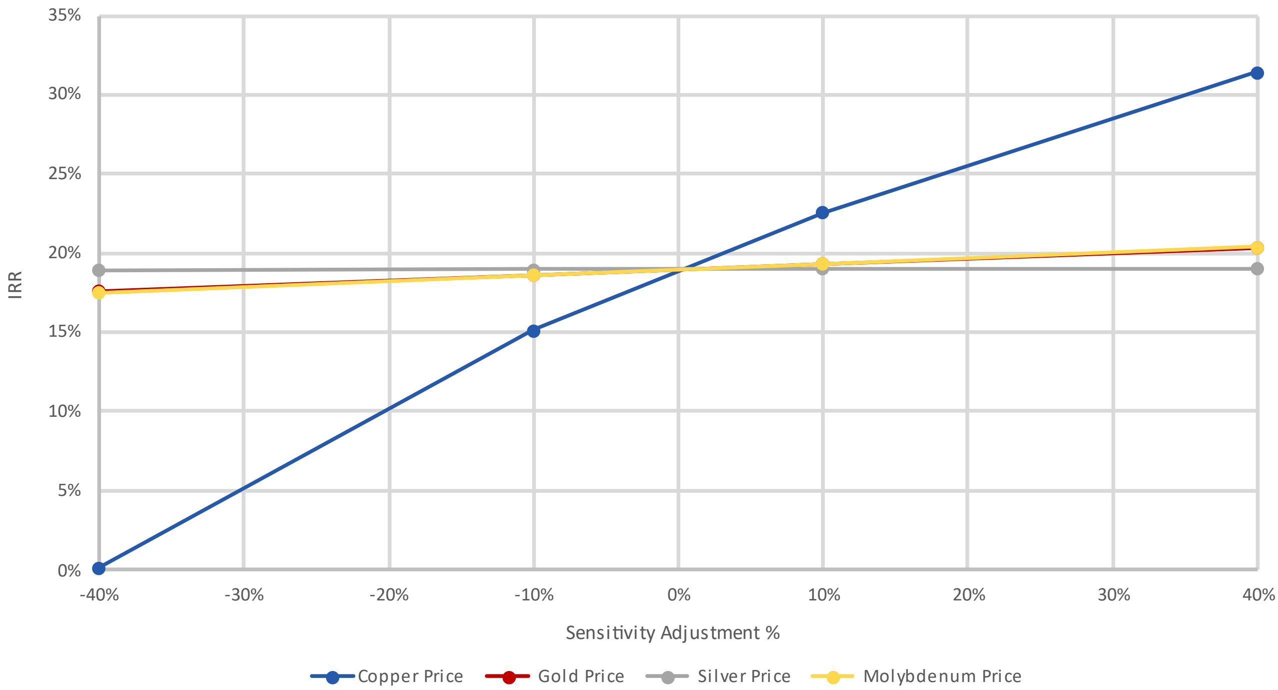


Figure 22.10 : Project IRR Sensitivity Spider Graph – Process Grade or Recovery

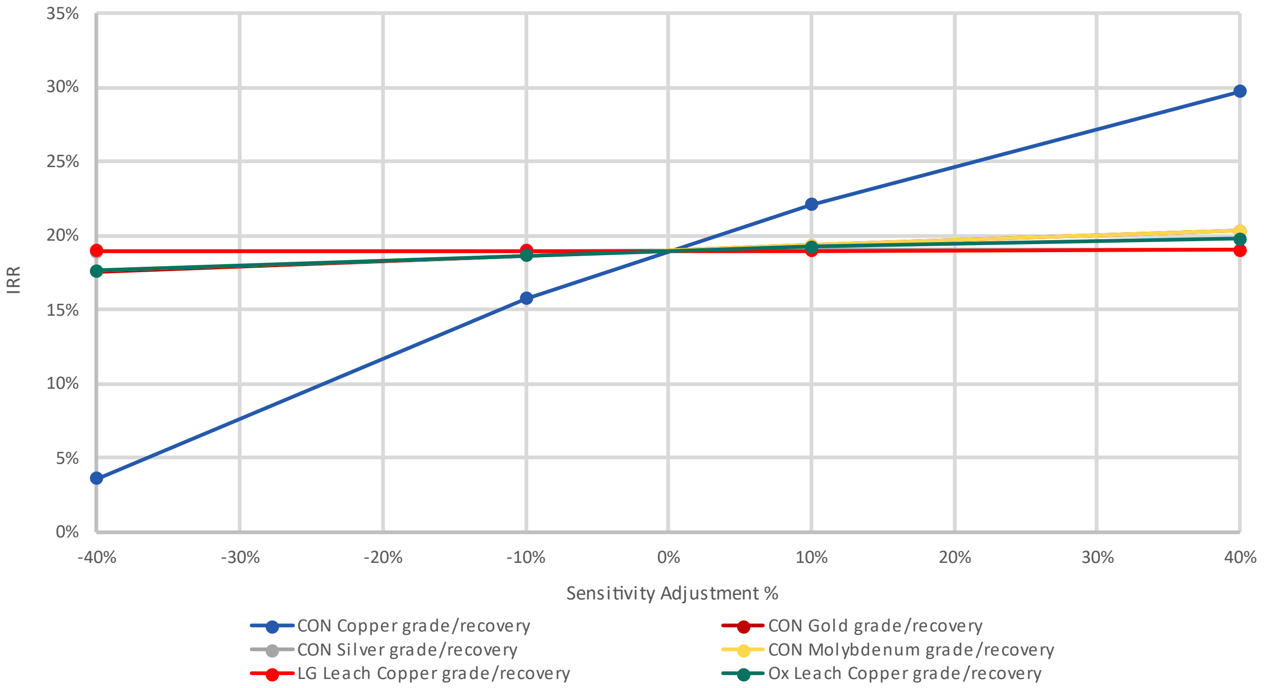
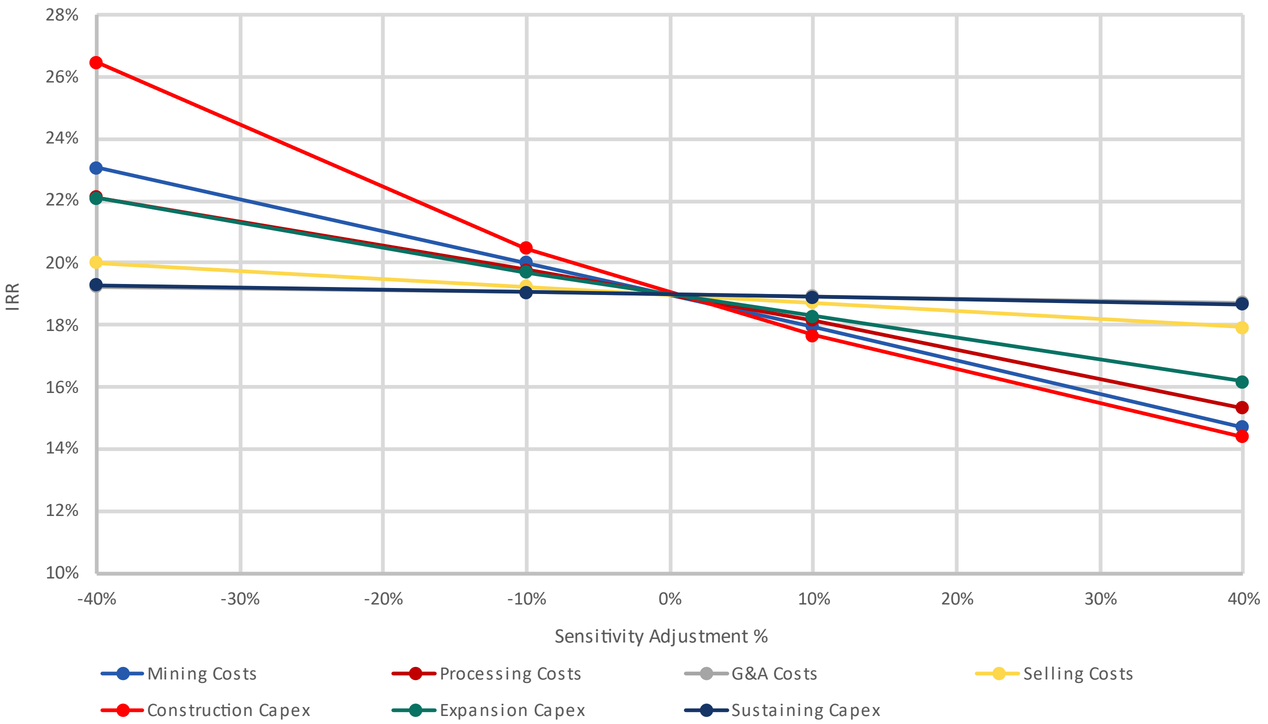
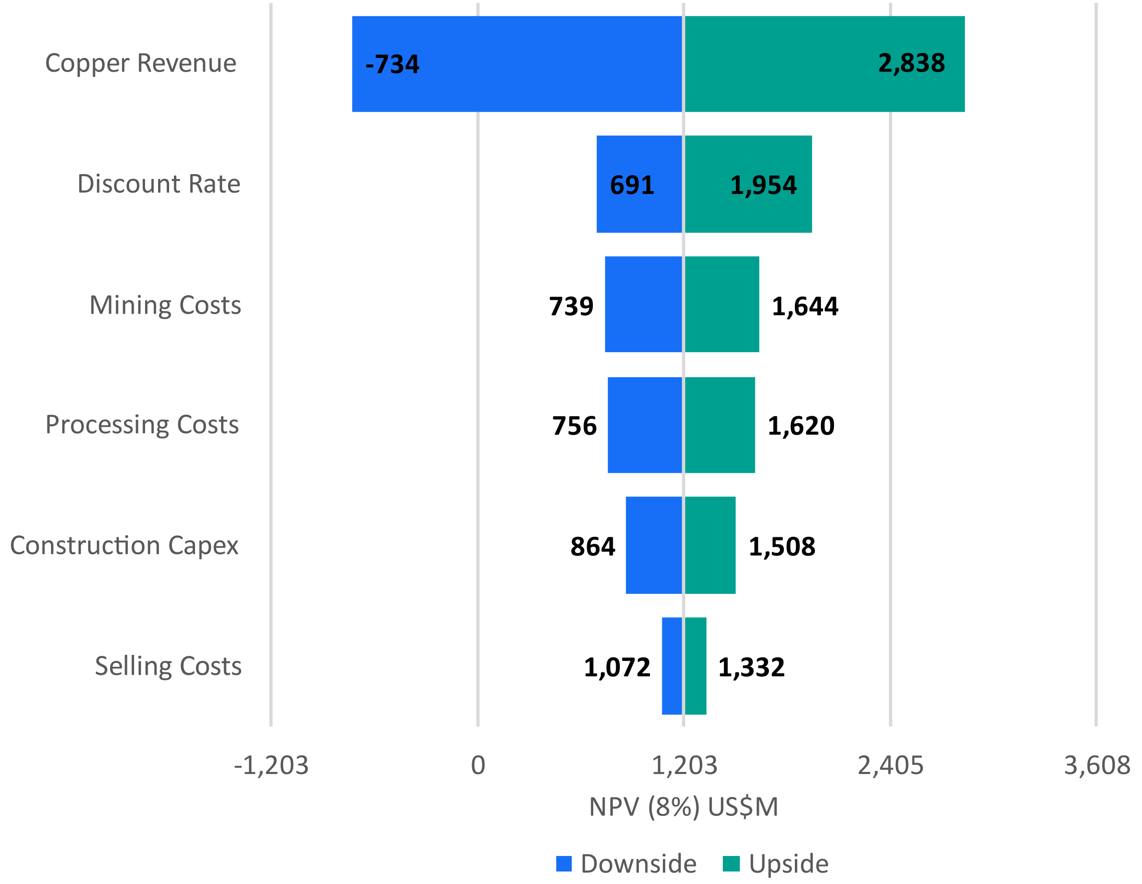


Figure 22.11 : Project IRR Sensitivity Spider Graph – Capital and Operating Costs



The tornado chart in Figure 22.12 shows the Project's post-tax NPV sensitivity to a +/-40% change in the most significant factors identified in the sensitivity analysis, arranged from most to least impactful.

Figure 22.12 : Project NPV8 Tornado Chart



The Project NPV and IRR are both sensitive to changes in copper revenue, one factor of which is copper price. Figure 22.13 shows the Project's post-tax NPV8 and IRR at a range of copper prices.

Figure 22.13 : Project Post-Tax NPV (8%) and IRR as a Function of Copper Price

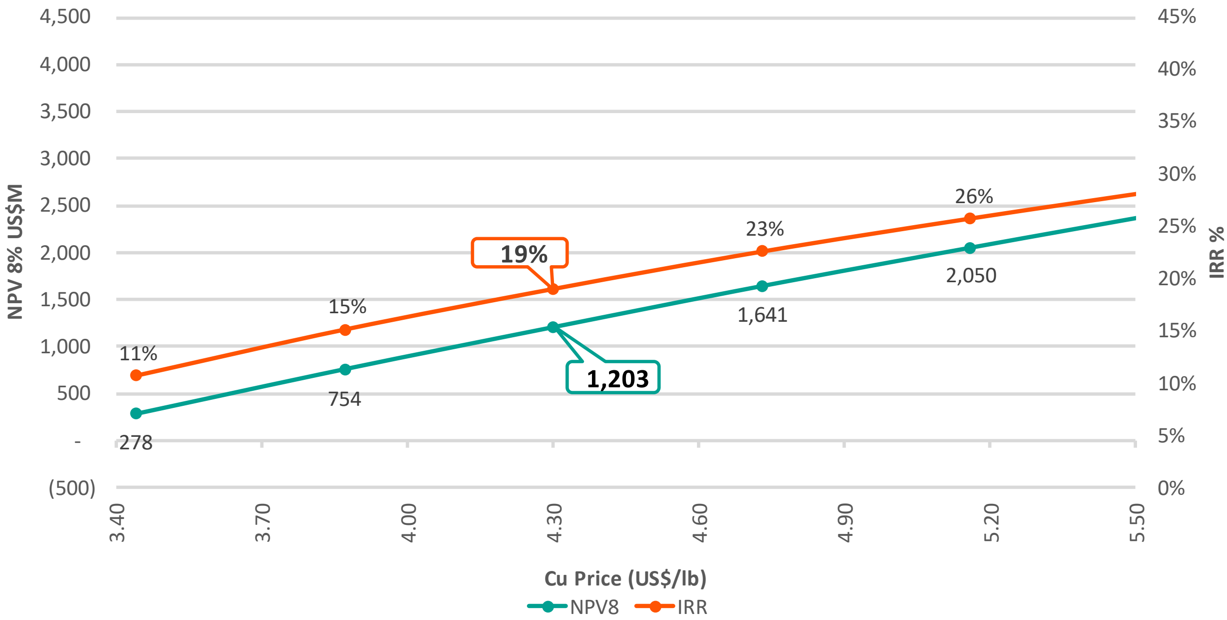


Table 22.14 shows financial metrics at lower and upper copper price sensitivity ranges (approximately ±10%).

Table 22.14 : Copper Price Ranges – Lower-, Base-, and Upper- Case Sensitivity Scenarios

Project Metric		Units	Copper Price		
			Lower (-10%) (US$3.90/lb)	Base (US$4.30/lb)	Upper (+10%) (US$4.70/lb)
Pre-Tax	NPV8%	US$M	1,157	1,712	2,263
	IRR	%	17%	22%	25%
Post-Tax	NPV8%	US$M	786	1,203	1,612
	IRR	%	15%	19%	22%
Annual Average EBITDA		US$M	368	426	484
Annual Average Free Cash Flow		US$M	148	191	233
Payback Period (From First Production)		yr	6.0	4.5	4.0
Post-Tax $NPV_{8\%}$ /Start-up Capital			0.6	0.9	1.3

23 Adjacent Properties

HCH knows of no immediately adjacent properties which might materially affect the interpretation or evaluation of the of the Project.

24 Other Relevant Data and Information

24.1 Project Execution Plan

24.1.1 Execution Summary

Figure 24.1 confirms the proposed project execution approach following completion of this PFS. A series of optimisation studies will be completed prior to proceeding into a feasibility study (FS). A risk management plan will also be developed based on the outcomes of the PFS and will include a strategy for regular risk review updates.

The timeline for completion of the optimisation studies is approximately twelve (12) months and will include initial funding approval and scoping of the following aspects:

- Metallurgical amenability studies - concentrator and leaching.
- Leaching of pyrite concentrate.
- Materials of construction studies.
- Infrastructure geotechnical drilling.
- Large-scale leaching testwork programme.
- Expanded flotation testwork programme.
- Pilot plant and molybdenum flotation testwork.

Completion of the optimisation studies will include transition into the FS and submittal of the EIA for the Project. It is anticipated that further regulatory approvals (including EIA updates) will be progressed in parallel to a twenty-four (24) month duration proposed for the completion of the FS. Planning for execution of the Project will be on the following basis:

- A twenty-four (24) month construction period including hot commissioning and ramp-up
- Detailed design to commence in parallel to the close out of the FS and completion of the class 3 capital estimate to be delivered with the FS.
- Finance investment decision (FID) for the execution of the Project will be confirmed prior to commencement of construction on site. Sufficient design information will be sourced during the FS to ensure that the detailed design can seamlessly commence.

The transition from FS into the detailed design for the execution phase of the Project will thus include the following:

- Geotechnical and site survey design information for the Project area (including interfaces with associated existing infrastructure)
- Route and access surveys with respect to delivery of equipment to site
- Confirmation of contractor laydown areas for all site erection activities for mining, process plant and infrastructure areas (including tailings storage facility)
- Procurement optimisation covering the following aspects:

- Mechanical supply packages – The focus will be on major packages (for both the 'dry' and 'wet' process circuits) as well as items that will need the placement of orders to secure certified drawings for key structures on the critical path
- Electrical supply packages – key packages identified during the FS
- Control and instrumentation – key packages identified during the FS
- Site erection packages - Bulk Earthworks, Concrete, Infrastructure, Liner Supply & Installation, Mechanical, Structural & Piping and E&I.

In addition to this, the enquiry for the mining contract will be issued (on a binding tender basis). The bulk earthworks scope of work will be then rationalised / finalised between the mining contract and the bulk earthworks / concrete contracts.

The transition between FS and Project execution will include further development focus on the Project execution plan (PEP) and the operational readiness plan.

24.1.2 Project Execution Plan (PEP) Deliverables

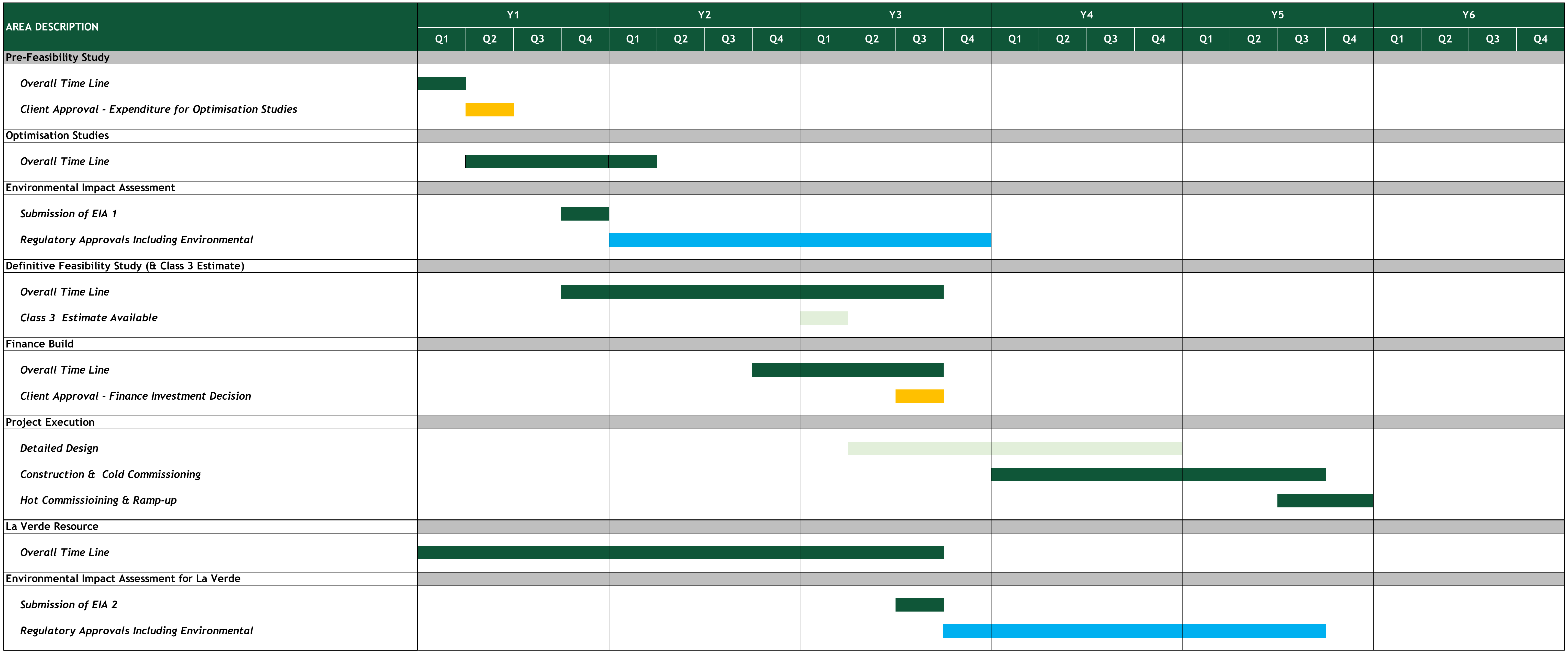
Update of elements of the PEP delivered as part of the FS will include:

- Project Execution Organogram
- Project Execution Schedule
- Work Breakdown Structure
- Procurement Operating Plan
- Risk Management Plan
- Index of Policies & Procedures.

The PEP will be updated during the transition phase and will also include the following additional management plans:

- Procurement & Expediting Plan
- Cost Management Plan
- Responsibility Assignment Matrix (RACI)
- Construction Management Plan
- Commissioning Plan
- Communications Management Plan
- Quality Management Plan
- Operational Readiness Plan
- Design & Engineering Management Plan
- Materials Handling Management Plan

Figure 24.1: High Level Project Execution Plan



24.1.3 Operational Readiness

The objective of operational readiness planning is to prepare for potential operational and maintenance difficulties which may be experienced in the execution/ ramp up phase of the proposed execution project. This way, they can be mitigated (or avoided) through implementation of the operational readiness strategy during the development and implementation stage.

The strategy behind the ORP will take cognisance of the nature of the Project scope:

- Optimisation of mine plan and process plant operation
- Long term and short-term planning for operation of the TSF
- Implementation of infrastructure changes / upgrades.

