Hot Chili Limited

Costa Fuego Copper Project

NI 43-101 Technical Report Mineral Resource Estimate Update

Effective Date 26 February 2024

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8th April 2024

Date and Signature Page

The undersigned prepared this technical report titled “Costa Fuego Copper Project - NI 43-101 Technical Report Mineral Resource Estimate Update”. The effective date of this Technical Report is February 26, 2024, and the report date is April 8, 2024.

Qualified Persons for this report are:

- Elizabeth Haren, MAIG, FAusIMM – Haren Consulting
- Anton von Wielligh, FAusIMM – ABGM Pty Ltd
- Edmundo J Laporte, PE, PEng CPEng – High River Services LLC
- David Morgan, MAusIMM CPEng – Knight Piésold Pty Ltd
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- Jeffrey Stevens, Pr.Eng – Wood Australia Pty Ltd
- Piers Wendlandt, PE – Wood USA Inc

Qualified Person Certificates are included in Appendix A.

Important Notice

This report was prepared as a National Instrument 43-101 Technical Report for Hot Chili Limited (Hot Chili) by Wood Australia Pty Ltd (trading as Wood). The quality of information, conclusions, and estimates contained herein is consistent with the terms of reference, constraints, and circumstances under which the report was prepared by Haren Consulting, ABGM Mining, Knight Piésold, High River Services, and Wood. The report is based on i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report.

This report is intended to be used by Hot Chili subject to terms and conditions of its contract with Wood. That contract permits Hot Chili to file this report as a technical report with Canadian securities regulatory authorities pursuant to provincial and territorial securities law. Except for the purposes legislated under Canadian provincial and territorial securities law, any other use of this report by any third party is at that party's sole risk.

Table of Contents

List of Tables

List of Figures

1 Summary

This section (except 1.1 to 1.11, and 1.23 to 1.25) has been taken directly from the technical report titled, "Costa Fuego Copper Project: NI 43-101 Technical Report & Preliminary Economic Assessment", dated August 14, 2023 with an effective date of June 28, 2023 (the "2023 PEA"). The Mineral Resource Estimate update, as described in this Report, does not materially change the Mineral Resource inventory that formed the basis of the 2023 PEA and no new scientific or technical information has been developed that would materially affect the outcome of the 2023 PEA and, therefore, the results and conclusions of the 2023 PEA are considered current and have been restated for this Report. This report replaces and supersedes the 2023 PEA report, which should no longer be relied upon.

1.1 Introduction

At the request of Hot Chili Limited (HCH or Company), Wood Limited (Wood) has prepared this Mineral Resource Estimate Update Technical Report (the "Report") for the Costa Fuego Copper Project (the Project), as announced by HCH on February 26, 2024 and to re-iterate the results of the Costa Fuego Copper Project: NI 43-101 Technical Report & Preliminary Economic Assessment (PEA) dated August 14, 2023 with an effective date of June 28, 2023.

The Report was prepared in accordance with the Canadian disclosure requirements of National Instrument 43-101 (NI 43-101) and in accordance with the requirements of Form 43-101 F1.

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 engineering and geological consultants are as follows:

- Haren Consulting is an independent geological consulting firm based in Perth, Australia
- 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
- High River Services is an independent environmental consulting company, based in Kentucky, USA.

The Qualified Persons for this assignment are as follows:

Table 1.1 : List of QP's		
Qualified Person	**Company**	**Section(s)**
Elizabeth Haren	Haren Consulting	1.6, 1.9, 1.10, 1.12, 1.23, 1.24, 1.25, 7, 8, 9, 10, 11, 12.1 – 12.6, 12.9, 12.11, 14, 25.3. 25.12, 25.13 and 26.2
Anton van Wielligh	ABGM Pty Ltd	1.13, 1.14, 12.8, 15, 16, 23, 24, 25.4 and 26.3
Dean David	Wood Australia Pty Ltd	1.1 - 1.5, 1.7, 1.8, 1.11, 1.15, 1.16, 1.20, 2, 3, 4, 5, 6, 12.7, 13, 14.7, 17, 18, 21.3, 21.4, 25.5, 25.6, 25.10, 25.12, 25.13, 26.4, 26.5 and 27
David Morgan	Knight Piésold Pty Ltd	18.6
Piers Wendlandt	Wood Group USA, Inc.	1.18, 1.21, 1.22, 12.7, 19, 22, 24, 25.8, and 25.11
Jeffrey Stevens	Wood Australia Pty Ltd	1.19, 21.1, 21.2, 21.5, and 25.9
Edmundo J Laporte	High River Services LLC	1.17, 12.10, 20, 25.7, and 26.7

1.2 Terms of Reference

This Report has been prepared following additional exploration, development (metallurgical and geotechnical), and resource expansion drilling in 2022 and 2023 at the Cortadera, Productora, San Antonio and Alice deposits. Drilling previously completed at Alice in 2016 and 2017, results of which were returned after the previous estimates cut-off date, was also incorporated.

The 2023 Preliminary Economic Assessment (2023 PEA) was completed after the Productora 2016 study (2016 study), following the addition of the Cortadera and San Antonio deposits, and included additional infrastructure required to transport these Mineral Resources to the proposed centralised process plant at Productora. This expanded Mineral Resource provided an opportunity to lift the scale of development for a combined development hub (Costa Fuego), with optimised infrastructure servicing all deposits.

The 2023 PEA of the Costa Fuego Project includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied that would enable them to be categorised as Mineral Reserves and there is no certainty that the 2023 PEA will be realised. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Measured or Indicated Mineral Resource with continued exploration. The 2023 PEA presents a materially different project to that contemplated in the 2016 Study.

The 2016 study was undertaken at a nominal plant throughput of 14 Mt/a for a copper sulphide process plant and a 3.3 Mt/a nominal throughput for the oxide heap leach process facility. This PEA considers a 20 Mt/a nominal plant throughput for a copper sulphide process plant and 3.3 Mt/a of oxide feed to be processed via a crushing-agglomeration-heap leach circuit coupled with a SX/EW plant producing up to 10 kt/a of copper cathode (increasing to 12 kt/a in Year 8).

The 2023 PEA contemplates conventional open pit and underground block cave mining, feeding a conventional process plant to produce copper and molybdenum concentrates and copper cathode.

A Mineral Resource Technical Report was issued with an effective date of 31 March 2022, that was filed on 16 May 2022 (referred to as the 2022 Resource Report).

The 2023 PEA had no impact on the 2022 Resource Report. The 2022 Resource Report is the basis of the key assumptions and parameters used in the 2023 PEA.

The 2024 Mineral Resource Estimate, as described in this Report, does not materially affect the 2022 Mineral Resource inventory that formed the basis of the 2023 PEA and no new scientific or technical information has been generated that would materially affect the outcome of the 2023 PEA and, therefore, the results and conclusions of the 2023 PEA are considered current and have been restated in this Report.

The term "Property" is used in reference to the overall mineral tenure holdings.

Units used in the 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).

Mineral Resources 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 NI43-101.

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 Project is located 17 km south of the regional township of Vallenar, in the Atacama region of Chile, in the low-altitude coastal range belt (at ~800 m elevation).

The Project contains four deposits with current Mineral Resources within a 10 km radius: Productora, Alice, Cortadera and San Antonio. Note that in the 2022 Mineral Resource Report, Productora and Alice were reported together.

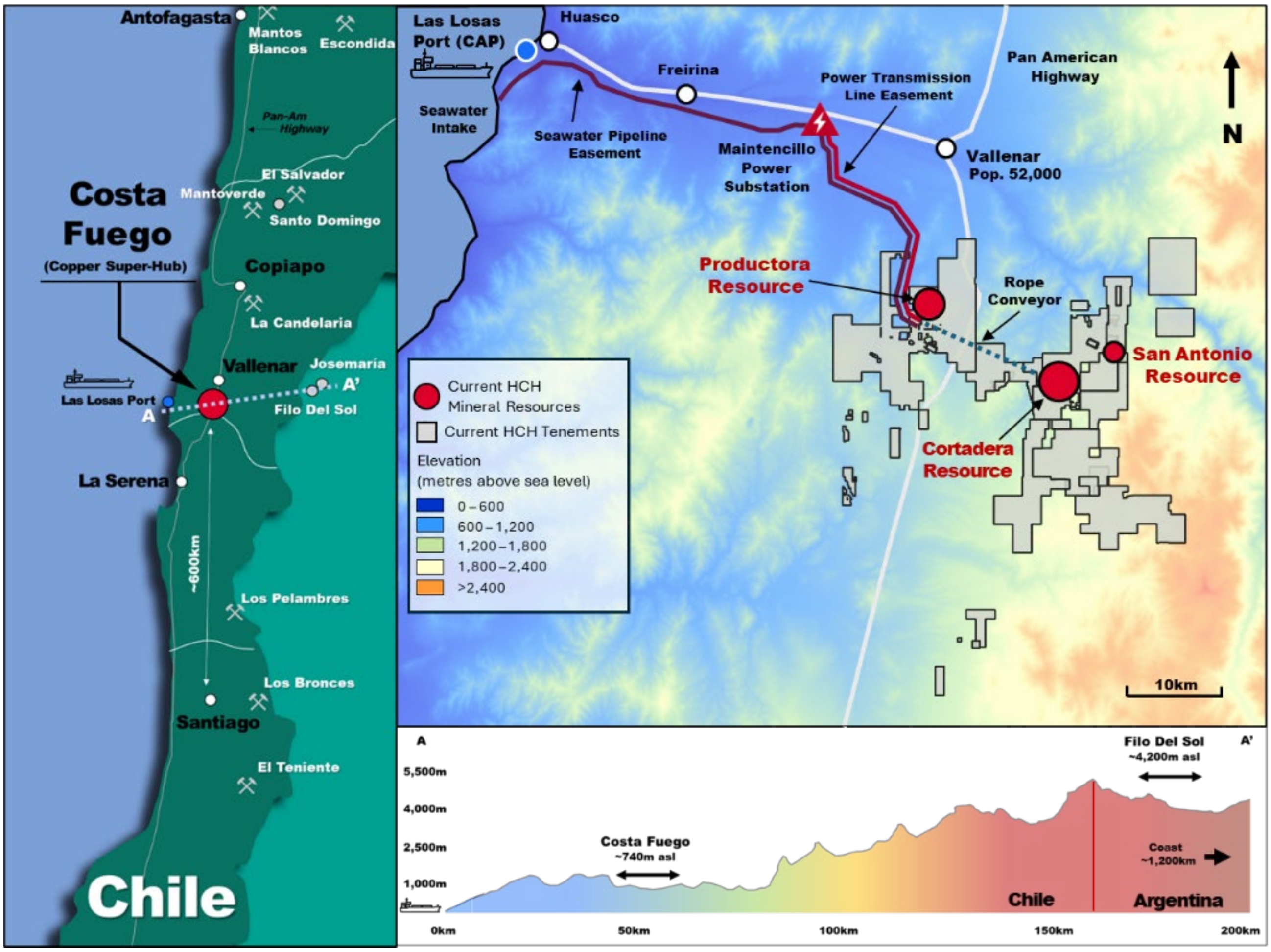
This study contemplates conventional open pit truck and shovel operations and an underground block cave mine, feeding a conventional copper sulphide mineral process plant and an oxide heap leach and solvent extraction and electro-winning (SX/EW) plant to produce copper and molybdenum concentrates and 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, 2024)



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 Hot Chili), 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.

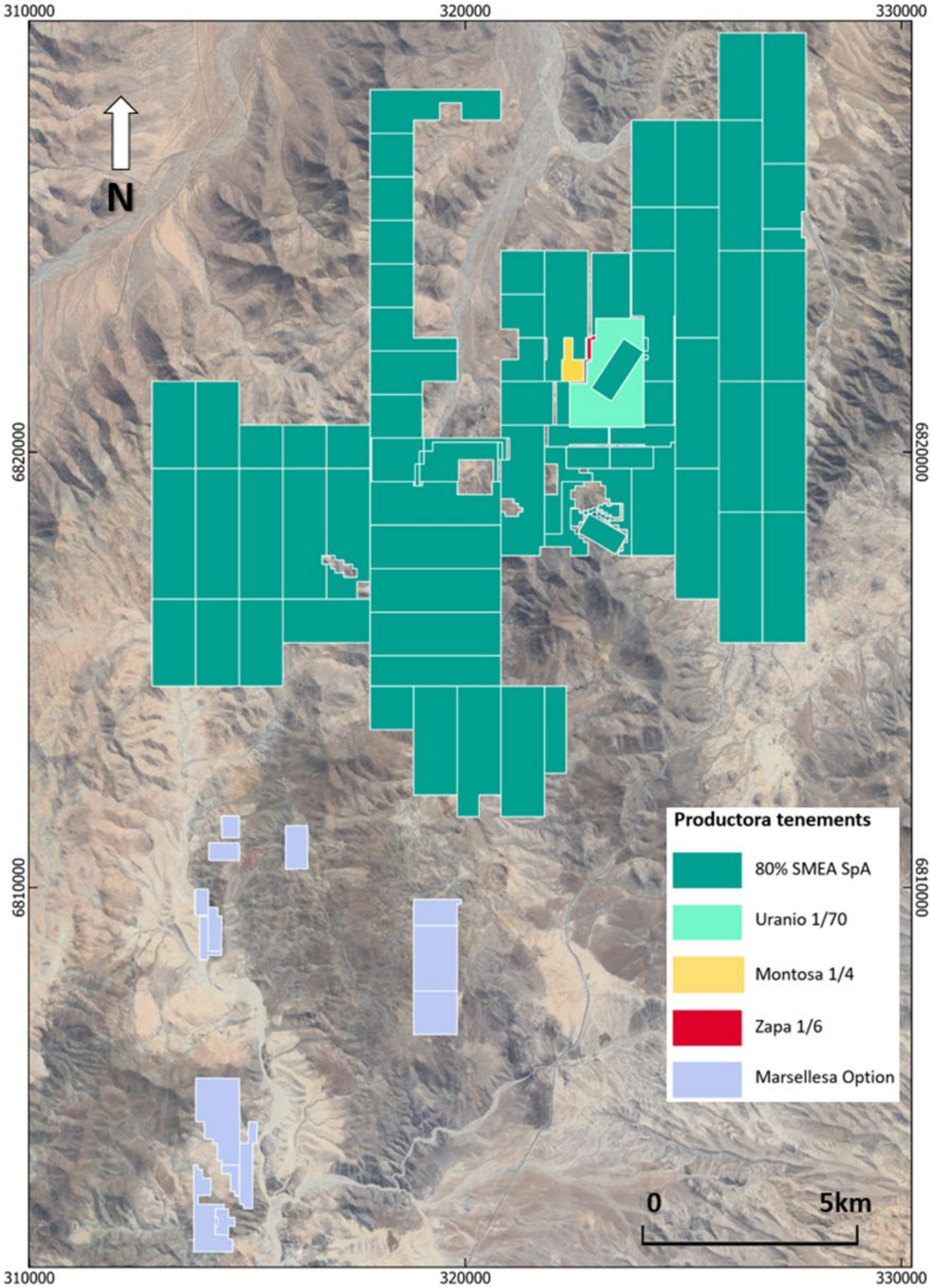
A 30-year lease agreement exists for the mining right Uranio 1-70 between the Chilean Commission of Nuclear Energy (CCHEN) and SMEA, dated 22 August 2012. This agreement incurs an annual lease payment of US$250 000 per year (paid no later than 31 August) and expires on 22 August 2042.

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 the Company, 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.

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

Figure 1.2 : Productora Mining Rights Under WGS84 (HCH, 2024)



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 20 000 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:

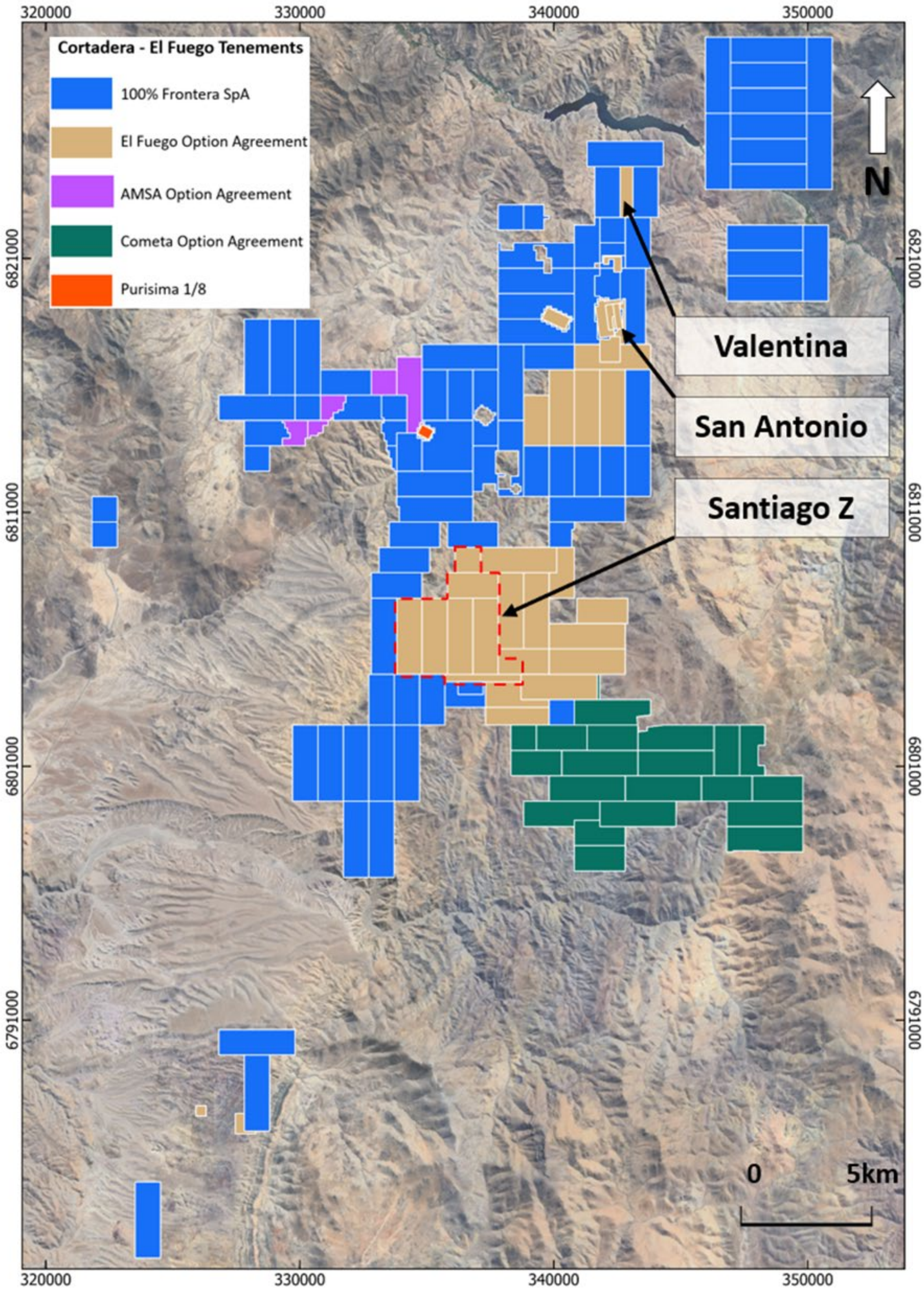
- ‘CORTADERA 1/40’ (374 ha). Mining tax (or cost per year to keep the mining right) US$2673. Such mining right 1/40 is owned 100% by Frontera (wholly owned by HCH).
- ‘Purísima 1/8 (1/2-5/6)’. (20 ha). Mining tax (or cost per year to keep the mining right) US$142. Such mining right is owned 100% by Frontera (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 Prokurika 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 below (Figure 1.3).

Figure 1.3 : Location of HCH's Cortadera Mining Rights and Surrounding Option Agreements under WGS84 (HCH, 2024)



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.

1.4 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.

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 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, 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.

1.5.3 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 seawater. SMEA currently holds a valid Maritime Concession Licence for the extraction of saltwater from a location south of the Huasco area.

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.

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

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.

1.6 Geology and Mineralisation

1.6.1 Productora and Alice

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.

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 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.

1.6.2 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.

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.

1.6.3 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.

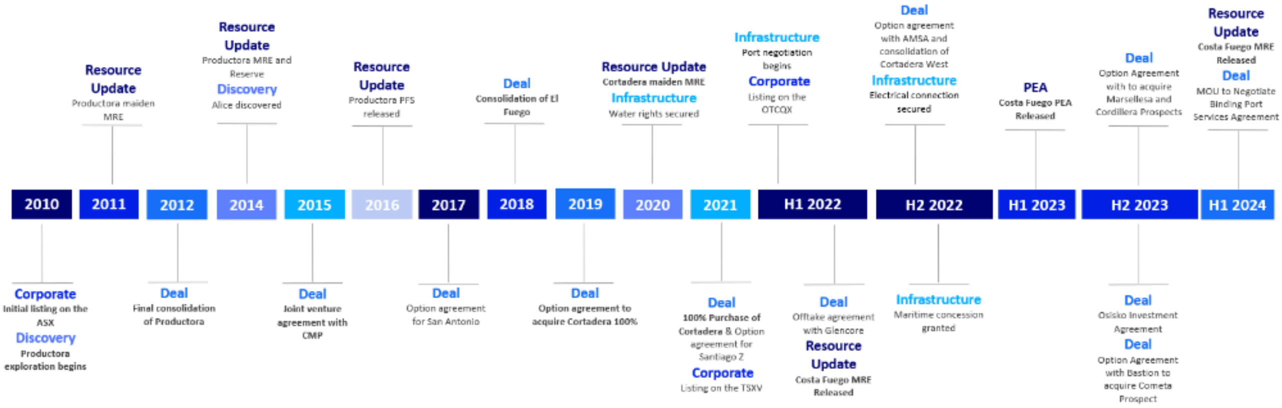
Mineralisation is focussed on the through-going San Antonio shear and associated fault zones (nominally less than 2 m width - striking between N30E and NS) and the cross-cutting mafic dykes. The intersection lineation between these structures is interpreted to plunge approximately 30° to the south and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

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 (HCH, 2024)



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, Remolin, 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.

1.7.3 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. Documentation regarding discovery and early exploration activities is limited for San Antonio.

1.8 Production History

1.8.1 Productora

Copper mining 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 ENAMI's Vallenar processing facility.

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 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.

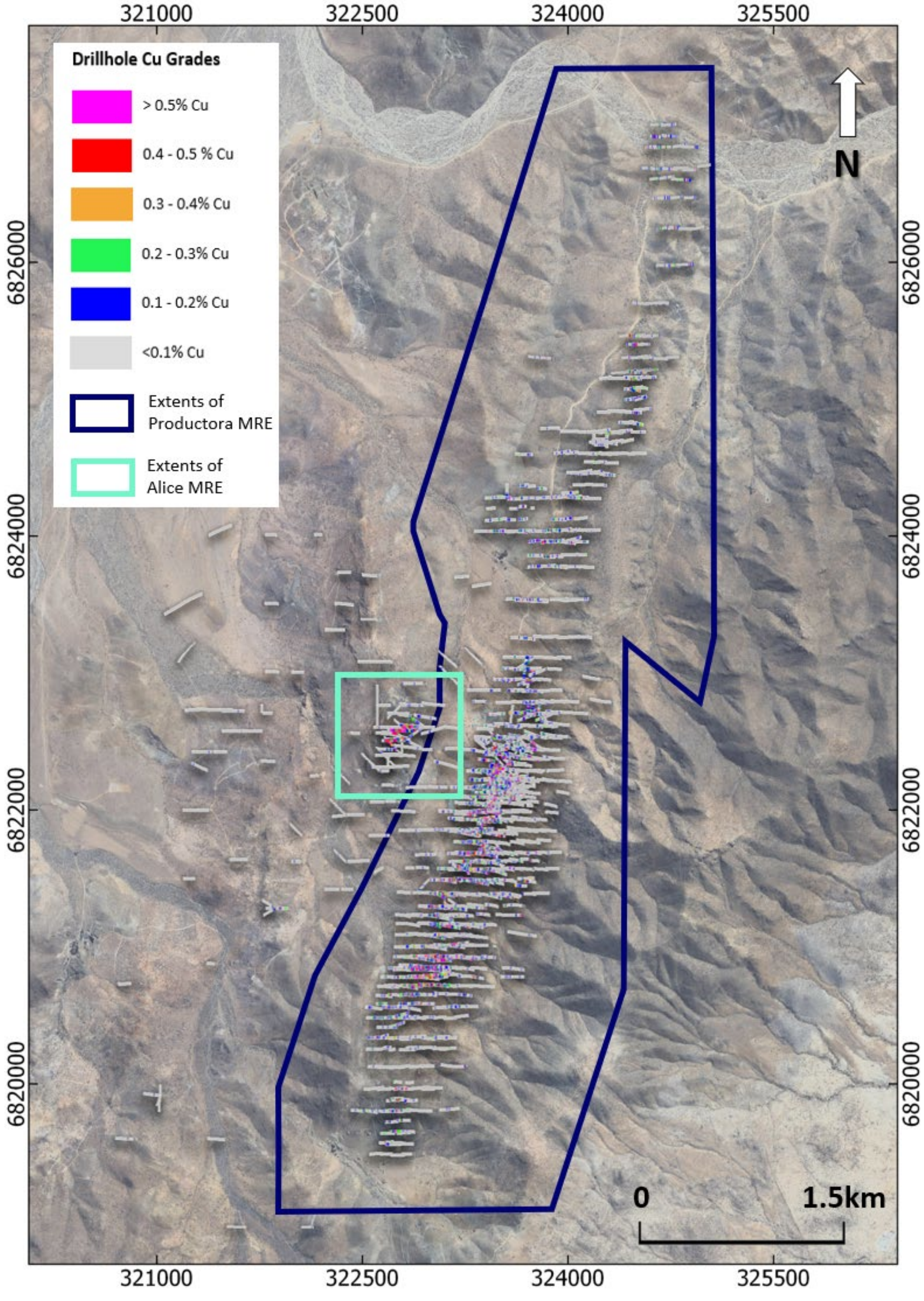
Change in drilling between the 2022 and 2024 Mineral Resources is included in Table 1.2 below.

Table 1.2 : Breakdown of the Change in Drilling Metres Between the 2022 and 2024 Mineral Resource Estimates									
Project	2022 Resource			2024 Resource			Change (2022 to 2024)		
	RC (m)	DD (m)	Total (m)	RC (m)	DD (m)	Total (m)	RC (m)	DD (m)	Total (m)
Productora	214 267	28 306	242 573	218 231	29 241	247 472	+3 964	+935	+4 899
Alice	15 845	1 388	17 203	17 156	1 802	18 928	+1 311	+414	+1 725
Cortadera	32 911	36 000	68 911	41 680	44 881	86 561	+8 769	+8 881	+17 650
San Antonio	4 922	0	4 922	6 931	495	7 426	+2 009	+495	+2 504

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



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.