This work-plan will highlight the actions required to amend / optimise the organization capabilities, the critical milestones to be achieved, who will execute the work, who will be involved as well as who carries accountability for which outcome. The ORP will comprises the following components, tailored to the actual scope and timing of execution of scope for the PFS:

- Technical design review
  - Mining
  - Metallurgical design
    - Metallurgical design basis
    - Metallurgical process envelope
    - Capacity and bottleneck analysis
    - Surge capacity
    - Redundancy
    - Process control.
  - Design standards
    - Technical standards and specifications
    - Legislation applicable to the design
    - Emergency routes and escapes
    - Ergonomics
    - Energy management.
  - Equipment and supplier support
    - Technology life
    - Key equipment failure characteristics
    - Key equipment supplier support
    - Warranty and vendor support
    - Maintenance management strategy (including supplier support)
    - Logistical support
    - Equipment accessibility

        - Equipment lifting strategy.
    - Services and utilities
    - Infrastructure.
- Organisational elements for operating mine
    - Alignment to the business plan and agreement on KPIs for ramp-up
    - Policies, procedures and work instructions
    - Risk management
    - Document control and information archiving
    - Change management.
- Business processes
    - Supply chain and logistics
    - Quality management.
- Human Resources needs and planning
    - Manpower plan and forecast
    - Training strategy.
- Safety, health and environment
    - Environmental Impact Assessment (EIA)
    - Social development compliance
    - Legal compliance
    - Safety management plan
    - Medical systems
    - Disaster recovery plan
    - Waste chemical handling and disposal
    - Baseline ergonomic review.
- Commissioning and ramp-up plan
    - Commissioning schedule and planning
    - Team management and controls
    - Commissioning material
    - Commissioning spares
    - Commissioning procedures.

24.2 Project Security

Security during the construction phase will be the responsibility of Hot Chili. Designated contractors site establishment and lay down area/s if located outside the proposed project security access areas, will be secured by a fence and 24-hour guarding. It is expected that all project personnel will adhere to a common Hot Chili security protocol.

24.3 Logistics

Logistics and supply chain management processes and procedures will be developed as part of the Project in the execution phase. This will include a detailed route survey for delivery of equipment and materials to the Project site.

24.3.1 Site location

The Project is located 29 km southeast of the regional mining centre of Vallenar. Permanent roads to the mine site would have to accommodate large traffic volumes to and from Vallenar each day and heavy vehicles carrying copper concentrate (along the same route) to the Huasco port facility (60 km from the mine site). The Project would also be accessed by the main sealed Pan-American Highway connecting Vallenar to Coquimbo in the south for 21 km. The Project haul road connecting Productora and Cortadera intersects the Pan-American, with a further 8km or 10km to be travelled to Productora or Cortadera respectively.

24.3.2 Logistics costs

Logistics (freight) costs for the Project have been estimated based on a percentage of the supply cost (from a similar reference project).

24.3.3 Insurance

Marine insurance and goods in transit (GIT) insurance for all shipments during the Project have been included in the CAPEX estimate as a percentage of the supply cost. In the execution phase of the Project, full insurance cover will be included as part of owner's costs.

24.4 Mine Closure

Refer to Section 20 for additional discussion on the conceptual mine closure approach.

The Project considers, within the PFS progressive closure approach, using the operational mining equipment fleet where practicable and specialised services where required. The Mine Closure Plan will be developed as a component of the EIA submission process and the development of this plan is expected to consider, amongst others:

24.4.1 ARD

The results of humidity cell tests for waste rock characterisation and tailings product testing will inform the level and manner of encapsulation, capping, lining or base preparation, and reshaping for permeant storage facilities including the tailings storage facility, leach pads and waste rock storage facilities for closure.

24.4.2 Water Treatment Facility

The water treatment facility is included within the PFS early in the Project life to allow for treatment of contact water, seepage, and other contaminated waters during the Project. Ascertaining the ongoing requirements for the water treatment facility will be informed by additional hydrological studies, and observations upon commencement of operations for the demand for the facility.

24.4.3 Surface Water Drainage

The PFS considers the eventual formation of pit lakes where the phreatic surface restores to baseline post mine dewatering activities, and the pits collect surface water during rain events. Hydrological studies will be updated to consider simulations where pits may be backfilled with waste or tailings products. These investigations would be completed prior to the EIA submission.

24.5 Alternative Tailings Storage Facilities

The PFS delivered a fit for purpose tailings storage facility for the scale of the Project. The company is assessing alternative TSF locations that may also satisfy the Project requirements. Alternative TSF locations may enable

- Additional capacity
- Reduced risk within the dam break assessment in relation to proximity to population centres

24.6 Project Geochemical Assessment

Assessment of ARD has a varied treatment within with PFS. Three streams of ARD geochemical assessment have been conducted being:

1. Waste Rock Classification ARD (PFS). This program is discussed in Section 20. 122 samples were collected across Productora and Cortadera and comprise static tests for ARD. Results of this testwork allowed for a surrogate correlation between the ARD results and the broader geochemical (drill sample) database. This surrogate correlation was input into the mine schedule to evaluate the volumes of inert (NAF) material available for construction of the TSF embankments.
2. Waste Rock Classification ARD (EIA). This program is discussed in Section 20. 122 additional samples were taken across Productora, Cortadera and San Antonio and comprise static and humidity cell tests for ARD. Results of this testwork were pending at the time of the PFS and these will be reviewed within the EIA program. Humidity cell tests provide longer time scale ARD results and validation to static tests.
3. Tailings ARD. One sample was available to inform the PFS TSF design, with additional samples taken for statics pending at the time of the PFS completion. Humidity cell tests are planned for tailings samples prior to the EIA submission.

Additional ARD sampling and testwork identified during the PFS includes:

- Waste Rock Classification ARD for the Alice deposit
- Tailings ARD static samples analysis and humidity cell tests

Treatment of ARD risk within the PFS is summarised as:

- Intentionally wet areas, being the dump leach pad, heap leach pad and supernatant pond and drainage paths of the TSF are lined with impermeable HDPE liners to control ARD product seepage risks.

- Dry areas, such as waste rock storage and stockpiles are designed with ARD controls of earthworks and surface water management into contact ponds. Humidity cell tests are expected to inform whether additional controls such as encapsulation or surface preparations are required.

24.7 Cortadera Large Open Pit Option

The Project currently considers a combination of open pit and underground block cave mining at Cortadera. With an increasing copper price in mind, an alternative scenario would be the option to replace the block cave with a single large scale open pit mine.

The company is currently investigating this alternative to assess the economics. Advantages of this scenario would be:

- Removal of capital (from years 5 to 8) required to establish an underground caving operation.
- Access to economic material currently located outside of the current PEA pit shells and block cave presented in Section 16. This particularly impacts high-value material located in the gap between Cuerpo 3 and the block cave, as well as below Cuerpo 2
- Increased selectivity of material in the block cave that would be accessed by a large open pit. This could potentially be 50% of the current caved production feed
- Material increase in production feed inventory and mine life

Two pseudo pit surfaces were generated marking the breakeven limit of mining, whereby the Project's direct costs equal the revenue. These two surfaces illustrate (Figure 24.2) the growth potential at the Costa Fuego Project when a higher copper price is applied, resulting in a larger pit surface volume and greater quantity of production feed.

Table 24.1 Optimisation commodity price deck for single open pit at Cortadera

Item	Unit	2025 PFS	Pit-Only Scenario
Copper	US$/lb	3.50	3.97
Gold	US$/oz	1,700	1,751
Molybdenum	US$/lb	14	14.42
Silver	US$/oz	20	20.6

Figure 24.2 Comparison of 2025 PFS Open Pit and Underground Designs against Preliminary Alternate Option of Single Open-Pit optimisation

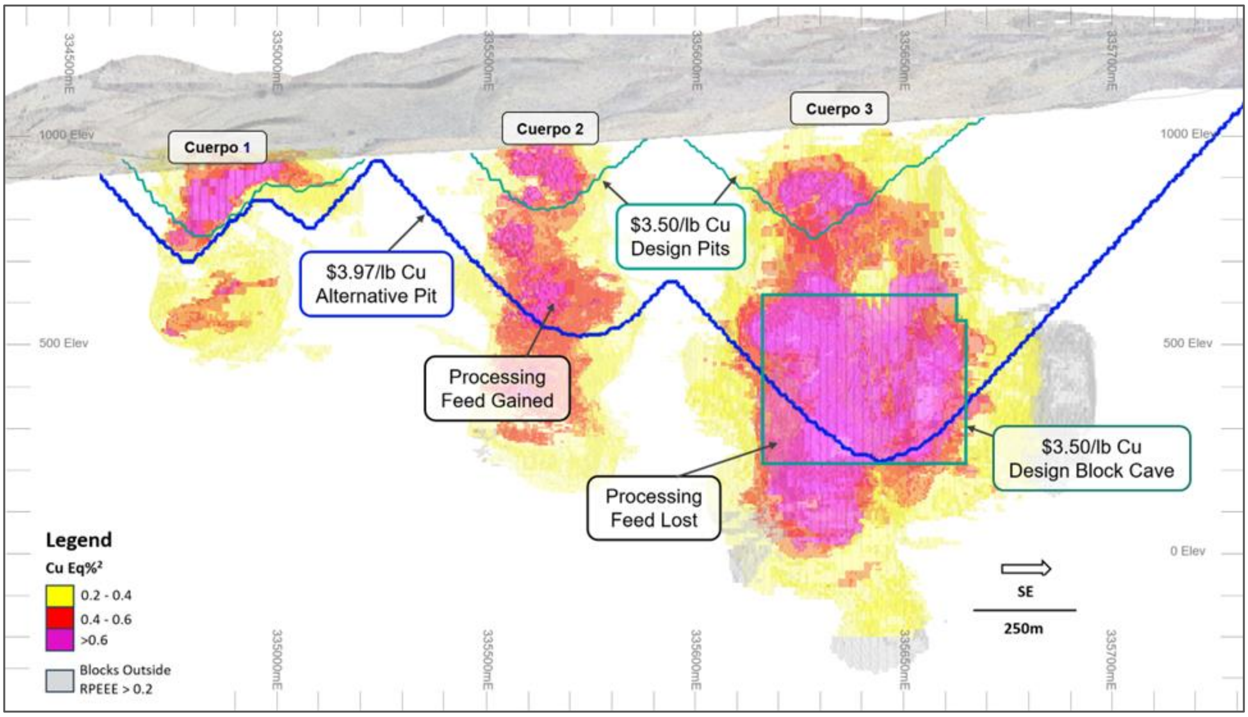
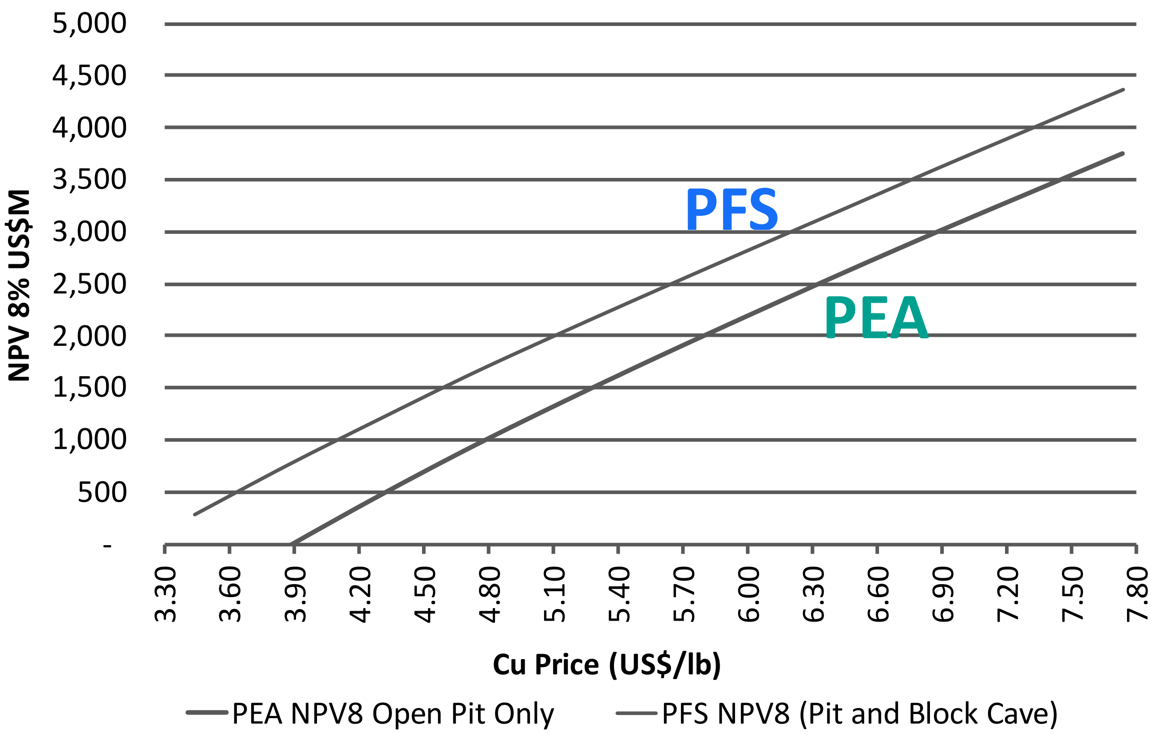
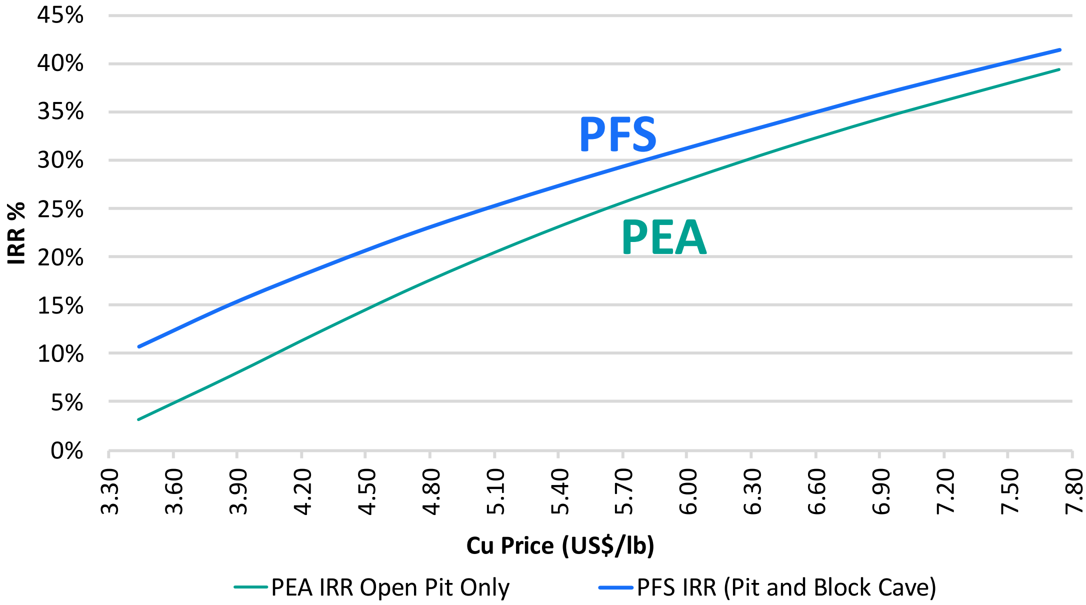


Figure 24.3 Comparison sensitivity charts for NPV (above) and IRR (below) for the PFS design (Pit and Block Cave) and a PEA level design (Big Pit Option)





24.8 Rope Conveyor – Pan American Highway Underpass

The PFS design delivered the rope conveyor passing over the Pan American Highway, with a 10 m clearance, and then lowering through the use of a trench for the rope conveyor foundations to achieve a the required (approx. 10 m) clearance to existing regional high voltage power transmission lines.

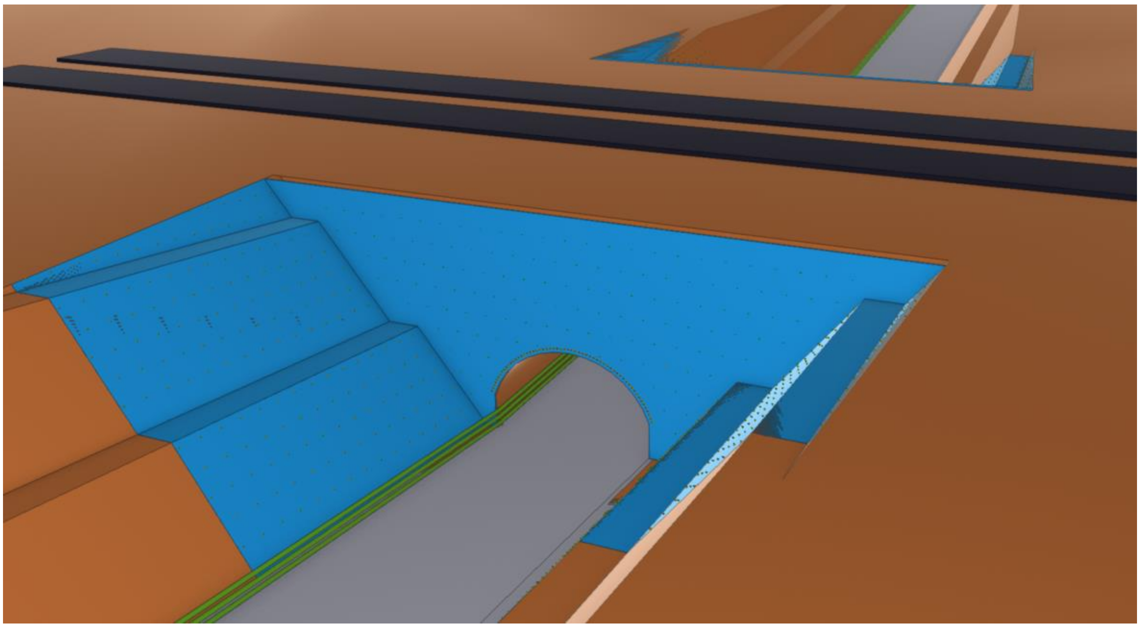
Risk mitigations considered within the design of the overhead passing rope conveyor included input from wind monitoring stations along the proposed pathway and the inclusion of under-belt shielding for sections passing over the highway.

An alternative design, being an underpass construction for the rope conveyor beneath the Pan American highway and existing regional high voltage power transmissions lines, will be reviewed prior to the commencement of an FS.

A trade off study was completed during the PFS to quantify alternative infrastructure corridor features between Productora and Cortadera, which included an underpass for the Pan American highway. The underpass was sized to allow for a mining truck Caterpillar 793 (15.1m x 8.7m tunnel cross section dimension), and it was noted that the capital expenditure would be reduced by approximately 50% if the sizing were reduced to a light vehicle or small service truck (6.5m x 6.5m tunnel cross section dimension). The underpass started approximately 18 m before the intersection with highway and would be 71 m long. The tunnel would have an overburden of 12.25m being 1.4 times the tunnel height of 8.75 m.

The total calculated cost for the underpass within the trade-off study was US$10.7M.

Figure 24.4 : 3D Model of the tunnel design passing beneath the Pan American Highway (HCH, 2025)



24.9 Pyrite Opportunity

24.9.1 Overview

An estimation of contained pyrite (FeS_2) has been completed for the Cortadera and Productora mining areas to allow for a determination of pyrite tonnage available for flotation.

This estimation of contained pyrite uses the block model values for both Fe and S, the estimation of which is further detailed in Section 14.

Flotation of pyrite has potential benefits, including the ability to render the tailings non-acid-forming.

The treatment of recovered pyrite concentrate opens the potential for revenue increase by:

- Allowing the recovery of cobalt which is bound within the pyrite lattice
- Recovering additional copper currently sent to tailings
- Enhancing the recovery of other metals, some of which remained in the copper-molybdenum flotation tailings, including gold, silver and molybdenum.

No benefit estimates have been completed for Alice or San Antonio due to insufficient dataset availability. These deposits will be assumed as not containing any pyrite and represent an opportunity for further studies.

24.9.2 Pyrite Recovery

Pyrite flotation has been included in most of the test programs conducted by Auralia and performed in more than 220 individual flotation tests. Its inclusion has taken two forms, the first is as a simple rougher float, sometimes followed by a scavenger stage, to remove all the pyrite. The second approach is as a rougher/cleaner float (with or without regrinding) to make a high-grade Pyrite concentrate in the vicinity of 50% S. However, combining the cleaner concentrate and cleaner tail in these tests gives effectively an equivalent result to the rougher plus scavenger test recoveries and grades.

The tests performed using development flowsheets or with alternative reagents were removed from the set leaving 87 tests for review. These were analysed by ore deposit and the average results for each deposit (and two ore oxidation variations) are given in Table 24.2.

Table 24.2 : Pyrite Recovery by deposit and for Transition and Oxide Ores

Item	# Tests	Wt%	%Cu	%S	g/t Au	g/t Ag	Co ppm	Mo ppm	Rec Cu	Rec S	Rec Au	Rec Ag	Rec Co	Rec Mo
Cort OP	42	4.01	0.71	30.5	0.73	7.43	277	69	6.01	63.5	20.4	16.5	64.5	18.1
Cort UG	31	3.00	0.49	30.8	0.71	4.15	337	162	3.05	26.9	12.4	9.04	72.6	3.54
Prod	9	4.76	0.75	34.0	0.43	1.74	2015	319	6.01	65.6	18.7	12.16	71.8	4.67
Alice	2	2.56	0.33	16.3	0.10	1.61	676	59	1.69	39.2	8.1	6.0	38.6	3.22
Trans	2	1.98	0.69	16.4	0.89	4.81	245	87	3.47	44.5	10.8	29.9	39.9	7.03
Oxide	1	1.29	0.42	3.6	0.64	0.54	59	78	2.31	39.7	10.9	1.4	7.6	2.88
Overall	Weighted Avg	3.62	0.61	31.1	0.63	4.40	560	140	5.11	53.8	18.0	12.2	68.3	9.93

The results confirm that pyrite concentrate carries, on average, over 5% of the Cu in the plant feed, 18% of the Au, 12% of the Ag and almost 10% of the Mo. Apart from these PFS pay metals, it also carries almost 70% of the Co in the ore. Even if only a portion of these metals from the pyrite were recoverable as products it suggests that pyrite concentrate would be a net generator of value in the project. Confirming such a possibility remains a matter of testwork aimed at developing a process pathway able to produce commercial products.

Pyrite flotation will be achieved with a bank of flotation cells of the same size as specified for Cu/Mo flotation. The scaled-up residence time required is about 30 minutes. Pyrite flotation feed is the Cu/Mo scavenger tailings mixed with a much smaller stream that is high in sulphides, the Cu/Mo cleaner scavenger tailings. The flotation collector Potassium Amyl Xanthate (PAX) is used in small quantities (5 to 10 g/t), no pH adjustment is made and frother is only used if required.

About 80 to 90% of the pyrite in pyrite flotation feed is recovered to flotation concentrate. Although sulphur recovery can be used as an indicator for pyrite recovery, Only 54% of the sulphur in the ore (overall) is recovered to pyrite concentrate as shown in Table 24.2. This low recovery value does not mean that significant pyrite remains in the tailings. Approximately 30% of the sulphur in the feed has, earlier in the flowsheet, been recovered to the Cu/Mo concentrate. A further 5 to 10% of the sulphur is locked in tailings as unrecoverable sulphides. About 65% of the sulphur is recovered to pyrite concentrate for Productora and Cortadera OP ore.

The Cortadera underground case has a much lower recovery of sulphur because that deposit's ore is pervaded with the non-floating mineral anhydrite ($CaSO_4$). Being non-floatable, the anhydrite sulphur reports to tailings under all circumstances. However, sulphur in anhydrite is in the sulphate form (SO_4) and is not environmentally damaging because it does not break down to form acid.