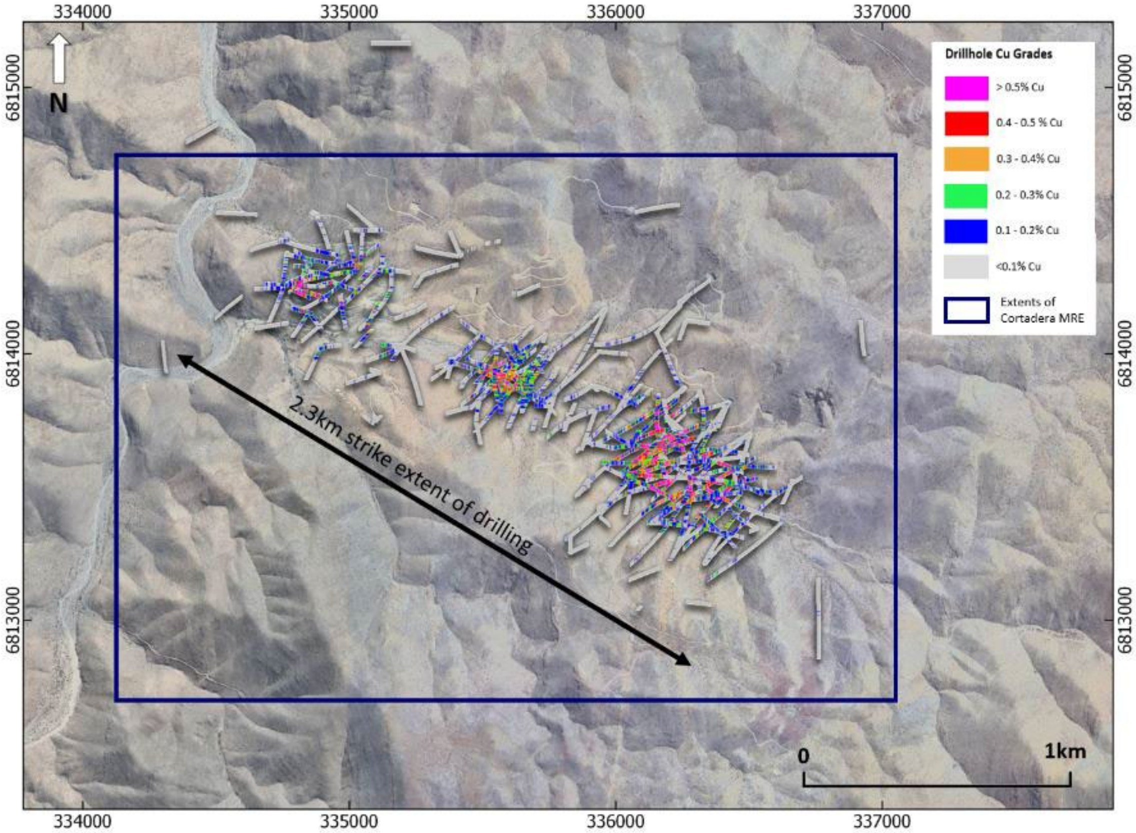
1.9.2 Cortadera

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 representivity 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



1.9.3 San Antonio

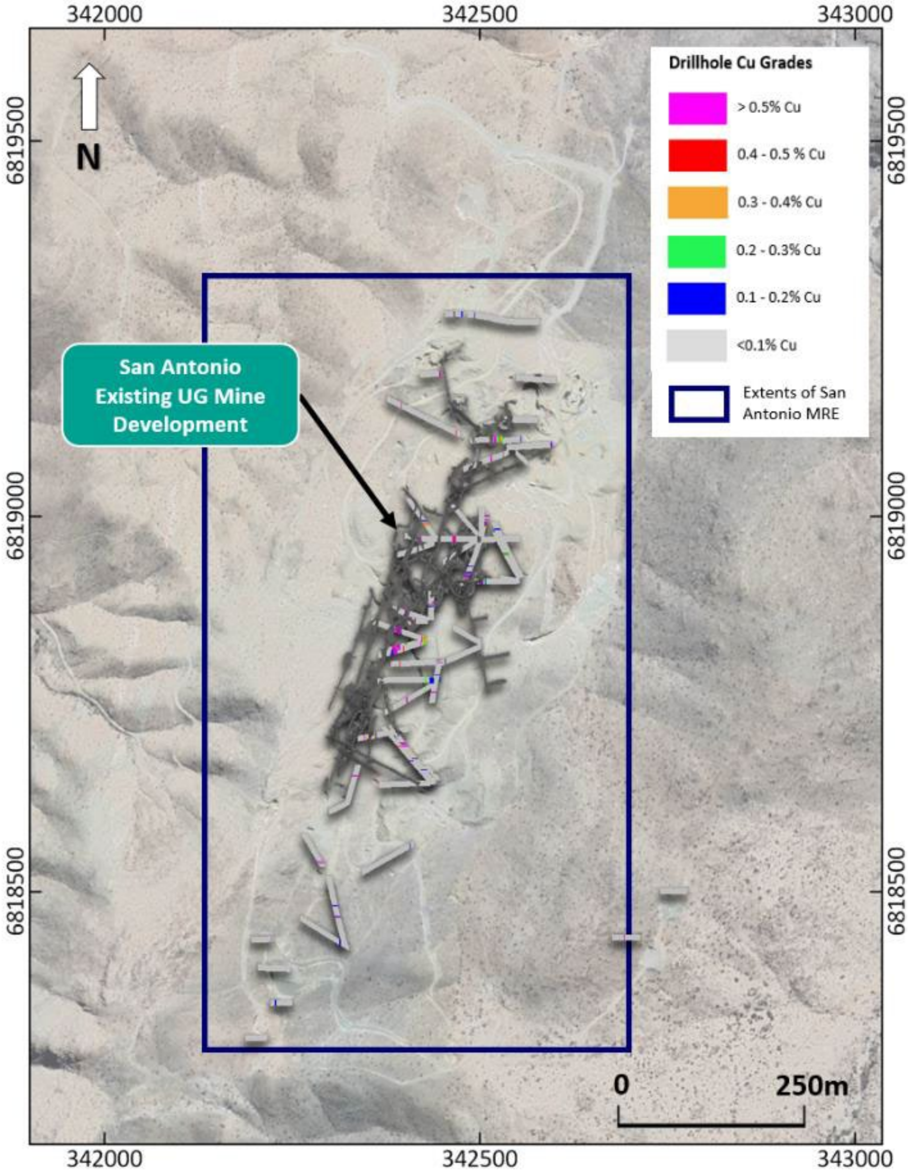
Since executing the San Antonio option agreement in November 2017, HCH has undertaken a range of exploration activities including field mapping, soil and rock sampling, underground mapping, underground chip sampling, an RC drill program, and a survey of accessible mined excavations to verify depletion 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 RC Drill Program Hole Locations Relative to Existing Underground Mine Development



1.10 Data Verification

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.

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 estimation of the Mineral Resources.

1.11 Metallurgical Testwork

Metallurgical diamond drilling was completed across Productora (21 drillholes), Alice (2 drillholes), Cortadera (6 drillholes) and San Antonio (3 drillholes) to provide appropriate coverage of the various mineralisation styles encountered across the Costa Fuego Project.

Metallurgical testwork was conducted at:

- ALS Laboratories in Perth Western Australia (flotation and comminution testwork)
- ALS Laboratories in Santiago, Chile (leaching and comminution testwork)
- Outotec in Perth, Australia (sulphide concentrate thickening and filtration)
- HydroGeoSense (oxide testwork) in the USA
- Auralia Metallurgy, Australia (flotation and comminution)
- Independent Metallurgical Operations Pty Ltd Laboratory, Perth (oxide testwork)
- Nova Mineralis (amenability testwork).

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-grinding circuit throughput, and metallurgical recoveries for each mining area.

The metallurgical testwork indicates the sulphide mill feed can be processed by conventional crushing, grinding and flotation technologies to recover copper, gold, and silver from the copper concentrate, and molybdenum into a separate concentrate.

Importantly, the copper concentrate produced from five locked-cycle tests completed for the Costa Fuego Project indicated very low arsenic in the fresh water washed concentrate. Negligible deleterious elements were reported in concentrate testwork, and it would be considered a high specification clean concentrate.

Oxide feed is amenable to heap leach extraction, with copper cathode produced through solvent extraction and electrowinning. Testwork completed by Nova Mineralis on seven low-grade sulphide bulk-sample composites from Productora and Cortadera indicated positive amenability for dump leach extraction, with copper recovery assumptions of 40% confirmed.

Average anticipated recoveries for sulphide and oxide material, broken down by mine area, are shown below in Table 1.3 and Table 1.4 respectively.

Table 1.3 : Average[2] Recoveries Applied to Sulphide Material for Each Deposit Area

Deposit	Average Recovery to Concentrate (%)				# Samples
	Copper	Gold	Silver	Molybdenum	
Productora	87	56	-	52	19
Alice	91	51	-	67	5
Cortadera Open Pit	77	44	27	50	19
Cortadera Block Cave	90	58	38	69	25
San Antonio	93	70	65	50	1
Average	**87**	**56**	**37**	**58**	

[2] Average for 'Recovery to Concentrate' weighted by proportion of copper metal production feed.

Table 1.4 : Average[3] Recoveries Applied to Oxide Material for Each Deposit Area

Deposit	Copper Recovery (%)	# Samples Bottle Roll	# Samples Column	% of Production Feed
Productora	56	22	5	80
Alice	46	3	0	8
Cortadera Open Pit	50	4	0	12
Average	55			

[3] Average for 'Copper Recovery %' weighted by proportion of copper metal production feed.

1.12 Mineral Resource Estimates

The Costa Fuego Mineral Resource Estimate (MRE) includes four MREs: Productora, Alice, Cortadera and San Antonio.

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

San Antonio has had two MREs completed; the maiden resource in 2022 which informed the 2023 PEA, and the current 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 work was conducted under the supervision of Elizabeth Haren, Director at Haren Consulting, Qualified Person, and the author responsible for the MRE in this Report.

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.

The combined Costa Fuego MRE can be seen in Table 1.5, with Figure 1.8, Figure 1.9, Figure 1.10, and Figure 1.11 illustrating the individual block models informing the 2024 MRE.

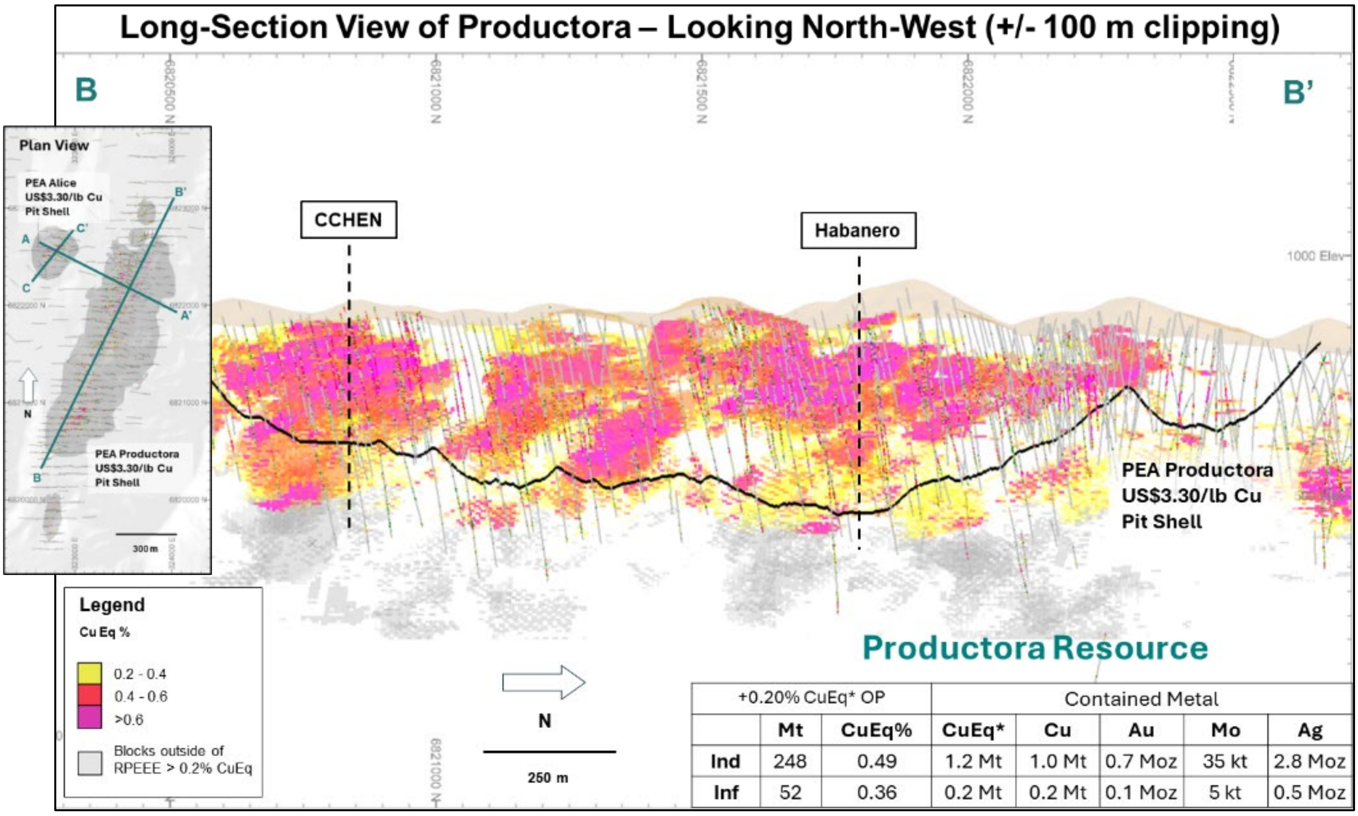
Table 1.5 : Costa Fuego Project Mineral Resource Summary – Reported by Classification (26 February 2024)

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 (November 29, 2019) 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.

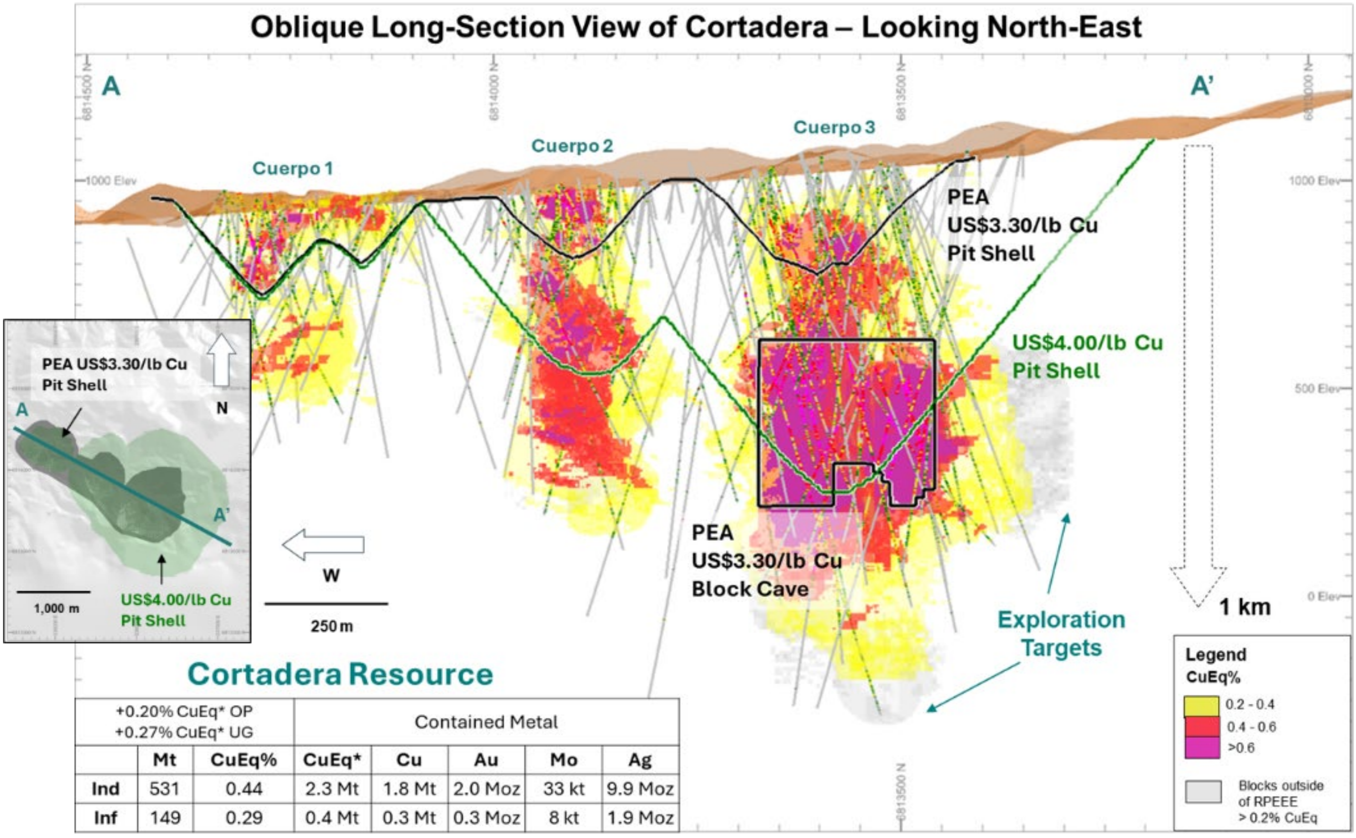
[2] The Productora deposit is 100% owned by Chilean incorporated company Sociedad Minera El Aguila SpA (SMEA). SMEA is a joint venture (JV) company – 80% owned by Sociedad Minera El Corazón Limitada (a 100% subsidiary of Hot Chili Limited), and 20% owned by Compañía Minera del Pacífico S.A (CMP).
[3] 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 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.
[6] All Mineral Resource Estimates 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 Productora, Alice and San Antonio.
[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 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.
[10] The effective date of the estimate of Mineral Resources is February 26th, 2024. Hot Chili 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.
[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 as disclosed in this Report. A detailed list of Project risks is included in 25.12.

Figure 1.8 : Long Section View of the Productora MRE Blocks, with the 2023 PEA Pit shape shown for reference, Model coloured by CuEq Grade, Drillholes shown within a window of +/- 100 m of the long-section plane.



+0.20% CuEq* OP			Contained Metal				
	Mt	CuEq%	CuEq*	Cu	Au	Mo	Ag
Ind	248	0.49	1.2 Mt	1.0 Mt	0.7 Moz	35 kt	2.8 Moz
Inf	52	0.36	0.2 Mt	0.2 Mt	0.1 Moz	5 kt	0.5 Moz

Figure 1.9 : Long Section View of the Cortadera MRE blocks, with the 2023 PEA Pit and UG shapes shown for reference, Model coloured by CuEq Grade, Drillholes shown within a window of +/- 100 m of the long-section plane.



+0.20% CuEq* OP +0.27% CuEq* UG			Contained Metal				
	Mt	CuEq%	CuEq*	Cu	Au	Mo	Ag
Ind	531	0.44	2.3 Mt	1.8 Mt	2.0 Moz	33 kt	9.9 Moz
Inf	149	0.29	0.4 Mt	0.3 Mt	0.3 Moz	8 kt	1.9 Moz

Figure 1.10 : Long Section View of the San Antonio MRE blocks, Coloured by CuEq Grade

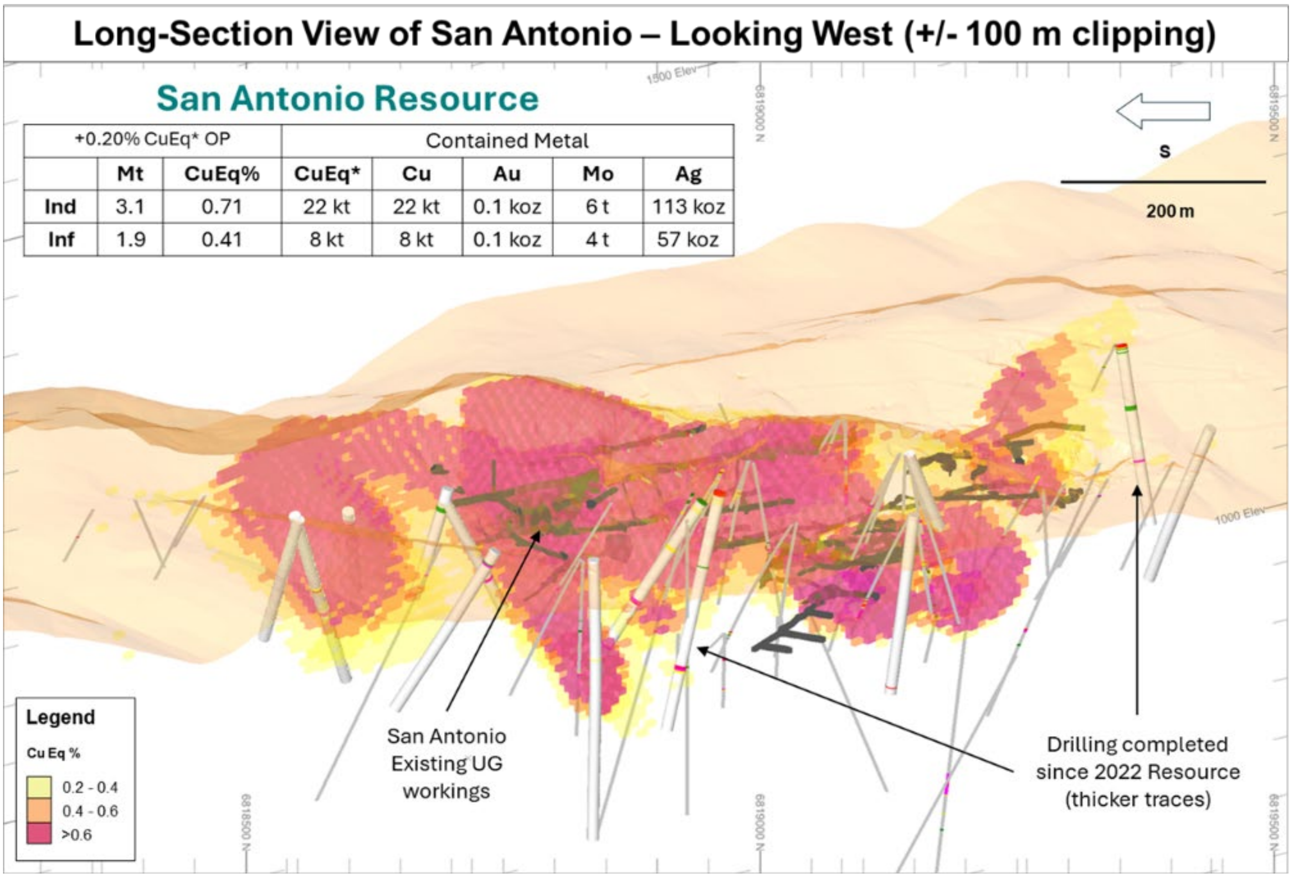
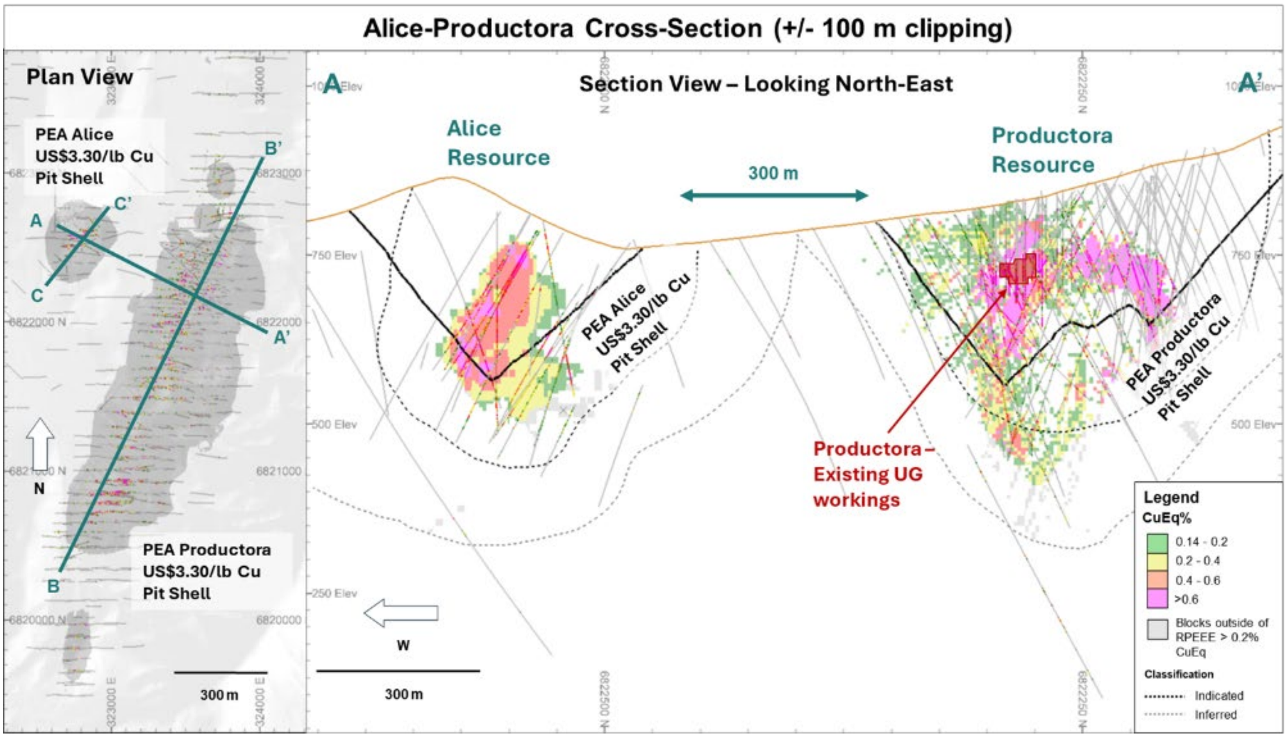


Figure 1.11 : Cross-section of Alice and Productora MREs, with the 2023 PEA Pit shape shown for reference, Model coloured by CuEq grade, Drillholes are shown within a window of +/- 100 m of the cross-section plane.



1.13 Mineral Reserve Estimation

As no current feasibility or pre-feasibility studies have been completed to the standard of NI 43-101, no Mineral Reserves have been estimated for the Costa Fuego Project on or before the submission date.

1.14 Mining Methods

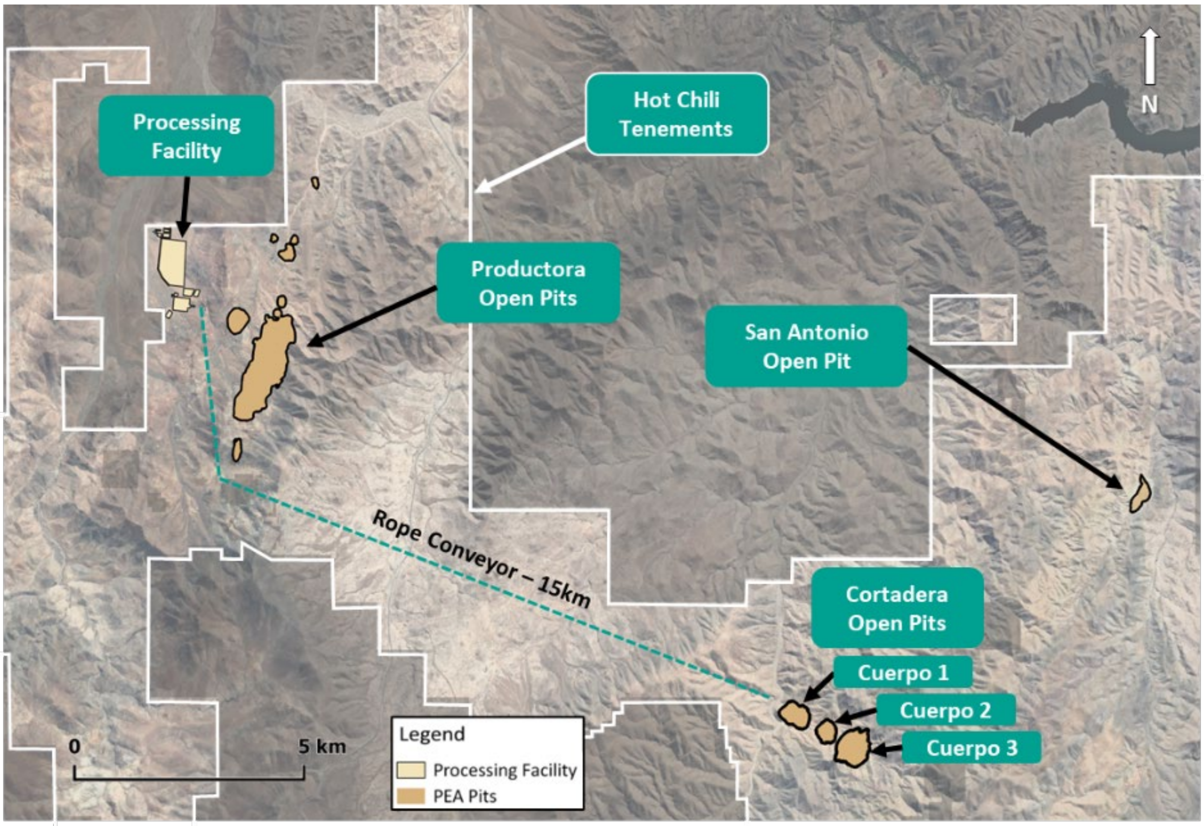
1.14.1 Setting

The Project includes three principal mining areas (Productora, Cortadera and San Antonio) with conventional open pit truck and shovel, and an underground block cave operation (Cortadera) planned. The study included five mining areas (four open pit operations and one underground block cave operation, shown in Figure 1.12).

The five mining areas are:

- Productora main open pit and several smaller oxide open pits (Productora)
- Alice open pit (Productora)
- Cortadera open pits (Cuerpo 1, 2 and 3) (Cortadera)
- Cuerpo 3 underground block cave (Cortadera)
- San Antonio open pit (San Antonio).

Figure 1.12 : Costa Fuego Mining Areas



Economic limits to mining and schedules were developed by ABGM Pty. Ltd. (ABGM) using the Costa Fuego Mineral Resource with an effective date of 31 March 2022. The 2024 MRE does not materially change the MRE, so the economic limits to mining and schedules remain valid.

The mine is 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 mining strategy selected for the Project is to operate the mine with a mining contractor, assuming owner activities only for supervising and mine support services.