24.9.3 Potential Impacts of Pyrite Disposal on Main TSF

As floated pyrite no longer reports to the tailings dam, tailings solids would be rendered non acid-forming (NAF). The tonnage of tailings to be disposed into the main TSF over the life of the Project would also be reduced. Assuming pyrite flotation is practiced, 15 Mt of pyrite concentrate (at about 31% S grade) is removed from the tailings stream, reducing the gross siliceous tailings quantity to 415 Mt, down from the current 430 Mt considered in this PFS.

24.9.4 Potential Impacts of Pyrite in Leaching

The pyrite concentrate presents opportunities for recovery of additional metals and for the generation of sulphuric acid. There are a number of possible methods for releasing the value from the pyrite. A conventional approach would be to burn the pyrite as fuel in a roaster and produce sulphuric acid, power and steam. The sulphuric acid would be used in the Project and excess sulphuric acid would be sold.

The residue from the roaster (cinder) would contain all the valuable metals, mostly in acid-soluble form, and would be leached with sulphuric acid to recover them. Although a proven pathway, a pyrite smelter is capital intensive and cinder leaching will generate a separate and complex liquor stream making metal recovery difficult. Once the liquor has been recovered from it, the barren leached cinder (dominantly iron oxide) would then need to be pumped to the tailings thickener or TSF.

A novel alternative has been proposed for recovery of values from pyrite concentrate that does not require significant capital expenditure because it integrates with the leaching approaches described in this PFS. The proposal is based on the acid hypersaline leaching process developed and tested by Nova Mineralis, NOVAMINORE® Leaching Technology. Both heap and dump leaching for the PFS use the NOVAMINORE® approach and have achieved encouraging recovery levels of all byproduct metals as described elsewhere. The proposal is to add and mix pyrite concentrate to each of the leach feeds (heap and dump) on the assumption that the pyrite mineral will react and release metals in the NOVAMINORE® environment.

There are already strong indications that pyrite is being broken down in the dump leaching testwork performed on fresh Productora ore, and this provides encouragement that the pyrite concentrate leaching testwork may be successful.

Using available information, it is possible to make conservative recovery estimates of the metals extracted into leach solution from the PFS dump leaching testwork. The test sample was selected to emulate dump leaching as closely as practical and had a top particle size of 300 mm. The coarse size will severely limit the ultimate achievable recovery of all metals, as is expected in a true dump leach with particles as coarse as 1,000 mm. The sample was free of oxidation and all copper and byproduct metals can be assumed to be contained in sulphide minerals. It was concluded that some form of pyrite breakdown was contributing to the recovery of metals into the leach solution as described in Table 24.3. Average pyrite concentrate grade is provided for comparative purposes.

Table 24.3 : Dump Leach Results for Potentially Payable Metals and Pyrite Feed Grade for Comparison

Metal	Cu	Au	Ag	Co	Mo
Assays in Dump Leach Test Feed	0.29%	0.08 g/t	<1 g/t	112 ppm	82 ppm
Actual Dump Leach Test Recoveries to Solution	40%	60%	-	20%	25%
Assays in Pyrite Concentrate	0.61%	0.63 g/t	4.4 g/t	560 ppm	140 ppm

Pyrite concentrate is higher grade in all metals compared to the dump leach feed sample and it is sand sized with a maximum P80 of about 150 µm. Adopting the dump leach recoveries for pyrite concentrate leaching is currently considered conservative, with further testwork required to provide higher-confidence recoveries.

The minerals present in the Dump Leach feed samples were measured by XRD, and the sulphide components are:

Chalcopyrite 0.72% (90.3% of 0.8% Total Cu Minerals)
Chalcocite 0.07% (8.4% of 0.8% Total Cu Minerals)
Pyrite 2.9% of ore

Pyrite represents 79% by mass of the sulphides in the dump leach feed sample and pyrite has a much higher Co content than chalcopyrite. The recovery of 20% of the Co and 60% of the Au (Table 24.3) suggests that significant pyrite breakdown may have occurred.

The Dump Leach was performed in a series of nine IBC cubes representing a 9 m high leach pad. The test was conducted for one year and then stopped. The results have been extrapolated to two years, a more realistic time frame in operation. In addition, the Au recovery has been capped at 40% after one year and 45% after two years, rather than the 60% recovery achieved in the test. These actual and interpreted results are summarised in Table 24.4.

Table 24.4 : Dump Leach Interpreted Recovery Results for Potentially Payable Metals (except Cu)

Element	1-year recovery (actual)	2-year projected recovery
Au	40%	45%
Mo	25%	30%
Co	20%	25%
Ag (none present in feed)	-	-

It has also been recommended by Nova Mineralis that the plant recovery is further discounted by 5% compared to the adopted or measured test result.

These results provide encouragement that the NOVAMINORE® process can improve the recovery of payable metals. Note the following:

- Co recovery of 20% is encouraging as cobalt is held almost exclusively inside the pyrite mineral lattice and can only be released into solution if the pyrite is breaking down
- The Co recovery level could be interpreted to say that 20% of the pyrite in the test feed has been chemically degraded, sufficient for Co release.

- Mo recovery was significant

Testwork to demonstrate pyrite leaching is currently under evaluation at Auralia Metallurgy in Perth.
The first step of this program is to evaluate the activity of the pyrite in the concentrate and determine if it requires modification, such as finer grinding, to make it more available for degradation. Unground pyrite flotation concentrate samples have been sourced and this testing has commenced (Q2 2025).

In the absence of the test results and using available information, it is possible to make conservative recovery estimates of the metals extracted from pyrite concentrate into leach solution in both heap and dump leaching. As both leaching environments are similar, the dump leach recovery results can be applied to any fresh sulphides in both systems. The calculation combines the recoveries by flotation into pyrite concentrate with the estimated sulphides leach recoveries from the IBC dump leach trial. A 95% efficiency is incorporated to allow for scaling up from test to operations. The estimates are contained in Table 24.5.

Table 24.5 : Conservative Recovery of Metals from Pyrite Concentrate into Solution by Heap and Dump Leaching

Item	Rec Cu	Rec Au	Rec Co	Rec Mo
Pyrite Flotation. Metal Recovery (%) from Plant Feed to Concentrate	5	18	68	10
Adopted Dump/Heap leach Recoveries (%) Capped results from Dump Leach IBC test	39	40	20	25
Combined Recovery (%) from Metal in Ore Into Pyrite Concentrate then into Dump Leach Solutions	2	8	16	3

An increase in overall copper recovery of 2%, compared to the current aggregate copper recovery across the Project, has a significant potential revenue benefit. The recovery of this copper from the leach liquor would not require any additional capital expenditure above the estimated PFS levels for the SX and EW stages.

The recoveries of Au, Co and Mo are also promising but none of these elements are recoverable from the pregnant Cu leach solution using the PFS flowsheet. In addition, the PFS flowsheet makes no provision for recovering each element into products and ultimately preparing those products for sale. The ongoing testwork will feed into flowsheet development and cost estimation to evaluate the economic potential for recovery of Au, Ag, Mo, and Co.

24.9.5 By-products Recovery from Heap and Dump Leaching

During the metallurgical testwork program executed during 2024 using the NOVAMINORE® technology, in both heap and dump leaching processes the extraction into solution of the potentially payable byproducts (Au, Ag, Co, and Mo) was confirmed.

The summary results for all feed materials, after allowing for scale-up losses, are presented in Table 24.6. Note that Cu extraction from these leach feed materials is already accounted for in the PFS and is not presented here.

Table 24.6 : Extraction of by-products from Oxide and Dump leaching testwork – 5% Losses Applied

Type of Ore/Process	Au (%)	Ag (%)	Mo (%)	Co (%)
Productora Heap (Oxides)	48	86	57	76
Cortadera Heap (Oxides)	38	67	43	48
Productora Dump (Sulphides) 1 year	38[1]	-	24	19
Productora Dump (Sulphides) 2 years	43[1]	-	29	24

[1] Regardless of the result achieved in the test (60% Au Recovery), it is recommended that the adopted gold extractions do not exceed 40% for the first year and 45% for the entire 2-year period, and then both have been discounted by 5%, as recommended by Nova Mineralis.

Getting the byproduct metals into solution is the first step of the process. Recovering the metals from those solutions may be challenging when the solution strengths of each are relatively low. Three different concepts that allow for the recovery of byproducts from process solutions have been identified for further investigation:

- Direct electrowinning (EMEW) cells: An electrolytic process that recovers metals, such as gold and silver, directly from PLS solutions, producing a dual gold-plus-silver plate (doré). This mixture must then be melted, and doré anodes can be prepared for further refining or direct sale to the market. It is estimated that a doré metal recovery rate of over 90% would be possible.

- Ion exchange (IEX) resins: A technology that captures ions of interest using solid resins. Once charged, these resins must be discharged to recover the metals of interest. This technology is frequently used in uranium recovery, water purification, and lithium solution purification, always with yields exceeding 90% in the capture levels of the metals of interest, which can become an attractive alternative for the recovery of molybdenum and cobalt.

- Solvent extraction (SX) reagents: Similar to the process used for copper, this technology allows for the extraction of other ions present in solution using specific reagents for each ion or metal of interest. However, this alternative requires a slightly higher investment than resins, comparable to that required for copper solvent extraction steps.

It should be noted that the overall recovery of byproducts would not be greatly affected by which of these technologies is chosen. The process solutions, after being treated in the plant that allows the recovery of one or more byproducts, returns to the leaching process and is not lost. This implies that the recovery of these byproducts depends on the continuity of the leaching process and any losses of impregnating moisture at the end of the leaching process. Note that there will also be misreporting losses where other metals are recovered as minor contaminants in a target metal (for example, Co recovered in Cu metal).

To assess the potential volumes of byproduct metals generated, a value of 95% recovery to metal product from process solutions is assumed and, by modifications to the mining schedule, the metals productions estimated in Table 24.7 may be achievable. Note that there was no Ag in the Dump Leach feed and this has resulted in no associated production estimate.

Table 24.7 : Indicative Recovery of by-product metals from Pyrite Concentrates via Oxide and Dump leaching

	Cu	Au	Ag	Co	Mo
Recovery from Plant Feed (%)	2	8	4.2	12	2.1
Indicative LOM Recovered Metal	29 kt	93 koz	0 koz	4,700 t	600 t

An economic evaluation will be carried out on the leaching of metals from pyrite concentrate and recovering those metals from solution into saleable products.

The following points describe the methodology to verify methods for recovery of saleable by-product metals released by the PLS treatment technologies for the Project:

- Obtain solutions from column or IBC leaching trials with representative byproduct metal concentrations
- Initiate laboratory-scale testing, first with each metal recovery technology provider
- Increase the testwork scale by providing more liquor from larger scale leaching tests. This stage may also be possible using artificially generated liquors
- Review and analyse technical results to advance the existing methods, pursue new activities or discard alternatives
- Evaluate business cases, quantifying the costs and benefits of the most attractive technical alternative
- Initiate larger-scale (pilot) testing with the selected technology provider, providing a basis for inclusion in the flowsheet and preparing cost estimates.

24.9.6 Schedule of Pyrite Concentrate Production and Potential Revenue

Using the combination of flotation recoveries to pyrite concentrate, leaching recovery to leachate and refining recovery to metal product, a schedule of estimated metal recovered and revenues from pyrite concentrate alone has been derived from the PFS schedule (Figure 24.5). Note that the analysis excludes minor pyrite contributions from Alice and San Antonio.

Revenue would be generated at the cost of additional capital and operating expenses that can only be estimated after process selection and design.

Figure 24.5 : Pyrite Concentrate and Contained Metals produced by the PFS schedule, based on Flotation Testwork

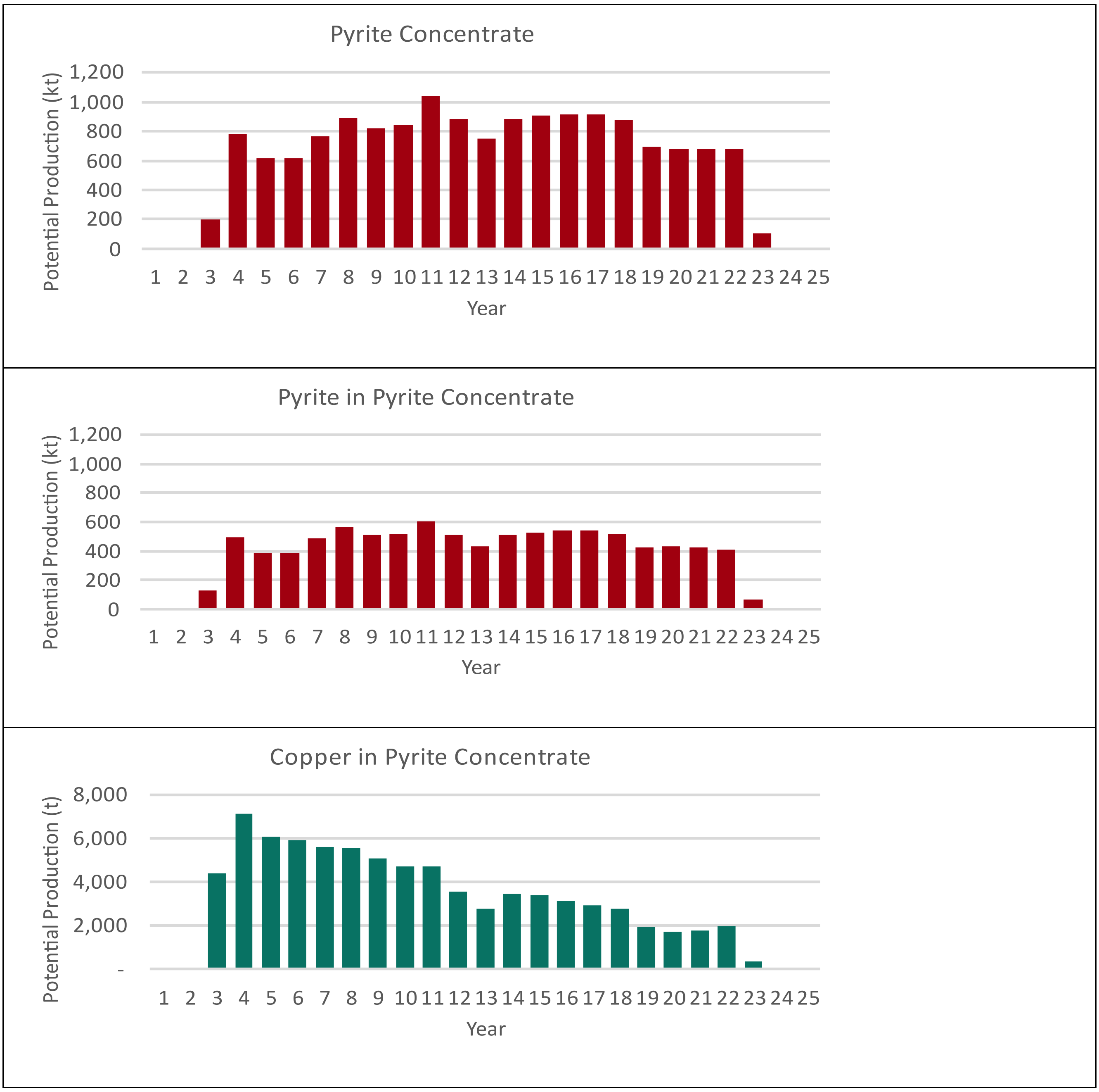


Figure 24.5 : Pyrite Concentrate and Contained Metals produced by the PFS schedule, based on Flotation Testwork

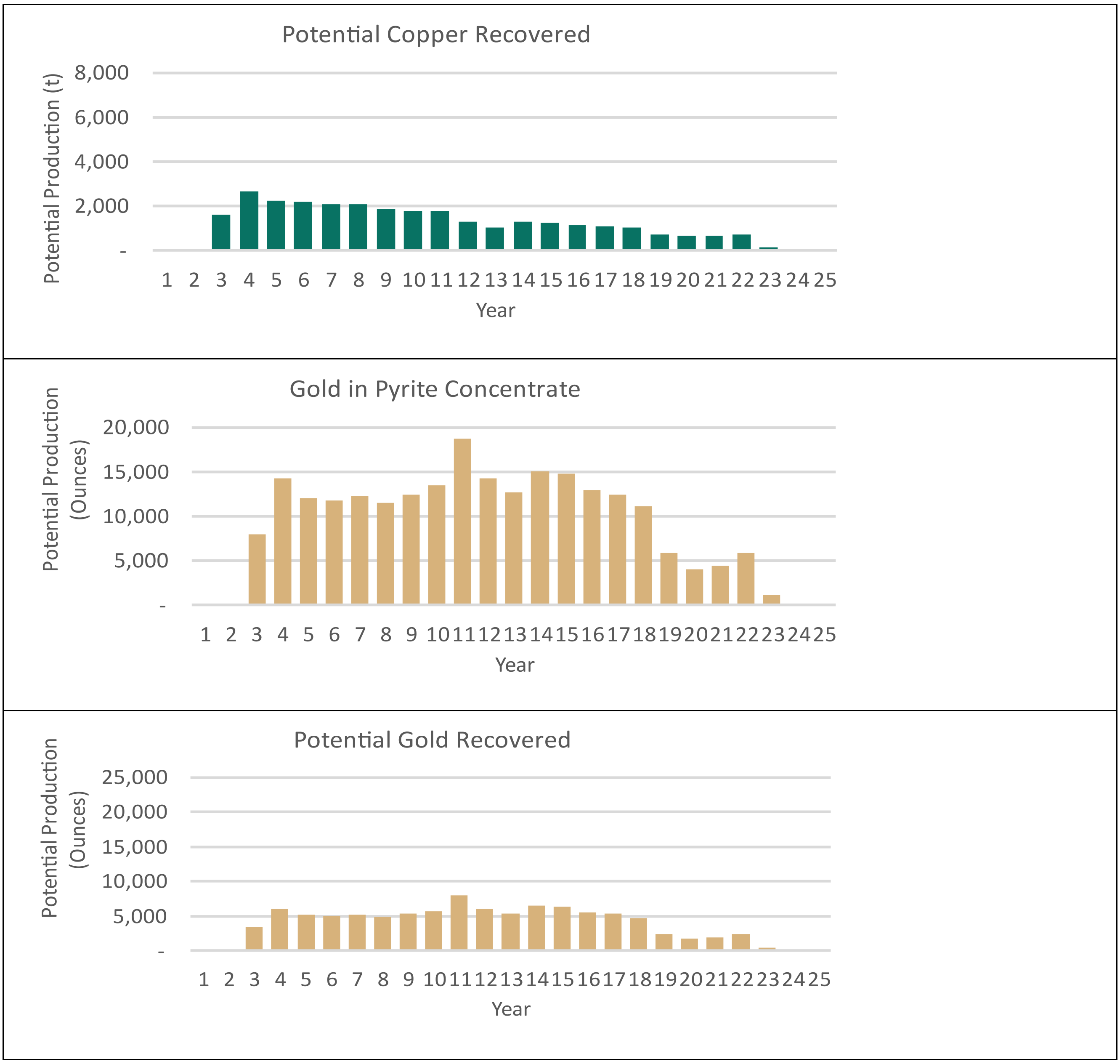
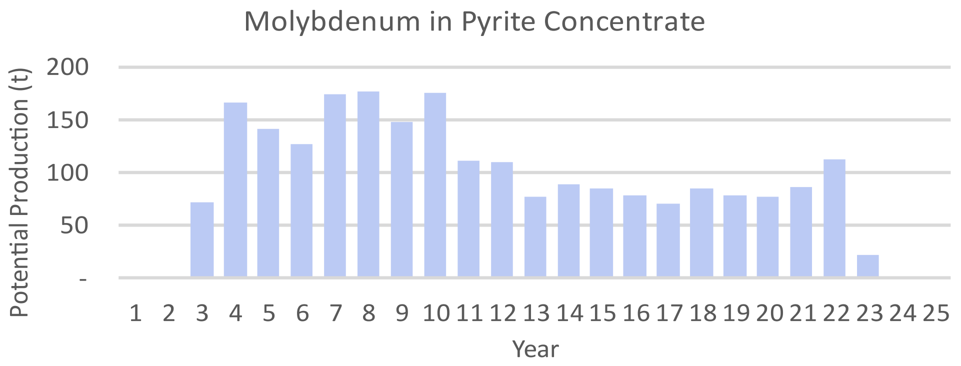
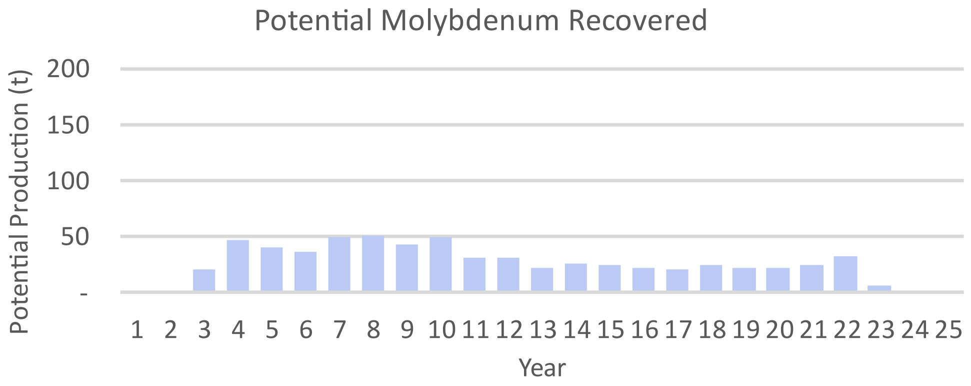


Figure 24.5 : Pyrite Concentrate and Contained Metals produced by the PFS schedule, based on Flotation Testwork





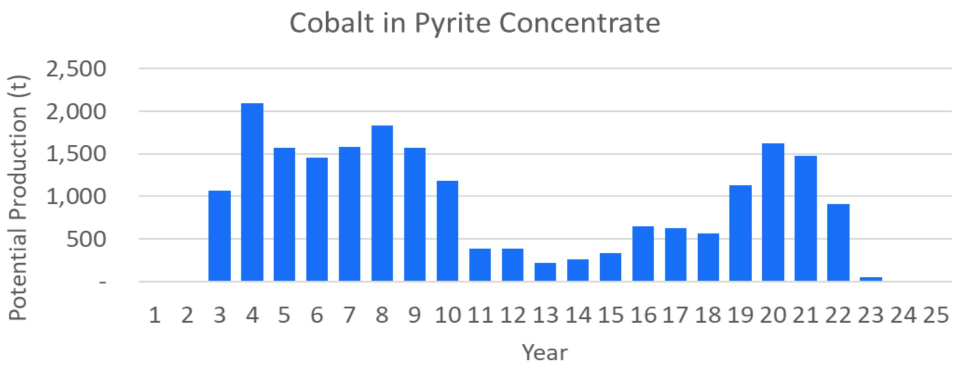
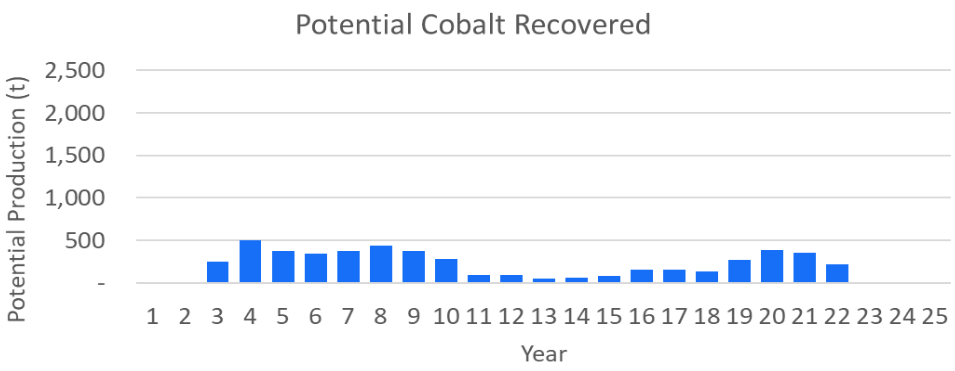


Figure 24.5 : Pyrite Concentrate and Contained Metals produced by the PFS schedule, based on Flotation Testwork



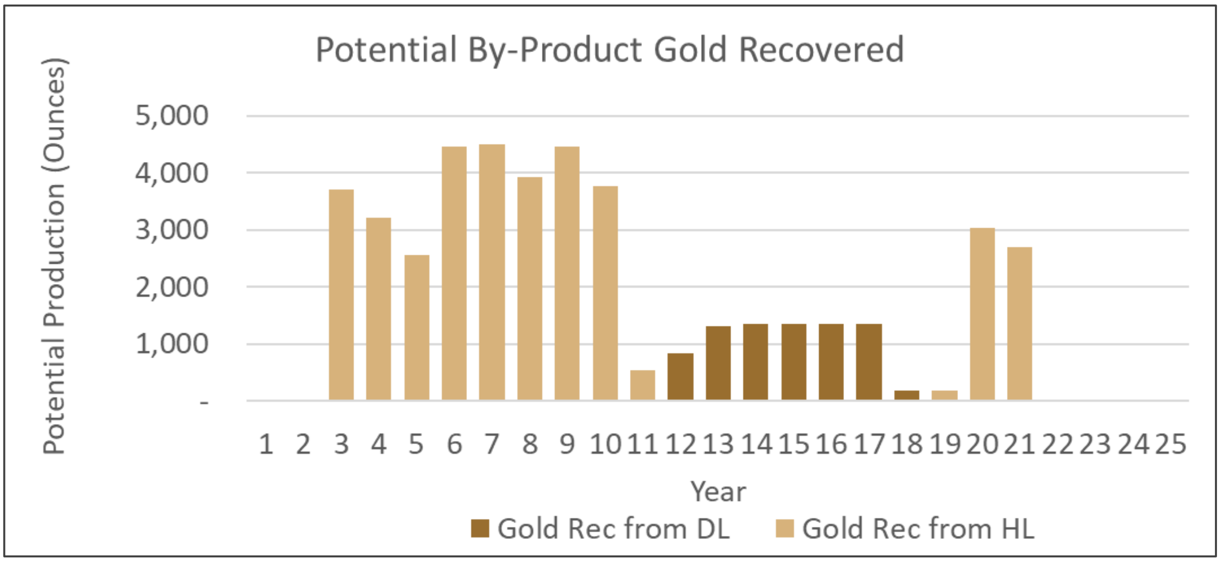
24.10 By-Product from Oxide and Dump Leach

The dump leach testwork that underpins the above pyrite concentrate analysis, and the heap leach testwork was performed on examples of the normal feeds to those processes as intended in the PFS. However, the PFS only assumes revenue from Cu arising from dump leach and heap leach liquors. The liquors from those tests contain potentially recoverable byproduct metals which can also contribute to project revenue.