The mine plan consists of 97% Indicated Resources, mining a total of 1098 Mt of material, comprising 334 Mt of processing plant feed, 37 Mt of oxide plant feed, 100 Mt of low-grade sulphide feed, and 627 Mt of waste over a 14-year mine production life and 16-year mine processing life, including stockpile reclamation (Table 1.6).

Table 1.6 : Production Feed[1] Breakdown		
Material	**Units**	**Total**
Oxide Leach	Mt	37
Cu Grade	%	0.42
LG Sulphide Leach	Mt	100
Cu Grade	%	0.14
Sulphide Concentrator	Mt	334
CuEq*	%	0.52
Cu Grade	%	0.44
Au Grade	g/t	0.12
Mo Grade	ppm	117
Ag Grade	g/t	0.45
Waste Rock Tonnes	Mt	627

[1] All figures are rounded, reported to appropriate significant figures. Production feed consists of 97% Indicated Resources, 3% Inferred Resource.
* The copper-equivalent (CuEq) grade was based on the combined processing feed (across all sources) and used long-term commodity prices of: Copper US$3.85/lb, Gold US$1750/oz, Molybdenum US$17/lb, and Silver US$21/oz; and estimated metallurgical recoveries for the production feed to the following processes: Concentrator (87% Cu, 56% Au, 37% Ag, 58% Mo), Oxide Leach (55% Cu only), and Low-grade Sulphide Leach (40% Cu only).

The current Life of Mine (LOM) focuses on mining higher grade, open-pit material early, with a resultant strip ratio for the open pit operations of 1.8:1. The operation will optimise production feed for a 20 Mt/a throughput sulphide flotation circuit (concentrator) and a 10 kt/a Solvent Extraction – Electrowinning (SX/EW) and associated leaching operations (oxide and low-grade sulphide material).

An elevated variable cut-off grade is applied throughout the mine life, with low-grade process feed material stockpiled and processed toward the end of the mining operations either as a low-grade sulphide leach or concentrator feed.

Indicated and Inferred Mineral resources were considered for processing. Within the PEA open pit shells and underground block cave, the classification breakdown of the processing feed material is 97% Indicated Mineral Resource and 3% Inferred Mineral Resource.

1.14.2 Open Pit Mining

Conceptual open pit (OP) mining of the near-surface mineralised material envisions a conventional drill-blast-load-haul method with 15 m 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.

The open pit slope angles were developed based on geotechnical logging of drill core plus rock quality evaluation and compressive strength testing of core samples. The amount of geotechnical data available is sufficient to support PEA-level input for the pit wall slopes.

The Productora deposit comprises the largest volume of open-pit mineralisation, with six pit pushbacks that are phased throughout the LOM. Pit-design and pit wall slopes are defined by data collected for the 2016 PFS.

Pre-stripping of the Productora main pit is completed in unison with project construction, resulting in pre-production stockpiles of 3.4 Mt of oxide production feed and 1.1 Mt of sulphide production feed following the two-year construction phase.

The Cortadera deposit consists of three separate pits, with the mining sequence commencing with the Cuerpo 1 pit, which has the largest volume of higher-grade, near-surface mineralisation.

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 211 Mt of processing plant feed, 37 Mt of oxide plant feed, 100 Mt of low-grade sulphide feed, and 622 Mt of waste will be mined, over a production life of 11 years.

LOM strip ratio for the open pit operation is 1.8:1 (including capitalised pre-stripping).

1.14.3 Underground Mining

Conceptual underground (UG) mining comprises an underground block cave, centred on the higher-grade core to the Cuerpo 3 mineralised body at Cortadera.

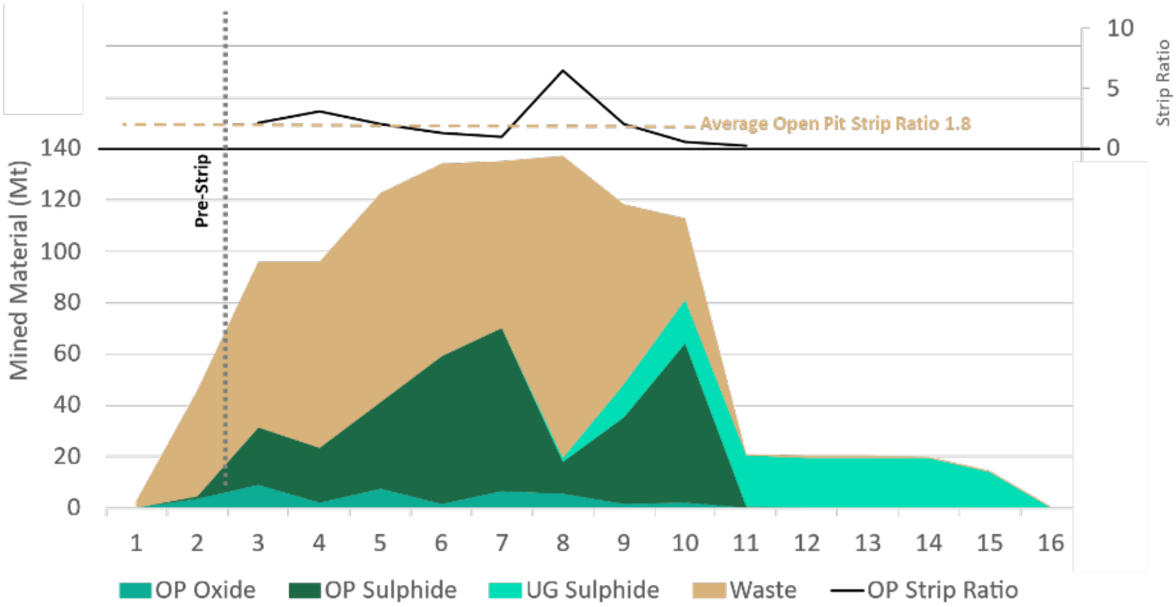
Indicative optimisations have been run to investigate the block cave potential, with the optimal block cave shape, footprint and geometry developed using Geovia's Footprint Finder software.

Cave development is planned to commence in Year 3, with a three-year lead time until commencement of revenue generation. Once opened, the block cave is expected to have a mine life of seven years, producing a total of 123 Mt of processing sulphide plant feed and 5 Mt of waste.

1.14.4 Combined Schedule

The following graph (Figure 1.13) depicts the combined (open pit and underground) mine production schedule for the 20 Mt/a throughput option. Testwork indicates variable LOM throughput ranges from 19.4 Mt/a to 22.6 Mt/a for a weighted average of 21.5 Mt/a.

Figure 1.13 : Mine Production Schedule Based on 21.5 Mt/a Processing Plant Average LOM Throughput



1.15 Recovery Methods

The Project will produce saleable flotation concentrates of copper and molybdenum and will also produce copper cathode from the heap leach (both oxide and sulphide). The Project has potential to generate other products, but costs and revenues associated with these products are not considered in the PEA.

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

The proposed processing facilities, located at Productora, are designed to process sulphide and oxide material and are suitable for all deposits within the Project. The sulphide concentrator is the centrepiece of the facility and is designed to process nominally 20 Mt/a of sulphide process feed. Concentrator capacity will vary by deposit based on comminution properties.

The deposits will also produce 3.3 Mt/a of oxide feed to be processed via a crushing-agglomeration-heap leach circuit coupled with a SX/EW plant producing up to 10 kt/a of copper cathode (increasing to 12 k/a in Year 8).

Process flowsheets for the sulphide flotation and leach circuits are show in Figure 1.14 and Figure 1.15 respectively.

Figure 1.14 : Sulphide Process Plant Flowsheet

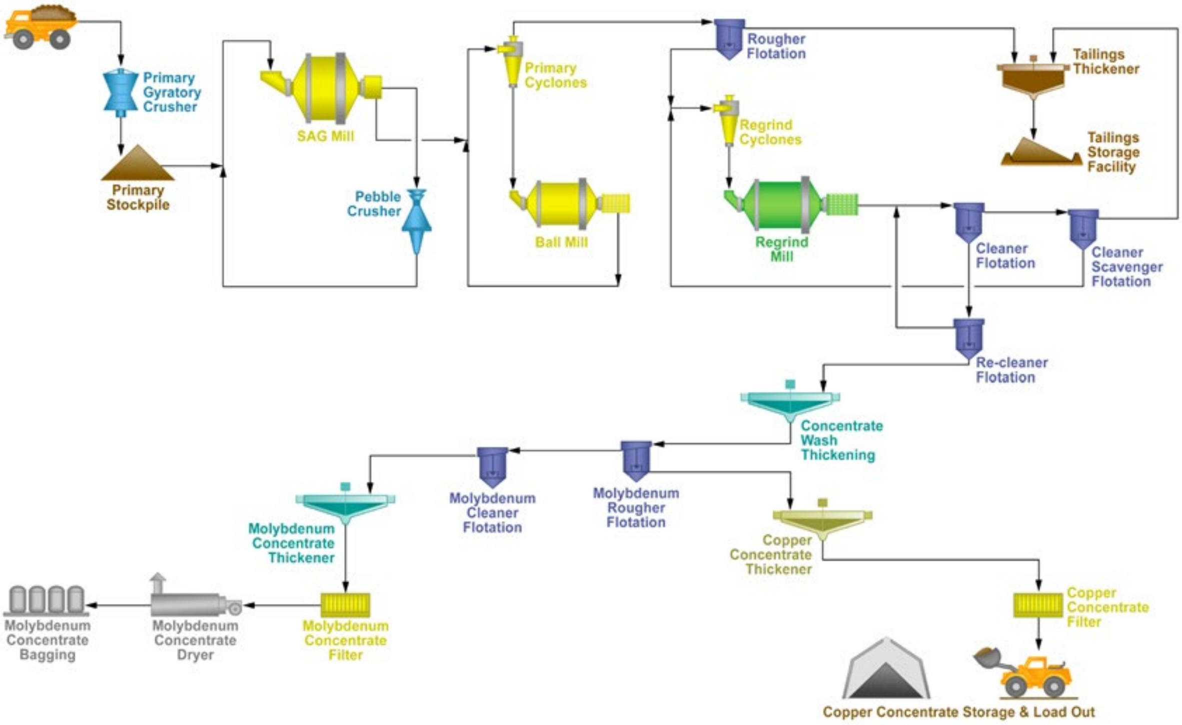
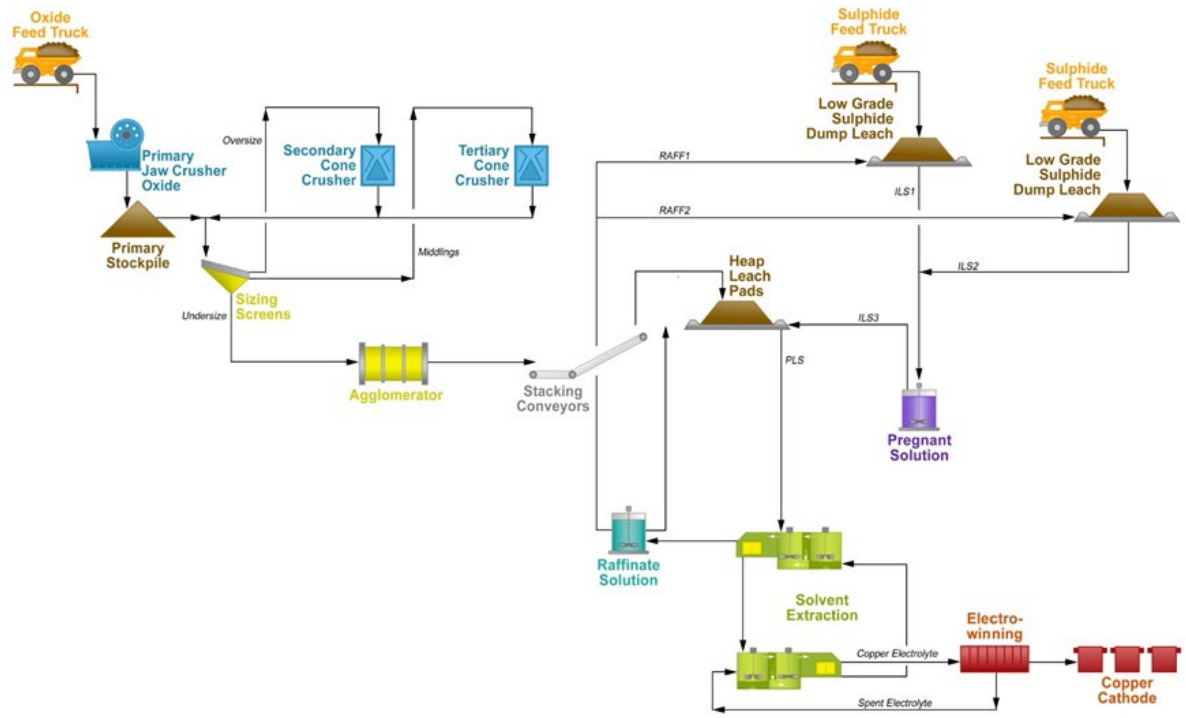


Figure 1.15 : Copper Oxide Process Plant Flowsheet



The Project has a processing ramp-up time of one year for both the concentrator and oxide heap leach with annual throughput for the sulphide flotation circuit averaging 21.5 Mt/a for the Project life (Table 1.7).

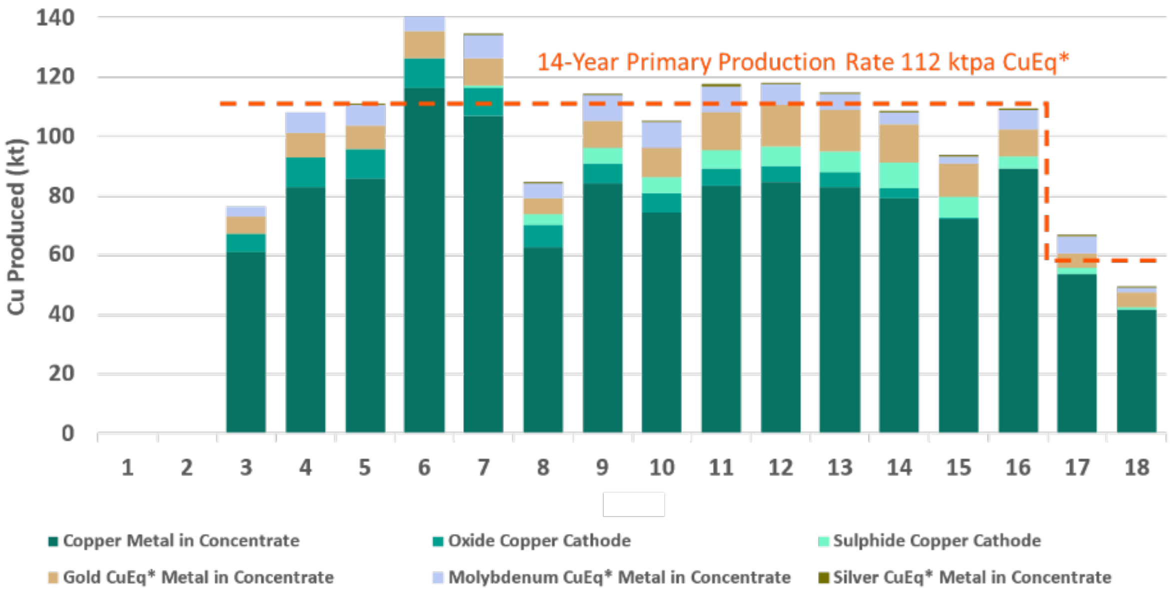
Table 1.7 : Average Processing Variable Throughput Rates by Mine Area

Deposit	Concentrator (Mt/a)	# Samples	% of Production Feed
Productora	22.3	27	46%
Alice	23.2	3	2%
Cortadera Open Pit	24.2	4	12%
Cortadera Block Cave	19.4	22	37%
San Antonio	19.4	1	3%
Average	**21.5**		

Sulphide material below variable mill cut-off grade is stockpiled to be processed via a low-grade sulphide dump leach. The low-grade sulphide leach option replaces tapering oxide production and helps maintain consistent copper metal production through the SX/EW plant to the end of mine-life.

Annual metal production across the three processing streams averages 95 kt Cu, 49 koz Au, 121 koz Ag and 3 294 klb Mo for the primary production period of the first 14 years. LOM annual metal production across the 16-year mine life averages 88 kt Cu, 45 koz Au, 121 koz Ag and 2 999 klb Mo. Figure 1.16 Figure 1.16 shows a breakdown of the yearly copper equivalent production over the life of the Project.

Figure 1.16 : Yearly Copper-Equivalent* Production Over Life-Of-Mine



1.16 Project Infrastructure

The proposed Project would be able to use existing infrastructure and services in the Vallenar/Huasco region. The township of Vallenar (with a population of approximately 52 000) is located 17 km from the processing facility and could provide accommodation, mining services, and logistical support to the Project. Major regional cities of La Serena and Copiapó are located 160 km south and 180 km north, respectively.

Key infrastructure of the Vallenar and Huasco valley region includes:

- Existing Las Losas port facility in Huasco bay near the city of Huasco
- Pan American Highway (sealed dual lane, located 5 km east of the proposed central processing facilities at Productora)
- Access roads from the Pan American Highway, and from Maitencillo for access to the Productora mine site and processing facility
- Main sealed road from Vallenar to Huasco for transportation of copper concentrate to the port facility at Los Losas
- Regional Vallenar airport (3 km south of Vallenar)
- A 220 kV electrical substation located at Maitencillo, connected to the Chilean electrical grid and 23 kV power supply in Huasco.

Hot Chili's subsidiary company, SMEA, owns easements to establish critical water and electrical infrastructure to the Project as well as surface rights to develop the mine footprint.

The proposed plant site is at Productora to the west of both the Alice and Productora pits. The Run-of-Mine (ROM) pad and primary crusher is to be located adjacent to the main haul road.

The following infrastructure will be established as part of the Project:

- Water for the Project will be supplied from a new seawater pipeline and associated pumping stations with the intake from the coast at Huasco to storage ponds at the process plant
- A new 220 kV overhead line will supply the Project from the Maitencillo substation to a new switchyard and power distribution within the Project
- Roads, buildings, fuel storage, waste storage, communications, and security.

Within the mining infrastructure areas, the Mining Contractor will provide the majority of the open pit infrastructure, however the Project has allowed for the sulphide processing feed from Cortadera to undergo primary crushing at Cortadera and then be conveyed approximately 15 km via rope conveyor to the main processing plant at Productora. In addition, as part of the expansion the Project has allowed for underground crushing.

The Tailings Storage Facility (TSF) would be located 5 km north-east of Productora processing facility and in conjunction with Productora Main pit and the Alice pit and has a planned capacity more than sufficient to accommodate the proposed tailings output.

Water, power, and road access would be established between the Productora, Cortadera and San Antonio deposits.

Figure 1.17 and Figure 1.18 detail planned and existing infrastructure for the Costa Fuego Project.

Figure 1.17 : Costa Fuego Planned and Existing Regional Infrastructure

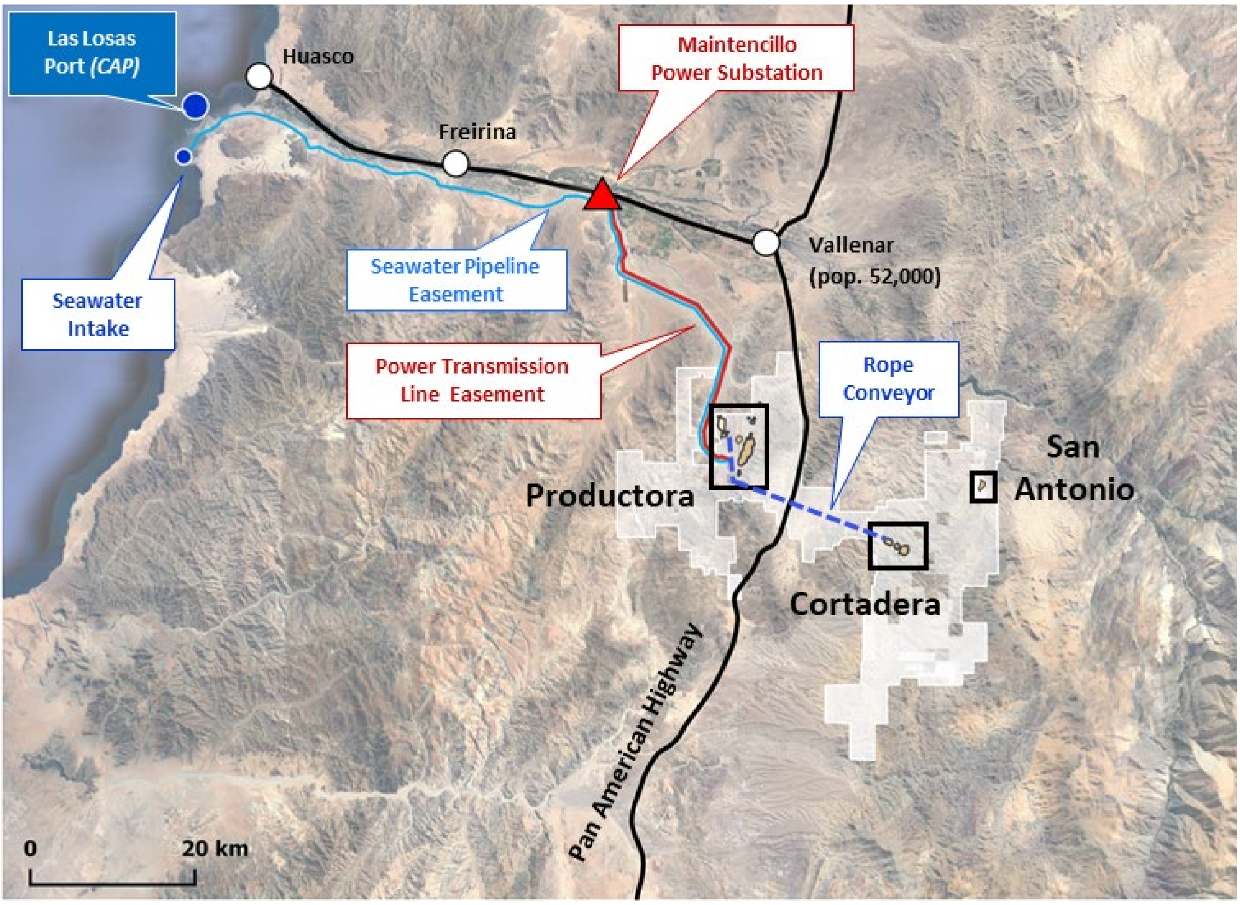
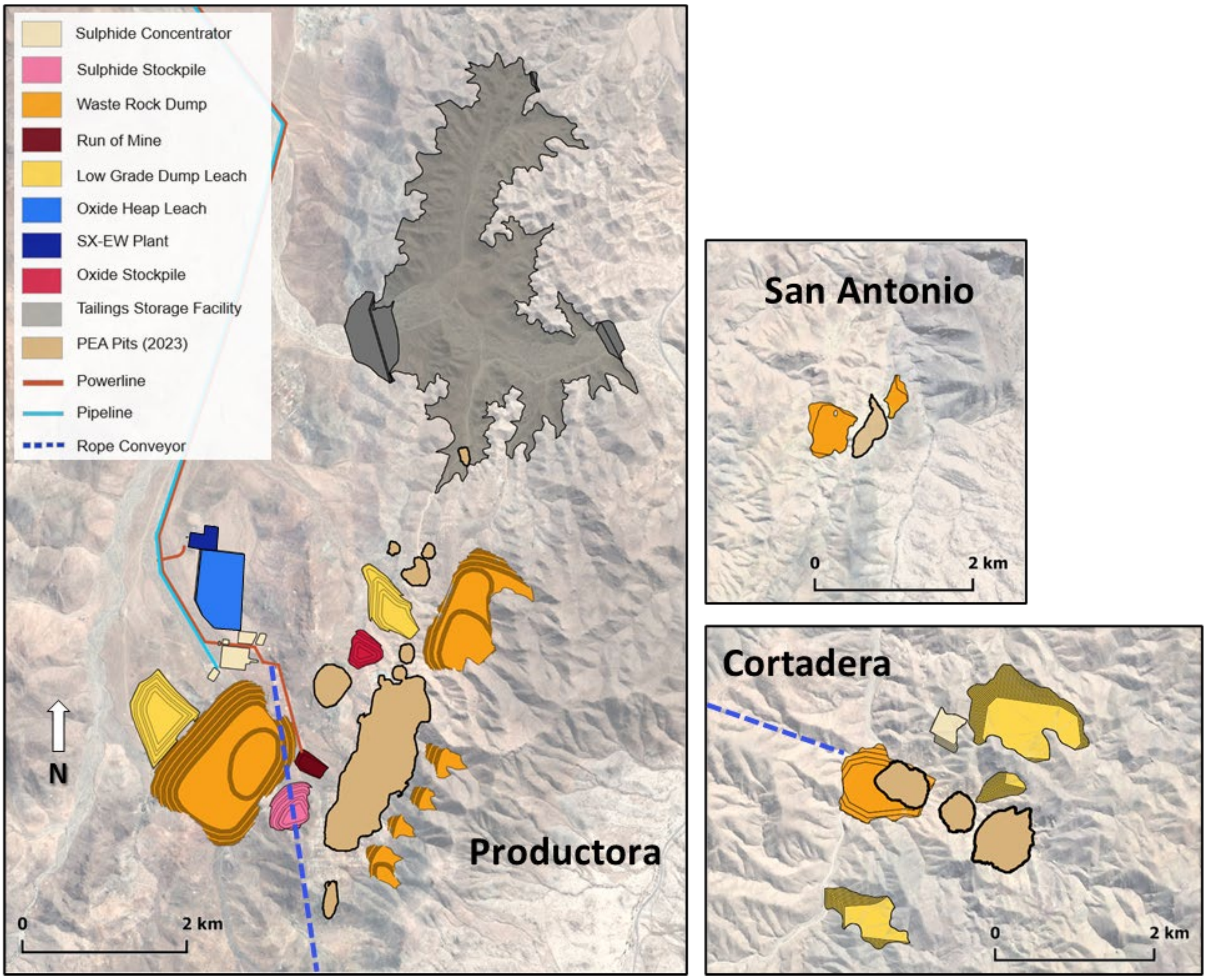


Figure 1.18 : Costa Fuego Planned and Existing Area Infrastructure



1.17 Environmental, Permitting and Social Considerations

1.17.1 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. In addition, diverse environmental information was generated from HCH exploration projects. For seasonal baselines (for instance flora and fauna), four campaigns have been organised to cover passive and active seasons' changes. 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 mine areas, so the baseline information on flora, fauna, archaeology, noise and vibration, and landscape in these areas gained additional information.

Additional flora information has been collected for obtaining the permits to disturb vegetation with the Forest Authority. This means that there is a thorough understanding of the biological characterisation of the mine area, which gives a solid basis for the assessment of a key environmental component – flora – in the Atacama region.

1.17.2 Tailings Storage Facility

Studies evaluating wind erosion control measures from the basin and walls will be required. This will consider the dried condition of tailings and its material consolidation because of water evaporation.

The tailings surface could be covered by a suitable cover of local borrow and waste rock to prevent erosion of the tailings surface and reduce infiltration.

Studies evaluating seepage control measures from the basin will be required. Two hydrogeological monitoring wells have been constructed, one upstream and one downstream of the TSF embankments. A monthly monitoring program gathers groundwater data, including water level and water quality, with more than 30 elements tested.

1.17.3 Water Management

The operation will utilise raw seawater for processing, with water extraction (maritime concession) and coastal land access rights already secured. The use of seawater reduces the energy intensity of the Project (no large-scale desalination plant required) and preserves the limited groundwater resources available in the region.

The hydrogeological baseline study is being prepared for the PFS to address water management at the Project. A hydrogeological bore network has been in place at Productora gathering information since 2012, and development of a similar network at Cortadera and additional bore holes at Productora is complete, with monitoring ongoing.

1.17.4 Closure and Reclamation Planning

All mining projects in Chile which have a production capacity above 10 000 t/month are required to present an official closure plan to be approved by the Mining Authority. Any associated infrastructure affected by a closure plan also needs to be developed to scoping engineering level.

The closure plan will specify the technical measures and activities at Costa Fuego designed to prevent, minimise, or control risks that may affect people and the environment. In Chile, the law governing the closure of mining sites (Law 20.551 and DS 41) does not require mining operators to rehabilitate the site where the mine infrastructure and facilities used to be located. However, a rehabilitation plan will be assessed during closure discussions.

1.17.5 Permitting Considerations

The Project has an EIA study being developed; some figures related with the EIA are outlined below.

- Around 11 000 ha covered with baselines studies during four campaigns
- More than eight volumes of information generated
- More than 50 specialists involved.

1.17.6 Social Considerations

Over the past decade the Company has forged strong community engagement (including with indigenous communities) and contributed positively to the region.

The Company has and will continue to:

- Provide ongoing support for the local regional communities through the Company's mental health program
- Recruit locally, wherever possible, to provide employment and training opportunities
- Preferentially procure local goods and services
- Provide ongoing support for two orphanages in Freirina and Vallenar
- Provide fresh water to local families in Agua Amarga for irrigation
- The Company is recognised as a leader in ongoing social support programs within the Vallenar and Huasco Valley region.

The key aspect of engagement at this stage of the Project is to make sure the EIA considers the opinion of stakeholders through a robust consultation process. Any issue that is indirectly related to the EIA will be addressed once the baselines, the impact assessment and the mitigation measures have been discussed with stakeholders.

Once the anticipated consultation activities with the relevant stakeholders are carried out, the Project will be well positioned for the EIA official and mandatory engagement process, which follows EIA submission.

1.18 Markets and Contracts

A rapidly accelerating energy transition supporting a faster uptake of copper-intensive industries like electric vehicles and renewable energy generation, would increase copper demand globally. The Project is well situated geographically to serve both Asian and North American markets, where China is the leading consumer of copper. Other Asian and American markets are also significant consumers of copper.

Long-term metals prices assumed for the economic analysis are shown in Table 1.8. The long-term price guidance for copper, molybdenum, gold, and silver are what Wood considers to be an industry consensus on the forecast and are supported by Wood's quarterly guidance for long-term metal prices. Exchange rate assumptions are shown in Table 1.11.

Table 1.8 : Metals Prices

Metal	Unit	Price (US$)
Copper	US$/lb	3.85
Molybdenum	US$/lb	17.00
Gold	US$/oz t	1 750.00
Silver	US$/oz t	21.00

HCH has negotiated an offtake agreement with Glencore for early copper concentrate production from the Project. The Glencore offtake agreement covers 60% of copper concentrate from the Project for a period of eight years from commercial production and was negotiated on arms-length commercially competitive benchmark terms.

1.19 Capital Cost Estimates

The capital cost estimate meets the requirements consistent with AACE® International cost estimating guidelines for a Class 4 estimate. The estimate accuracy range of ±25% is defined by the level of project definition, amount of engineering inputs, the time available to prepare the estimate and the amount of project cost data available.

The capital cost estimate is based on Q4 2022 assumptions, in US dollars.

The estimate has been compiled by Wood with inputs from consultants for their responsible scope of work:

- ABGM Plus: Mine production, mine footprint and decline development, ventilation, dewatering and underground infrastructure, underground crushing and materials handling
- Knight Piésold: Heap leach, tailings storage, surface water management
- Wood: Process plants, surface infrastructure, tailings pipelines, power supply and distribution, services and utilities and concentrate pipelines based on the 2016 Study by Mintrex
- HCH: Low grade leach, Owners costs
- All: Indirect costs, EPCM
- Wood: Contingency allowance.

The indicative cost increase of the estimate from the study base date of 2016 is 20.4% which represents an average escalation rate of 2.69% per annum.

The capital cost estimate for the Project is summarised in Table 1.9

Table 1.9 : Capital Cost Summary Update – Q4 2022	
	Total (US$ M)
Stage 1 Capital	
Area 01 Bulk Earthworks and Drainage	46.37
Area 02 Site Services	3.13
Area 03 Sulphide Process	332.66
Area 04 Oxide Process	83.77
Area 05 Molybdenum Process	13.03
Area 07 Infrastructure (Excluding TSF)	181.82
Area 07 Infrastructure – TSF Only	31.57
Area 09 Mining (Excluding Prestrip)	32.97
Total Direct Construction Costs	**725.34**
Area 06 EPCM Construction Stage 1 Costs	118.2
Area 08 Owners Costs	102.97
Total Indirect Construction Costs	**220.13**
Total Construction Project Costs – Stage 1	**945.47**
Capitalised Expenses	
Area 10 Preproduction Mining Cost	99.65
Total Capitalised Expenses	99.65
Total Pre-Start Capex	**1 045.11**
Expansion CAPEX	
Plant Upgrade	-
Stage 2 - Cortadera Infrastructure	70.99
Stage 2 - Rope Conveyor	165.37
Stage 3 - Block Cave Development	405.90
Stage 3 - Block Cave Infrastructure	66.05
EPCM for Expansion	
Total Expansion CAPEX	**708.31**
Total Project Costs	**1 753.42**
Sustaining Capital	
Tailings	59.1
Sulphide Process	183.6
Molybdenum Process	1.8
Oxide Process	44.4
LG Leach Process	47.4
Waste Stripping	629.6
Total Sustaining Capex	**965.94**
Closure	48.1
Total Capital Expenditure	**2 768.77.**
Salvage Value	-48.01
Life of Project Capex	**2 719.77**

1.20 Operating Cost Estimates

Operating costs were estimated for mining, sulphide and oxide plant processing, administration, concentrate transport and seawater supply areas as part of the 2016 study. These costs have been escalated by 20.4% and are presented in US dollars as of the second quarter 2022 (2Q22) to an accuracy level of +/- 40%.

Exchange rate assumptions (as provided by HCH) used in the estimate are as follows:

- Australian dollar (A$) to US dollar (US$) 0.75000
- Canadian dollar (C$) to US$ 0.80000
- Euro (EUR) to US$ 1.16000
- Chilean peso (CLP) to US$ 0.00125.

Financial analysis utilised a power cost of $0.065/kWh based on current power supply negotiations and a diesel cost of $0.73/L in line with independent recommendations and current long-term forecasts, respectively.

The estimate has been compiled by Wood with inputs from consultants for their responsible scope of work:

- ABGM : Mining, rehandling
- Knight Piésold: Heap leach, tailings storage, surface water management
- Wood: Process plants, surface infrastructure, tailings pipelines, power supply and distribution, services and utilities and concentrate pipelines based on the 2016 Study by Mintrex
- Wood: General and Administration (G&A) based on escalation of the 2016 Study by Mintrex with inputs from HCH
- HCH: Low grade sulphide leach

The open pit mining operating cost estimate is based on a mining contractor quote. The quote excludes costs for minor items such as contractor supervision, blast hole sampling, RC drilling for process feed control, seawater supply and pit dewatering. For these activities not included in the mining contractor scope, HCH prepared the operating cost estimate. A 2023 PEA level benchmark operating cost estimate was used for the block cave operation.

Average Project operating costs are summarised in Table 1.10.

Table 1.10 : Operating Costs Summary		
Operating Costs	**Unit**	**Life of Mine**
Mining (Average)	US$/t mined	2.87
Open Pit	US$/t mined	2.21
Underground	US$/t mined	6.55
Processing		
Sulphide Concentrator – Cu/Au/Ag Concentrate	US$/t Process Feed	6.04
Sulphide Concentrator – Mo Concentrate	US$/t Mo in Conc	0.56
Sulphide Leach – Front End Processing	US$/t	1.03
Sulphide Leach – Back End Processing	US$/lb Cu	0.26
Oxide Leach – Front End Processing	US$/t	4.62
Oxide Leach – Back End Processing	US$/lb Cu	0.26
G&A	US$M/quarter	3.28

1.21 Economic Analysis

The results of the economic analysis in the 2023 PEA, which were carried forward into the current Report on the basis that the assumptions therein have not materially changed, 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.

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.

Using an 8% discount rate, the Project delivers a base-case, post-tax NPV of US$1.10 Billion and an Internal Rate of Return (IRR) of 21% (based on metal price assumptions of US$3.85/lb copper (Cu), US$1750/oz gold (Au), US$21/oz silver (Ag), and US$17/lb molybdenum (Mo). On a pre-tax basis, the Project delivers a base-case NPV of US$1.54 Billion and an IRR of 24%.

Infrastructure layout was optimised to reduce capital cost and improve operational efficiencies, processing a total of 471 Mt from a combined open pit and underground operation for 95 kt Cu and 49 koz Au annually during the first 14 years of a 16-year mine-life.

Three processing streams (conventional concentrator, oxide heap leach and low-grade sulphide leach) will generate a total of 1.41 Mt Cu and 718 koz Au over a 16-year processing life-of-mine (LOM).

The PEA financial model generates total life-of-mine revenue US$13.52 Billion and post-tax free cashflow of US$3.28 Billion.

Average LOM Total Cash Cost1 of US$1.43/lb Cu (including royalties and net of by-product credits) and All-In Sustaining Cost1 (AISC) of US$1.74/lb Cu is estimated after including by-product credits (Au, Mo, Ag).

Start-up and expansion capital costs are estimated at $1.05 Billion and $708 Million, respectively, with LOM sustaining capital costs (including reclamation and closure) estimated at $1.01 Billion.

Payback of start-up capital is expected 3.5 years after commencement of production, with post-tax cashflows funding expansionary capital projects, including the development of underground infrastructure and construction of a rope-conveyor to transport Cortadera sulphide mill feed to the processing facility at Productora.

The Project economics are strongly leveraged to further resource growth and copper price appreciation.

Preliminary outcomes indicate strong economics for Costa Fuego as a combined open pit and underground mining hub utilising centralised processing for a conventional 20 Mt/a sulphide plant and a 4.5 Mt/a SX/EW plant, producing both copper concentrate and cathode as well as significant revenue streams from by-product credits of gold, silver, and molybdenum. These preliminary outcomes are sufficient to recommend advancing the Project to completion of a preliminary feasibility study.

Table 1.11 shows the base exchange rate for the Project.

Table 1.11 : Exchange Rates

Currency	Exchange Rate
US$/AU	1.33
US$/CLP	800

Table 1.12 shows a summary of the financial analysis results. Undiscounted Cashflow and Production profiles over the LOM are shown in Figure 1.19 and Figure 1.20.

Table 1.12 : Summary of the Financial Analysis				
Project Metric	**Units**		**Value**	
Financial Measures				
Pre-Tax	Cu US$3.85/lb	NPV8%	US$M	1 540
		IRR	%	24
Post-Tax	Cu US$3.85/lb	NPV8%	US$M	1 100
		IRR	%	21
Payback period (from start of operations)			years	3.5
Open Pit Strip Ratio			W/P	1.8
NPV/Capex			Ratio	1.1
Capital Costs				
Total Pre-production Capital Expenditure			US$M	1 046
Expansion			US$M	708
Sustaining			US$M	1 014
Total			US$M	2 768
Operating Costs				
C1			$/lb Cu	1.33
Total Cash Cost (net by-products and including royalties)			$/lb Cu	1.43
All-in-Sustaining Cost			$/lb Cu	1.74
All-In Cost LOM			$/lb Cu	2.31
Mine Life and Metal Production				
Primary Mine Production Including Ramp-up			years	14
Mine Life (Life of Mine Processing)			years	16
Primary Mine Production – Average Annual Copper Equivalent Metal			kt	112
Primary Mine Production – Average Annual Copper Metal			kt	95
Primary Mine Production - Average Annual Gold Metal			koz	49

Figure 1.19 : Undiscounted Cashflow over Life-of-Mine

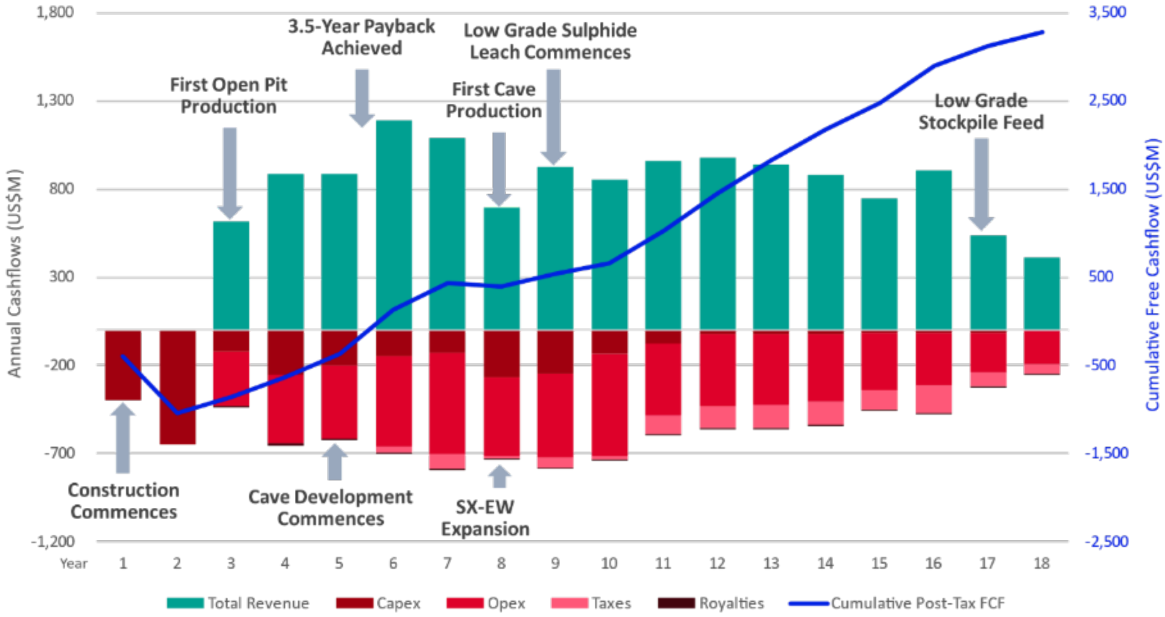
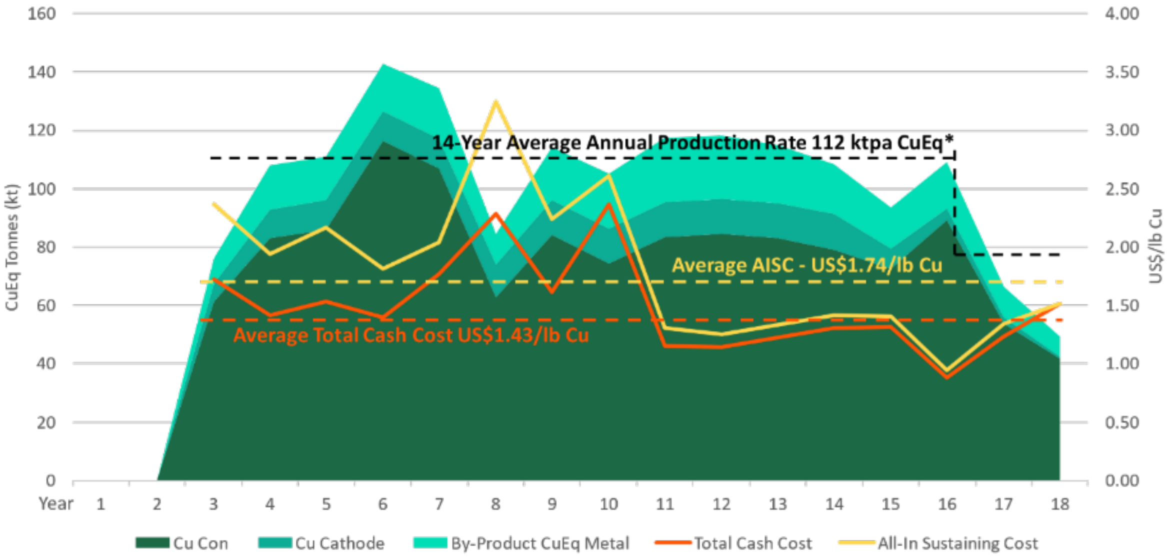


Figure 1.20 : Cost and Production Profile over Life-of-Mine



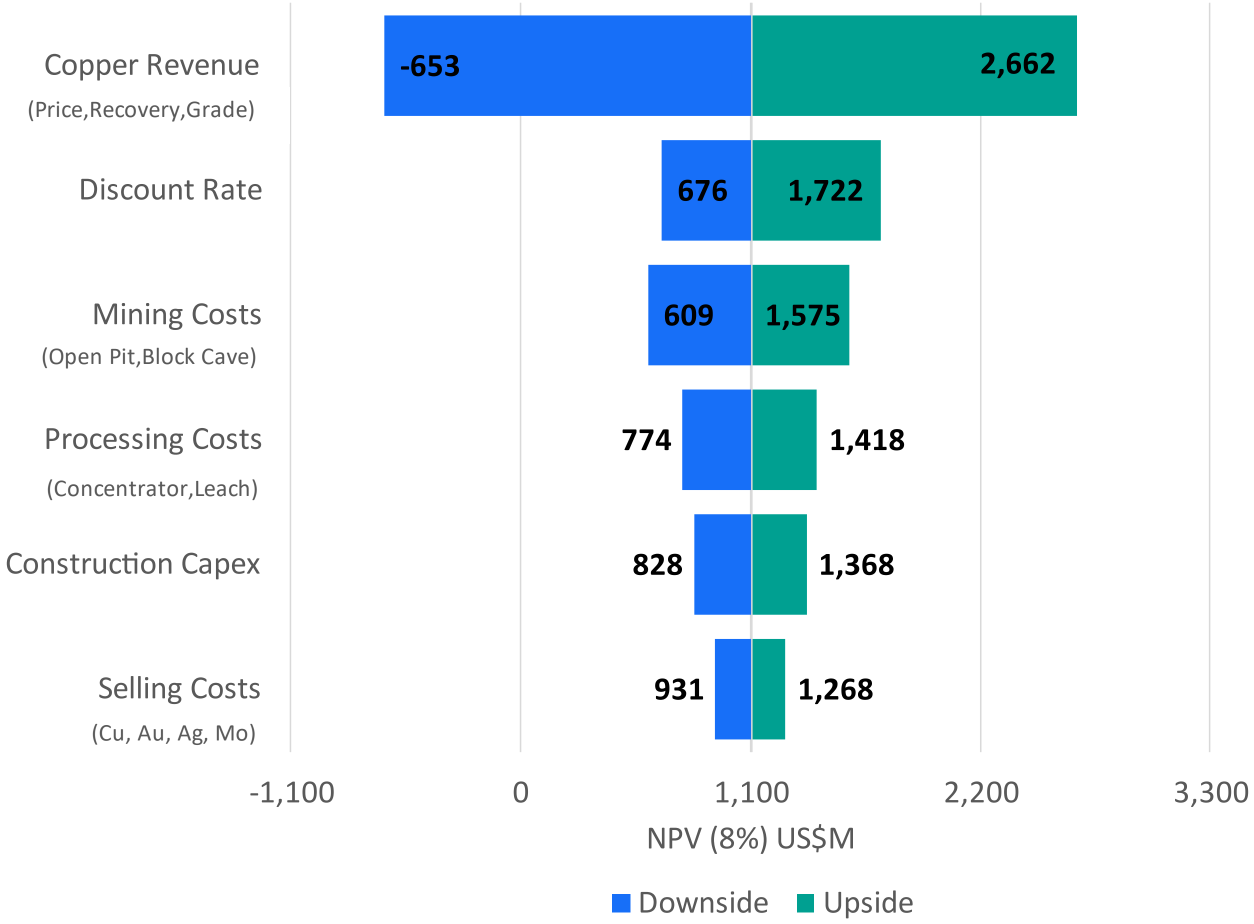
1.22 Sensitivity Analysis

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 tornado chart in Figure 1.21 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 1.21 : 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 1.22 illustrates the Project's post-tax NPV8 and IRR at a range of copper prices whilst Table 1.13 shows financial metrics at lower and upper copper price ranges (±10%).

Figure 1.22 : Project Post-Tax NPV(8%) and IRR as a Function of Copper Price

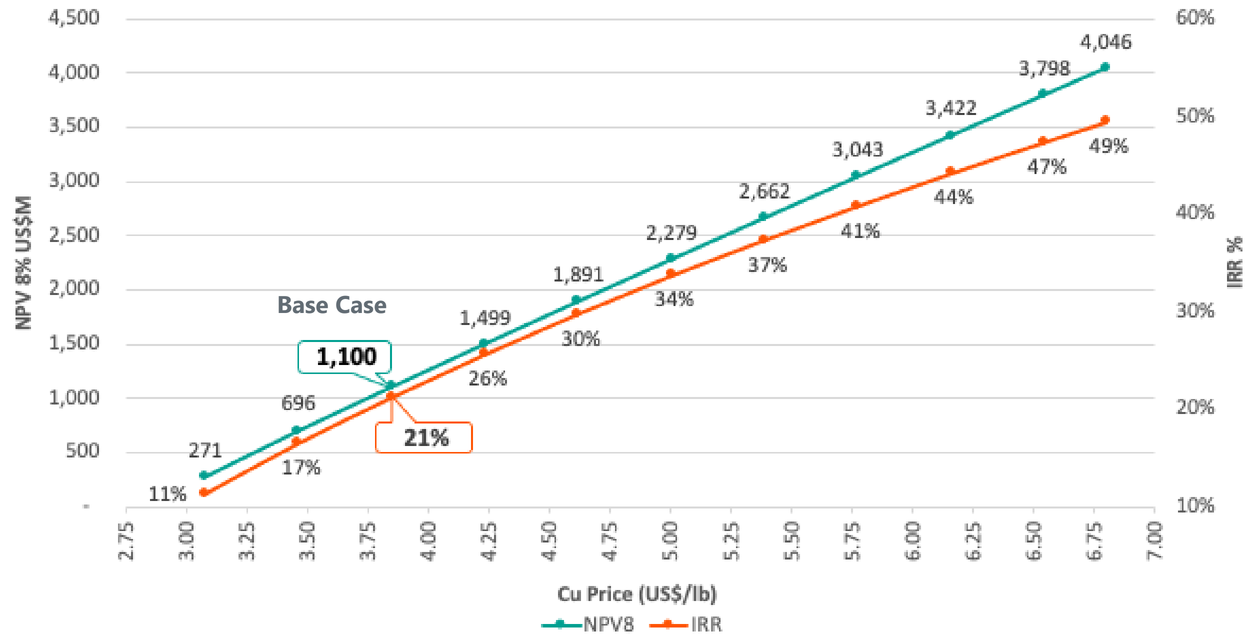


Table 1.13 : Copper Price Ranges – Lower-, Base-, and Upper-Case Scenarios					
Project Metric		**Units**	**Copper Price**		
			Lower (-10%) (US$3.50/lb)	**Base (US$3.85/lb)**	**Upper (+10%) (US$4.20/lb)**
Pre-Tax	NPV8%	US$M	1 040	1 540	2 030
	IRR	%	19%	24%	29%
Post-Tax	NPV8%	US$M	731	1 100	1 460
	IRR	%	17%	21%	25%
Annual Average EBITDA		US$M	384	445	506
Annual Average Free Cash Flow		US$M	161	205	250
Payback Period (From First Production)		yr	4.25	3.50	3.25
Post-Tax $NPV_{8\%}$ /Start-up Capital			0.7	1.1	1.4

1.23 Risk and Opportunity

The following risks and opportunities associated with development of the Project have been identified by the Qualified Persons.

During the pre-feasibility study phase, several risks will need to be investigated further and possibly reduced or eliminated. Similarly, further investigation and evaluation of opportunities may allow their incorporation in the Project.

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 in a risk review in 2022 and 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
- 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
- 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.

No additional material risks have been identified since the 2022 risk review. A description of the main risks and opportunities for this Project is presented in Chapter 25.12 of this Report.

1.24 Interpretation and Conclusion

The 2024 Costa Fuego MRE update increased the proportion of total equivalent copper metal in Indicated classification to 85%, following focused development drilling (metallurgical and geotechnical) designed to support a Pre-Feasibility Study (PFS) and targeted exploration and resource extension drill programs. The expansion of the Costa Fuego Indicated Resources, without material impact to reported metal grades, increases confidence in the reliability of the MRE and its ability to inform further studies on the Project.

The results of the 2023 PEA and the current Report indicate that the Project demonstrates favourable economic potential that warrants further work toward the completion of a pre-feasibility study.

The mine plan is appropriate to the mineralisation and adequately reflects the deposit type, dimensions, and host rock characterisation.

No fatal flaws were identified during the Project study. The recommendations are based on conventional metallurgical and other development testwork which would be part of project development.

Based on the economic analysis, the Project generates positive before and after tax discounted cashflows. The after-tax NPV8 for the Project is US$1 100 M with 21% IRR and 3.5-year payback period.

Opex and Capex considerations used for the Project represent those expected for a project of this type exhibiting average mineral abrasiveness and hardness characteristics, and grades and rock type characterisations as indicated in the geological section.

1.25 Recommendations

The following figure (Figure 1.23) illustrates the proposed project development pathway.

Figure 1.23 : Costa Fuego Project Development Pathway

1.25.1 Costa Fuego Prefeasibility Study

Based on the results of the 2023 PEA, which remain valid in the current Report, it is recommended that the Project be advanced to the Pre-feasibility study (PFS) stage. The estimated work program budget for the delivery of the intended PFS is summarised below.

Table 1.14 : Future Work Program

Work Program	Cost (US$M)
G&A	1.5
Exploration and Resources	3.0
Development Studies	2.7
Contingency and Opportunity	0.3
Total	**7.5**

1.25.2 Costa Fuego Environmental Works

The Costa Fuego Environmental Impact Assessment (EIA) will be managed as a separate study and will be submitted to Chilean authorities for their consideration utilising the outcomes of many of the workstreams required for the PFS.

Valuable information related to baselines, impact assessment and management plans for environmental and social elements will be integrated into the PFS. The key issues determining the EIA timing are the development of the hydrogeology model, getting preliminary agreements with families to be resettled and running the stakeholder engagement plan.

1.25.3 Mineral Resources

1.25.3.1 Introduction

Drilling of a 30 000 m program across Costa Fuego is underway to test regional targets to determine whether they can be incorporated into the Project.

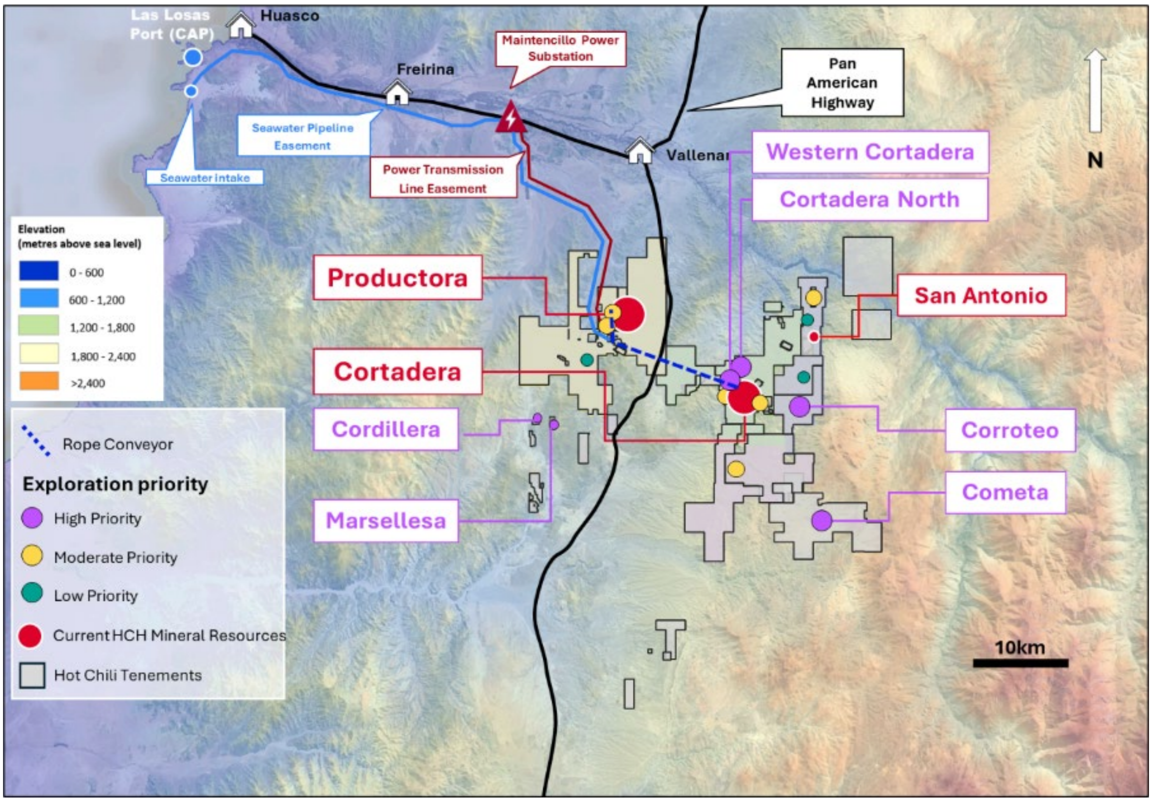
Ongoing access track and drill pad construction is required with subsequent drilling using a combination of RC and DD.

A complementary program of detailed geological mapping and geophysical programs are being planned across multiple project areas, including:

- Early-stage foundational exploration activities including geological mapping and geochemical soil sampling progressing across tenement holding
- Deep penetrating, high resolution MIMDAS geophysical survey at Productora and Cortadera providing coverage over multiple exploration targets
- Ground magnetic survey along the San Antonio trend.

Figure 1.24 shows the priority growth targets across the Costa Fuego Project, in relation to existing resources and key infrastructure (existing and planned).