This section estimates byproduct metal volumes arising from the oxide ore in heap leach and the low-grade sulphide ore in dump leach.

The heap leach testing demonstrated high Au recoveries from oxide ore. The reason Au is recovered in a heap leach without the presence of cyanide is related to the hypersaline leaching environment, but the mechanism is not fully understood. Dump leach testing was not able to recover as much Au. In addition, the Au grade of dump leach feed was lower than the gold grade in the oxide ores. The estimated volumes of Au recovered from each leaching facility are shown in Figure 24.6.

Figure 24.6: Schedule of potential recovered Au from Dump Leach and Heap Leach

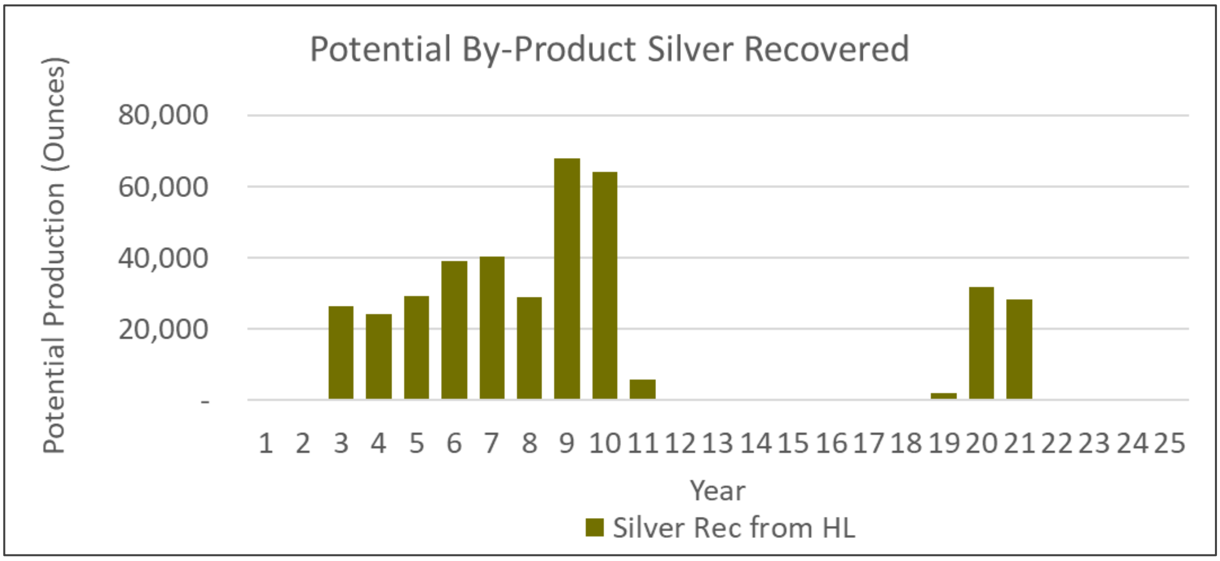


The LOM gold recovered from the PFS feeds delivered to both leach facilities is estimated at 43 Koz (1.3 t).

In the heap leach testwork the Ag recoveries were very high but, as there was <1 g/t Ag in the dump leach test sample the reported Ag recovery for dump leaching was 0%. This likely has a false effect of showing that for a period there is no recovery of Ag from dump leaching in the schedule, but this would only occur if all dump leach feed was <1 g/t Ag.

The estimated volume of Ag recovered from the heap leaching facility is shown in Figure 24.7

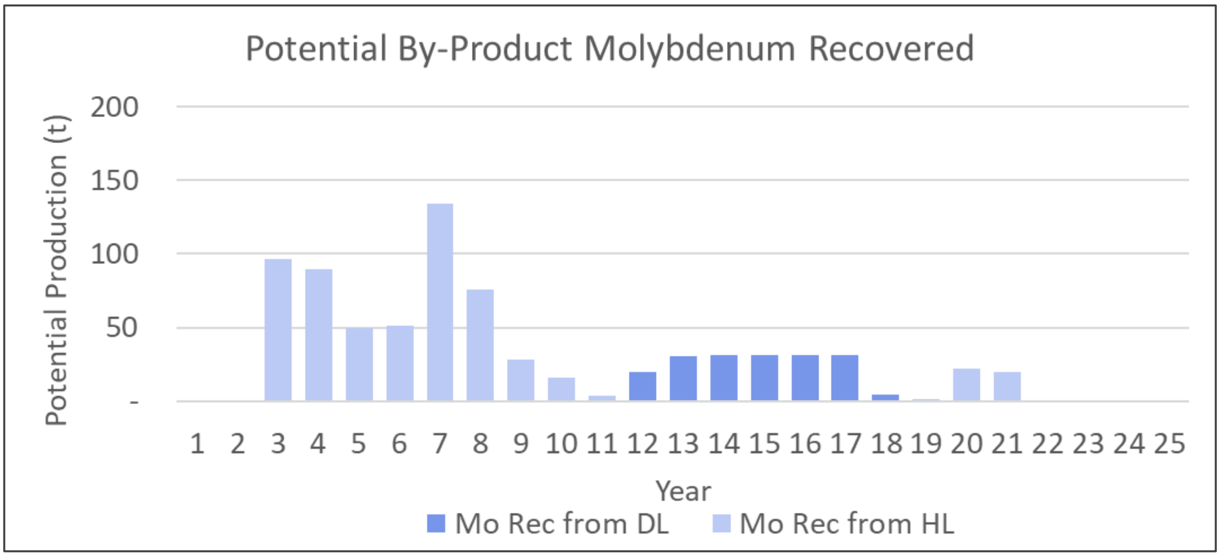
Figure 24.7: Schedule of potential recovered Ag from Dump Leach and Heap Leach



The LOM silver recovered from the PFS feed delivered to the heap leach facility is estimated at 370 Koz (11 t).

In the heap leach testwork the Mo recoveries were moderate and for the dump leach the recovery was much lower. The estimated volumes of Mo recovered from the leaching facilities are shown in Figure 24.8.

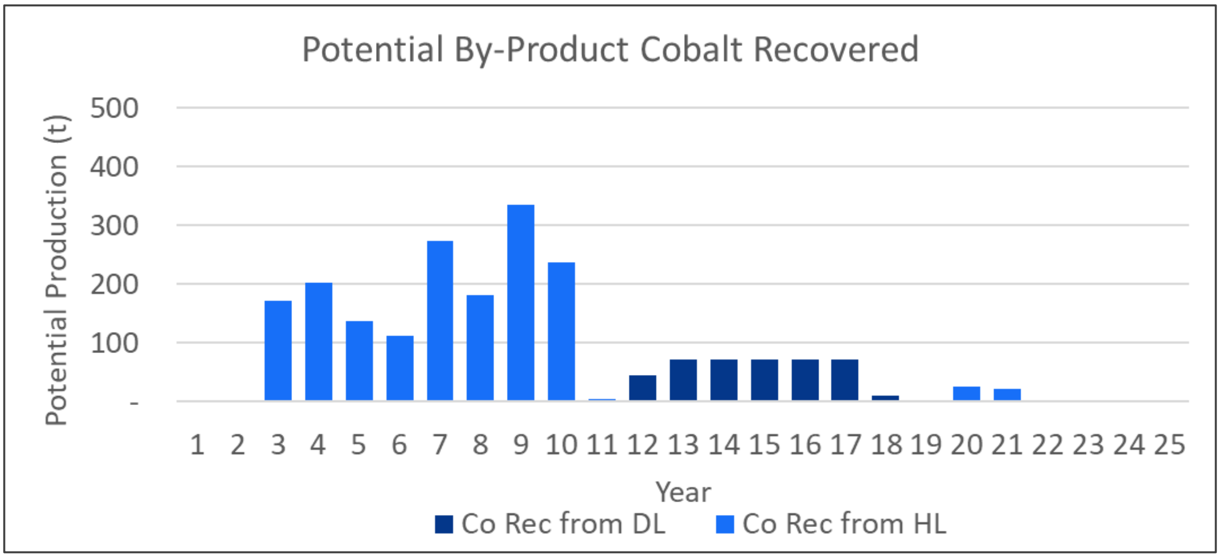
Figure 24.8: Schedule of potential recovered Mo from Dump Leach and Heap Leach



The LOM Mo recovered from the PFS feeds delivered to the heap leach facilities is estimated at 730 t.

In the heap leach testwork the Co recoveries were moderate and for the dump leach testing the recovery was much lower. The estimated volumes of Co recovered from the leaching facilities are shown in Figure 24.9.

Figure 24.9: Schedule of potential recovered Co from Dump Leach and Heap Leach



The LOM Co recovered from the PFS feeds delivered to the heap leach facilities is estimated at 2,000 t.

24.11 Improved Leaching Potential because of Pyrite Degradation

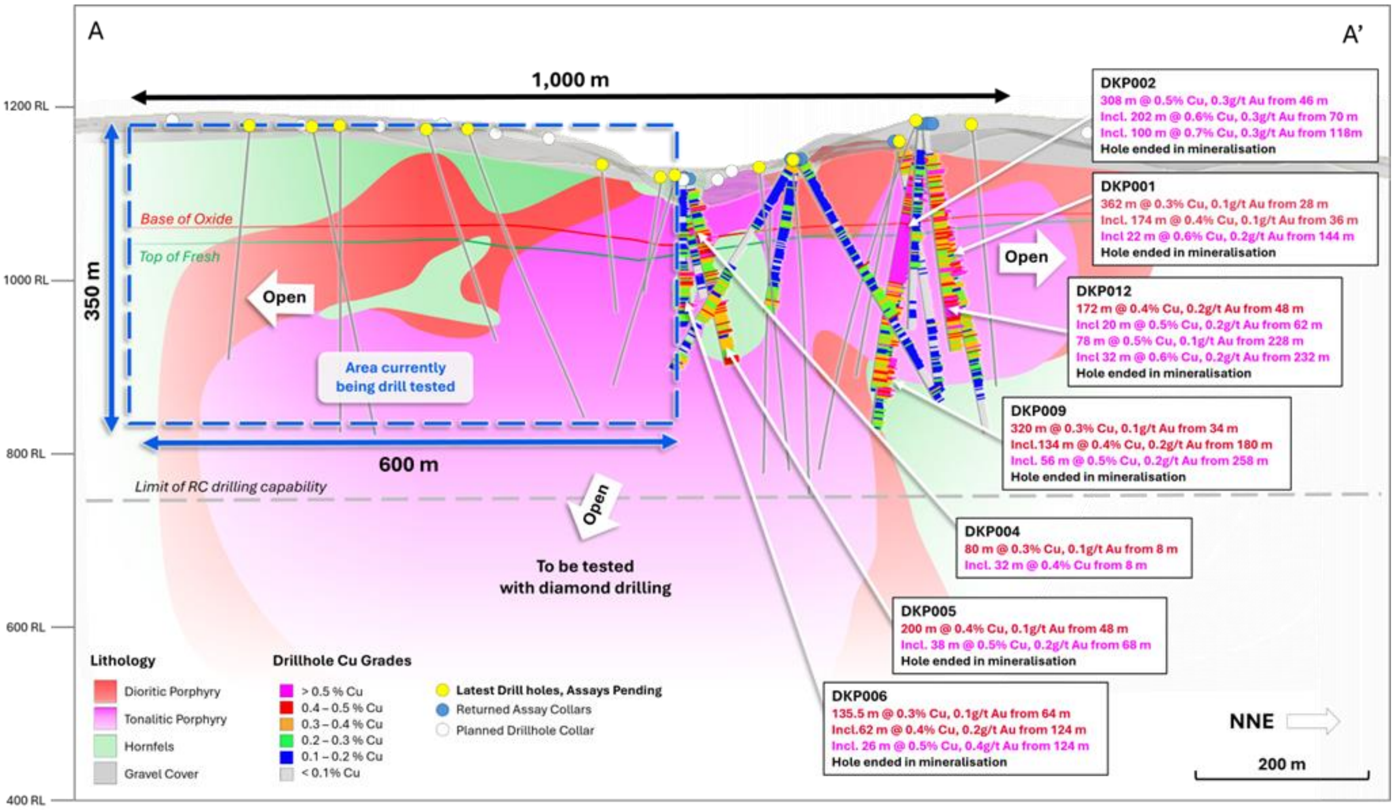
Due to the unusual high chloride environment, it is currently unclear if the breakdown of pyrite will generate sulphuric acid, or other assisting species such as iron sulphate, during leaching. Amenability testwork is underway to establish the chemistry of degradation of pyrite, and the associated potential to release payable metals. In the event acid is being produced then it is estimated that conversion of less than 20% of the sulphur in the pyrite concentrate to acid will make the leaching circuit self-sufficient with respect to fresh acid demand.

24.12 La Verde Exploration

Hot Chili announced the discovery of a significant copper-gold mineralised porphyry system at La Verde in December 2024. The La Verde discovery is located 35 km, or 50km by road from the planned Productora processing facility.

Wide, shallow mineralisation at La Verde currently extends over a 1000 m x 550 m footprint and remains open in all directions. Gravel cover masks a potentially much larger system with a number of geophysical targets to be tested. Depth potential also remains with near surface mineralisation extending to more than 300 m below surface (Figure 24.10).

Figure 24.10 : La Verde drilling showing opportunity along strike and at depth (HCH, 2025)



La Verde shares key characteristics with the Company's nearby Cortadera Resource, including a consistent geophysical signature (Figure 24.11) Four-dimensional modelling, which considers timing relationships between the host rocks, pre-, syn- and post-mineralisation intrusions has commenced.

Figure 24.11 Comparison between geophysical signatures at La Verde (left) and Cortadera (right) (HCH, 2025)

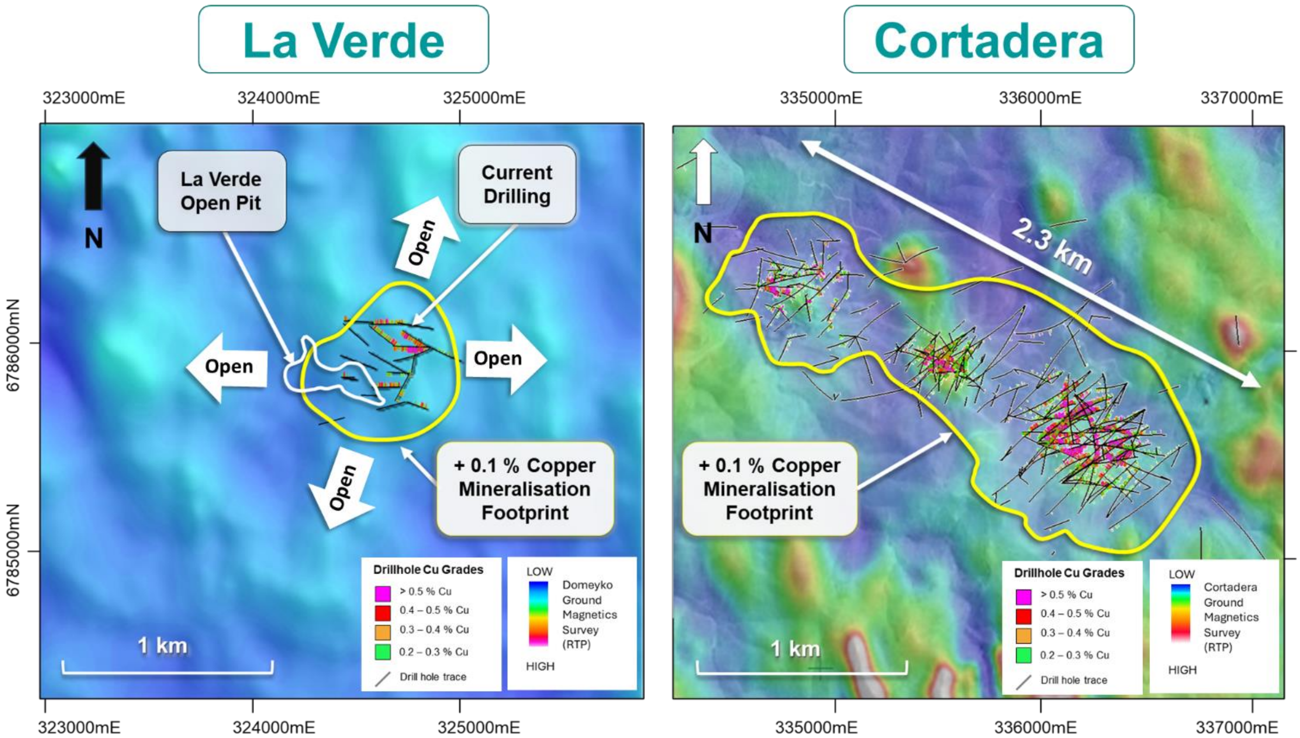
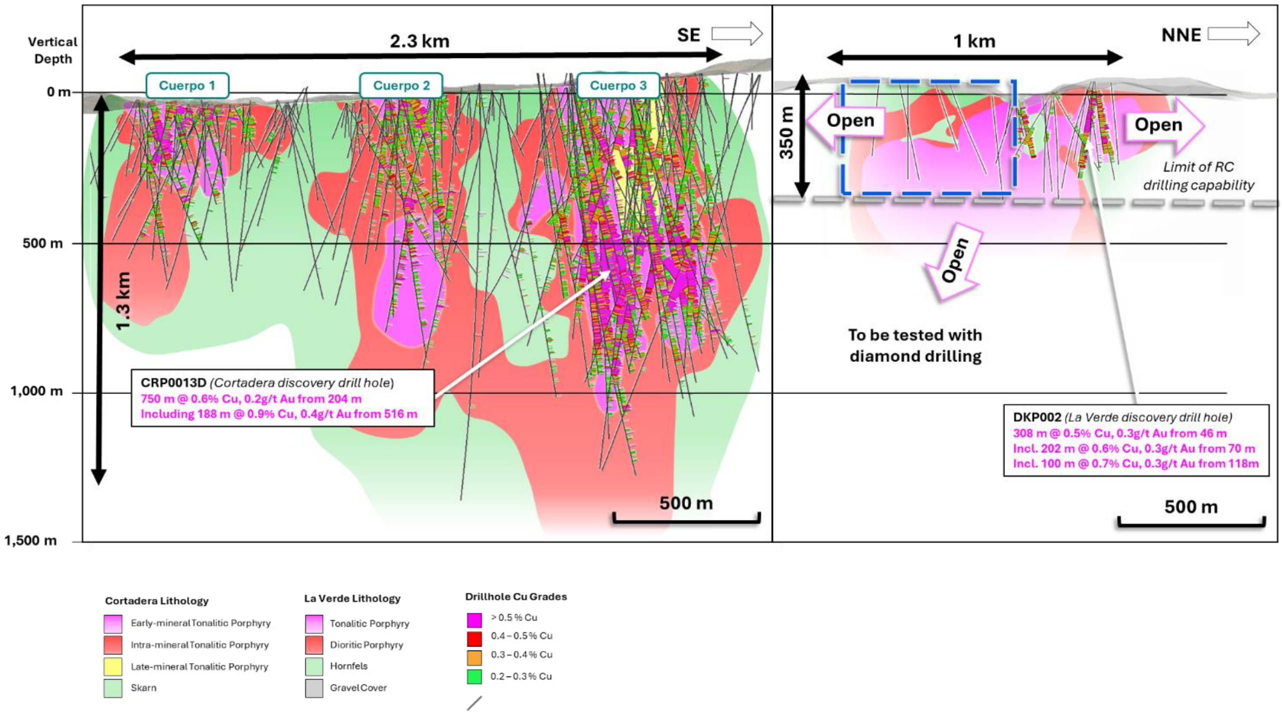


Figure 24.12 Long Section comparison between Cortadera (left) and La Verde (right) (HCH, 2025)



24.12.1 Additional Priority Exploration Targets

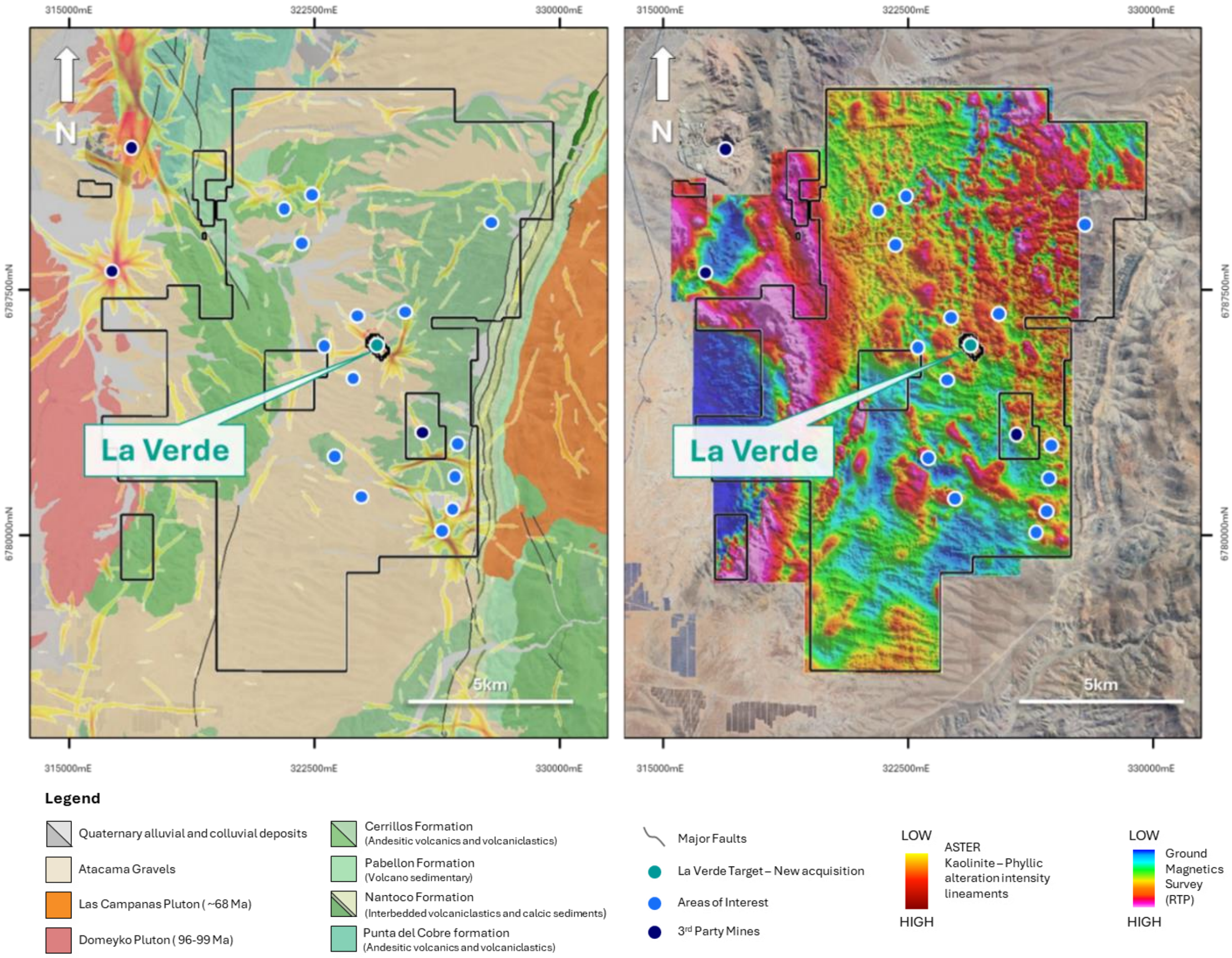
The Company is constantly progressing targets that show potential, as well as identifying and acquiring new project opportunities. In all cases, the primary exploration objective of the Company is to continue to add to its Mineral Resource base and build on previous successes at Productora and Cortadera, and more recently at La Verde.

24.12.1.1 Domeyko Cluster

The Domeyko Cluster, within the historic Domeyko mining district, is located 30 km SE of the Productora deposit. Historic copper-gold mining in the Domeyko area largely exploited oxide mineralisation with very limited exploration undertaken for copper sulphide mineralisation. This is partly due to the gravel cover which is present over much of the region and to the lack of consolidated ownership of the tenement package.

Mapping and soil sampling across the Domeyko cluster is ongoing, and a regional ground magnetic survey has been conducted. This work has helped define a series of targets (Figure 24.13) which will be further refined by follow-up planned geophysical programs and the use of artificial intelligence (in the form of machine learning algorithms) on the data sets.

Figure 24.13 Left - Regional Geology, ASTER lineaments and soil sampling in the Domekyo Cluster. Right - Ground magnetic survey across the Domeyko Cluster (HCH, 2024)



24.12.1.2 Productora Near-Mine

The steeply dipping Productora breccia mineralisation is currently open at depth and laterally in several places, representing straightforward opportunities for resource expansion. Provenance of the mineralisation also remains uncertain, and deeper anomalism within the drilling data and existing geophysical datasets point to an underlying porphyry source of the mineralising event.

Similarly, immediately to the west of the Productora deposit, prominent silica ridges trend north-south. The large extent of the ridges and the surrounding alunite clay alteration zone indicates that neither Productora nor Alice can be singularly responsible for their formation. Observation of an easterly dipping silica alteration zone in drilling below the central silica ridge supports the possibility of a link between the extensive lithocap alteration and the main Productora mineralising system at depth. An easterly dip of lithocap feeder structures may also present further targets below the other silica ridges; the typical discrete lenses of massive sulphides in an epithermal high sulfidation deposit could be identified as small chargeable and conductive anomalies within broader resistive horizons in IP surveys.

Other discrete targets exist immediately adjacent to the future Productora open pit, with a notable molybdenum soil geochemical anomaly over an IP high chargeability ring feature suggestive of an Alice sized porphyry at 'Productora Central'. Some 200 m to the south, a co-incident zone of MIMDAS survey chargeability and conductivity similar in character to the main Productora mineralisation is observed just outside the open pit design limit. The proximity of these targets presents convenient near-mine expansion drilling opportunities once the open pit is in development.

24.12.1.3 Cortadera Near-Mine

Resource development drilling at Cortadera paused in 2021 leaving the mineralisation open in some directions, particularly at depth in Cuerpo 3. Modelling suggests areas of best mineralisation may also extend laterally due to the presence of certain types of host stratigraphy, these areas together present targets for drill testing.

A recent high-powered MIMDAS survey undertaken by HCH has identified significant further targets along strike to the north-west in the prospective Serrano fault corridor.

25 Interpretations and Conclusions

25.1 Introduction

The QPs and authors have provided the following interpretations and conclusions in their respective areas of expertise, based on the review of data available for the Report.

25.2 Mineral Tenure, Surface Rights, Water Rights, Royalties and Agreements

SMEA controls Productora primarily through direct ownership, except for one exploitation concession (Uranio 1/70), in which a 30-year lease agreement has been executed with The Chilean Nuclear Energy Commission (CCHEN), which commenced in 2012.

All mining rights at Productora are exploitation concessions with no risk of expiring if the mining taxes are duly paid annually. Surface rights are 100% owned by SMEA, as are the maritime concession to extract sea water from the coast and the corridor of easements to construct a pipeline and electrical transmission line to Productora.

HCH Limited owns the Cortadera deposit through Frontera and controls an area measuring approximately 22,600 ha at the deposit through various 100% purchase option agreements with private mining title holders and 100% owned tenure.

All mining tenements at Cortadera are in good standing and all mining requirements have been met for the exploration phase. The area covered by the surface rights are sufficient for any potential open pit and underground mining operation together with the potential area for waste disposal and potential dump leach pads.

HCH, through Frontera, renegotiated the option agreement in Dec 2023 of the three now terminated options for Valentina, San Antonio and Santiago Z, now known as the El Fuego option agreement. The proposed JV involves an option agreement over 27 exploitation leases (~4,727 ha), whereby full ownership of 100% of the mining rights of the deposit will be transferred upon satisfaction of agreed upon payment.

Additional mining rights were also negotiated resulted in an option agreement with Antofagasta Minerals to earn a 100% interest in the AMSA leases over a two-year period. The proposed JV involves an option agreement over five leases (~555 ha), whereby full ownership of 100% of the mining rights of the deposit will be transferred upon satisfaction of an agreed upon payment. An option agreement with private parties to acquire 100% of the historical copper mine area Cordillera, located approximately 10 km southwest of Productora was also completed by HCH.

In 2024 the Company, through Frontera, acquired the Domeyko cluster of tenements, which includes the Companies active exploration project La Verde. The Domeyko option agreement with a series of private parties involves 78 leases whereby full ownership of 100% of the mining rights will be transferred upon satisfaction of agreed upon payment.

The exploitation of the Project contemplates operating with sea water. SMEA currently holds a valid Maritime Concession Licence for the extraction of saltwater from a location south of the Huasco area.

25.3 Geology and Mineralisation

The Costa Fuego Project includes four deposits; Productora, Alice, Cortadera, and San Antonio, which exist within a 10 km radius.

Drill spacing across the Costa Fuego Project is mineralisation style dependent, with the complex nature of Productora necessitating close spaced (80 m x 40 m) drilling across the majority of the deposit, resulting in over 274 km of samples collected over an initial five-year resource drill-out. Multi element assaying was completed on all drilling, resulting in a large dataset which contributed to the completion of a geochemistry-focused PhD, copper speciation analysis and incorporation of geometallurgical parameters in the Mineral Resource Estimate.

The scale of the Cortadera porphyry deposit meant the majority of the 87 km of drilling since 2019 was completed on an average of 80 m x 150 m spacing. Multi-element geochemistry was completed on all drilling, enabling improved understanding of the porphyry system and host rock geology, resulting in a higher grade and expanded model with subsequent drilling campaigns.

San Antonio has been mined historically for high grade copper, so the focus of drilling was to test for along strike and depth extension of the narrow, faulted mineralisation, as well as delineate the underground workings. Over 7 km of drilling has been completed by HCH since 2018 at varying drill spacing, generally down to 40 m x 40 m.

The current drilling density at Cortadera, Productora, Alice and San Antonio provides sufficient information to support a robust geological and mineralisation interpretation as the basis for Indicated and Inferred classified Mineral Resources for the majority of the deposit. A cut-off grade of 0.20% CuEq was used to report the open pit Mineral Resource, while 0.27% CuEq was used to report the underground Mineral Resource at Cortadera.

All deposit resources were reported within constraining surfaces to model Reasonable Prospects for Eventual Economic Extraction (RPEEE). Commercial software is utilised to generate constraining surfaces for open pit mining and block cave underground mining.

QP Statement: It is the opinion of the Qualified Person responsible for the Mineral Resource Estimate that the 2024 Mineral Resource Estimates for the Cortadera, Productora, Alice, and San Antonio deposits were prepared using industry standards and best practices by qualified professionals and may be relied upon for public reporting.
It is the opinion of the Qualified Person responsible for the Mineral Resource Estimate that the 2024 Mineral Resource Estimates for the Cortadera, Productora, Alice, and San Antonio deposits informing the 2025 PFS were prepared using industry standards and best practices by qualified professionals and may be relied upon for public reporting and for use in the preliminary economic analysis contained.

25.4 Mine Plan

The PFS included the parameters and procedures used for the designs of the Cortadera, Productora, Alice and San Antonio mines as conventional open pits and a block cave mine, estimates of the Mineral Reserves within the open pit and block cave mine plan, and establishes a practical mining schedule for the Costa Fuego Project PFS.

Open pit mining methods have been selected as the key exploitation technique for the Costa Fuego Project deposits. These methods are supported by near surface mineralisation which allows for low waste to processing feed strip ratios and associated lower mining costs. There is also underground block cave exploitation potential at Cortadera's Cuerpo 3 deposit.

The PFS delivered a first principles approach to the definition of the mine plan across

- Mine Operations and Equipment selection
- Open pit mining methodology, hydrogeological and geotechnical considerations, and pit design optimisation.
- Underground mining methodology, geotechnical analysis, design specific to a block cave methodology, fragmentation, crushing and materials handling
- Surface infrastructure including waste rock storage and stockpiles, acid rock drainage and slope stability, and
- Strategic mine scheduling

25.4.1 Mine Operations and Equipment Selection

The mining strategy assumption was one of a seven days per week, 365 days per year operation. Each day will consist of two 12-hour shifts with four mining crews required to cover the operation. The mines are envisaged to be operated by a mining contractor.

The PFS considered suitable production equipment: typically, a ~26 m^3 bucket excavator (for use in waste zones) and a 16 m^3 hydraulic excavator (for handling processing feed). Both would be expected to load 190 t trucks, with the smaller excavators also loading 90 t haul trucks in the deeper areas of the open pits where smaller access ramps would be introduced.

Underground mine equipment is assumed to be a rubber tyred, diesel-powered fleet. Underground development equipment would typically include major front-line equipment such as face drills (Twin Boom Jumbo), bolting drills, cable bolters, loaders (load haul dump units, or LHD) (~17 t to 21 t capacity), trucks (~60 t capacity), ground support equipment (shotcrete sprayers, agitator trucks), charge-up rigs, secondary breaking rigs, integrated tool carriers, boxhole rigs (for drawbell development), and other auxiliary support equipment. The underground production fleet would typically include loaders (material movement from drawpoint to crusher), secondary breakage equipment (drills and charge-up), integrated tool carriers and water cannons.

The best suited loader for the underground operations is either the 17 t LHD or 21 t LHD, with the 21 t LTD likely the preferred option. The reasonably long LHD hauling distances due to the large block cave/draw point footprint requires larger LHDs to reduce operational and production/productivity risks.

The main ore handling system with the underground mine is LHD tipping into two underground jaw-gyratory crushers (each of a name plate capacity of ~13 Mtpa) and feeder conveyor belts discharging via a discharge chute onto the main conveyor belt.

25.4.2 Open Pit Mining

Conceptual open pit mining of the near-surface mineralised material envisions a conventional drill-blast-load-haul method with 15-metre-high benches.

Optimisations were assessed across a series of revenue factors to find the optimal balance of NPV contribution, footprint requirements and strip ratio. The assessment culminated in an optimised economic value for each block, which was then combined with wall angles and assessed by an implementation of the Lerchs-Grossmann algorithm.

Geotechnical stability analysis informed the open-pit slope angles, incorporating geotechnical logging of drill core, rock quality evaluation, and intact rock strength testing. The amount of geotechnical data available is considered sufficient to support PFS-level input for the pit wall slopes.

The Productora deposit comprises the largest volume of open-pit mineralisation, with six pit pushbacks phased throughout the LOM. Pit-design and wall slopes are defined by data collected for the 2016 Productora PFS and reviewed by the Geotechnical QP for the 2025 PFS.

Pre-stripping of the Productora main pit is completed in parallel with project construction, resulting in pre-production stockpiles of 1.6 Mt of oxide leach feed, 3.0 Mt of sulphide concentrator feed, and 1.9 Mt of low-grade sulphide leach feed.

Open pit mining at Cortadera deposit comprises three separate pits, with the mining sequence commencing with the Cuerpo 1 pit, which has the largest volume of higher-grade, near-surface mineralisation. Cuerpo 2 and Cuerpo 3 are mined thereafter, in that order.

Satellite pits at Productora as well as pits at Alice and San Antonio are mined in a single phase. Across the combined open pits, it is anticipated that 296 Mt of concentrator feed, 41 Mt of oxide leach feed, 22 Mt of low-grade sulphide leach feed, and 537 Mt of waste will be mined, over a mining life of 14 years.

25.4.3 Underground Mining

Underground mining comprises an underground block cave, centred on the higher-grade core of the Cuerpo 3 deposit at Cortadera.

Underground optimisations have been run to investigate the block cave potential, with the optimal block cave shape, footprint and geometry developed using Geovia's Footprint Finder software. The final cave shape and block cave draw schedules were developed using Geovia's Long Term Planner software (PCBC).

Development to access the cave is planned to commence in Year 3, with a 4-year lead time until the cave is established. Once opened, the block cave is expected to have a mine life of 14 years.

Open-pit interaction is evaluated by analysing the development of the cave shape over time and its interaction with the open-pit bottom surface. The findings suggest that a crown pillar width of at least 300 m is necessary to maintain the integrity of the rock mass beneath the pit floor. The model also highlights the significant influence of geological structures on caving propagation.

Subsidence, fragmentation and standoff distances for the crusher chamber were all modelled as a component of the block cave geotechnical assessment.

The ground support assessment for caving infrastructure inside the footprint, including undercut and extraction level excavations, and for the crusher chambers located outside the footprint was conducted using benchmarking of caving operations. For declines, GMT employed the empirical Q-System, which has been adapted for mining applications by Potvin et al. (2015). The benchmarking for the undercut and extraction levels involved collecting data from 17 mines across 43 productive sectors, encompassing rock quality, in situ stress, drift size and spacing, undercut methodologies, and ground support types. Additionally, the benchmarking for the crusher chambers drew on data from seven different caving mines.

Standard excavation profiles were utilised based on typical cave operations. Larger extraction drives are implemented around the world in competent ground conditions, allowing the use of larger 21 t range LHDs. The decision was made for this study to use the more common 17 t LHD size and drive sizes, especially common for South American operations. This also provides marginally larger and more stable pillars on the extraction level layout due to smaller tunnel requirements for a given Drawpoint layout.

The block cave uses the El Teniente footprint layout, selected for large footprint caves with multiple tipping locations along the extraction drive.

Undercutting is a critical part of block caving and is required to create an initial void large enough to initiate and sustain caving of the intact rock above the intended extraction footprint. Given relatively shallow depth and manageable geotechnical and stress conditions, and the addition of hydro-fracture preconditioning, a post undercut method was selected for this study. To reduce risk, bit an apex and undercut level was added in such a location relative to the drawbells that the undercut method can be switch to a post undercut at a later stage. This is not a typical approach as it adds an extra undercut level, but it reduces risk at this early study stage by increasing the blasted undercut height, allowing high blast preconditioning to be completed from the Apex level during undercutting, and allows the undercutting sequence to be switched from post to pre undercutting.

25.4.4 Surface Infrastructure

Waste rock storage were designed in 15 m lifts. Each lift is constructed at an approximate angle of repose of 37°. A 15 m setback between each lift maintains the overall angle at 25° to facilitate reclamation and long-term stability. A constant 2.0 t/m^3 loose density was assumed. All the dumps avoid the quebrada (surface water course) boundaries, except for the Cortadera Cuerpo pit valley's where there are approvals to mine and disturb that area.

The main ROM pad is located in a strategic location reasonably close to the Productora and Alice open pits, where the main ROM primary crusher will feed a transfer (overland) conveyor belt to the Productora plant comminution circuit. There is also a ROM pad design with a primary crusher at Cortadera. The ROM pad at Cortadera was optimally located to connect the underground block cave ore via the Cortadera surface primary crusher to the RopeCon loading point.

All stockpiles and waste rock storage sites were specifically designed at an angle of repose of 36 degrees with sufficient berms to provide for stable impoundment designs. The geotechnical team reviewed the waste storage and stockpile designs and simulated the factors of safety to ensure they are deemed reasonable with lower risk of failures.

At Cortadera, waste rock will be used from Cuerpo 1 and 2 open pits to establish the RopeCon loading area for ore to be transported to the concentrator near Productora. This area will also serve as the platform for the underground portal development for the Cuerpo 3 block cave mine. The waste from the Cuerpo 3 open pit and any waste from Cuerpo 3 the underground development will be backfilled into/ deposited onto the Cuerpo 1 and 2 open pits and surface waste rock storage sites.

25.4.5 Mine Scheduling

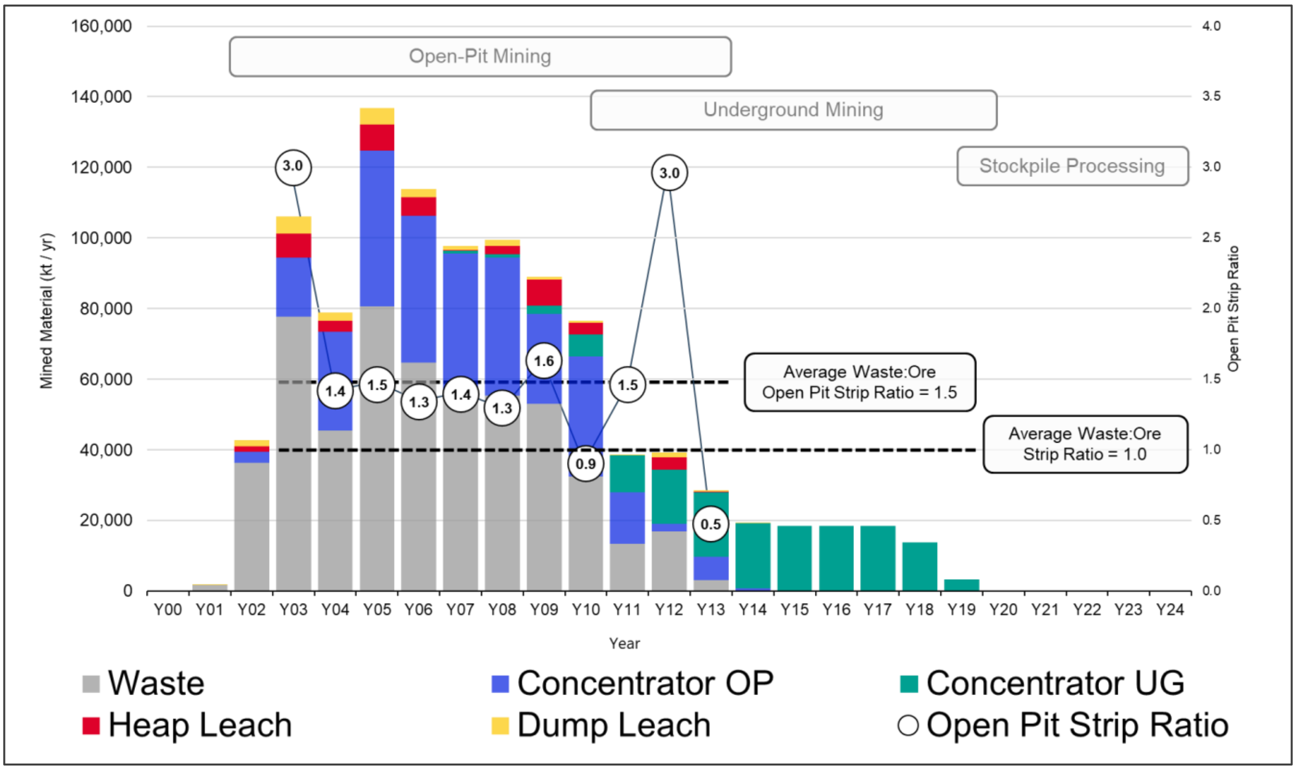
For the Costa Fuego Project, scheduling was conducted using the strategic scheduling software Minemax.

The mine plan consists of 100% Indicated Resources, mining a total of 1,042 Mt of material, comprising 439 Mt of sulphide concentrator feed, 41 Mt of oxide leach feed, 22 Mt of low-grade sulphide leach feed, and 540 Mt of waste over a 19-year mining life and 20-year processing life, including stockpile reclamation

The current Life of Mine ("LOM") plan focuses on mining higher grade, open-pit material early, with a waste to ore strip ratio for the open pit operations of 1.5:1 (including capitalised pre-stripping). The plan optimises metal delivered to the concentrator, heap leach and dump leach.

An elevated cut-off grade is applied throughout the mine life, with low-grade ore stockpiled and processed toward the end of the mining operations through either the low-grade sulphide leach or sulphide concentrator.