Figure 1.24 : Exploration Growth Targets Across the Costa Fuego Project



1.25.3.2 Mineral Resource Development Drilling and Estimation

Resource Development drilling completed at Cortadera through 2023 and Q1 of 2024 aimed to test targets remaining following the pause in drilling in early 2021. Drilling provided critical information on target areas, which are now being reviewed by the HCH geology team. Planned geophysical surveys will be used in conjunction with completed drilling, surface mapping and soil geochemistry to determine additional targets across the highly prospective Cortadera tenements.

The Productora Resource and mineralisation remains open at depth in several places as well as laterally. While it is understood the width of mineralisation at Productora decreases with depth, additional drilling may add incremental mineralised tonnage, and will also aid in the upgrade of existing Inferred resource to Indicated classification.

The updated 2024 Alice resource estimate has identified multiple target areas for potential resource extension along the NE-SW striking Alice trend.

Work at San Antonio will focus on improving confidence in the depletion shapes used for the San Antonio MRE. Currently, the spatial uncertainty of the depletion shapes has been mitigated by depleting a generous amount of volume and ensuring a lower resource classification was applied in the vicinity of the depletion.

1.25.3.3 Costa Fuego Development Studies

A specific metallurgical and geotechnical drilling program of approximately 2600 m has been completed for a better definition of geotechnical domains and the drill core will also supply further variability and composite samples to the metallurgical testwork program. This metallurgy program is predominantly assessing comminution, flotation, filtration and thickening at Cortadera. A smaller program at Productora has added to the substantial work already completed for this resource.

The development studies for the Project are ongoing, having commenced in June 2021. A mining study will be required, comprising the entire open cut mining studies involving pit optimisation, mine designs, mining schedule/sequence, waste dump designs, mine fleet analysis, and operating cost and capital cost estimates. The mining study will cover open pit and underground mining, geotechnical and groundwater and surface water studies.

Recent geotechnical drilling at Cortadera will allow better definition of geotechnical domains for both the open pit and block cave. Additional engineering will be undertaken to prepare detailed pit slope stability analysis for the first mining phase and the final pit as a minimum. It will also include rock mass classification for the block cave and detailed stability analysis for waste dumps.

Groundwater and surface water studies currently underway will include hydrogeological assessment and complementary field works to improve the information quality for a robust PFS.

The Qualified Persons (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

Some sections, text and figures have been taken directly from the technical report titled, "Costa Fuego Copper Project: NI 43-101 Technical Report & Preliminary Economic Assessment (PEA)" dated August 14, 2023 with an effective date of June 28, 2023. The Mineral Resource estimate update, as described in this Report, does not materially affect the mineral resource inventory that formed the basis of the 2023 PEA and no new scientific or technical information has been developed that would materially affect the outcome of the 2023 PEA and, therefore, the results and conclusions of the 2023 PEA are considered current and have been restated in this Report.

2.1 Introduction

This Technical Report has been prepared to provide a Mineral Resource Estimate Update for the Productora, Alice, Cortadera and San Antonio deposits that comprise the Costa Fuego Copper Project (the Costa Fuego Project or the Project), located near Vallenar in Chile, South America.

At the request of Hot Chili Limited (HCH or Company), Wood Limited (Wood) has prepared this Mineral Resource Estimate Update (the "Report") for the Costa Fuego Copper Project (the Project), as announced by HCH on February 26, 2024 and to re-iterate that the results of the Costa Fuego Copper Project: NI 43-101 Technical Report & Preliminary Economic Assessment (PEA) dated August 14, 2023 with an effective date of June 28, 2023 remain current.

The Report was prepared in accordance with the Canadian disclosure requirements of National Instrument 43-101 (NI 43-101) and in accordance with the requirements of Form 43-101 F1.

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 engineering and geological consultants are as follows:

- Haren Consulting is an independent geological consulting firm based in Perth, Australia.
- 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
- High River Services is an independent environmental consulting company, based in Kentucky, USA

Hot Chili Limited is a Perth-based copper-gold exploration and development company that undertakes exploration and development of its various copper-gold projects, with projects located in Chile's Atacama Region.

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

2.2 Terms of Reference

This Report (2024 Resource) has been prepared following additional exploration, development (metallurgical and geotechnical), and resource expansion drilling in 2022 and 2023 at the Cortadera, Productora, San Antonio and Alice deposits. Drilling previously completed at Alice in 2016 and 2017 which returned after the previous estimates cut-off date was also incorporated.

The 2023 Preliminary Economic Assessment (2023 PEA) was completed after the Productora 2016 study (2016 study), following the addition of the Cortadera and San Antonio deposits, and included additional infrastructure required to transport these Mineral Resources to the proposed centralised process plant at Productora. This expanded Mineral Resource provided an opportunity to lift the scale of development for a combined development hub (Costa Fuego), with optimised infrastructure servicing all deposits.

The 2023 PEA of the Costa Fuego Project includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied that would enable them to be categorised as Mineral Reserves and there is no certainty that the 2023 PEA will be realised. It is reasonably expected that the Inferred mineral resource can be upgraded to an Indicated mineral resource through additional exploration. The 2023 PEA presents a materially different project to that contemplated in the 2016 Study.

The 2016 study was undertaken at a nominal plant throughput of 14 Mt/a for a copper sulphide process plant and a 3.3 Mt/a nominal throughput for the oxide heap leach process facility. This PEA considers a 20 Mt/a nominal plant throughput for a copper sulphide process plant and 3.3 Mt/a of oxide feed to be processed via a crushing-agglomeration-heap leach circuit coupled with a SX/EW plant producing up to 10 kt/a of copper cathode (increasing to 12 kt/a in Year 8).

The 2023 PEA contemplated conventional open pit and underground block cave mining, feeding a conventional process plant to produce copper and molybdenum concentrates and copper cathode.

The term "Property" is used in reference to the overall mineral tenure holdings.

A Mineral Resource Technical Report was issued with an effective date of 31 March 2022, that was filed on 16 May 2022 (referred to as the 2022 Resource Report).

The 2023 PEA had no impact on the 2022 Resource Report. The 2022 Resource Report was the basis of the key assumptions and parameters used in the 2023 PEA.

The 2024 Mineral Resource Estimate, as described in this Report, does not materially affect the 2022 Mineral Resource inventory that formed the basis of the 2023 PEA and no new scientific or technical information has been developed that would materially affect the outcome of the 2023 PEA and, therefore, the results and conclusions of the 2023 PEA are considered current and have been restated in this Report.

Units used in the 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).

Mineral Resources 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 NI43-101.

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.

2.3 Qualified Persons

The following are the qualified persons (QP) responsible for the contents of this Technical Report as that term is defined in National Instrument 43-101, Standards of Disclosure for Mineral Projects and for preparing the Technical Report in compliance with Form 43-101:

Table 2.1 : List of QP's

Qualified Person	Company	Section(s)
Elizabeth Haren	Haren Consulting	1.6, 1.9, 1.10, 1.12, 1.23, 1.24, 1.25, 7, 8, 9, 10, 11, 12.1 – 12.6, 12.9, 14, 25.3. 25.12, 25.13 and 26.2
Anton van Wielligh	ABGM Pty Ltd	1.13, 1.14, 12.8 15, 16, 23, 24, 25.4 and 26.3
Dean David	Wood Australia Pty Ltd	1.1 - 1.5, 1.7, 1.8, 1.11, 1.15, 1.16, 2, 3, 4, 5, 6, 12.7, 13, 17, 18, 21.2, 21.4, 25, 26, and 27
David Morgan	Knight Piésold Pty Ltd	18.6
Piers Wendlandt	Wood Group USA, Inc.	1.18, 1.21, 1.22, 12.7, 19, 22, 24, 25.8, and 25.11.
Jeffrey Stevens	Wood Australia Pty Ltd	1.19, 1.20, 21.1, 21.3. 21.5, and 25.10
Edmundo J Laporte	High River Services LLC	1.17, 12.10, 20, 25.7, and 26.7

2.4 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. Two small underground copper mines were in operation within the central mining lease of Productora from late 2006 until mid-2013.

The Mineral Resource QP Elizabeth Haren visited the site in May 2022. The purpose of the QP's site visit was to appraise the standards of core drilling and core logging, the sampling methods, and the database integrity of these samples in support of the Mineral Resource Estimations (MRE).

2.5 Effective Dates

The Report has several effective dates as follows:

- Date of Mineral Resource estimate informing Preliminary Economic Assessment: 31 March 2022
- Date of the Preliminary Economic Assessment: 28 June 2023.
- Date of supply of last assay data used in the updated Mineral Resource Estimate: 21 December 2023
- Date of updated Mineral Resource estimate: 26 February 2024, any subsequent drilling will not materially change the MRE.

2.6 Information Sources and References

This Report relies on historic and recent data generated by HCH, 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.

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

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

2.7 Previous Technical Reports

HCH has filed the following technical reports on the Project:

- Von Wielligh, A, Haren, E, Resource Report for the Costa Fuego Copper Project Located in Atacama Chile, Technical Report NI43-101 prepared by Haren Consulting and ABGM, Effective date 29 October 2021.
- Von Wielligh, A, Haren, E, Resource Report for the Costa Fuego Copper Project Located in Atacama Chile, Technical Report NI43-101 prepared by Haren Consulting and ABGM, Effective date 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 NI43-101 prepared by Haren Consulting, ABDM, Wood, Knight Piesold, and High River Services, Effective date 28 June 2023.

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

3 Reliance on Other Experts

Some sections, text and figures have been taken directly from the technical report titled: "Costa Fuego Copper Project: NI 43-101 Technical Report & Preliminary Economic Assessment (PEA)" dated August 14, 2023 with an effective date of June 28, 2023. The Mineral Resource estimate update, as described in this Report, does not materially affect the mineral resource inventory that formed the basis of the 2023 PEA and no new scientific or technical information has been developed that would materially affect the outcome of the 2023 PEA and, therefore, the results and conclusions of the 2023 PEA are considered current and have been restated in this Report.

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 Technical 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.

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 the 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 the 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 the Report. It is also used in Section 25.8 and Section 25.11 of the Report.

4 Property Description and Location

Some sections, text and figures have been taken directly from the technical report titled, "Costa Fuego Copper Project: NI 43-101 Technical Report & Preliminary Economic Assessment (PEA)" dated August 14, 2023 with an effective date of June 28, 2023. The Mineral Resource estimate update, as described in this Report, does not materially affect the mineral resource inventory that formed the basis of the 2023 PEA and no new scientific or technical information has been developed that would materially affect the outcome of the 2023 PEA and, therefore, the results and conclusions of the 2023 PEA are considered current and have been restated in this Report.

4.1 Introduction

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

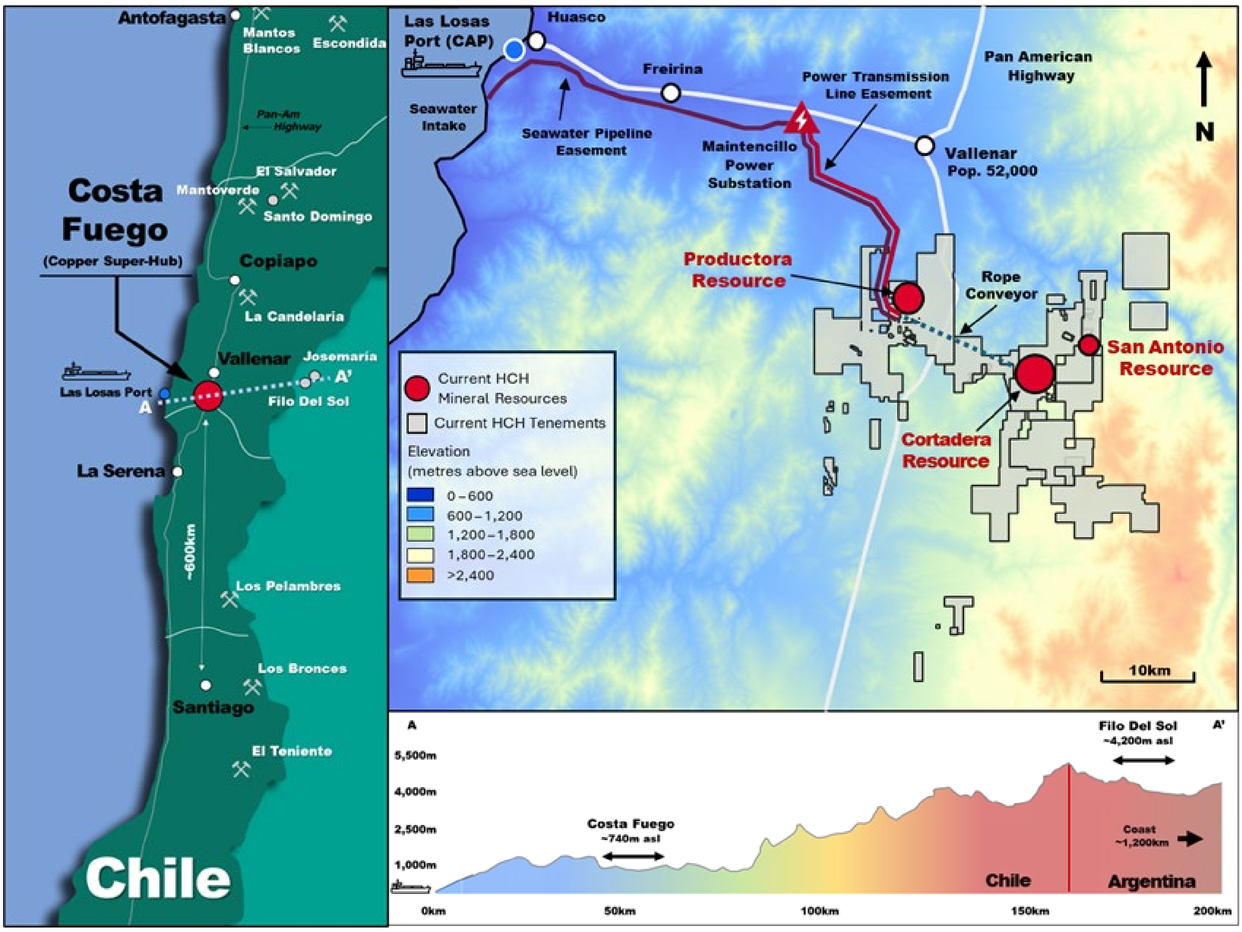
Productora lies 5 km off the main sealed Pan-American Highway which connects Vallenar to La Serena in the south and can be accessed via the Quebrada Arenas gravel road 2.2 km towards the west. From that point, the Project can be accessed by continuing along the Quebrada Verde track for 5 km. Alice sits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

Sociedad Minera El Aguila SpA (SMEA) is a joint venture company – 80% owned by Sociedad Minera El Corazón SpA (a 100% subsidiary of Hot Chili Limited (HCH) – ASX: HCH), currently holds a valid Maritime Concession Licence for the extraction of saltwater from a location south of the Huasco area.

Figure 4.1 : Costa Fuego Copper Project Location (HCH, 2024)



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, Chilean 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 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 SpAda, 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

4.4.1 Productora

The Project is approximately 18 km long by 14 km wide, with a total landholding of 13,605 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.

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 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
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
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

Table 4.1 : SMEA SpA Mining Tenement Holding for Productora

Licence ID	Holder	% Interest	Licence Type	Area (ha)
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
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

Table 4.1 : SMEA SpA Mining Tenement Holding for Productora

Licence ID	Holder	% Interest	Licence Type	Area (ha)
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
TOLTÉN 1/14	SMEA SpA	80%	Exploration concession	70
URANIO 1/70	CCHEN	100%	Exploration concession	350
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

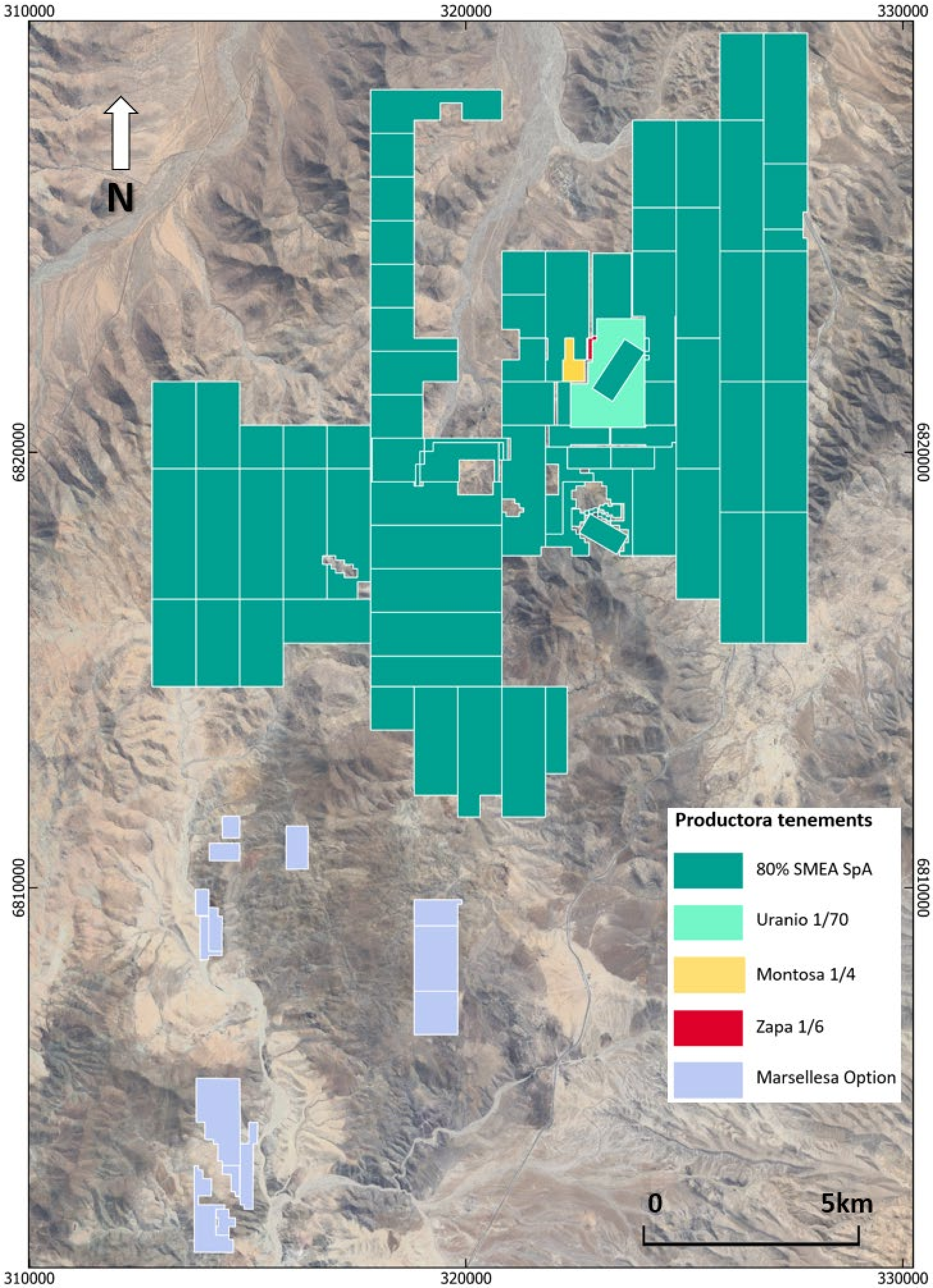
* Subject to a 3% Net Smelter Return (NSR).
**Subject to a 1% Gross Royalty.

All mining rights 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 are the maritime concession to extract sea water from the coast, and most of the corridor of easements to construct a pipeline and electrical transmission line to Costa Fuego.

A map of HCH´s mining rights at Productora under WGS84 is shown in Figure 4.2.

Figure 4.2 : Productora Mining Rights Under WGS84 (HCH, 2024)



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 20 000 ha at the deposit through various 100% purchase option agreements with private mining title holders and 100% owned tenure.

Figure 4.3 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 Option Agreement) for 100% of the mining rights expiring in September 2026
- Frontera has a 2-year Option Agreement over the mining rights coloured in purple (AMSA Option Agreement)
- Frontera has a 4-year Option Agreement over the mining rights coloured in light blue (Cordillera Option Agreement)
- Frontera has a 3-year Option Agreement over the mining rights coloured in light blue (Marsellesa Option Agreement) - Figure 4.2
- Frontera has Option Agreement over the mining rights coloured in light blue (Cometa Agreement) which expired alternatively in 12, 18 or 30 months - Figure 4.2

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

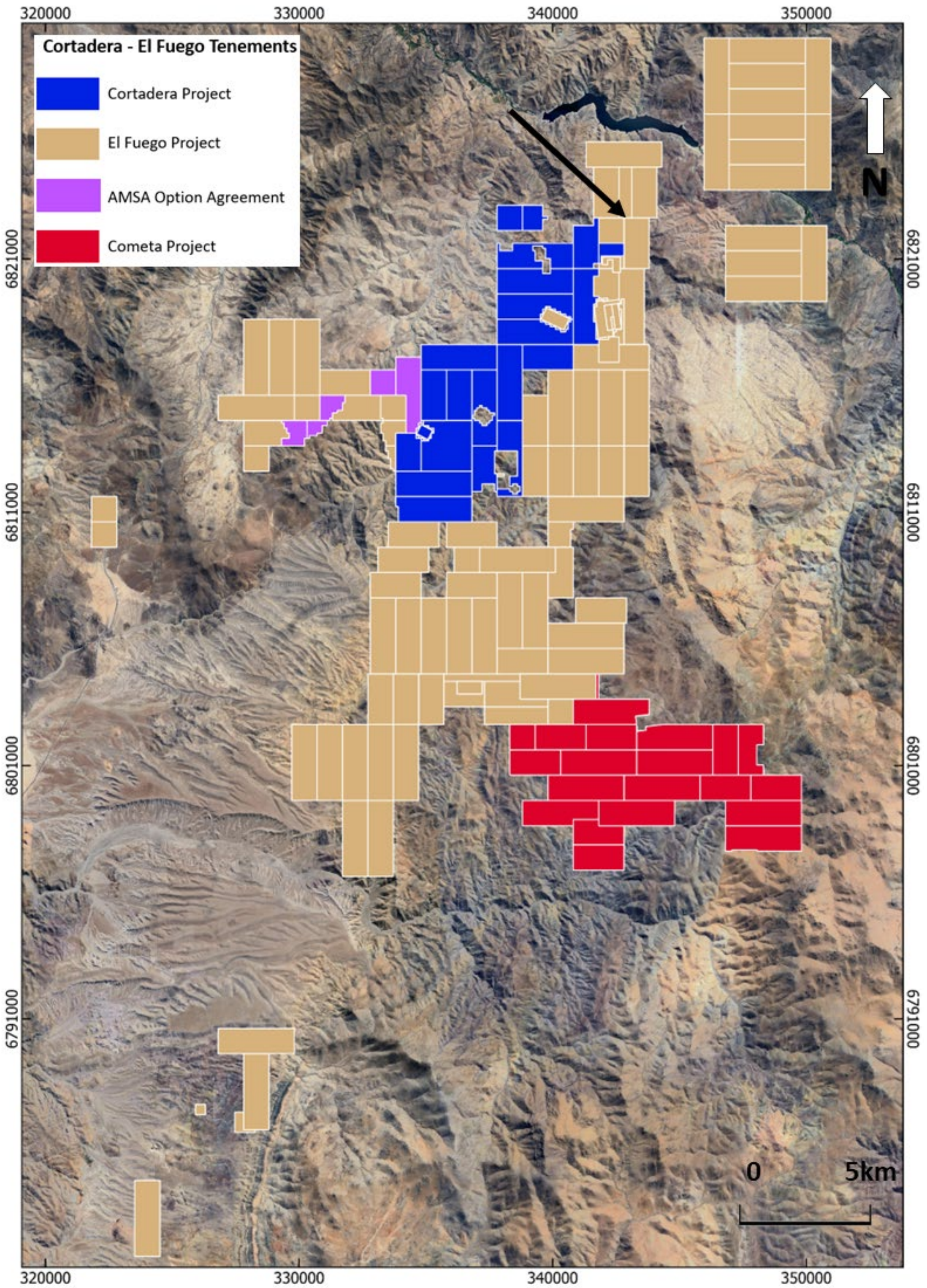
Surface land access for exploration activities has also been granted by the owner, Mr. Pedro Prokurika Morales by agreement acknowledged and approved by the local Court of Vallenar on 30 March 2022.

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 4.3 : 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.

4.4.3 Cortadera

Figure 4.3 shows in dark blue colour, Frontera 100% full 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)
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
MAGDALENITA 1/20	Frontera SpA	100%	Exploitation concession	100
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 1 2 5 y 6*	Frontera SpA	100%	Exploitation concession	20

*Subject to a 1.5% NSR

4.4.4 El Fuego

Figure 4.3 shows, in brown colour, the El Fuego mining rights which include three now terminated Options for Valentina, San Antonio and Santiago Z. 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 El Fuego 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$2,000,000M 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 January 1st, 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 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 September 30th, 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 31st December 2025.

Table 4.3 details the El Fuego agreement mining rights.

Table 4.3 : Summary of El Fuego Agreement Mining Rights

Licence ID	Holder	% Interest	Licence Type	Area (ha)
KRETA 1/4	Del Campo Family	100%	Exploitation concession	16
MARI 1/12	Del Campo Family	100%	Exploitation concession	64
MERCEDES 1/3	Del Campo Family	100%	Exploitation concession	50
PORFIADA A 1/40	Del Campo Family	100%	Exploitation concession	200
PORFIADA C 1/60	Del Campo Family	100%	Exploitation concession	300
PORFIADA E 1/20	Del Campo Family	100%	Exploitation concession	100
PORFIADA F 1/60	Del Campo Family	100%	Exploitation concession	300
PORFIADA IX 1/60	Del Campo Family	100%	Exploitation concession	300
PORFIADA VII 1/60	Del Campo Family	100%	Exploitation concession	300
PORFIADA VIII 1/60	Del Campo Family	100%	Exploitation concession	300
PRIMA DOS	Del Campo Family	100%	Exploitation concession	2
PRIMA UNO	Del Campo Family	100%	Exploitation concession	1

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 1/14 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	236
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
CF 1	Frontera SpA	100%	Exploration concession	300
CF 2	Frontera SpA	100%	Exploration concession	300
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
CHAPULIN COLORADO 1/3	Frontera SpA	100%	Exploitation concession	3
CHILIS 1	Frontera SpA	100%	Exploration concession	200
CHILIS 10 1/40	Frontera SpA	100%	Exploitation concession	200
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	Frontera SpA	100%	Exploration concession	300
CORTADERA 7 1/20	Frontera SpA	100%	Exploitation concession	93
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
ELEANOR RIGBY 1/10	Frontera SpA	100%	Exploitation concession	100
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
MARI 1	Frontera SpA	100%	Exploration concession	300
MARI 6	Frontera SpA	100%	Exploration concession	300
MARI 8	Frontera SpA	100%	Exploration concession	300
PEGGY SUE 1/10	Frontera SpA	100%	Exploitation concession	100
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
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
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

4.4.5 AMSA Option Agreement

Figure 4.3 shows, in purple colour, the mining rights of 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 a payment of USD 1 500 000 by November 2024. Also, 6 000 metres of drilling of any type is required as an expenses commitment. AMSA has the right to buy-back a 55% interest in the AMSA mining rights within 120 days of exercise of the Option, by repaying 55% of the Option exercise price and paying five times the exploration expenditure incurred during the option period.

Table 4.4 details the AMSA agreement mining rights.

Table 4.4 : Summary of AMSA Agreement Mining Rights

Licence ID	Holder	% Interest	Licence Type	Area (ha)
ARBOLEDA 7 1/25	AMSA	100%	Exploitation concession	234
MONICA 2 1/40	AMSA	100%	Exploitation concession	85
MONICA 4 1/52	AMSA	100%	Exploitation concession	39
NAVARRO 1 41/60	AMSA	100%	Exploitation concession	119
NAVARRO 2 21/37	AMSA	100%	Exploitation concession	78

4.4.6 Cometa Option Agreement

Figure 4.3 shows, in red colour, the mining rights of an Option Agreement with Bastion Minerals for the right to acquire 100% of Bastion's Cometa Project, located approximately 15 km southeast of Cortadera. The exploration and mining concessions cover approximately 5 600 ha. If the Option is exercised within 18 months from the date of grant of the Option (February 2024), the consideration payable to Bastion is US$2 400 000, and if the Option is exercised within 30 months from the date of grant of the Option the consideration payable to Bastion is US$3 000 000.

Table 4.5 details the Cometa agreement mining rights.

Table 4.5 : Summary of Bastion Cometa Agreement Mining Rights				
Licence ID	**Option Holder**	**% Interest**	**Licence Type**	**Area (ha)**
COMETA 1 1/60	SCM Constelación	100%	Exploitation concession	300
COMETA 2 1/60	SCM Constelación	100%	Exploitation concession	300
COMETA 3 1/60	SCM Constelación	100%	Exploitation concession	300
COMETA 3D	SCM Constelación	100%	Exploration concession	200
COMETA 4A	SCM Constelación	100%	Exploration concession	300
COMETA 4B	SCM Constelación	100%	Exploration concession	200
COMETA ESTE 1B	SCM Constelación	100%	Exploration concession	200
COMETA ESTE 2B	SCM Constelación	100%	Exploration concession	200
COMETA ESTE 3B	SCM Constelación	100%	Exploration concession	300
COMETA ESTE 4B	SCM Constelación	100%	Exploration concession	300
COMETA IV D	SCM Constelación	100%	Exploration concession	300
COMETA NORTE 1 B 1/40	SCM Constelación	100%	Exploitation concession	200
COMETA NORTE 1D	SCM Constelación	100%	Exploration concession	200
COMETA NORTE 2 B 1/40	SCM Constelación	100%	Exploitation concession	200
COMETA NORTE 2 D	SCM Constelación	100%	Exploration concession	200
COMETA NORTE 3D	SCM Constelación	100%	Exploration concession	300
COMETA NORTE 4 D	SCM Constelación	100%	Exploration concession	200
COMETA NORTE 5D	SCM Constelación	100%	Exploration concession	100
COMETA OESTE I D	SCM Constelación	100%	Exploration concession	200
COMETA OESTE IID	SCM Constelación	100%	Exploration concession	200
COMETA SUR DOS D	SCM Constelación	100%	Exploration concession	200
COMETA SUR UNO D	SCM Constelación	100%	Exploration concession	200

4.4.7 Marsellesa and Cordillera Option Agreements

Figure 4.2 shows, in light blue colour, the mining rights of an Option Agreements to acquire 100% with private parties for the historical copper mine areas Marsellesa and Cordillera, located approximately 10 km southwest of Productora.

The Option Agreement for Marsellesa with Hermanos Pefaur SpA, the holder of a 100% interest in the concession comprising Marsellesa, may be exercised within 36 months of the date of grant of the Option for a final non-refundable cash payment of US$1 000 000. Pefaur will also be granted a 1% NSR royalty over the Marsellesa concession on exercise of the Marsellesa Option. Frontera will have a right of first refusal to buy-back the NSR royalty.

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 4.6 details the Marsellesa and Cordillera agreement mining rights.

Table 4.6 : Summary of Marsellesa and 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
MARSELLESA 1/5	Hermanos Pefaur SpA	100%	Exploitation concession	50

4.5 Surface Rights

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

Articles 120 to 125 of the Chilean 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. SMEA currently holds a valid Maritime Concession Licence for the extraction of saltwater from a location south of the Huasco area.

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. 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.

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 2011 (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.

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 Chapter 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

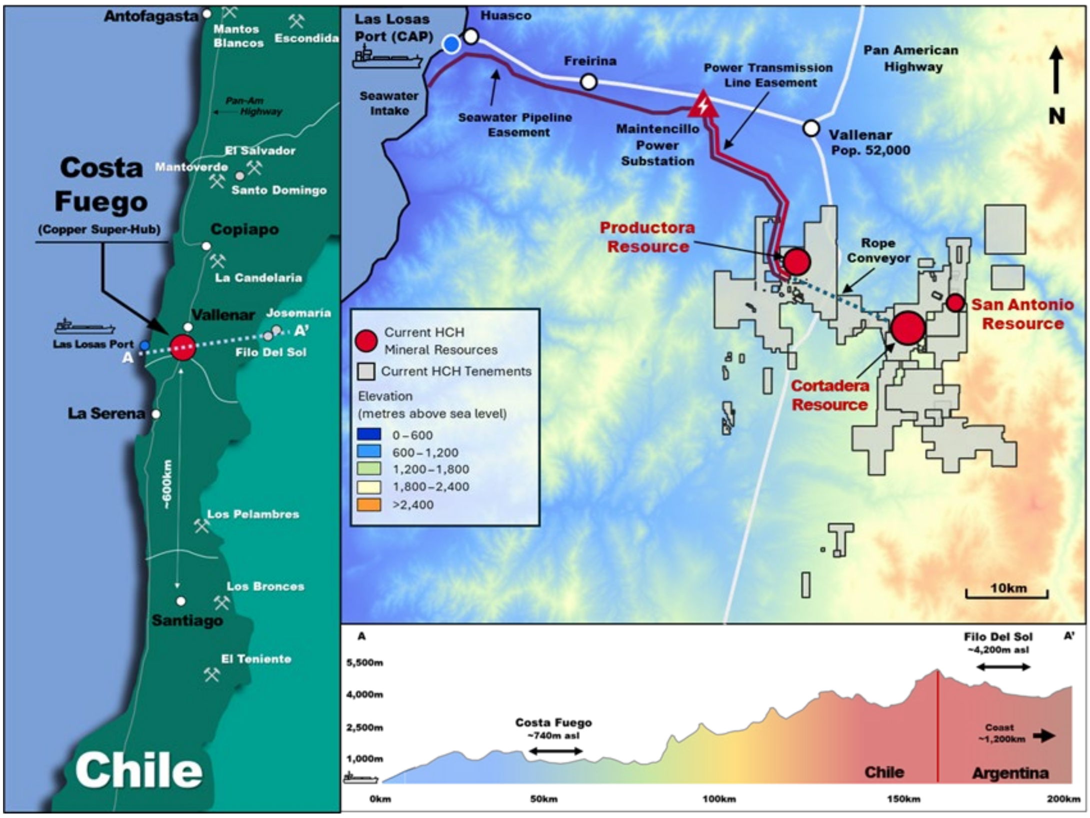
Some sections, text and figures have been taken directly from the technical report titled, "Costa Fuego Copper Project: NI 43-101 Technical Report & Preliminary Economic Assessment (PEA)" dated August 14, 2023 with an effective date of June 28, 2023. The Mineral Resource estimate update, as described in this Report, does not materially affect the mineral resource inventory that formed the basis of the 2023 PEA and no new scientific or technical information has been developed that would materially affect the outcome of the 2023 PEA and, therefore, the results and conclusions of the 2023 PEA are considered current and have been restated in this Report.

5.1 Accessibility

The Project has a favourable location, surrounded by infrastructure which can be utilised for developing a greenfield copper project. Figure 5.1 displays the location of the following infrastructure items:

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

Figure 5.1 : Infrastructure Proximal to the Costa Fuego Project (HCH, 2024)



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 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.

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

HCH, through its Chilean subsidiary company Sociedad Minera El Aguila (SMEA) SpA, was granted a Maritime Concession for extraction of sea water 60 kilometres from the Costa Fuego Project in December 2020.

These water extraction rights represent a critical infrastructure requirement for the Project, with sufficient water supply secured to support a large-scale conventional copper-gold operation.

In addition to this, the Chilean Naval Authority granted HCH access to the physical land of its maritime concession in December 2022.

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, 50km 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 Feasibility Study 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 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

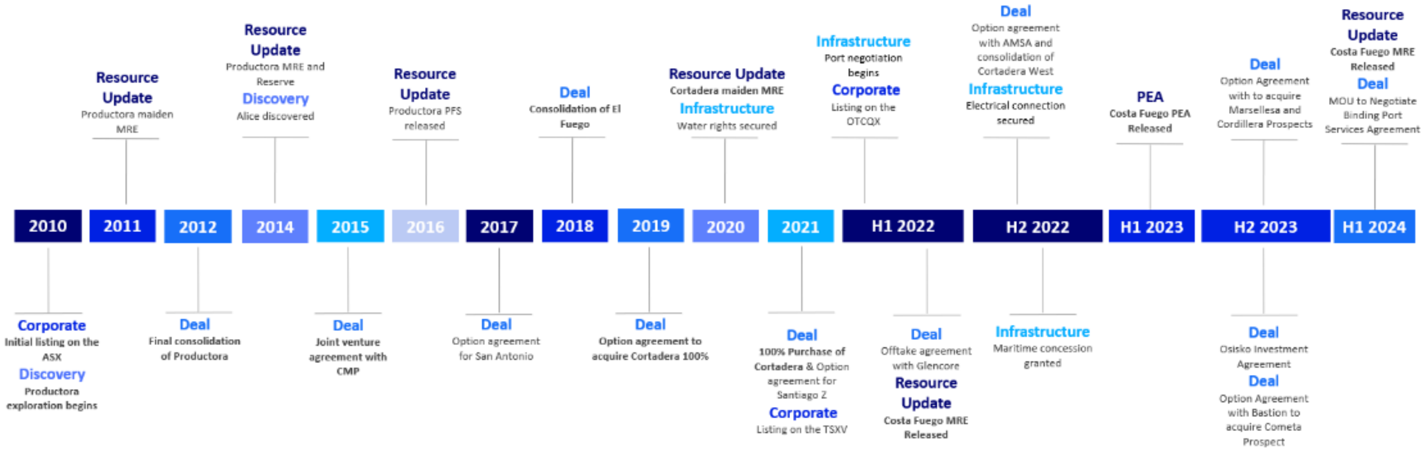
In the QP's opinion:

- 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, 2024)



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.
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.

Table 6.2 : Cortadera Ownership and Activity History		
Year	**Activity**	**Result**
<1993	Empresa Nacional de Minería (Enami) completed four percussion drill holes.	Defined near-surface oxide mineralisation.
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 (Empresa Nacional de Mineria) 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 completed drilling campaigns in 2019 and 2022 to define the San Antonio Resource extent, verify and infill data from previous operators.

6.4 Production

6.4.1 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 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.

6.4.2 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 Previous Mineral Resources

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.

6.5.1 Productora JORC 2004 Resource, HCH, September 2011

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".

6.5.2 Productora JORC 2004 Resource, HCH, February 2013

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".

6.5.3 Productora JORC 2012 Resource, HCH, March 2014

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).

6.5.4 Costa Fuego JORC 2012 Mineral Resource, October 2020

The first combined Costa Fuego 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".

6.5.5 Costa Fuego NI 43-101 Resource, HCH, March 2022

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 NI 43-101 Mineral Resource, March 2022

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'

6.6 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.

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 JORC 2012 Ore Reserve, March 2016

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) provides 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 Bandurias 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 overly 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 calarenite and fossiliferous limestone, capped by volcaniclastic rocks
- Totorolillo 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 stratigraphic succession exposed have 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 sub-parallel, linked (presumably) fault systems, 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).

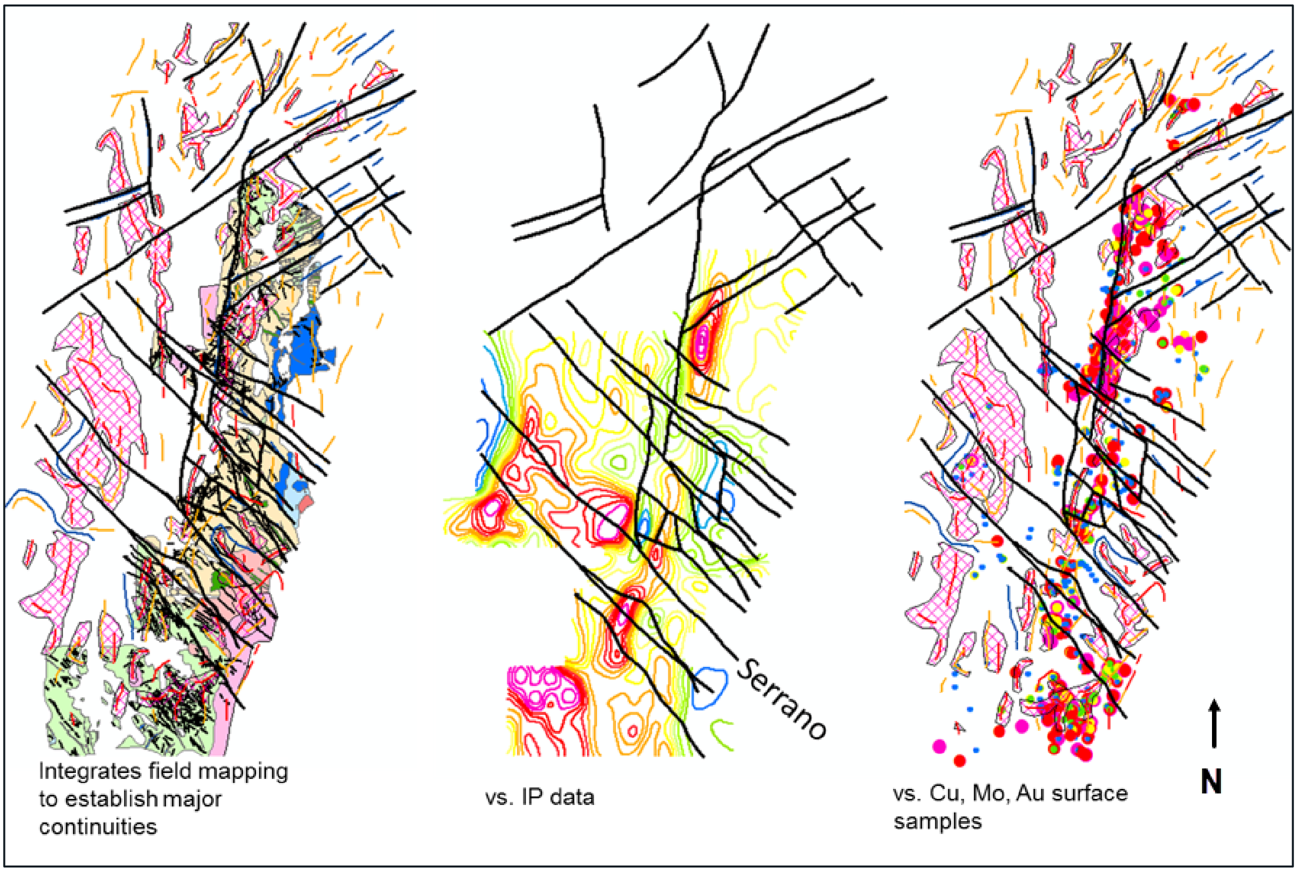
7.2 Property Geology

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



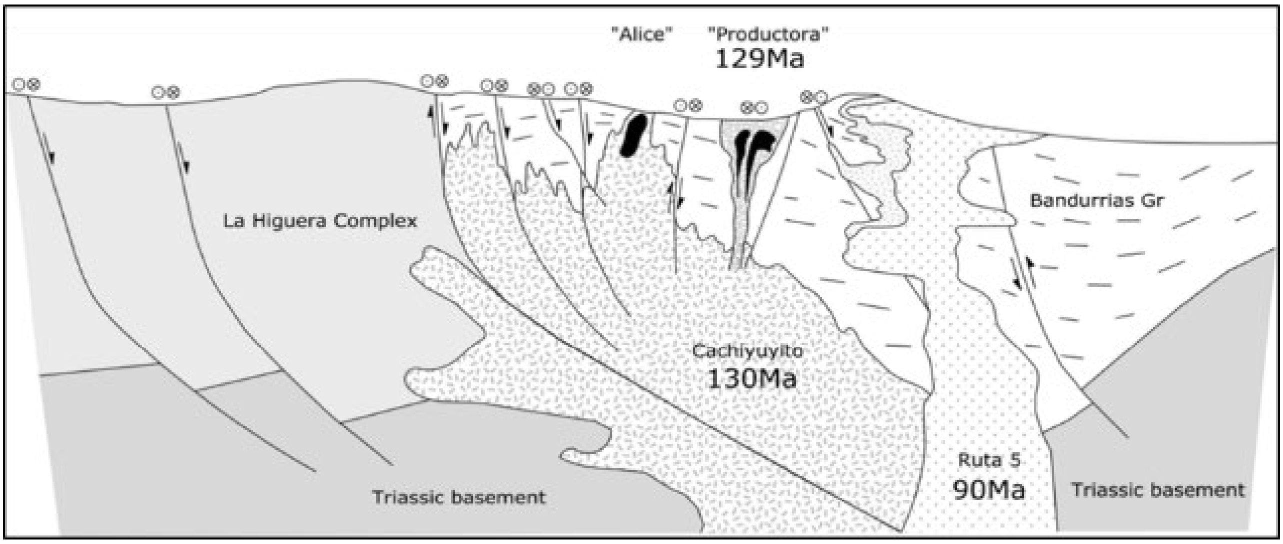
Note : Left; Compared to Ground Mapping, Centre; Compared to Previous IP Data, Right; Compared to Rock Chip Sampling. (HCH, 2016)

The following sections summarise the work described by Fox (2000), Beeson et al. (2012) 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 volumetrically minor and comprise well-bedded volcanic sandstones. 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 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.

Tobey (2011a, 2011b, 2012a, 2012b, 2012c, 2013a, 2013b) provides a comprehensive discussion regarding mineralised porphyry-style vein systems at Cortadera. At Cortadera, the porphyry-style vein systems are associated with a multi-phase tonalitic intrusive system showing hydrothermal alteration zonation typical of such mineralised systems. 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 cross-bedding. 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.

In structurally conducive 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.

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 Deposits

The Costa Fuego Copper Hub being developed by HCH encompasses several different deposit types and mineralisation styles. Cortadera mineralisation forms a classic copper-gold-molybdenum porphyry deposit, 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, which suggests mineralisation at Productora may have porphyry sources.

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)

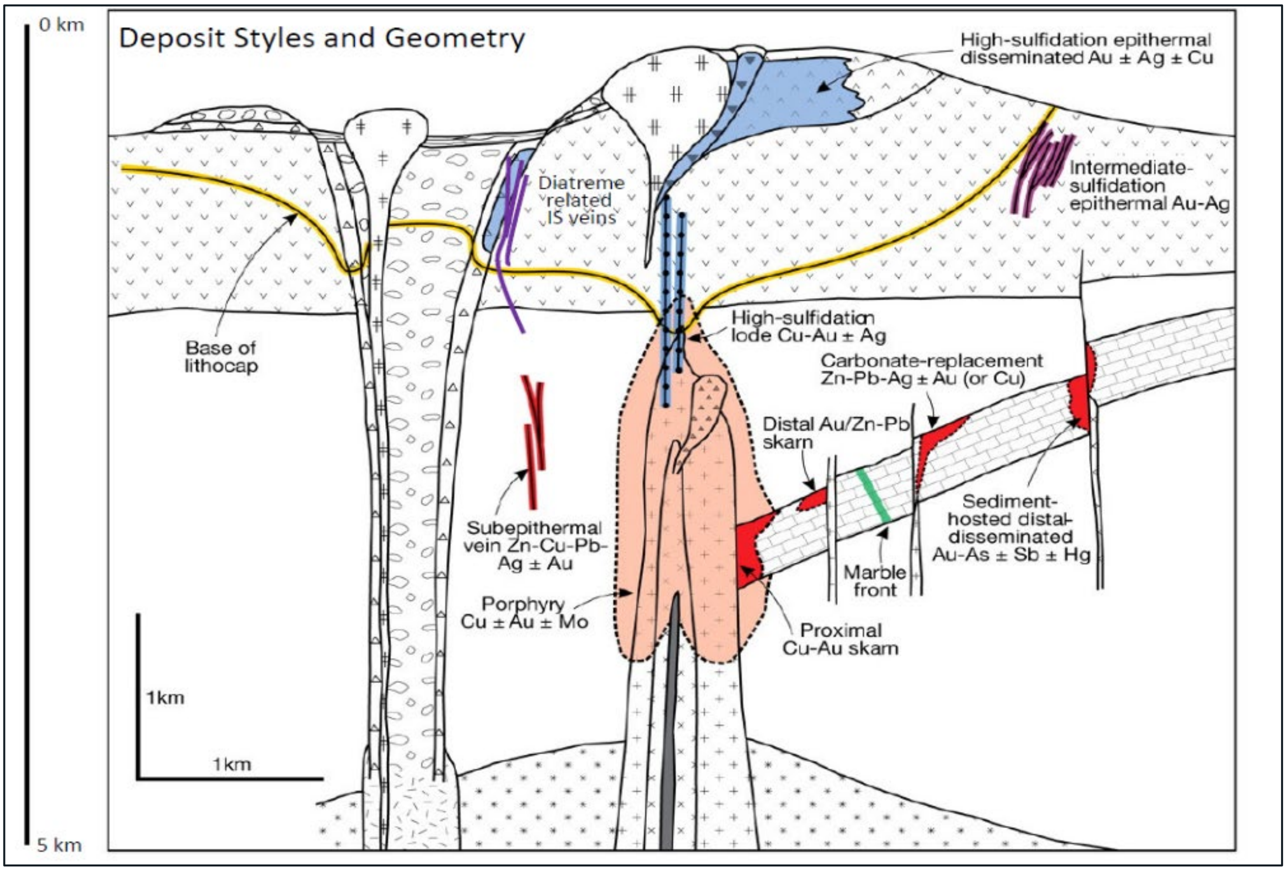
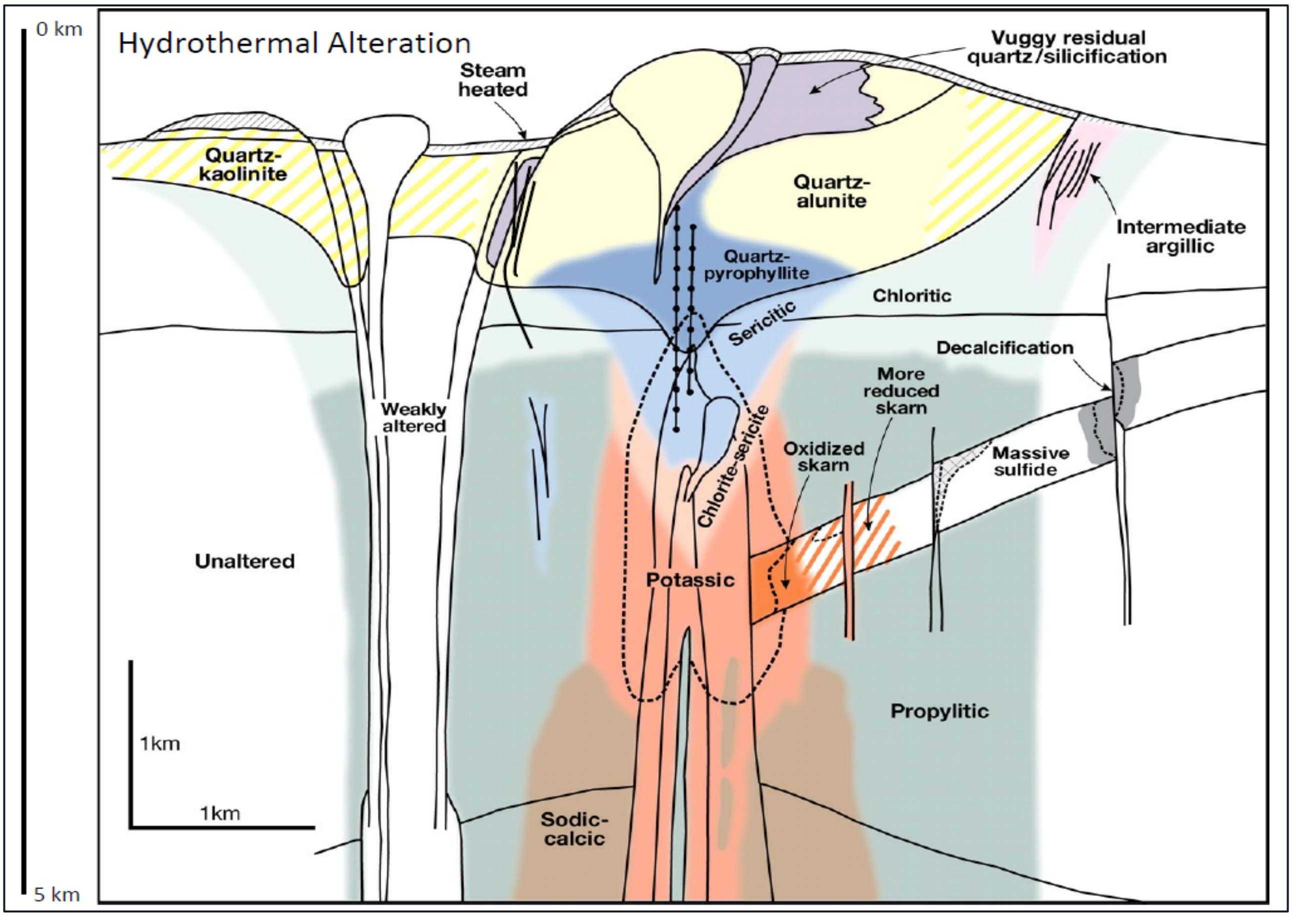


Figure 8.2 : Example Cross Section of Porphyry Systems in an Arc Setting, Showing Alteration Variations (Sillitoe, 2010)



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 Topografia 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 Costa Fuego by HCH since 2010, this is summarised in Table 9.1.

Table 9.1 : Summary of Geological Mapping Completed at Costa Fuego 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.
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.
	Cortadera (Cuerpo 1-3 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

Year	Project Area	Activity	Result
			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 dyke corridor trending immediately north of Cuerpo 3.	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, 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.
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.

Year	Project Area	Activity	Result
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. A number of 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 main structure was interpreted to extend to surface.	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).

9.3 Geochemical Sampling

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

Table 9.2 : Summary of Geochemical Sampling Completed at Costa Fuego 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	Geochemical sampling demonstrated that significantly elevated copper-gold-molybdenum grades, together with other elevated pathfinder elements, were evident within soils. Molybdenum in soils appeared to define an anomaly immediately above the Productora mineralisation. Where uranium

Year	Project Area	Activity	Result
		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.	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 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.

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 three geophysical surveys across Costa Fuego, they are summarised in Table 9.4.

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

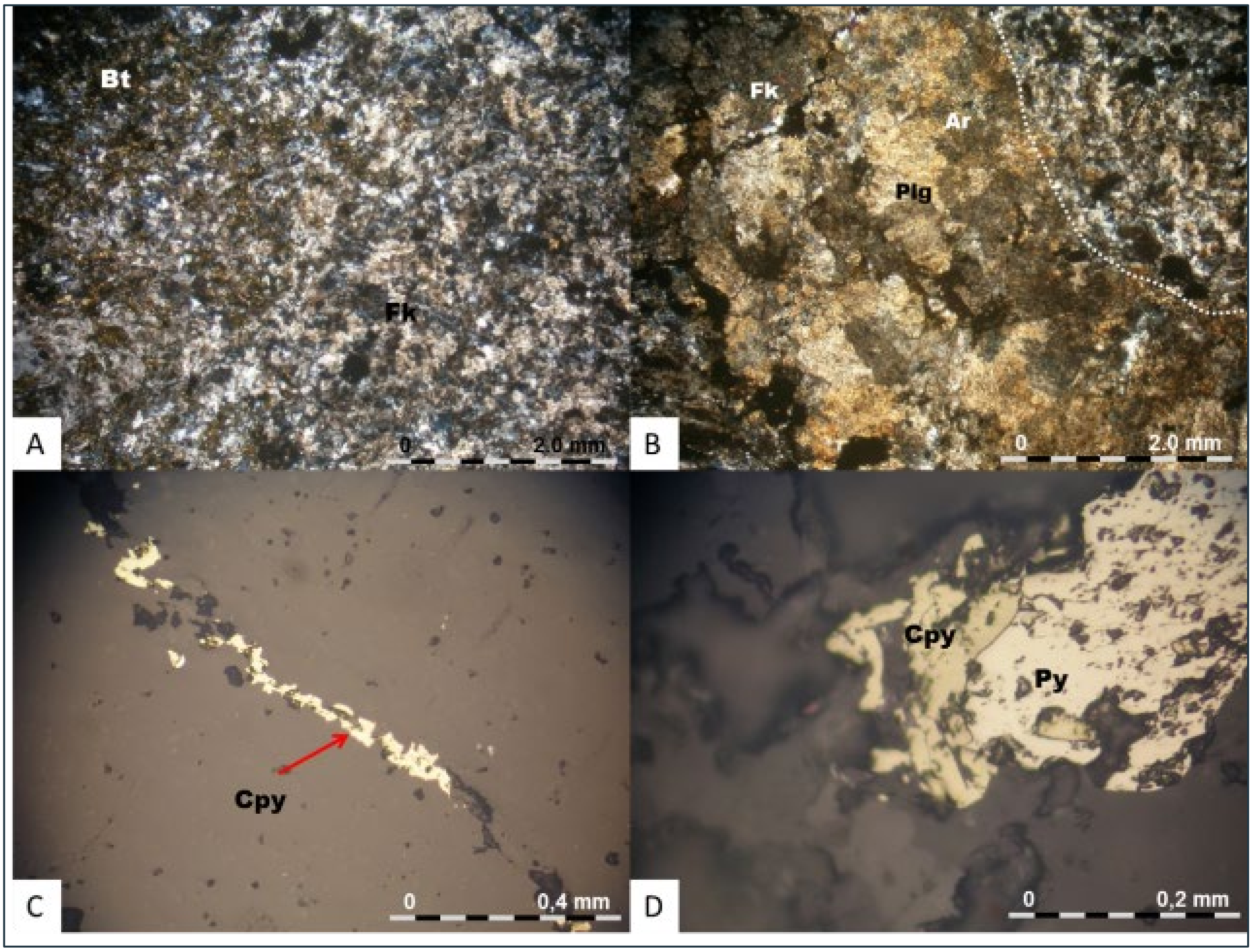
Year	Activity	Result
2010	An airborne geophysical survey was conducted at Productora by Geodatos and flown by helicopter with an average sensor height of about 145 m, on 100 m spaced east-west flight lines, and 1 000 m spaced north-south tie lines. Data collected included standard flight height, magnetic 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.