Figure 25.1 Mine Production Schedule based on 21.7Mtpa Processing Plant Average LOM Throughput



25.5 Metallurgical Testwork

25.5.1 Comminution

The comminution testwork has demonstrated that each of the Costa Fuego Project ores have defining hardness values and understanding this differences between deposits has provided a basis for selection of equipment in the preferred SABC (SAG mill, ball mill and pebble crusher circuit). Variations in competence and grindability mean that different ores will bottleneck in different mills in the flowsheet. Therefore, to predict throughput rate it is necessary to predict unconstrained SAG mill and Ball mill throughputs for a given mill feed then take the smallest of the two throughput values. To make these predictions it is necessary to have values for the competence parameter (DWi) and the grindability parameter (BWi). Correlations were developed to allow prediction of these two comminution parameters from the set of geological information which is carried into mine modelling.

The SAG limiting properties of the Cortadera UG ore (due to high competence) results in free ball milling capacity when Cortadera UG ore is treated alone. The project will take advantage of this by fine crushing a portion of the feed such that it takes up much of the unused ball mill capacity without placing further limitations on the SAG mill. This opportunity exists because, for several years, the Cortadera UG ore is projected to be the sole feed source for the concentrator.

25.5.2 Flotation

The following conclusions were reached from the flotation testwork throughout the history of the Project:

- Seawater is the preferred process water option for comminution and Cu/Mo flotation.
- Generally, grinds of 106 µm P_{80} were needed for acceptable liberation and rougher flotation recovery. A 106 µm P_{80} laboratory grind corresponds to a 125 µm grind in the plant due to differential SG effects on silicates and sulphides arising from cyclone classification.
- As chalcopyrite is effectively the only Cu mineral in all deposits, the RTD2086 collector is appropriate for use with sulphide ores from all deposits. RTD2086 is highly selective in floating chalcopyrite (and molybdenite) away from pyrite and gangue, but it also has poor to zero ability to recover copper minerals other than chalcopyrite if they were to be present.
- No pH adjustment or depressants are required for Cu/Mo flotation.
- Copper concentrate grades of 25% Cu were achievable in almost all samples tested. For most samples, regrinding to the range 25 to 40 µm P_{80} is essential to the achievement of 25% Cu grade at high recovery.
- Mo, Au and Ag generally float strongly with the chalcopyrite to produce a bulk Cu/Mo concentrate.
- To reduce chloride levels in the final Cu concentrate, and allow Mo flotation to proceed under optimal conditions, the mixed Cu/Mo concentrate is washed with low chloride water in a two stage CCD.
- The washed Cu/Mo concentrate is then processed by selective flotation to remove the Mo. The chalcopyrite is depressed using NaSH while the molybdenite remains strongly floating. A rougher Mo concentrate is produced and this is cleaned in an RO water circuit to 50% Mo, ready for sale. Most (90% or greater) of the Mo recovered in Cu/Mo flotation is recovered to the final Mo concentrate.

- The removal of the Mo leaves a copper concentrate which still contains Au and Ag, is of acceptable grade for sale and is essentially free of all penalty elements.
- The final step to make the Cu concentrate saleable, in respect to chloride content, is to dilute it with RO water and thicken it before filtration.
- For Cortadera OP ore, the rougher concentrates can report to a rougher-cleaner stage without the need of regrinding as it will still be possible to achieve target concentrate grade. For Productora ore and Cortadera UG ore, it is necessary to regrind all rougher and scavenger concentrates before cleaning to achieve the target grades.
- Testwork demonstrated that a pyrite flotation circuit could recover some of the copper and gold that was not recovered in the Cu/Mo flotation circuit. Pyrite flotation also has the benefits of leaving the plant tailings effectively free of sulphides, and providing a potential pathway for Co recovery.. Further testwork will be done to confirm the viability of this circuit.

25.5.3 Concentrate and Tailings Dewatering Testwork

Dewatering testwork conclusions:

- Thickening testwork has demonstrated that 65-70% solids underflows are possible with both concentrate and tailings using fresh ore at grinds of 150 and 106 um P80.
- With the same samples the flocculant consumption is only 10–15 g/t for both concentrate and tailings and the tested overflow clarities were below 150 ppm.
- With transition ores in the feed blend and at a 106 um P80 grind, the tailings thickener underflow densities reduced to about 60% solids, flocculant addition rates increased to 15-20 g/t and overflow clarities worsened to 250 to 300 ppm
- Centrifugal pumping of tailings is possible at densities of 70% (w/w).
- Filtration of copper concentrates with both membrane and pressure filtration resulted in moisture contents of less than 10% and chloride levels well below the 500ppm target.

25.5.4 Leaching

The hydrometallurgy QP notes that the work completed for this Report has been developed based on a robust flowsheet based on column pilot with ROM samples, and lab tests for heap and dump leaching using crushed drill cores, plus the Novaminore expertise in hypersaline process, in a number of testing campaigns of other copper minerals.

Additionally, SX reagent tests with high chloride PLS content were evaluated to demonstrate the feasibility of this reagent to process hypersaline copper solution.

The database generated was enough to develop a Cu recovery and acid consumption model to be applied in the Project evaluation.

Representative samples were collected by Hot Chili Geologists considering the Productora and Cortadera distribution of high-grade copper processing feed for Heap leaching and low-grade ore for Dump leaching.

25.5.5 QP Statements

The concentrator QP concludes that the concentrator design is at a state of development suitable for, or in advance of, the needs of the PFS. The Project complexity resides mostly in the logistics of moving ores to a single concentrator and not so much in the unit operations that the ores will be subject to. The ores have been defined to a suitably high extent and the concentrator flowsheet can be altered during operations to suit the changing flotation needs of the ores and blends.

There has been strong interaction between the process and mining disciplines as the means for predicting recoveries improved and as the implications of the comminution characteristics of the deposits became clearer over time. The ability to combine coarse and fine feeds to the plant and maximise production (while treating underground ores) was a major collaborative opportunity that has made its way into the PFS mine schedule.

While additional testing is recommended, the PFS outcomes are sound and the concentrator design is appropriate.

25.6 Infrastructure

The Project benefits from its proximity to existing regional infrastructure including population centres with aerodromes and ports, grid electrical connections, dual lane sealed highways, and service providers accustomed to industrial customers, including mining projects.

The study includes all associated infrastructure for providing industrial water, potable water, electricity, diesel, access roads, warehouse and other facilities required by the mine and production feed processing plants.

Offsite infrastructure includes a seawater supply system and power transmission line; both facilities are dedicated to the Project. The water supply system shall capture ocean water from the coast to the south of the existing Las Losas port and transport water to the Project site through a dedicated pipeline of 62 km length. The proposed Las Losas port facility will be utilised for receipt, storage, reclaim and ship loading of copper concentrate. The existing facility has environmental approval to upgrade to a copper terminal.

25.6.1 Rope Conveyor

Process feed from Cortadera will be primary crushed at Cortadera and then conveyed approximately 15 km via a rope conveyor to the main processing plant at Productora. The rope conveyor design includes 4 segments with a dual discharge to stockpile at Productora, allowing for the handling of both oxide and sulphide processing feed at a capacity of 25 Mtpa.

The PFS delivered a rope conveyor design, including foundation loads assessment, and capital and operating cost estimates. A trade off study was conducted to determine the approach for the rope conveyor for crossing of the Pan American Highway, where the overpass design was included within the design achieving clearance of 10m from the highway, and then incorporating a trench to lower the rope conveyor below regional high voltage powerlines. The trade off study included an option for a 71m long tunnel beneath the Pan American Highway, with sizing options from a light vehicle scale to a mining truck Caterpillar 793.

The roofing structure of the rope conveyor would host the electrical cabling, connecting Cortadera to the Productora electrical supply. The conveyor belt includes raised corrugated sides to maximise material containment, and shielding plates are able to be fitted where additional protection is required.

25.6.2 Access Roads

An access road has been aligned to follow the rope conveyor, providing access between Productora and Cortadera and intersecting the Pan American Highway.

An existing access road is in place connecting Cortadera and San Antonio. This road would be modified to accommodate haulage trucks delivering the San Antonio processing feed to the Cortadera rope conveyor loading station.

25.6.3 TSF

The proposed TSF utilises in-valley storage with three multi-zoned embankments. The TSF would be located approximately 5 km northeast of the Project plant site at Productora.
The topography of the proposed TSF area is steep and hilly, with the basin located in a valley that is bounded by ridges to the north and south. Principal embankments will be located at the western and eastern sides of the valley, with four smaller saddle embankments north of the basin.

The TSF is designed to accommodate 386 Mt of tailings, which is supplemented by an additional 114 Mt to be contained within the mined-out Productora Open Pit, for a total of 500 Mt of storage. This is sufficient for the current Costa Fuego Project.

Tailings would be delivered to the TSF via a delivery pipeline contained in a HDPE-lined trench, with an access road constructed adjacent to facilitate regular inspection by process personnel. Emergency preparedness design includes telemetry and automatic shut offs in the event of pipeline failure.

A TSF underdrainage system is included in the design to reduce head pressure near the embankments, while the basin itself is partially lined (across the gravel areas) with a waterproof membrane to reduce seepage, increase tailings densities, and improve the geotechnical stability of the embankments.

Other seepage management features (used for containment and recovery of seepage at the main embankment location) include the use of a cut-off trench, waterproof liners on upstream embankment faces, and abstraction bores downstream of the main embankment which will capture and return seepage to the process plant from the TSF. Seepage monitoring will be through the use of shallow and deep monitoring bore stations, with eight stations already established up- and downstream of the planned TSF location.

The TSF design also considered further risk mitigation measures that exceeded the regulatory requirements so that potential additions in the FS could be quantified at this early stage but were not included in the PFS. These included the use of additional waste rock in the embankment buttresses (+ USD 6M), extending the waterproof membrane beyond the gravels to cover the entire basin (+ USD 8M) and the addition of a second liner across the basin utilising Geosynthetic Clay (+ USD 28M).

The facility will be designed to attenuate the probable maximum flood ("PMF"), which is in addition to the capacity of the target operational pond. Excess stormwater capacity is significant, and the containment of the PMF is considered practicable with minimal additional freeboard being applied.
For closure, it is proposed that the final tailings profile be shaped to direct runoff to the north of the facility, where a closure spillway will be excavated in the northern ridge so that any rainfall runoff will run over the tailings surface to a sediment control area before discharge downstream.

25.6.3.1 Beach Slope

The design is based on an average tailings beach slope of 0.67% (150H:1V). However, the beach slope is heavily dependent on the grind size and the ore blend. Thus, small changes in plant performance or design, ore type, or the ore blend have the potential to change the tailings beach slope.

There are a number of approaches which can be used in response to measured beach slopes that are consistently different to the beach slope used for design. One advantage of staging construction on an annual basis is the ability to modify the design each year based on measured data obtained from the TSF. In these cases, the timing and height of the subsequent embankment raises can be modified to bring the schedule back into line with the design, and the subsequent lifts will be on an annual basis essentially as per the design raised heights.

Steeper Beach Slope

If the measured beach slope is steeper than the design slope, the tailings rate of rise against the TSF embankment will be faster than expected, and the Stage 1 TSF will reach its tailings storage capacity earlier than the design. If this were to become an issue, the response will be to move Stage 2 construction of the TSF forward. Commencing the construction one or two months earlier will not have a significant impact as the construction will still be predominantly in the dry season. It should be noted Stage 1 capacity is 18 months, which provides a high level of flexibility for the construction schedule, if required. In addition, the deposition line could be extended to the eastern valley to provide additional tailings storage capacity without impacting the operation significantly.

It should be noted that for steeper beach slopes the potential tailings storage will be reduced, but the storm water storage capacity will be increased accordingly.

Flatter Beach Slope

If the measured tailings beach slope is flatter than the design slope, the capacity of the Stage 1 TSF to store tailings will be increased. The overall TSF stormwater storage capacity will not be affected, unless Stage 2 construction is deferred beyond the original construction schedule.

25.6.3.2 Achieved Densities

The staged TSF embankment crest elevations are based on the ore blend and throughput used for the water balance modelling. Changes in these characteristics and/or throughput will result in changes in the achieved densities in the TSF. Similar to the variations in tailings beach slope, this may result in an adjusted construction schedule for the first raise, either earlier or later than the design timing. It is recommended that monitoring of throughput, ore blend, rate of rise and achieved densities be undertaken so that suitable planning and staging of the future embankment construction can occur.

The densities presented in this Report are based on a single tailings sample from the Productora open pit, therefore a change in density is possible. The density modelling will be reviewed at the completion of the physical tailings testing to determine the implications on the TSF design.

25.6.3.3 Life of Mine Planning

Any changes to the life of mine plan or throughput may impact the tailings management requirements for the site. Any significant increases in throughput may result in lower tailing densities being achieved within the TSF, thus increasing construction costs. Any decrease to the total tonnage may require reconsideration of the proposed closure plan, as the closure spillway may become prohibitively deep.

In addition to the impacts on the TSF design, any changes to the operating throughput and percent solids of the tailings may impact water demands.

25.6.3.4 Engineered Soil Cover

After decommissioning, the final soil cover for the tailings surface will be confirmed during operation based on ongoing operational tailings geochemistry testing results. The closure capping profile has been assumed for this Study and will be reviewed after the tailings geochemistry testing is completed. Based on the arid Project climate, it is anticipated that a store-and-release cover will be the most appropriate cover solution.

25.6.3.5 Operating Embankment Downstream Profile

If the TSF embankment Zone C material comprises coarse, clean rockfill of high strength sourced from the open pit, the stability of the TSF embankment downstream face using a steeper slope may be considered, subject to confirmation by a stability assessment once the rock fill properties are known. The final profile of 4H:1V (overall) is required for rehabilitation purposes.

Due to the liquefaction potential of the embankment foundation, this is not considered to be likely.

25.6.3.6 Availability Of Mine Waste

Design of the TSF is based on structural fill material being sourced from the open pit mining operations for Stage 1 and construction of future raises. If waste is not readily available during the Stage 1 construction, additional borrow areas will be required in proximity to the TSF. Although this is possible, the capital cost will increase significantly. Utilising a civil earthworks fleet to win material from the Open Pit footprints may prove to be uneconomical due to the long haul distances. In this scenario, material may be sourced from within the TSF basin area, which may offset some of the increased costs by providing additional capacity within the TSF (thus reducing the embankment fill volumes). Based on the current mining schedule there will be sufficient benign mine waste for all stages of TSF construction. Material placement should be carefully planned as part of mining operations to ensure waste placement is sufficiently advanced to allow stage construction of the upstream embankment zones and avoid double handling of mine waste.

Likewise, suitable low permeability fill material may be stockpiled by the mining operation at locations in close proximity to the TSF embankment, for use by civil contractors in future stages. This may reduce civil earthworks rates during future raise construction. The availability of low permeability material sourced from the open pit is not expected, however it will be reviewed when data are available.

25.6.3.7 Tailings Geochemistry

Static geochemical testing was carried out on a single tailings sample in 2015. Further static geochemical testing results were received at the end of the PFS and will be incorporated into the design prior to the commencement of a FS. The static results demonstrated a mix of PAF (potentially acid forming) and NAF (non acid forming) tailings products. Kinetic testing will be completed to supplement the understanding of PAF material and inform any treatments in the design, operation or closure of the facility.

Further geochemical testing of the tailings should be conducted at points throughout the life of the facility (nominally within the first year of operation and then every 2 years thereafter) to ensure that initial testing remains valid. Measurements will need to continue as part of ongoing operations to ensure information is available on the geochemical and physical behaviour of the tailings.

If tailings are confirmed to be potentially acid generating (PAG) and/or metal leaching (ML), then the proposed seepage control measures should be reviewed. It may be possible to separate the PAG +/- ML material and non-acid generating (NAG) non-metal leaching (NML) materials into a separate tailing stream that can be stored in separate cells. Alternatively, the PAG material could be stored on the heap leach pad, which will be lined with HDPE and could provide incremental metal recovery. The necessity of feasibility of separating the tailings into a PAG/ML and NAG/NML stream, if required, should be reviewed by HCH in the next phase of design.

25.6.3.8 Survey Data

Inaccurate base survey is a common cause of variations between expected and actual quantities, particularly in reference to bulk fill earthworks volumes. Topographical contours at times can be generated with small amounts of survey pickup, and as such there is significant interpolation by computer programs.

Accurate basin pickup is required as this will have a significant impact on both bulk fill volumes and the storage capacity of the facility. If there is less storage capacity than currently designed, an earlier start to Stage 2 construction may be required in order to continue to provide the required stormwater storage capacity. It is recommended that a stripped ground survey be completed during construction of the TSF to reconcile the required embankment heights prior to embankment construction. This may result in some cost reductions (or increases).

25.6.3.9 Low Permeability Fill (Zone A) Availability

Limited naturally occurring low permeability materials have been identified across the Project area. Further geotechnical investigations will be conducted to identify potential sources of low permeability materials. The current base case assumption is that no materials will be available, however if a borrow location in close proximity to the Project can be identified it may be possible to reduce costs.

If low permeability materials are unavailable, screened alluvium won from the TSF basin could be considered as an alternative. A desktop evaluation of the laboratory results will be conducted to determine if screening of the basin alluvium achieves the target material specification. Screening trials will be required to determine if screening can achieve the target material specification. If the desktop evaluation demonstrates that this is possible, then this will be adopted as the base case for the Feasibility study.

After Stage 1, the use of tailings as a construction material should be considered. Further technical evaluation will be needed to confirm that the material will be suitable as a construction material.

It is noted that the requirement to line the TSF with low-permeability material will significantly increase TSF construction costs. The use of geomembranes as the basis to reduce seepage from TSFs is not permitted in Chile, so they should not be considered as the primary seepage control measure even if adopted.

The current base case is that a composite liner system of HDPE liner and a GCL will be used due to the lack of naturally occurring low-permeability materials.

25.6.3.10 Seepage

The TSF was designed to capture seepage downstream of the TSF via a series of abstraction bores; this is the primary seepage control measure for the facility. In addition, a composite basin liner comprising a GCL overlain by HDPE is proposed as a secondary seepage measure, which will work in conjunction with a cut-off trench, toe drain and underdrainage to reduce seepage. Seepage is still expected from the facility and will be managed by the abstraction bores downstream.

A groundwater model will need to be developed as part of subsequent design to specify the bore locations and installation depths to ensure an adequate level of performance is achieved. The impact to downstream users of groundwater in contact with TSF seepage needs to be evaluated and understood, to ensure adequate contingency is incorporated into the design. At this stage, analysis by HCH baseline studies confirmed that there are currently no dwellings present downstream of the TSF, with the closest being 14km to the northwest. Other receptors in the region include the Mariposa operation and other abandoned mining sites. Currently HCH is unaware of any users of the groundwater downstream, however, this will be further investigated in future study analysis.

Water quality from the TSF must be monitored closely during operation and closure to ensure that performance criteria are being met. Should performance criteria not be met, a response plan should be developed and implemented to address the issue.

25.6.3.11 Seismicity

A preliminary seismic hazard assessment has been completed for this phase of design. An in-country seismic specialist has been engaged to conduct a site-specific seismic hazard assessment, which is in progress. It is anticipated that further characterisation and site investigations will be required to refine the seismic loadings determined for the TSF as part of this initial assessment. If the seismic loadings are determined to be larger than initially estimated, the embankment slopes may need to be flattened, and additional buttressing could be required, which will increase costs.

25.6.3.12 HDPE Liner and GCL

Based on discussions with HCH, the installation of a geosynthetic clay liner overlain by an HDPE liner is proposed as a basin treatment within the TSF basin (3H:1V and flatter). It is noted that the incorporation of these will increase costs significantly; however, while it is not a primary seepage control measure, it will help lower seepage and improve water recoveries. Therefore, a cost-benefit analysis and risk assessment should be conducted to evaluate the advantages and disadvantages of the proposed basin treatments.

It is noted that the installation of just an HDPE liner or GCL may be appropriate, which will reduce costs, whilst achieving a similar design intent.

The requirement for HDPE liner and GCL should be reviewed as the updated tailings geochemical testing and groundwater modelling are completed.

25.6.3.13 Buttressing

A minimum crest width of 24 m and 4H:1V profile is required to achieve the target factors of safety. A final crest width of 27.5 metres and 5H:1V overall buttressing profile are proposed to improve the factor of safety of the facility.

If sufficient mine waste is available, further buttressing could be considered. The haul distance from the Productora open pit is 5 to 6 km. Therefore, the additional overhaul costs are deemed reasonable. HCH should consider the additional haul costs and an additional factor of safety to determine whether further buttressing is needed or desirable to manage the facility's risks.

If the width and slope of the buttress were to increase, then the downstream abstraction bores will need to be relocated further downstream or alternatively raised vertically. The embankment was adjusted to allow the positioning of the abstraction bores based on the better underlying foundation conditions; therefore, a change in the buttressing will impact this design (or the boreholes can be located within the buttress footprint).

25.6.3.14 Foundation Liquefaction

Due to the region's seismicity and nature of the alluvium and fractured bedrock, the liquefaction of the embankment foundation was considered possible under post-seismic loading conditions. This was assumed to occur where the alluvium below the embankment was saturated. Based on preliminary steady-state seepage modelling with no HDPE liner or GCL, the alluvium was shown to be partially saturated, and adequate factors of safety can be achieved with the design profiles discussed above. If HDPE liner and a GCL are incorporated into the seepage model the factors of safety improve. The seepage modelling will be reviewed following the second phase of site investigation to confirm the findings and ensure that adequate factors of safety are achieved. The methodologies adopted are considered conservative based on available information. However, further verification of the foundation conditions and site-specific seismicity is needed as part of the subsequent design. If slopes need to be flattened further or buttressing increased, to achieve compliant factors of safety, then costs will increase.

The removal of the liquefiable material within the embankment foundation could be considered. However, it is envisaged that this may not be practical due to the depth of alluvium, resulting in increased costs. An additional geophysical survey of the revised embankment alignments in conjunction with additional boreholes are planned to estimate the volume of alluvium and determine if this approach is practical. This is proposed as part of the Phase 2 investigation.

25.6.3.15 In Pit Deposition

If Pit deposition were not practicable, additional capacity within the conventional TSF will be needed to accommodate the entire 500 Mt of capacity, which will increase costs. A preliminary assessment indicated sufficient capacity is available within the current TSF location for the entire 500 Mt. However, further technical evaluation will be needed to assess the viability of this expansion. It is noted that expansion to 500 Mt will increase the height of the northern saddle dams, so the deposition plan will need to be amended to push the supernatant pond away from these saddles to reduce the dam breach risk to Vallenar.

A groundwater model will be developed in the next stage to assess the impacts of tailings deposition into the Productora Pit. The water balance model will be reviewed and updated as additional information is available to assess the implications of in-pit tailings management.

A programme to assess alternative TSF sites, that may also meet the Project's requirements including for potential increased capacity, is underway and will be completed prior to the commencement of a FS.

25.7 Environmental, Permitting and Social Considerations

Several environmental studies are currently underway at the Project site to support the development of the Environmental Impact Assessment System and ensure compliance with environmental and operational permits.

These studies have investigated various aspects, including soils, climate and meteorology, geology, geochemistry, biological resources, cultural and anthropological sites, socioeconomics, hydrogeology, and water quality.