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 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. A summary of all drilling that informs the 2024 Costa Fuego Mineral Resource Estimate (MRE) is detailed in Table 10.1.

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.

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 February 2024 MRE update.

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.

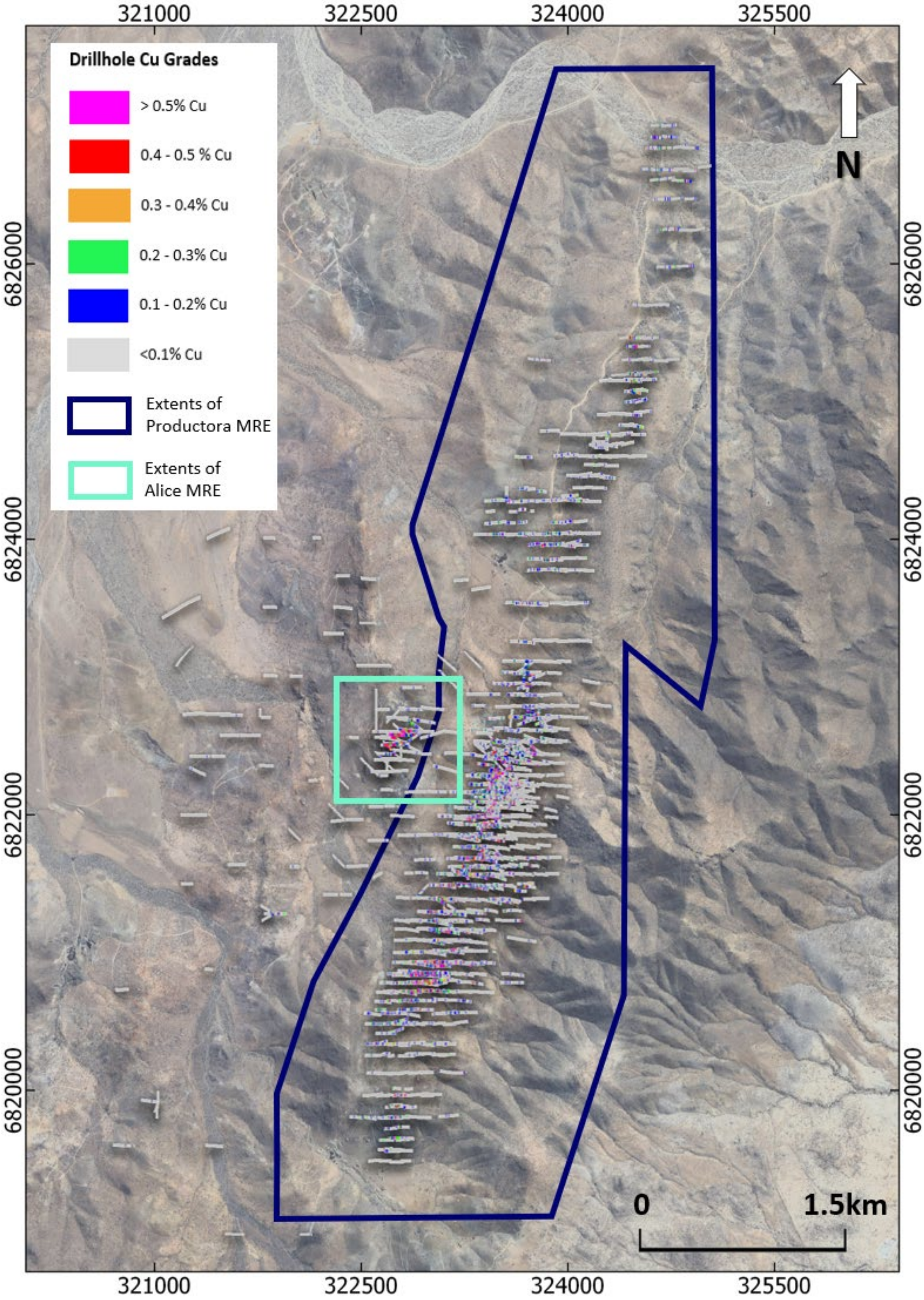
Table 10.1 : Breakdown of the Drilling Completed by HCH Across Costa Fuego Mineral Resources

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

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 Productora and Alice MRE extents, 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 representivity, 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.

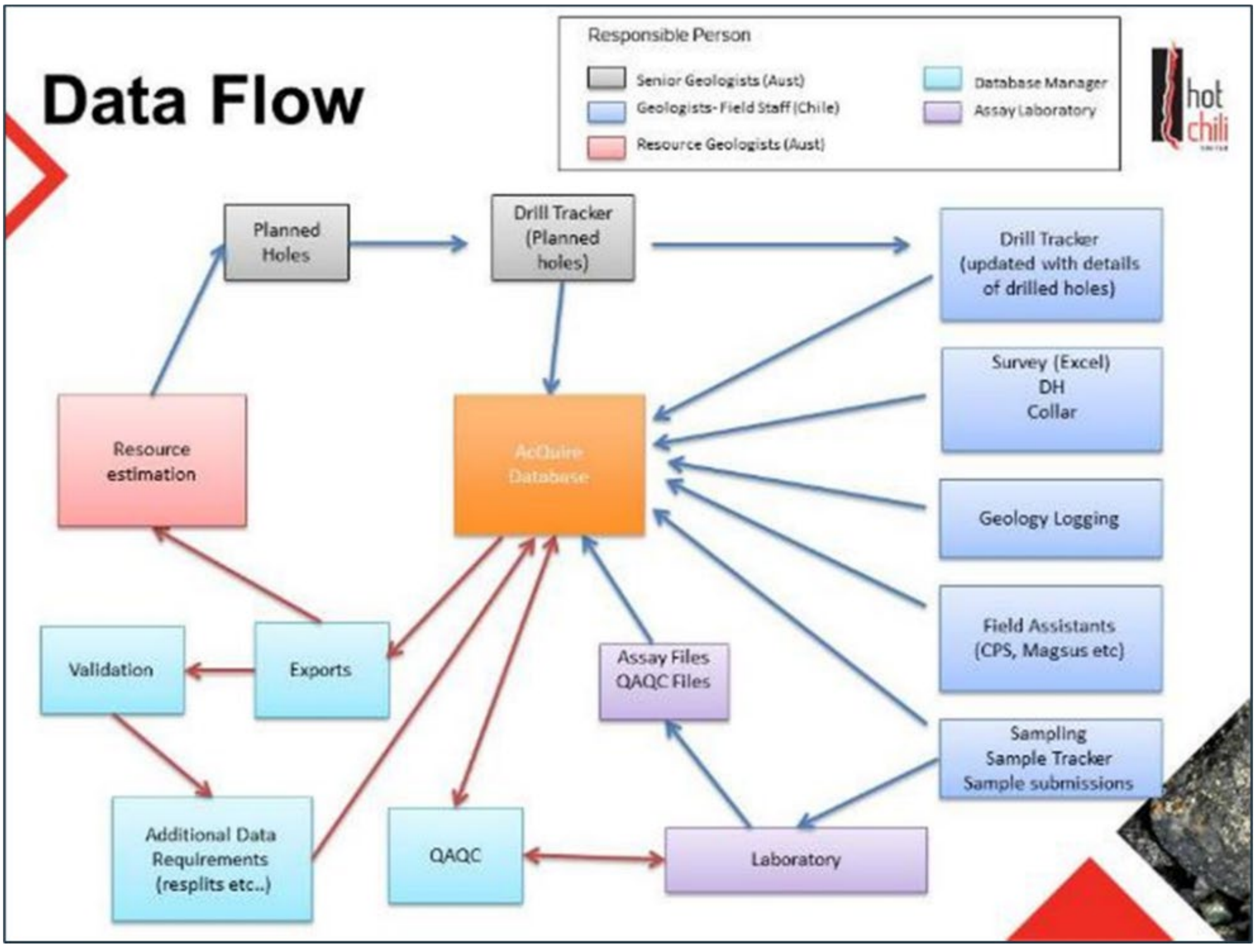
10.2.3 Geological Database

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 steps taken 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 : Drill Data Flow Showing Drill Planning, Drill Execution, Logging, Sampling, Data Entry and Capture



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.

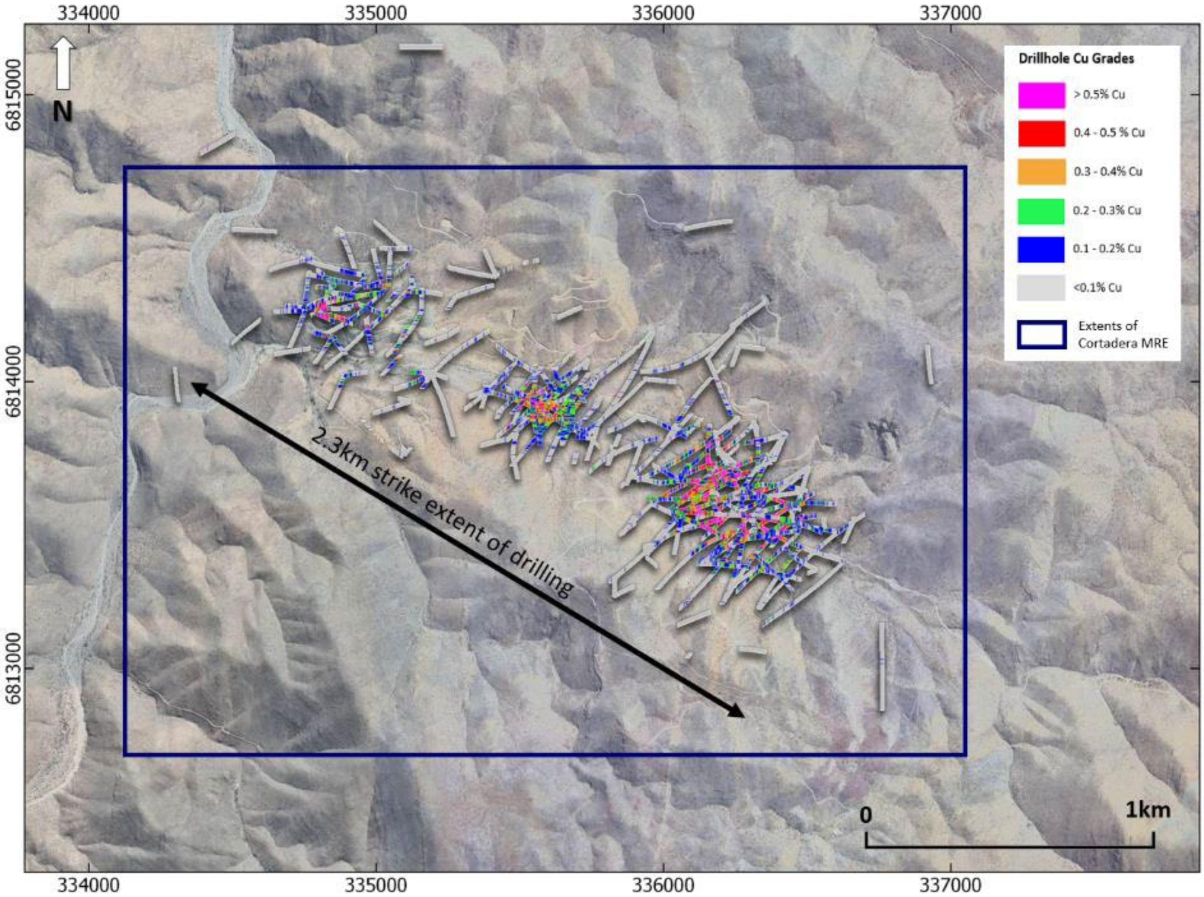
In addition to the bulk density analysis completed on drill core samples, pycnometer density tests by ALS (ALS Code OA-GRA08b) were also carried out on historic pulps for comparative purposes. Specific gravity (SG) analysis on pulps was carried out by placing 3 g of sample pulp into a pycnometer, which 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, Drill Holes Displayed by Copper Grade.



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'.

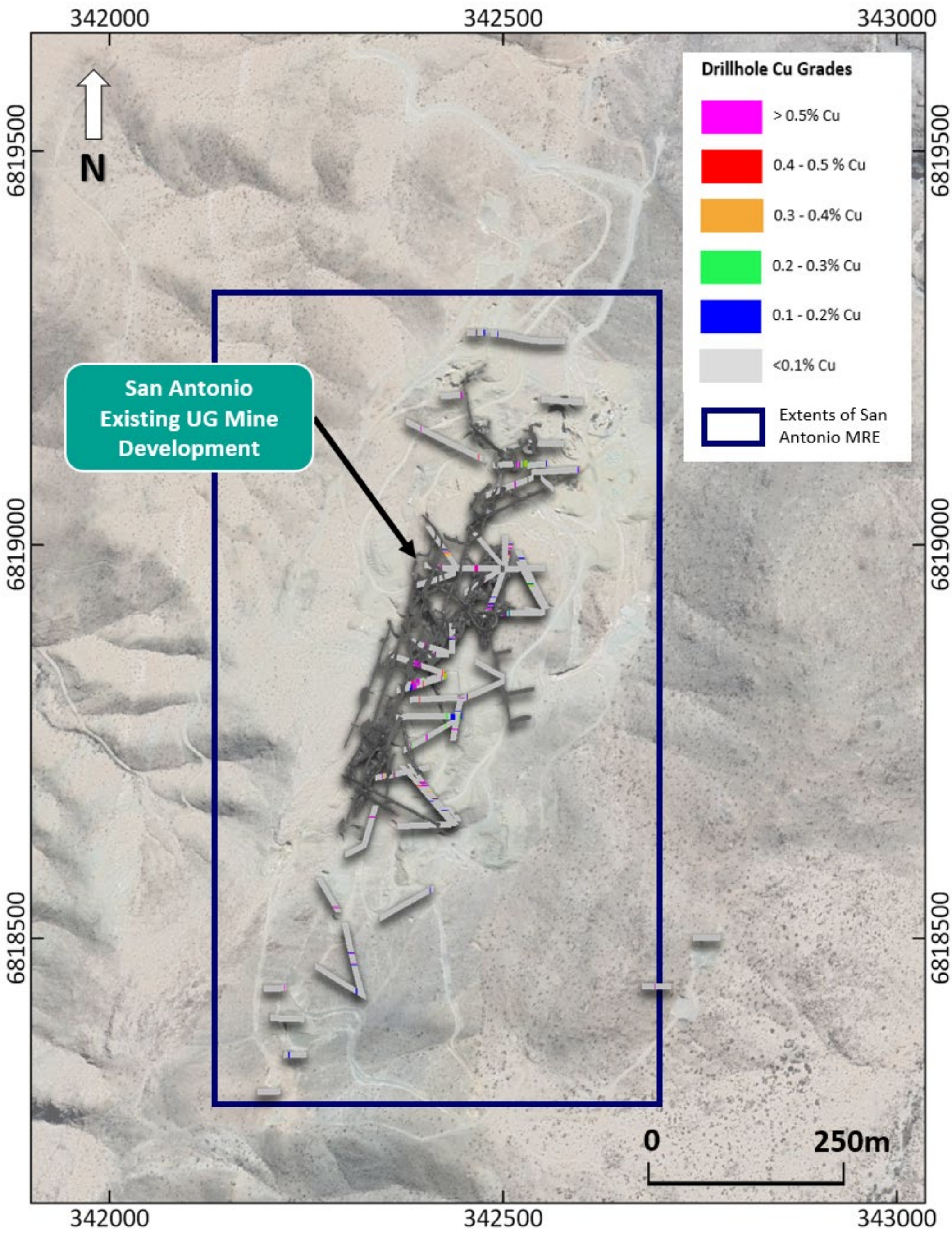
The testwork was undertaken on an approximate 10 cm piece of core taken from the designated sample bag and is used to determine the bulk density for the 1 m interval. The bulk density test consists of using the Archimedean water submersion method of analysis; a description of the ALS OA-GRA09 method 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 RC Drill Program Hole Locations Relative to Existing Underground Mine Development



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 Costa Fuego 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.

11 Sample Preparation, Analyses and Security Sampling Methods

11.1 Sampling Methods

11.1.1 RC Sampling Methods

RC drill hole sampling at Costa Fuego 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.

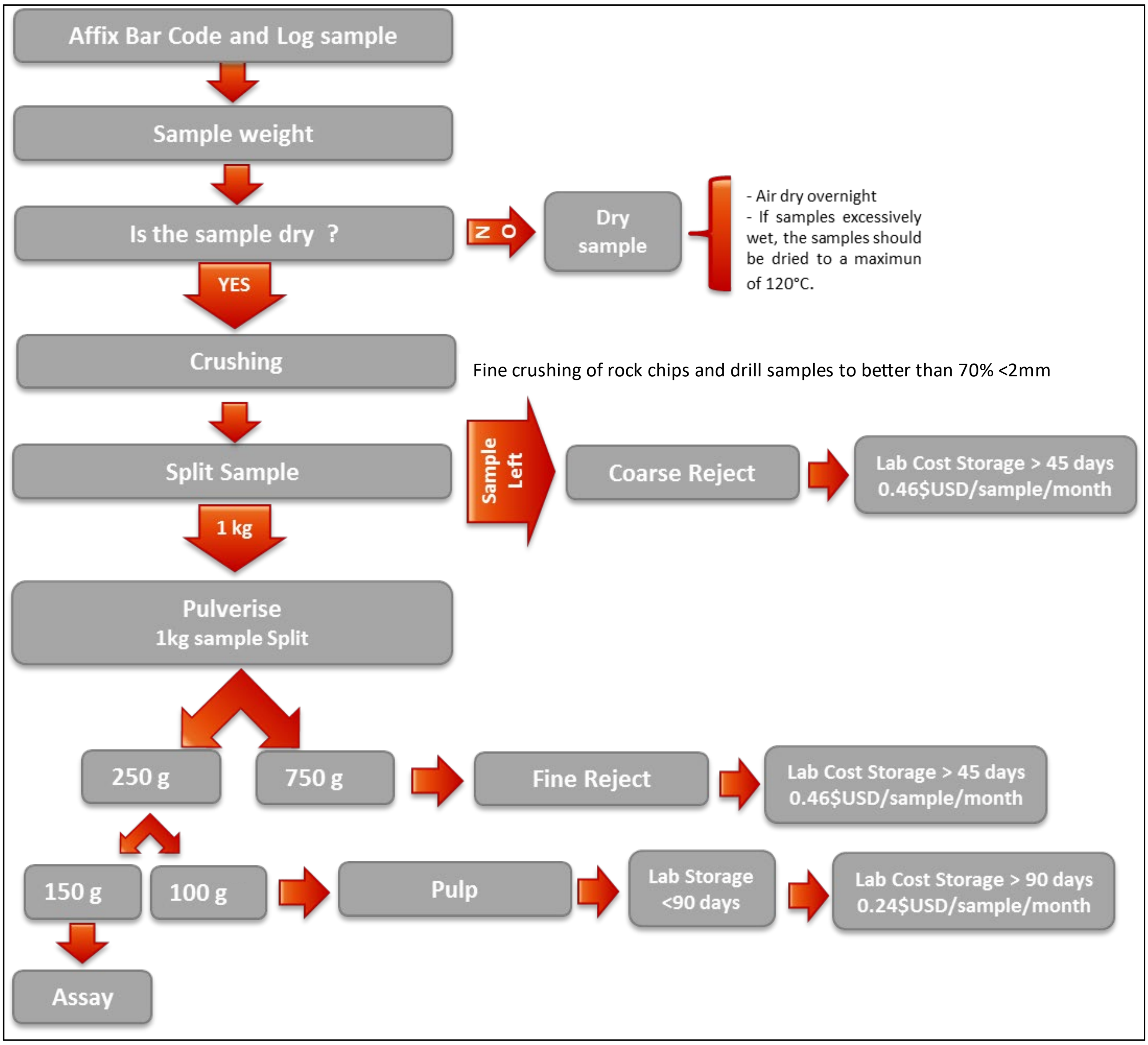
The Qualified Person, Elizabeth Haren, visited the primary sample preparation facility of ALS La Serena (Coquimbo), Chile and ALS Lima, Peru during 2022.

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.

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 (as 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 Comments on Section 11

11.7.1 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.

11.7.2 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 Mineral Resource Estimates was appropriate. A site visit was also completed by HCH employees in 2023, which included a detailed review of the RC sampling procedure at Cortadera.

12 Data Verification

12.1 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.

12.1.1 Certified Reference Materials Summary

Certified Reference Materials (CRMs) comprise 2% of the total Costa Fuego 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 Costa Fuego. Current drilling programs use OREAS series CRMs; OREAS-501b, OREAS-502, and OREAS-503b combined represent approximately half of the total CRM population (Figure 12.1 and Figure 12.2).

Table 12.1 : 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 12.1 : 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%	
Total Samples	**295,898**	100%	

Figure 12.1 : Percentage distribution of CRMs for Costa Fuego

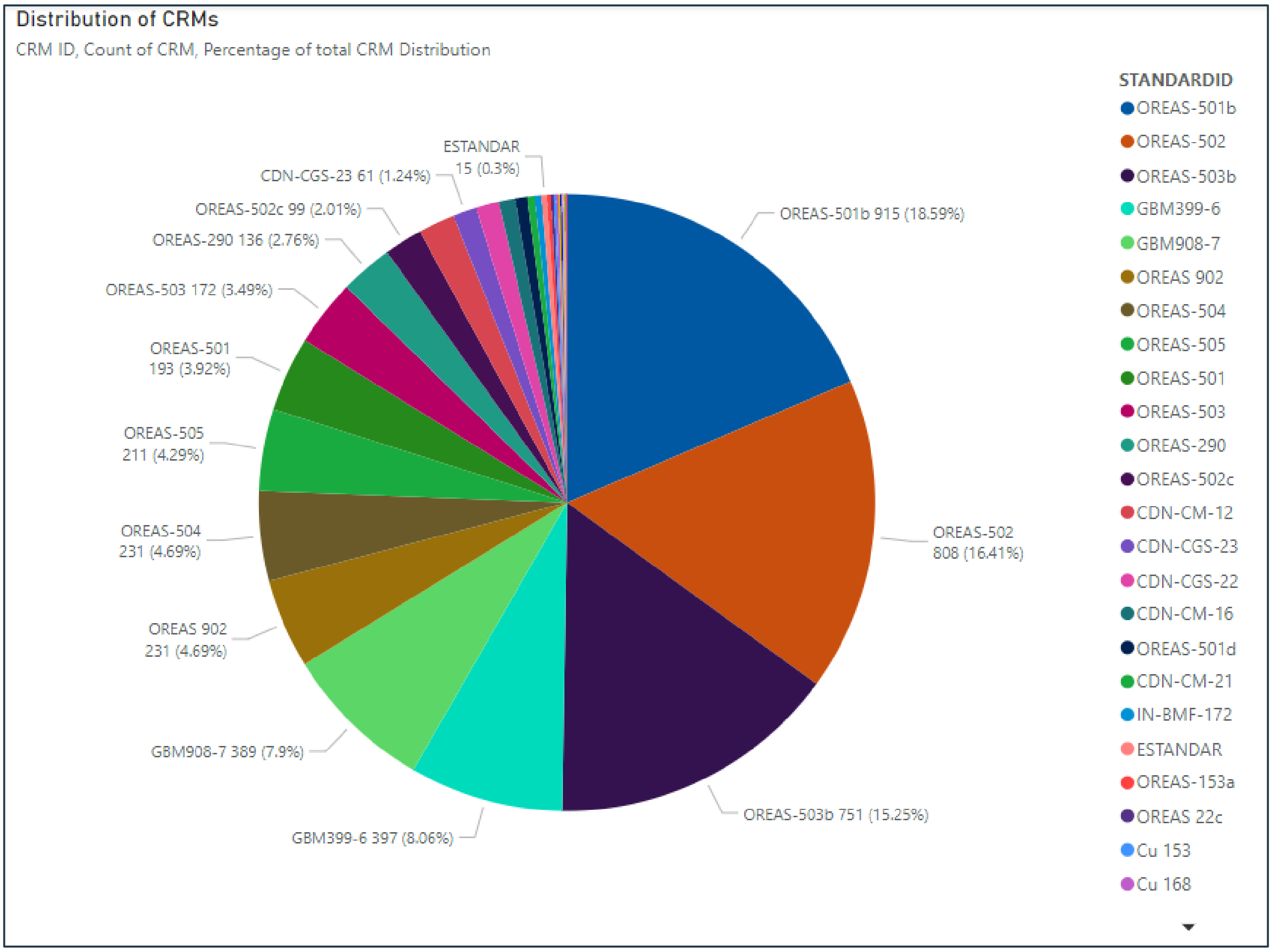
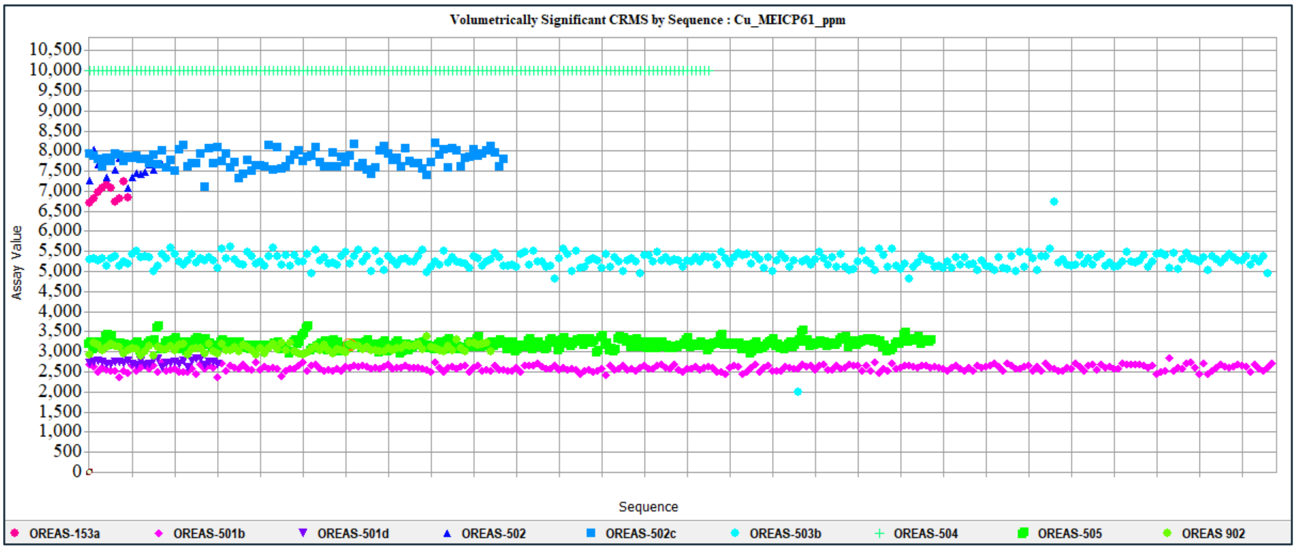


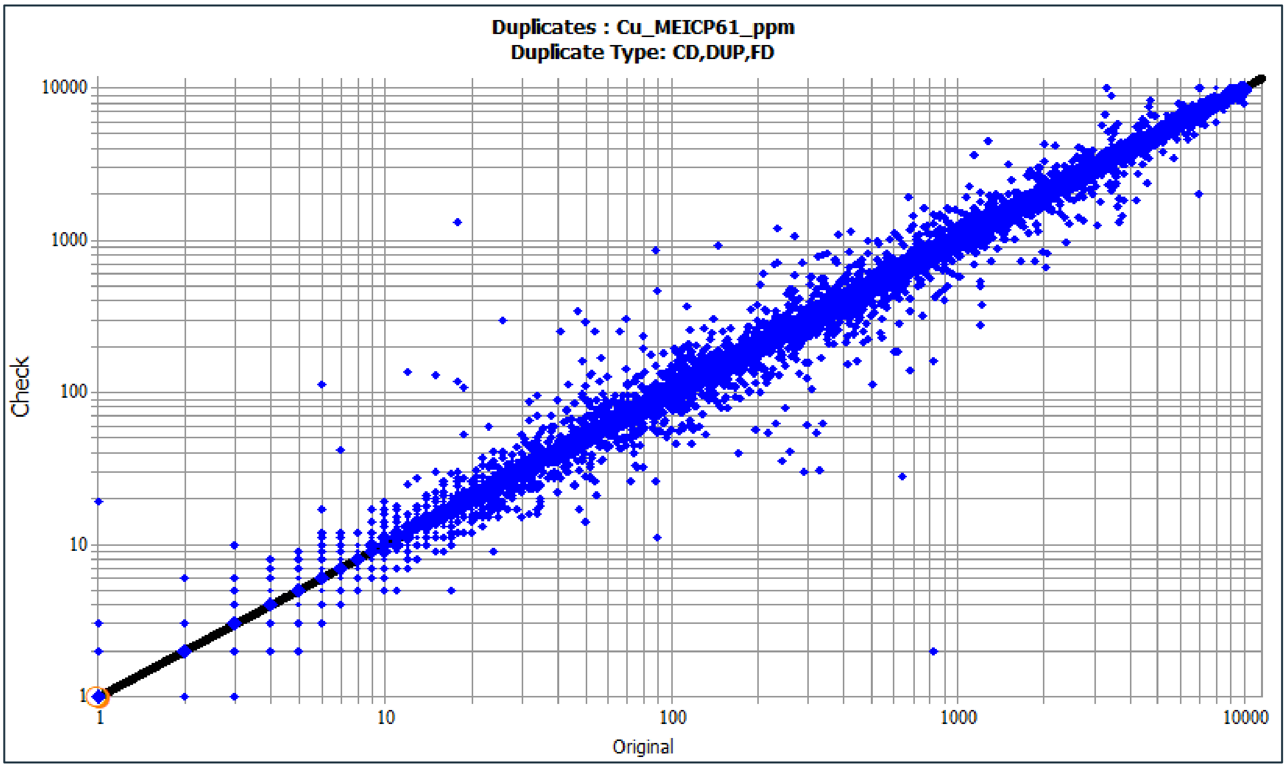
Figure 12.2 : CRM summary for all projects for most volumetrically significant CRMS showing coverage across a wide range of Cu ppm ranges.