Hydrogeological studies have determined a low permeability setting for the Project, with low recharge rates varying from 1L/s - 1.5L/s. Future phreatic surface modelling (mine dewatering modelling) was conducted on the mine designs and inflow estimates range from 3 to 25 L/s within the pits and 58 to 116 L/s in the underground excavations. Surface water is addressed with non-contact water diversion channels.

Waste rock characterisation, including Acid Rock Drainage studies with net acid generation modelling, found sufficient correlation to geochemistry to allow for an interim surrogate model using S% and Ca% to approximate acid generation and acid neutralisation respectively.
The Project places a strong emphasis on environmental monitoring, water management, and biodiversity protection, aligning with global sustainability goals. The Project is well-positioned to effectively mitigate environmental risks, meet regulatory requirements, protect sensitive ecosystems and promote Community Engagement and Social Responsibility.

Building strong stakeholder relationships through transparent communication with local communities and indigenous groups is a key priority. Encouraging meaningful participation in decision-making processes will foster trust and social acceptance. Furthermore, the Project's investment in long-term community development initiatives, such as support for children's residence houses and mental health programs, will leave a lasting positive legacy in the region, promoting both economic growth and social well-being.

A comprehensive approach to planning, monitoring, and risk assessment for the Tailings Storage Facility (TSF) and rock emplacements is essential to minimizing structural and environmental risks. By prioritizing stability and structural integrity of the TSF, effective water management and progressive rehabilitation strategies of the

Project infrastructure, the Project will enhance safety, reduce environmental impacts, and align with international best practices.

Exploring innovative solutions, such as utilizing seawater in processing and minimizing the environmental footprint of rock emplacements and the TSF, will further enhance the Project's sustainability. These efforts, combined with investments in long-term rehabilitation and closure planning, will ensure responsible resource management and contribute to a sustainable mining future.

The QP is not aware of any significant political, environmental, or other risks that could potentially impact the development of the Project.

25.8 Markets and Contracts

A copper price of US$4.30/lb has been applied to the calculations for the 2025 PFS, viewed as balanced when compared with market forecasts. A 25-bank assessment, provided by NBF in February 2025 has a long-term copper price range of US$3.45/lb to US$5.00/lb.

Gold is a key by-product for the Costa Fuego Project, with a price of US$2,280/oz being applied for the 2025 PFS, aligned with the long-term gold price forecast.

A discount rate of 8% has been used for net present value ("NPV") calculations.

The mine is expected to produce a clean copper-gold-silver concentrate to be sold to smelters in Asia with offtake terms reflective of that market.

25.9 Capital Cost Estimates

All costs are estimated in United States dollars as at Q4 2024 and are judged to have an accuracy of ±15% to 25% and are considered to be at PFS level in accordance with Wood's Estimating Procedures and Class 4 as defined in the AACE document 18R-97.

The estimates are based on a combination of direct quotes, benchmarking, inputs from consultants and QP's, and HCH supplied data.

Construction and expansion capital costs are estimated at $1.27 billion and $1.35 billion, respectively, with LOM sustaining capital costs (including reclamation and closure) estimated at $811 million.

25.10 Operating Cost Estimates

All costs are estimated in United States dollars as at Q4 2024 and are judged to have an accuracy of ±15% to 25% and are considered to be at PFS level in accordance with Wood's Estimating Procedures and Class 4 as defined in the AACE document 18R-97.

Processing operating cost estimates have been prepared for the Cortadera surface primary crusher, primary crusher and sulphide concentrator located at Productora, Productora oxide plant (ore preparation, heap leach, SX/EW), sulphide dump leach and port operations (copper concentrate storage and transhipment).

Open pit mining cost estimates were developed using a zero-based model, and all expenditures for the open put have been classified as operating costs. The open pit plans to employ a contractor for the entire LOM. Underground operating cost estimates include the use of a development contractor for the initial 30 quarters. During this period all expenditures will be capitalised.

The overall life of mine operating cost is US$8,650 M.

25.11 Economic Analysis

Based on the economic analysis, the Project delivers a base-case, post-tax NPV8% of US$1.20 Billion and an IRR of 19% (based on metal price assumptions of US$4.30/lb copper (Cu), US$2,280/oz gold (Au), US$28/oz silver (Ag), and US$20/lb molybdenum (Mo)). On a pre-tax basis, the Project delivers a base-case NPV8% of US$1.71 billion and an IRR of 22%, with a project life of 20 years and a payback period of 4.5 years.

Project after-tax NPV is most sensitive to factors that affect copper revenue - copper price, grade and recovery - and discount rate. NPV is also sensitive to changes in mining cost, processing cost and construction capital.

Tax calculations are based on the tax regime in effect as of the date of this Report.

Royalties in the financial model are applied according to the royalty agreements described in Section 4.

25.12 Risks and Opportunities

25.12.1 Introduction

The following risks and opportunities associated with development of the Project have been identified by the Qualified Persons.

Section 24 provides additional discussion on multiple risk mitigation or opportunities that are intended to be investigated between the PFS and commencement of a FS. Risks and opportunities will be continuously assessed and reviewed throughout the various phases of the design, construction and operation, in accordance with HCH's Risk Management Framework.

25.12.2 Risks

As part of ongoing risk management HCH monitors general mining risks such as changes in political risks and uncertainties affecting legislation, royalties, labour and market volatility. These risks are monitored and mitigated by HCH as part of their ongoing Project development.

Within the PFS process, multiple risk assessments and assurance processes were undertaken. The most significant risks identified include:

Table 25.1 : Key Risks

Risk	Explanation	Possible Mitigation
Environmental/Social Risk	All phases of the Project are subject to environmental regulation in the	Continued development of good relationships with the local

Table 25.1 : Key Risks		
Risk	**Explanation**	**Possible Mitigation**
	jurisdiction in Chile. Environmental legislation is evolving in a manner that will require stricter standards and enforcement, increased fines and penalties for non-compliance, more stringent environmental assessments of proposed projects and a heightened degree of responsibility for companies and their officers, directors and employees. Government environmental approvals and permits are currently required in connection with the Project operations. Delays in obtaining and preparing baseline information and in the approvals process for the EIA may result in changes to projected timelines and nominated construction dates. Failure to comply with applicable laws, regulations and permitting requirements may result in enforcement actions thereunder, including orders issued by regulatory or judicial authorities causing operations to cease or be curtailed, and may include corrective measures requiring capital expenditures, installation of additional equipment, or remedial actions. Amendments to current laws, regulations and permits governing operations and activities of mining companies, or more stringent implementation thereof, could have a material adverse impact on the Project and cause increases in capital expenditures or production costs, reduction in levels of production, or delays in development of the Project. Citizen participation is considered for those mining projects that are environmentally assessed by the SEIA (the EIAs and the DIAs when they	stakeholders, particularly with the nearby communities is critical to the success of the Project. Formal agreements with identified Indigenous Communities for the Project Commencement of the EIA preparation, including the PCT (community communication program), baseline surveys and environmental studies, and incorporation of findings into the PFS and future design and planning activities Early establishment of monitoring points, including hydrology monitoring around critical infrastructure., with the ability to upgrade technology to automated or integrate into adaptive management practices. The Mining and Society Department of the Ministry of Mining helps to strengthen the relations between mining companies and local communities. That department encourages the development of alliances between mining companies, local communities, and NGOs, promotes training and education opportunities for local residents to qualify for mining jobs and communicates policies and/or good social practices in regions/communities associated with mining operations, among other tasks. A close relationship with that department can represent an opportunity to strengthen the ties with the local communities and other stakeholders, which will benefit the Project.

Table 25.1 : Key Risks		
Risk	**Explanation**	**Possible Mitigation**
	generate "environmental burdens"). At that stage, any interested party can make observations on the Project, which must be reviewed by the authority and answered by the owner of the mining project, if applicable. Industry best practices and standards call for the assessment of environmental, social and local economic impacts related to the Tailings Storage Facilities (TSF) on a continuous basis, so that any material changes can be addressed using best practices in adaptive management. Failure to obtain or maintain, or a delay in obtaining necessary permits or approvals from government authorities	Permitting, in addition to the EIA preparation is managed and advanced in consideration for supporting the current and future activities of the company across the Project.
Cost Risk	Capital and operating cost escalation as Project plans and parameters change or are refined. Diesel fuel is a significant component of the mine operating costs. Higher fuel prices could impact project returns given the stripping ratio, pit depth, and corresponding long haulage profiles. Project capital and operating costs are sensitive to foreign exchange changes. A strengthening US dollar without an offsetting positive change in the copper price could render the Project economically unviable. Project capital and operating costs are sensitive to increased equipment and labour costs.	Cost estimation included quotes from qualified local and global suppliers. Local availability of labour, investigated within the PFS through review of projects by consultants within the region, and through local supplier quotations. Multiple trade off studies completed to quantify alternative design options or components of project plans.
Resource Risk	HCH does not consider there to be significant risks or uncertainties present which would affect the Mineral Resource Estimation results. HCH has conducted exploration and resource development work in the	The approach taken by the QPs to complete these Mineral Resource Estimates incorporated detailed data validation and analysis, and several tens of iterations were completed before the final estimation was finalised.

Table 25.1 : Key Risks		
Risk	**Explanation**	**Possible Mitigation**
	region and on the Project for over 10 years. The Cortadera, Productora, Alice and San Antonio Mineral Resource Estimates have been reported using realistic economic constraints to produce a CuEq calculation, as well as an estimate of the Reasonable Prospect for Eventual Economic Extraction for both surface (open pit) and underground (block cave) operations.	Additional data including metallurgical testwork, bulk density analysis and infill resource drilling information is required to increase the confidence in the Mineral Resource Estimate to Measured Classification.
Process and Metallurgy	The combination of multiple orebodies being treated in blends throughout the Project may result in reductions in recoveries and grades compared to treating ore from each deposit in isolation and under optimised conditions. Stockpiling of some ores for long periods (many years) may result in oxidation of the sulphides leading to poor flotation performance when eventually treated. The flotation tails are classed as acid-forming due to the presence of pyrite. This could result in the need to line the entire tailings impoundment. Acid consumption may be excessive with some of the heap and dump leach ores.	Additional testwork is required to explore the behaviour of blends in the concentrator and develop strategies for managing the treatment of blends. Minimise the time that high grade fresh ore remains stockpiled after mining. This will need to be addressed in mine scheduling. Remove the pyrite from the tails by flotation (proven) and store it in a separate lined tails impoundment, leaving the main TSF unlined. Pyrite concentrate may also be used as a means of recovering additional payable metals.
Infrastructure	TSF reaching design capacity; Investment may be required to increase TSF capacity, which could lead to delays in mine production. Dam Break Assessment result Extreme, based on proximity to Vallenar.	Alternative TSF locations with additional capacity to be reviewed prior to FS. The TSF was designed with a reduced capacity to minimise likelihood of a dam failure event, given the consequence category could not be

Table 25.1 : Key Risks		
Risk	**Explanation**	**Possible Mitigation**
	In pit tailings deposition within the Productora pit will interact with the groundwater levels upon cessation of mine dewatering. Tailings deposition and geochemistry of tailings.	reduced. Additional locations will be investigated prior to FS. Additional hydrogeological studies specific to the scenario of in pit tailings in Productora. TSF design (for in pit tailings deposition) to be completed prior to FS Water treatment facility installed to treat TSF seepage Incorporate additional geochemical testwork for tailings products to confirm geochemical properties and any impact(s) on TSF design.
	The PFS rope conveyor design includes an over head passing of the Pan American highway, at 10m clearance. Risk of falling objects into a publicly accessible zone, and increased risk for development application challenges.	Over head design includes a walled belt to contain material and shielding plates for additional protection. Configuration is installed and real world tested in other locations. Trade off study to implement an underpass option for passing beneath the Pan American highway, has been completed.
	A substantial critical infrastructure footprint will be developed as part of the Project. This includes the three mining areas and the central process plant location at Productora, a western corridor to Huasco for the seawater pipeline and concentrate transport to port. Intra-project corridors include the RopeCon between Productora and Cortadera, and the road access corridor between San Antonio and Cortadera.	Infrastructure Corridor and Land Access Assurance Review by Independent QP High River Services. Community engagement activities, negotiation with stakeholders for formal agreements where appropriate. Establishment of environmental, flora, fauna and cultural heritage baselines and ongoing continuous monitoring activities. Expanded direct land holding and securing of easements by the company. Continuous monitoring of

Table 25.1 : Key Risks		
Risk	**Explanation**	**Possible Mitigation**
		land ownership status, legal and regulatory requirements

25.12.3 Opportunities

As part of ongoing risk management HCH monitors general mining opportunities such as changes in metal prices, decrease in energy/reagent/diesel pricing, Project optimisation and new technology. These opportunities are monitored and mitigated by HCH as part of their ongoing Project development.

Opportunities that may improve the Project that should be studied in more detail are described in the following table.

Table 25.2 : Key Opportunities		
Opportunities	**Explanation**	**Possible Benefits**
Single pit option at Cortadera	Currently in future years of operation a block cave mining operation is considered. Investigate a large single open pit scenario for Cortadera (no underground block cave) with the potential to materially increase production feed inventory and mine life.	After completion of initial studies there is opportunity to optimise Capex and Opex. Open pit mining method carries lower technical risk in design and operation.
Additional revenue streams	Include cobalt in the process feed inventory.	After completion of initial test work, and assuming positive results, there is opportunity to add an additional revenue stream in the form of Co metal. This would need to be assessed against impact of Capex and Opex within the financial modelling parameters and developed in future stages of the study.
	Consider further recovery of acid via pyrite roasting.	After completion of initial test work, and assuming positive results, there is opportunity to add an additional revenue stream. This would need to be assessed against impact of Capex and Opex within the financial modelling parameters and developed in future stages of the study.
Autonomous electric mining equipment	For mining (open pit and block cave) and as an alternative for the proposed future Cortadera materials handling system consider rope conveyor.	After completion of studies, assuming a positive result the financial modelling parameters given that autonomous electrical mining equipment would likely entail a higher Capex and lower Opex. Further benefits to the Project if

Table 25.2 : Key Opportunities		
Opportunities	**Explanation**	**Possible Benefits**
		the assessment is positive is the potential to drive a lower carbon footprint and increased operation safety for mining operations.
Alternate materials handling systems for Cortadera.	RopeCon vs. conveyor vs. autonomous haulage options.	Opportunity to optimise Capex and Opex
Optimise process flowsheet	Optimise grinding mill sizes and SAG/ball mill split.	Minimise overall power consumption
	Investigate use of alternative grinding technology.	Minimise overall power consumption and CAPEX
	Investigate elimination of the trash screens ahead of the flotation circuit.	CAPEX saving
	Investigate use of alternative flotation cells instead of conventional flotation cells.	Improved recovery and grade
	Investigate the use of coarse particle flotation	Minimise overall power consumption and CAPEX
	Perform pyrite flotation on the Cu/Mo tailings stream and maximise the value of the pyrite to the Project	Render project tails non-acid forming, provide low cost acid to leaching, allow Co recovery via leaching, improve Cu (and possibly Au, Ag and Mo) recovery via leaching
Increase solids content of tailings	Investigate use of a paste thickener instead of conventional high-rate thickener given an increase in the tailings slurry density from 65% to 70% solids.	Reduced moisture within TSF, possibly increasing stability and reducing seepage outcomes
Development of La Verde as additional feed	Currently an exploration Project, if the Project progresses to a Mineral Resource and Mineral Reserve estimate, an increase in production feed within a reasonable distance to Productora processing facilities is possible.	Increased tonnage across LOM, and potential impacts to increased project life, and revenue.

25.13 Conclusions

Results of this Report supports progressing the Project to a Feasibility Study (FS) stage.

- The exploration program continues to demonstrate the potential for future growth of the Mineral Resource, which may further enhance Project economics and/or extend the operating life.
- The sample preparation, security, and procedures followed by HCH are adequate to support the Mineral Resource Estimates contained herein.
- Assay data provided by HCH was represented accurately and is suitable for use in the Mineral Resource estimation.
- There are no known factors related to metallurgical, environmental, permitting, legal, title, taxation, socio-economic, marketing, or political issues which could materially impact the ability to develop the Project.
- The mine design produced within the PFS is robust and adequately informed. Mineral Reserves comprise Indicated Resources.
- The TSF solutions produced within the PFS are adequate for the proposed quantity of tailings within the PFS, with appropriate levels of risk management.
- The Project delivers a robust, positive, economic outcome.
- The metallurgical testwork undertaken is reasonably extensive and suitable for this level of study. The design of the processing circuit is based in the majority on this testwork data and there is minimal use of assumptions.
- The mineralised material is of moderate competency and hardness, and amenable to grinding in a conventional SABC circuit. The mineralogy is fine grained compared to may projects in South America. Testwork indicates a requirement to re-grind to a fine particle size (25 µm P_{80}) to achieve adequate liberation for flotation to produce good copper concentrate grades.
- Overall recoveries from ores to flotation concentrates are estimated at 87% for copper and 71% for molybdenum.
- The Project has been designed to meet current social and environmental management practices.

26 Recommendations

26.1 Introduction

The QPs consider that growth and optimisation opportunities outlined in Section 25 should be explored to advance the Costa Fuego Project through the next study phases, which will include the submission of an Environmental Impact Assessment (EIA) and Feasibility Study (FS).

The FS would be based on the recommended case presented in the PFS, utilising additional technical work to further improve capital and operating cost estimation accuracy and provide increased confidence in the financial model outcomes.

The recommended work program has an estimated cost of US$50,000,000 (Table 26.1).

Further detail of the proposed workstreams is included below.

26.2 Exploration

26.2.1 La Verde Copper-Gold Porphyry Discovery

HCH announced the discovery of a significant copper-gold mineralised porphyry system at La Verde in December 2024. The La Verde discovery is located 35 km, or 50km by road from the planned Productora processing facility.

Wide, shallow mineralisation at La Verde currently extends over a 1,000 m x 550 m footprint and remains open in all directions. Gravel cover masks a potentially much larger system with a number of geophysical targets to be tested. Depth potential also remains with near surface mineralisation extending to more than 300 m below surface.

With the application of first principles geological techniques, including surface mapping, geochemical sampling, interpretation of geophysical surveys, and the use of considered drillholes, a broad geological understanding of the La Verde system should be developed. With this in place, it would be possible to estimate a maiden Mineral Resource on the La Verde deposit.

Refer to section 24.11 for additional information.

26.2.2 Costa Fuego Near-Mine

Near-mine opportunities at Productora and Cortadera should be further investigated. The steeply dipping Productora breccia mineralisation is currently open at depth and laterally in several places, representing straightforward opportunities for resource expansion. Provenance of the mineralisation also remains uncertain, and deeper anomalism within the drilling data and existing geophysical datasets point to an underlying porphyry source of the mineralising event.

Similarly, immediately to the west of the Productora deposit, prominent silica ridges trend north-south. The large extent of the ridges and the surrounding alunite clay alteration zone indicates that neither Productora nor Alice can be singularly responsible for their formation. Observation of an easterly dipping silica alteration zone in drilling below the central silica ridge supports the possibility of a link between the extensive lithocap alteration and the main Productora mineralising system at depth. An easterly dip of lithocap feeder structures may also present further targets below the other silica ridges; the typical discrete lenses of massive sulphides in an epithermal high sulfidation deposit could be identified as small chargeable and conductive anomalies within broader resistive horizons in IP surveys.

Other discrete targets exist immediately adjacent to the future Productora open pit, with a notable molybdenum soil geochemical anomaly over an IP high chargeability ring feature suggestive of an Alice sized porphyry at 'Productora Central'. Some 200 metres to the south, a co-incident zone of MIMDAS survey chargeability and conductivity similar in character to the main Productora mineralisation is observed just outside the open pit design limit. The proximity of these targets presents convenient near-mine expansion drilling opportunities once the open pit is in development.

At Cortadera, Resource development drilling paused in 2021 leaving the mineralisation open in some directions, particularly at depth in Cuerpo 3. Modelling suggests areas of best mineralisation may also extend laterally due to the presence of certain types of host stratigraphy, these areas together present immediate targets for drill testing.

A recent high-powered MIMDAS survey undertaken by HCH has identified significant further targets along strike to the north-west in the prospective Serrano fault corridor.

26.2.3 Domeyko Cluster

The Domeyko Cluster, within the historic Domeyko mining district, is located 30 km SE of the Productora deposit. Historic copper-gold mining in the Domeyko area largely exploited oxide mineralisation with very limited exploration undertaken for copper sulphide mineralisation. This is partly due to the gravel cover which is present over much of the region and to the lack of consolidated ownership of the tenement package.

Mapping and soil sampling across the Domeyko cluster is ongoing, and a regional ground magnetic survey has been conducted. This work has helped define a series of targets which will be further refined by follow up planned geophysical programs and the use of artificial intelligence (in the form of machine learning algorithms) on the data sets.

26.3 Mineral Reserve Drilling

The Costa Fuego Project MREs are well advanced, with 100% of the PFS mining inventory classified as Indicated.

It is recommended that to derisk the Project prior to the FS, ore that is scheduled to be processed during the payback period (at the time of the PFS) is upgraded to Measured classification. It is estimated that the total amount of drilling required to achieve this would be approximately 27,500 m (with 14,500 m drilled using reverse circulation and 13,000 m drilled using diamond techniques).

26.4 Geotechnical

Several key recommendations should be addressed prior to the FS, including:

26.4.1 Open Pits

- Enhancing the geotechnical database through additional drilling and regular updates to the geotechnical models.
    - A proposed workplan includes a drilling campaign comprising 2.5 km at Productora, 1.1 km at Alice, and 2.2 km at Cortadera pits. The objective of this campaign is to outline the pit walls on final pit designs for Productora, Alice and Cortadera.
- Refining structural models with more detailed data on domain boundaries and joint characteristics and conducting direct shear tests to better understand structural strength.
- Updating the hydrogeological model using all piezometric data collected, with consideration given to a range of water table scenarios for slope stability assessments.
- Completing on-site validation of key structures to confirm their geometry as well as identifying any additional structures.
- Carrying out a seismic hazard assessment to determine appropriate peak ground acceleration (PGA) values for operational and post-closure phases.
- Reoptimising the Costa Fuego Project open pits following updates described above.

26.4.2 Waste Dumps

- Implementation of a test pit program focused on areas within the waste dump designs where material is to be deposited onto steeper slopes.
    - This program comprises 8 test pits at Productora and 2 at Cortadera.
- Re-evaluation of the current conservative waste dump design criteria, considering the acceptance of lower Safety Factor (SF) values in areas with higher data confidence and lower failure consequence.
- Completion of a seismic hazard report to validate the horizontal seismic coefficient used in pseudo-static analysis, ensuring it reflects site-specific conditions.

Characterisation of waste dump materials through laboratory testing to better understand shear resistance parameters. This characterisation may also help identify variability in the shear strength of deposited materials and enable calculation of the Probability of Failure (PoF) in future analyses

26.4.3 Underground

26.4.3.1 Data Collection

- Implementation of a geotechnical drilling program to ensure comprehensive coverage within the planned cave footprint and extraction levels. This includes dedicated drillholes for evaluating areas designated for critical infrastructure such as access and conveyor declines, ventilation shafts, and crusher chambers.
    - The program comprises 9.4 km of diamond drilling at Cortadera.
- Inclusion of reliable stress determination methods (such as overcoring) as part of future drilling programs.
- Completion of intact rock testing on the hanging wall sedimentary unit, which hosts much of the block cave development.
- Identification of rock mass strength through point load testing (PLT).

26.4.3.2 Geotechnical Hazards

- Investigation of risks and hazards related to underground block caving—even if certain risks are considered unlikely or absent—including air blast, mud rush, rock burst, mine collapse, unplanned dilution, and instability of vertical developments.

26.4.3.3 Numerical Modelling

- Incorporate flow modelling into the FLAC3D simulations, including essential parameters such as drawpoint spacing, material fragmentation, and height of draw
- Development of a large-scale 3D model of the extraction and undercut levels to assess potential rock mass damage resulting from stress abutment caused by the advancing undercut front and drawbell incorporation
- Development of 3D models of the crusher chambers to evaluate their stability and assess ground support requirements.

26.5 Mining

Recommendations for further work on both open pit and underground optimisations are summarised below:

26.5.1 Open Pit

- Further optimising pit design stages to enable optimal scheduling, particularly for the Productora and Cortadera open pits
- Reviewing pit ramp locations against updated site layouts to ensure ramp exit positions optimise haulage requirements
- Further iterative refinement of mine schedules to ensure total rock smoothing, smoothing of ore mining, sequential pit stage depletion, mining practical mining periods and sustaining reasonable mining periods within mining stages
- Trade-off between smaller and larger mining equipment fleet for deeper pit benches to determine whether additional ore loss and/or waste rock addition would be offset by decreases in mining costs
- Reviewing dump and stockpile locations to ensure optimal haulage and mining costs, and potentially reducing equipment requirements

26.5.2 Underground

- Reviewing alternative crushers and layouts, including jaw crushers, gyratory crushers, jaw gyratory crushers, and rotary breakers

26.6 Metallurgy and Processing

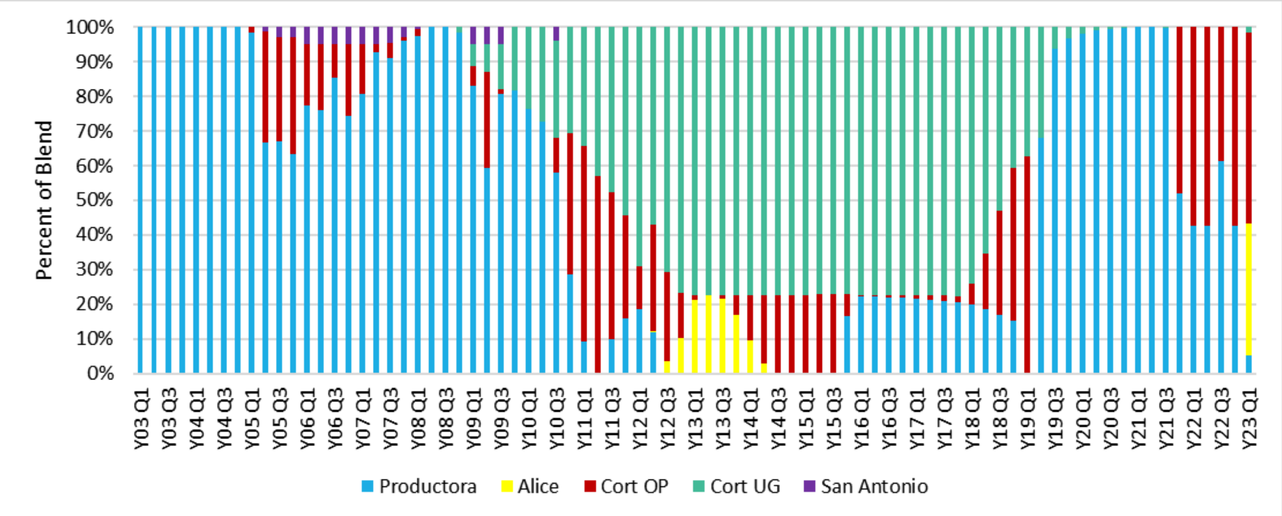
Additional metallurgical testwork is required to raise the process design to FS level. More variability testing is needed to ensure the ore properties are well understood, and more flotation testing is needed, especially in relation to blends from the deposits.

26.6.1 Sulphide Processing

26.6.1.1 Ore Blend Testwork

Ore blend testwork should be complete for sulphide ore reporting to the concentrator. Blends will be highly variable in terms of the deposits involved and the proportions of ore from each. The PFS ore delivery schedule will be used as a guide for planning future work, but the testwork requirements are likely to change before commencement of FS. The PFS delivery schedule of ore to the concentrator (by quarter) is shown in Figure 26.1.

Figure 26.1 : Ore Delivery Schedule Showing Deposit Proportional Contributions to Blends



Based on this schedule, the blends to be assessed by metallurgical testwork are:

- Productora with Cortadera OP (50:50, 75:25)
- Productora with Cortadera UG (75:25, 50:50, 25:75)
- Cortadera OP with Cortadera UG (50:50, 75:25)
- Cortadera OP with Alice (75:25)

The contribution of San Antonio to any blend in the schedule is less than 10% in all predicted quarters, so conducting blend testing is not justified.

The blend testing would focus on flotation behaviour, especially how the treatment of blends compares to the individual ore types. Both reagent additions and flowsheets change to varying degrees between deposits, so this testwork will provide practical operation guidance for dealing with changing blends. It will also inform the flotation circuit design with respect to stage residence times, reagent additions, stage concentrate and tailings destinations and sizing the regrind mill.

26.6.1.2 Sample Selection

A review will be conducted of the spatial representativity of the variability samples tested to date (in each of the deposits) to identify gaps where additional variability samples are needed. This review will also examine geological domains (where known) and identify gaps where additional variability samples are necessary to better define important lithological or alteration domains.

All new variability samples should be sourced through diamond drilling.

26.6.1.3 Comminution Testwork

Additional comminution testwork should be completed based on statistical and strategic requirements. While new samples must improve spatial representativity of all the deposits, there should be a bias towards samples representing ore from the early life of mine. These early ore focus time periods are commissioning ore, from year 1 of production, and payback ore, which is from the next five years of production. Further, the multiple deposits and staged project development introduce a need to bias towards those parts of the deposits that are likely to be involved in major blended processing (based on the most likely ore delivery schedule at the time of sample selection).

The additional comminution testwork should form part of a project geometallurgy strategy that continues into operations with the aim of providing robust comminution parameter databases. These databases will result in improved plant throughput equations to inform short, medium and long term mine and processing planning.

It is recommended ahead of the FS that 20 additional examples from each of Productora, Cortadera OP and Cortadera UG deposits be extracted and undergo comminution testwork. An addition three samples should also be obtained from each of Alice and San Antonio.

To continue this practice into operation, a geometallurgical budget should be allocated to all infill drilling programs and to ongoing exploration.

26.6.1.4 Flotation Testwork

Sufficient mass is required of each of the comminution variability samples to use the same materials in a flotation test program. Each variability sample will be tested individually and will also be used to prepare composites based on deposit, geomet time period (commissioning, Payback, Later), ore delivery blends and geological domains as necessary. A target variability sample mass of 40 kg minimum, and 50 kg per sample where possible, is recommended.

It is recommended that the FS flotation testwork program cover the following:

- Commissioning and payback ore composite testing
- Deposit master composite testing, open circuit and locked cycle, to confirm deposit responses

- Targeted blending composite testwork based on FS ore delivery schedule
- Flotation of each individual variability composite
- Exploration of additional opportunities for improving Cu/Mo flotation
    - Improving pyrite rejection through finer regrinding, froth washing and exploring additional regimes for depressing liberated pyrite if required
    - Additional trials of less selective copper collectors that will continue to reject pyrite while also recovering minerals such as bornite and chalcocite.
    - Explore residence time and collector addition variations
    - Evaluate coarse particle flotation options for Cu/Mo and for pyrite.
- Pilot testwork to demonstrate effects of stream recycling and to produce customer samples
    - Production of customer quantities of Mo-free Cu concentrate and Mo concentrate (if required)
    - Pyrite flotation testwork (pilot) to provide sample for associated values recovery testwork, especially incorporation of pyrite in heal leaching.
    - Pyrite stage water recycle system evaluation to determine if residual xanthate (PAX) will affect selectivity in Cu/Mo flotation
- Thickener and filter testwork on FS concentrates and tailings
- Chloride washing testwork on bulk concentrates

It is necessary to identify a replacement for the Tall Bennet collector RTD2086 as the vendor has now ceased operating. In addition, other reagent schemes are worth considering for flowsheet optimisation and for times when the ore is problematic.

- Possible replacement reagents for RTD2086 are
    - C4410 (SNF Flomin, Xanthate Ester)
    - C4403 (SNF Flomin, Xanthate Ester & Hydrocarbon blend)
    - C3302 (Syensqo, Xanthate Ester)
    - A3894 (Syensqo, Thionocarbamate, requires the use of sodium metabisulphite for selectivity against pyrite)
- For improving Mo recovery:
    - C4403 for improved selectivity against pyrite when treating oxidised ores
    - RTD2086 + kerosene for improved Mo recovery
- A8761 (Syensqo monothiophosphate) with or without RTD11A (Tall Bennet, Thionocarbamate) for improved selectivity of copper minerals over pyrite when Mo recovery is not essential (e.g Cortadera OP ores)
- A3894 as a less selective copper collector than RTD2086 which benefits from sodium metabisulphite use in cleaning. Also possible benefit from Diesel to improve Mo recovery.
- Other potential collectors from vendors, especially additional sources of Xanthate Esters.

26.6.1.5 Additional Testwork

Where applicable, further testwork on the following:

- Concentrate thickening testwork (including tests on samples from different domains)
- Concentrate filtration testwork (including tests on samples from different domains)
- Tailings thickening testwork (including tests on samples from different domains)
- Regrind mill testwork (including tests on samples from different domains)
- Rheology testwork
- Tailings characterisation testwork.

26.6.2 Pyrite Flotation

Pyrite flotation has been included in test programs conducted by Auralia, performed in more than 220 individual tests. Its inclusion has taken two forms, the first is as a simple rougher float to remove all the pyrite and the second approach is as a rougher/cleaner float (with or without regrinding) to make a high-grade Pyrite concentrate.

Flotation of pyrite has several potential material project benefits including the following:

- Rendering the tailings NAF (Non Acid-forming) as has been determined by testwork conducted by Knight Piesold
- Providing a source of sulphur for making sulphuric acid in a dedicated plant on site
- Providing a means of generating additional leaching extent within heap or dump leaching

The treatment of recovered pyrite concentrate opens up the potential for revenue increases by:

- Allowing the recovery of Cobalt which is bound within the pyrite lattice
- Recovering some of the Cu currently sent to tailings
- Enhancing the recovery of other metals that were unrecovered in Cu/Mo flotation, including Au, Ag and Mo.

26.7 Infrastructure

Major site infrastructure engineering and design has been performed to a PFS level, with significant advancements since the PEA materially derisking the Costa Fuego Project in key areas.

The QP notes that the infrastructure design is in line with the planned mining and processing rates and is appropriate for a development project within the region.

Recommendations for key infrastructure development items are detailed below.

26.7.1 Power

The Project has allowed for conceptual rerouting of existing overhead lines to avoid the planned TSF footprint. It is recommended to further develop this power line re-routing with the power authorities as part of the Project development.

26.7.2 TSF

26.7.2.1 TSF Design

- Conduct further geotechnical investigations in the next design phase to enhance understanding of subsurface conditions across the TSF footprint, including refinement of TSF embankment locations to avoid deep alluvial deposits
- It is recommended that a stripped ground survey be completed during construction of the TSF to reconcile the required embankment heights prior to embankment construction
- Complete further geotechnical investigations to identify potential sources of low permeability materials for use as Zone A low permeability fill.

26.7.2.2 Tailings Testwork and Geochemical Assessment

- Complete additional physical tailings test work on samples from each deposit and incorporate results into the next design phase
- Review recent geochemical test results (in progress) as part of the next design phase to refine tailings management and determine if further testing is needed.

26.7.2.3 TSF and Tailings Management

- Evaluate options based on the physical tailings testwork and geochemical assessment to reduce the risk of impact to the Quebrada La Higuera alluvial deposits and the Agua Verde wetland, e.g. liners, cut-off walls, grout injections, while a seepage collection system (SCS), based on a trench and/or pumping wells downstream of the West embankment
- Develop a groundwater model as part of subsequent design to specify the bore locations and installation depths to ensure an adequate level of performance is achieved. The impact to downstream users of groundwater in contact with TSF seepage needs to be evaluated and understood, to ensure adequate contingency is incorporated into the design
- Determine feasibility of shifting from annual to biennial embankment raises depending on mine scheduling
- Tailings densities included in design are based on a single tailings sample from the Productora open pit, therefore a change in density is possible. Density modelling will be reviewed at the completion of the physical tailings testwork programme to determine the implications on the TSF design
- Determine whether it will be necessary to separate potentially acid generating (PAG) and/or metal leaching (ML) material from non-acid generating (NAG) and/or non-metal leaching (NML) material to be stored in separate cells (or whether PAG material could be stored on the HDPE lined heap leach pad).

26.7.2.4 Productora In-Pit Tailings

- Additional hydrogeological studies are recommended for the Productora open pit to:
    - Assess seepage potential from tailings deposition in the final Productora open pit
    - Evaluate preparation and closure requirements for transitioning the final Productora open pit from mining to tailings deposition
    - Determine suitable locations for abstraction bores around the Productora open pit to intersect structures, allowing for groundwater quality to be monitored (and pumped, if required).

26.7.3 Rope Conveyor (RopeCon)

The PFS includes a trade-off study for an alternate underpass method to the Pan American Highway, discussed in Section 24, and a final design will be reviewed prior to the commencement of the FS.

26.7.4 Port

Port facilities for the Project are planned utilising an existing port area however the PFS has allowed for the construction of new storage and loading facilities with discussions ongoing to secure port access and services.

26.8 Environment and Social

HCH plans to complete the Environmental Impact Assessment (EIA) after finalising the PFS, ensuring that valuable baseline data, impact assessments, and environmental and social management plans are fully integrated into the FS.

Upon submission, the Environmental Assessment Service (SEA) has a statutory 120-day review period to issue a resolution on the application. However, in practice, the approval of project EIAs typically takes approximately 24 to 36 months. Therefore, timely completion of the EIA is critical to maintaining the Project schedule.

Upon completion of the administrative process, the SEA issues the Environmental Qualification Resolution (Resolución de Calificación Ambiental or RCA), which grants authorization for the construction, operation, or closure of the Project and certifies compliance with applicable environmental regulations.

The QP lists a number of recommendations for work to be completed prior to the submission of the EIA, and subsequent FS.

Technical workstreams contributing to EIA and FS:

- Expand coverage of hydrological studies to include the San Antonio deposit with the drilling of a monitoring well and development of a conceptual hydrogeological model, and in consideration of in-pit tailings deposition at Productora
- Complete additional studies on the ground water characterisation, to model flow, across the Costa Fuego Project
- Expand Acid Rock Drainage (ARD) studies, including the collection of additional samples from the Alice deposit. Review the expanded ARD sampling, including the longer-term humidity cell tests and tailings product ARD classification.

Social and Governance recommendations:

- Increase transparency in stakeholder communication by developing a transparent communication strategy that incorporates stakeholder feedback into decisions. Provide regular updates, distribute accessible ESG performance reports, use digital platforms for real-time engagement, and continue to hold community meetings to enhance trust and support
- Enhance environmental monitoring practices, with expanded monitoring, collection of comprehensive baseline data, increased monitoring frequency in high-risk areas, integration of data into a centralised platform, and engagement of third-party auditors for validation to ensure effective environmental risk management

- Formalise indigenous community relationships by establishing legally binding agreements with indigenous communities, outlining mutual benefits, cultural heritage protections, and capacity-building programs. Ensure access to legal representation and provide mechanisms for ongoing engagement and agreement updates. 75% of the Indigenous Communities have entered formal negotiations with the Company
- Improve ESG governance by including experts in emerging issues, incorporating stakeholder perspectives, regularly updating ESG policies, and enhancing transparency through detailed and accessible ESG reporting
- Expand Social Investment and Community Development Initiatives, continuing funding for community development, prioritising local procurement, expanding employment and training programs, implementing health and safety initiatives, and supporting cultural preservation projects to strengthen community relations and promote regional growth.

Other recommendations for Project advancement:

- Strengthen documentation practices by reviewing and updating to address gaps, ensuring all stakeholder feedback, decision rationales, and updates are thoroughly documented and accessible to stakeholders and authorities
- Revise and expand environmental monitoring plans by increasing monitoring frequency, allocation of sufficient resources, enhanced biodiversity monitoring, and adoption of adaptive management to promptly detect and address environmental impacts
- Implement formal risk assessment processes, establishing a structured risk assessment process integrated into project management, with regular reviews, documentation, and team training to evaluate and mitigate potential risks
- Develop a strategy for regular audits, integrate stakeholder feedback, and future planning workshops to align the Project with best practices and address future challenges
- Conduct legal audits and renegotiate land agreements with communities as needed to ensure compliance with evolving regulations and protect all parties' rights.

26.9 Work Plan

The estimated work program budget for the completion of Project FS is summarised below (Table 26.1).

Table 26.1 : Future Work Program

Work Program	**Cost (US$M)**
Reserve Upgrade Drilling	9
Geotechnical (including US$6M drilling)	7
Mining	1
Metallurgy (including US$8M drilling)	12
Infrastructure	4
Environmental and Social	4
Permitting and EIA Management	4
Legal/Commercial	1
Feasibility Study Management	8
Total to FS	**50**

The QPs have reviewed the proposed program of work and budget and find them to be reasonable and justified considering the observations made in this Report. The recommended work program and proposed expenditures are appropriate and well thought out. The proposed budget reasonably reflects the type and scope of the contemplated activities.

The QPs recommend that HCH conduct the planned activities subject to availability of funding and any other matters which may cause the objectives to be altered in the normal course of business activities.

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27.2 Abbreviations

Table 27.1 : Abbreviations and Units

Abbreviation	**Definition**
%	Percentage
°C	Degrees Centigrade/Celsius
μm	Micron
a	Year (annual)
Ag	Silver
Ai	Abrasion index
Au	Gold
cm	Centimetre
Co	Cobalt
Cu	Copper
dt	Dry tonne
dt	Dry tonne
F_{80}	Feed 80% passing particle size
g/L	Grams per litre
g/t	Grams per tonne
g/t	Grams per tonne
kg/t	Kilograms per tonne
kg/t	Kilograms per tonne
kt/a	Kilotonnes per annum
kW	kilowatt
kWh	Kilowatt-hour
L	Litres
L	Litre
m	Metre
m/h	Metres per hour
m^2	Square metre
m^3	Cubic metre
m^3/h	Cubic metres per hour
max	Maximum
mg/L	Milligrams per litre

Table 27.1 : Abbreviations and Units	
Abbreviation	**Definition**
min	Minutes/minimum
mm	Millimetre
mRL	Mean Reduced Level (Mean Sea Level)
Mt/a	Million tonnes per annum
Mo	Molybdenum
No.	Number
P_{80}	Product 80% passing particle size
ppm	Parts per million
ppm	Parts per million (by mass)
t	Tonne
t/a	Tonnes per annum
t/m^3	Tonnes per cubic metre
Wi	Work index

27.3 Glossary of Terms

Table 27.2 : Glossary of Terms	
Abbreviation	**Definition**
AP	Advance Payment
APC	Advance Payment Charge
AQSIQ	General Administration of Quality Supervision, Inspection and Quarantine, China; more commonly known as *China Customs*. Previously CIQ.
BL	Bill of Lading
BHOD	Best Height of Draw
BWi	Bond Ball Mill Work Index
CAGR	Compound Annual Growth Rate
CCHEN	The Chilean Nuclear Energy Commission
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CFR	Cost & Freight (Incoterms)
CIF	Cost, Insurance & Freight (Incoterms)
CIQ	See AQSIQ
CMP	Compañía Minera del Pacífico S.A
CoA	Contract of Affreightment
CSPT	Chinese Smelter Purchasing Team
Comex	(US) Commodity Exchange (a division of the New York Mercantile Exchange)
CuEq	Copper Equivalent
CY	Container Yard / Contract Year / Calendar Year
DAP	Delivered at Place
DD	Diamond Drilling
DL 600	Decree-Law 600
DMT	Dry Metric Ton or 2204.62 lbs avoirdupois equivalent
DWi	Drop Weight Index
EIA	Environmental Impact Assessment
EXW	Ex-Works
FCFM	Universidad de Chile laboratory

Table 27.2 : Glossary of Terms	
Abbreviation	**Definition**
FF	Footprint Finder
FO	Free Out (Buyer pays for discharge)
FOB	Free on Board (Incoterms)
FoS	Factors of Safety
Frontera	Sociedad Minera Frontera SpA
G&A	General and Administration
GAC	Gestión Ambiental Consultores
HCH/Company	Hot Chili Limited
HC/HTC	Holding Certificate / Holding & Title Certificate
ICSG	International Copper Study Group
ILZSG	International Lead ~ Zinc Study Group
IMO	International Maritime Organisation
Ingeroc	Ingeniería de Rocas Ltda
JV	joint venture
KP	Knight Piésold Consulting Engineers
LBMA	London Bullion Market Association
LIBOR	London Interbank Offer Rate
LME	London Metal Exchange
LOM	Life of Mine ()
MAO	Material Allocation Optimiser
MMS	Multimine Software
Mo	Molybdenum
MRE	Mineral Resource Estimate
NPV	Net Present Value
NPVS	NPVS software program NPVS developed ultimate and optimal final pit shell surfaces that identified the economic limit of mining, split up into revenue factor
NSR	Net Smelter Return
OBL	Original Bill of Lading
OCG	iron-oxide-copper-gold
OP	Open Pit
Ounce	Troy Ounce of 31.1035 gram(mes)
PP	Price Participation
PCBC	Geovia PCBC™ software
PEA	Preliminary Economic Analysis
PFS	Pre-Feasibility Study
PLT	Point Load Testing
QA/QC	Quality Assurance Quality Control
QPs	Qualified Persons
RC	Reverse Circulation/ Refining Charge
RCDD	Reverse Circulation pre-collar with Diamond Drill tail
rf	revenue factor
ROM	Run of Mine
RPEEE	Reasonable Prospects of Eventual Economic Extraction
SCM Carola	Sociedad Contractual Minera Carola
SLM Purísima	Sociedad Legal Minera Purísima Una Sierra La Cortadera
SMEA	Sociedad Minera El Aguila SpA
SMECL	Sociedad Minera La Frontera SpA
SMU	Selective Mining Units

Table 27.2 : Glossary of Terms	
Abbreviation	**Definition**
SOE	State-owned Enterprise
SOFR	Secured Overnight Financing Rate
ST	Stacking Test
SX/EW	Solvent extraction and electro-winning
TC	Treatment Charge
TC/RC	Treatment and refining costs
the Company	Hot Chili Limited
the Project	Costa Fuego Copper Project
TML	Transportable Moisture Limit
TSF	Tailings Storage Facility
WMT	Wet Metric Ton or 2204.62 lbs avoirdupois equivalent
WSMD	Weighing, Sampling and Moisture Determination
Battery Market Terms	
ESS	Energy Storage System
EV	Electric Vehicle
IEA	International Energy Agency
LiB/LIB	Lithium-ion Battery
Mn	Manganese
NCA	Lithium Nickel Cobalt Aluminium Oxide
NiCd	Nickel-Cadmium
NiMH	Nickel-Metal Hydride
NMC	Lithium Nickel Manganese Cobalt