12.1.2 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 Costa Fuego are shown in Figure 12.3, a strong correlation across all copper value ranges.

Figure 12.3 : Duplicate pair log-log scatter plot for Cu ppm across Costa Fuego



12.2 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 12.2). 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 12.2 : Summary of common copper CRM proportions and expected values used at Costa Fuego

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**				

12.2.1 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 12.4 to Figure 12.6).

Figure 12.4 : OREAS-501b results for Cu ppm for Cortadera

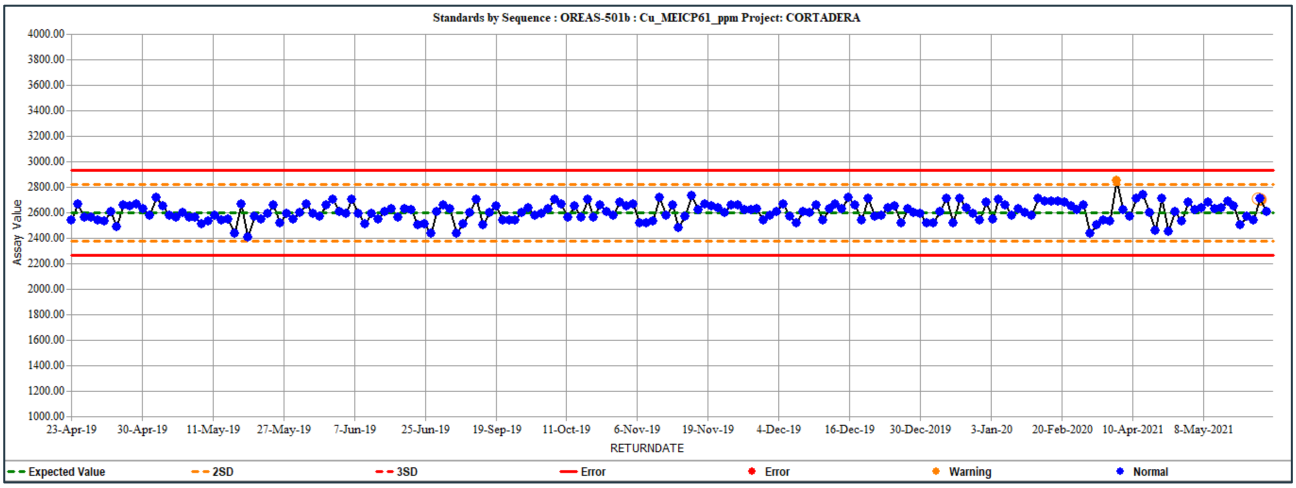


Figure 12.5 : OREAS-501b results for Cu ppm for Productora.

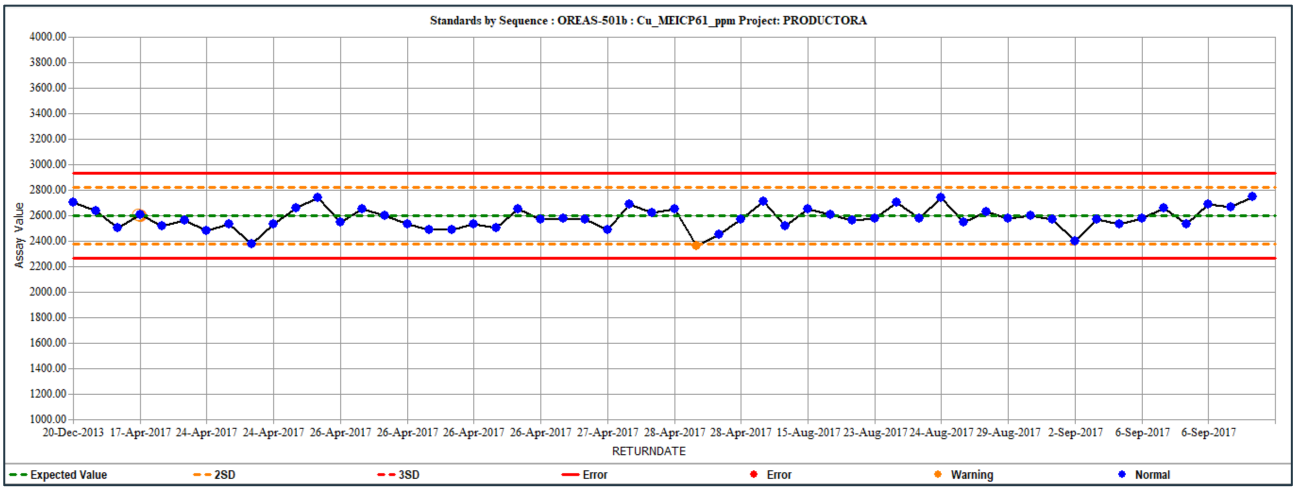
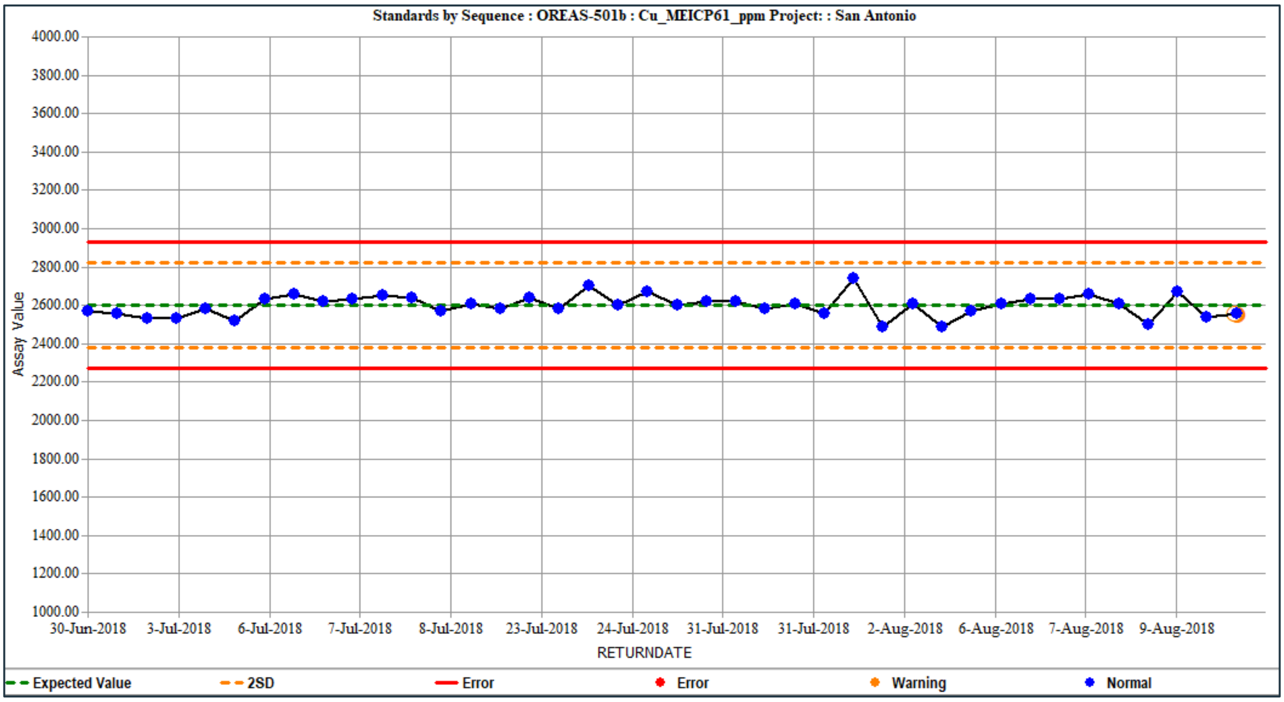


Figure 12.6 : OREAS-501b results for Cu ppm for El Fuego San Antonio



12.2.2 OREAS-502

Performance of this standard was excellent, with no results outside of the expected values at Cortadera and Productora (Figure 12.7 and Figure 12.8). No samples were used at El Fuego San Antonio.

Figure 12.7 : OREAS-502 results for Cu ppm for Cortadera.

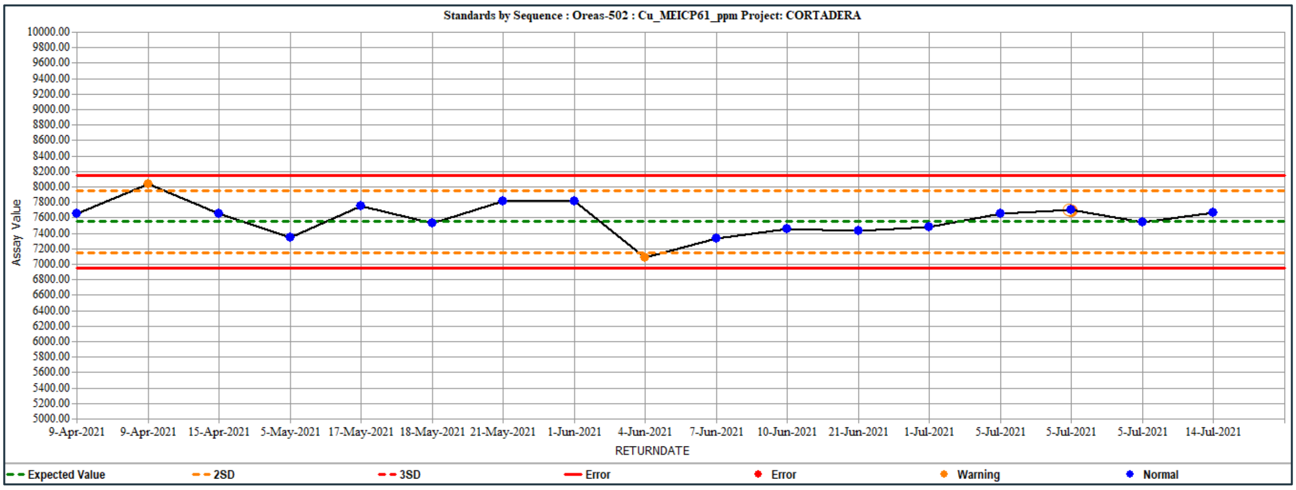
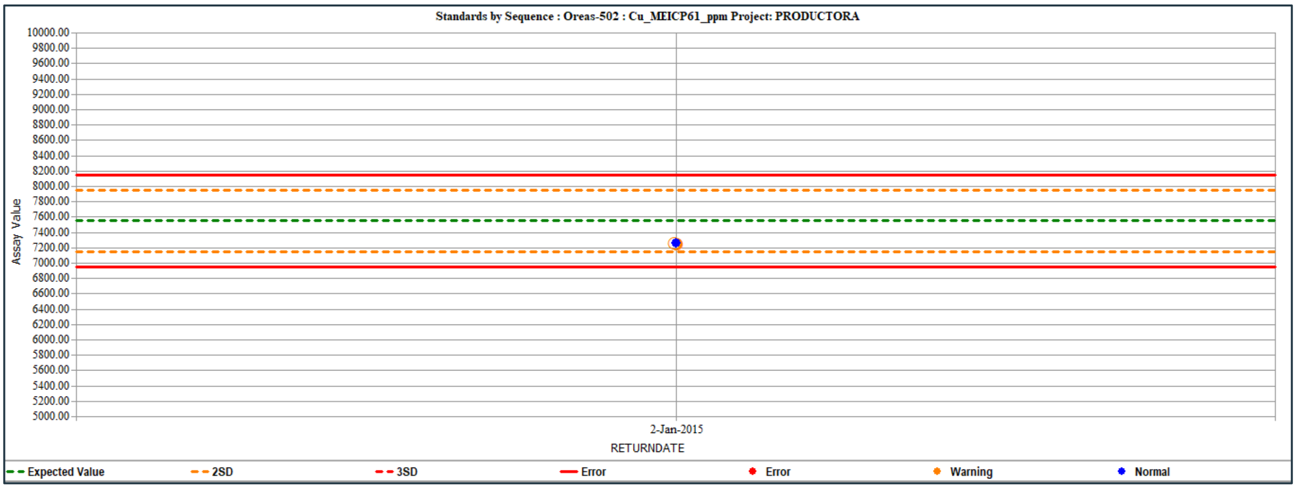


Figure 12.8 : OREAS-502 results for Cu ppm for Productora.



12.2.3 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 12.9 to Figure 12.11). All samples at Productora and El Fuego San Antonio also passed.

Figure 12.9 : OREAS-503b results for Cu ppm for Cortadera

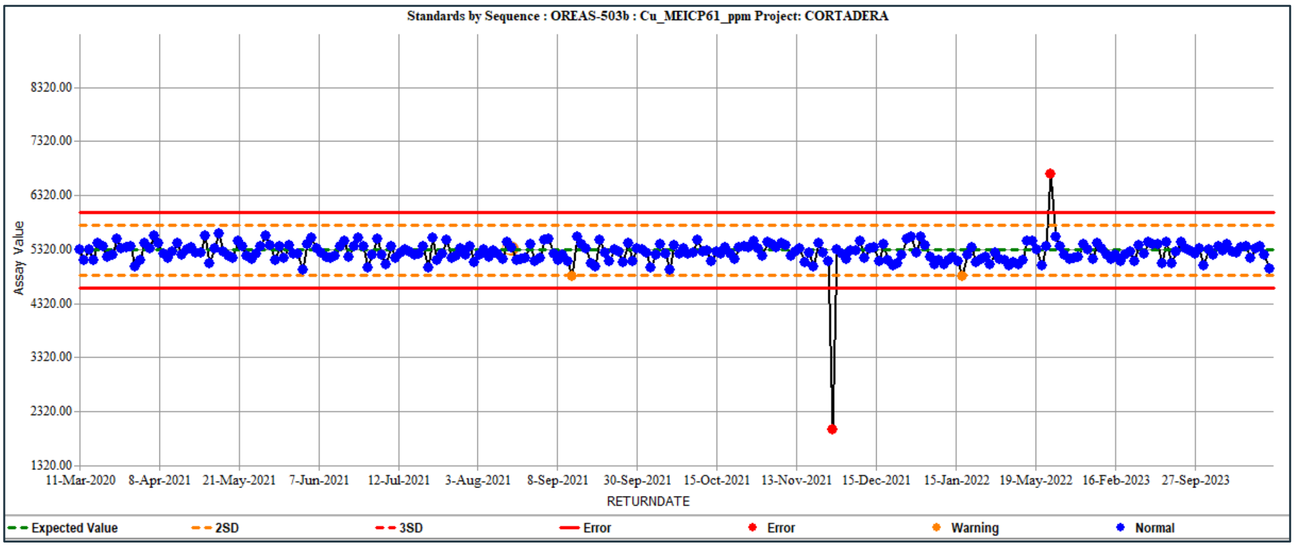


Figure 12.10 : OREAS-503b results for Cu ppm for Productora.

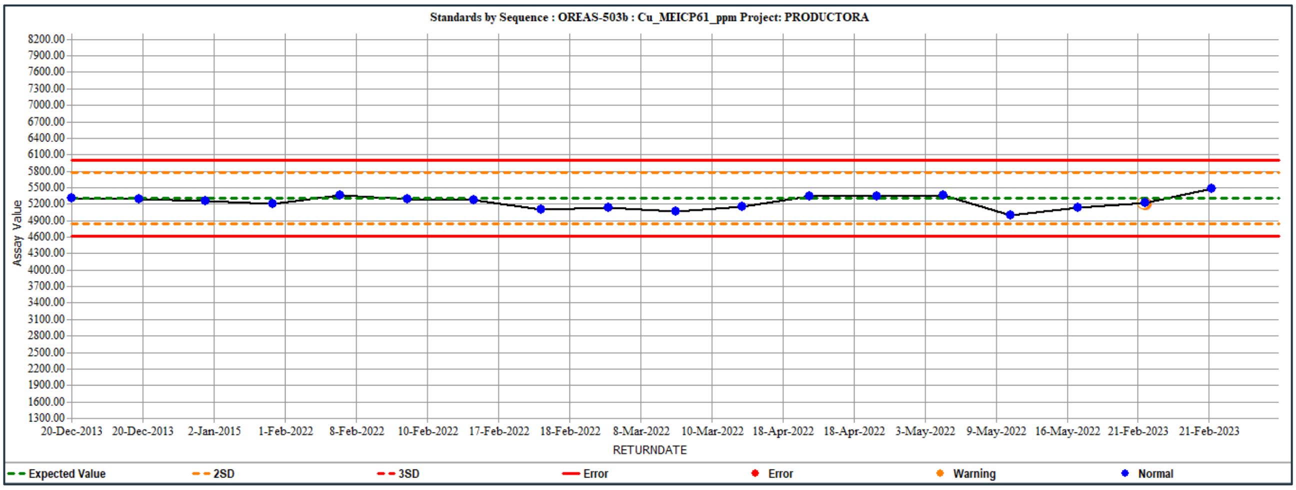
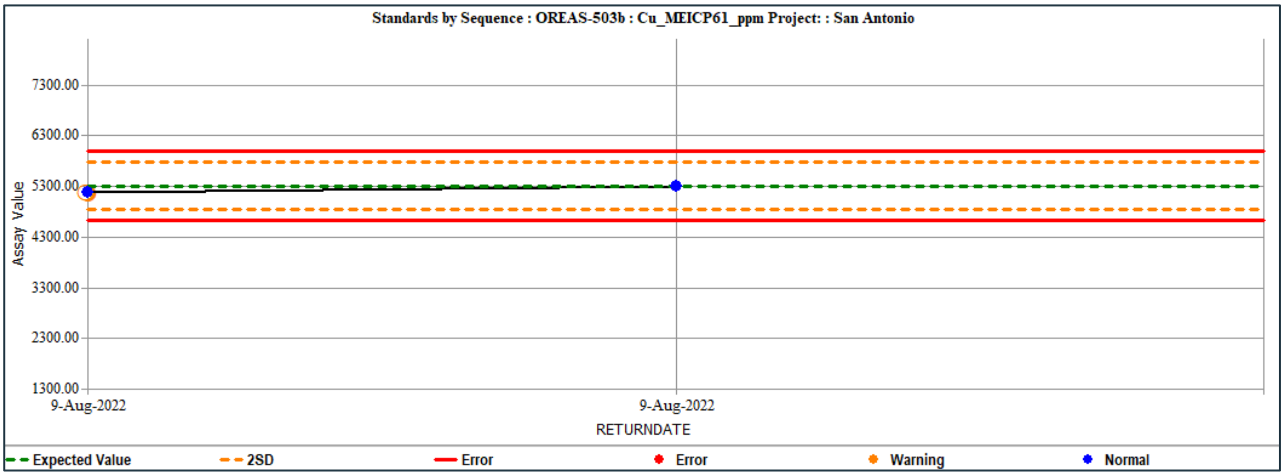


Figure 12.11 : OREAS-503b results for Cu ppm for El Fuego San Antonio



12.3 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 12.12 to Figure 12.14 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 12.12 : OREAS 22c (Blank) results by Cu ppm for Cortadera.

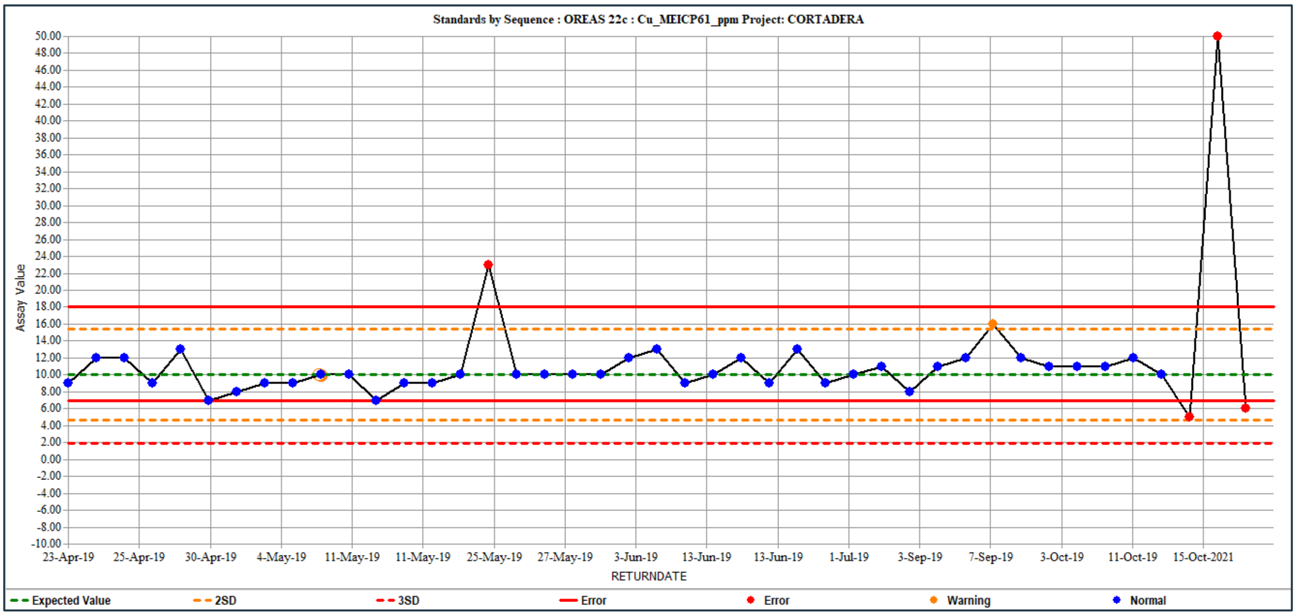


Figure 12.13 : OREAS 22c (Blank) results by Cu ppm for Productora.

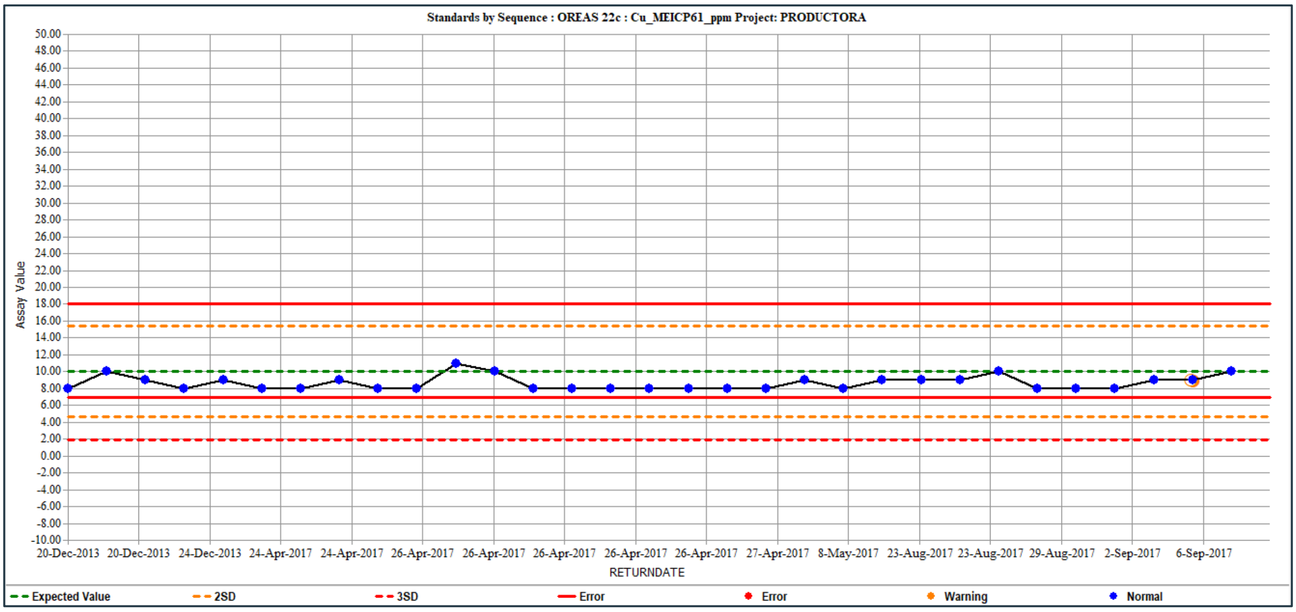
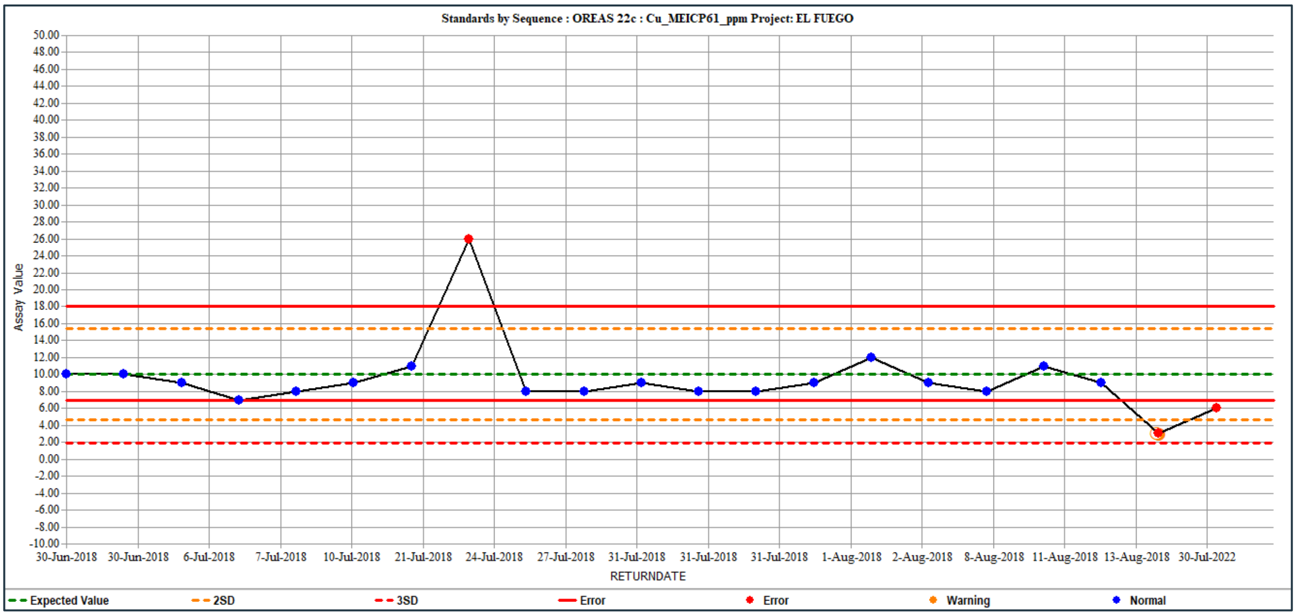


Figure 12.14 : OREAS 22c (Blank) results by Cu ppm for El Fuego San Antonio



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 12.15 to Figure 12.17). 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 12.15 : Quartz blank results for Cu ppm for Cortadera

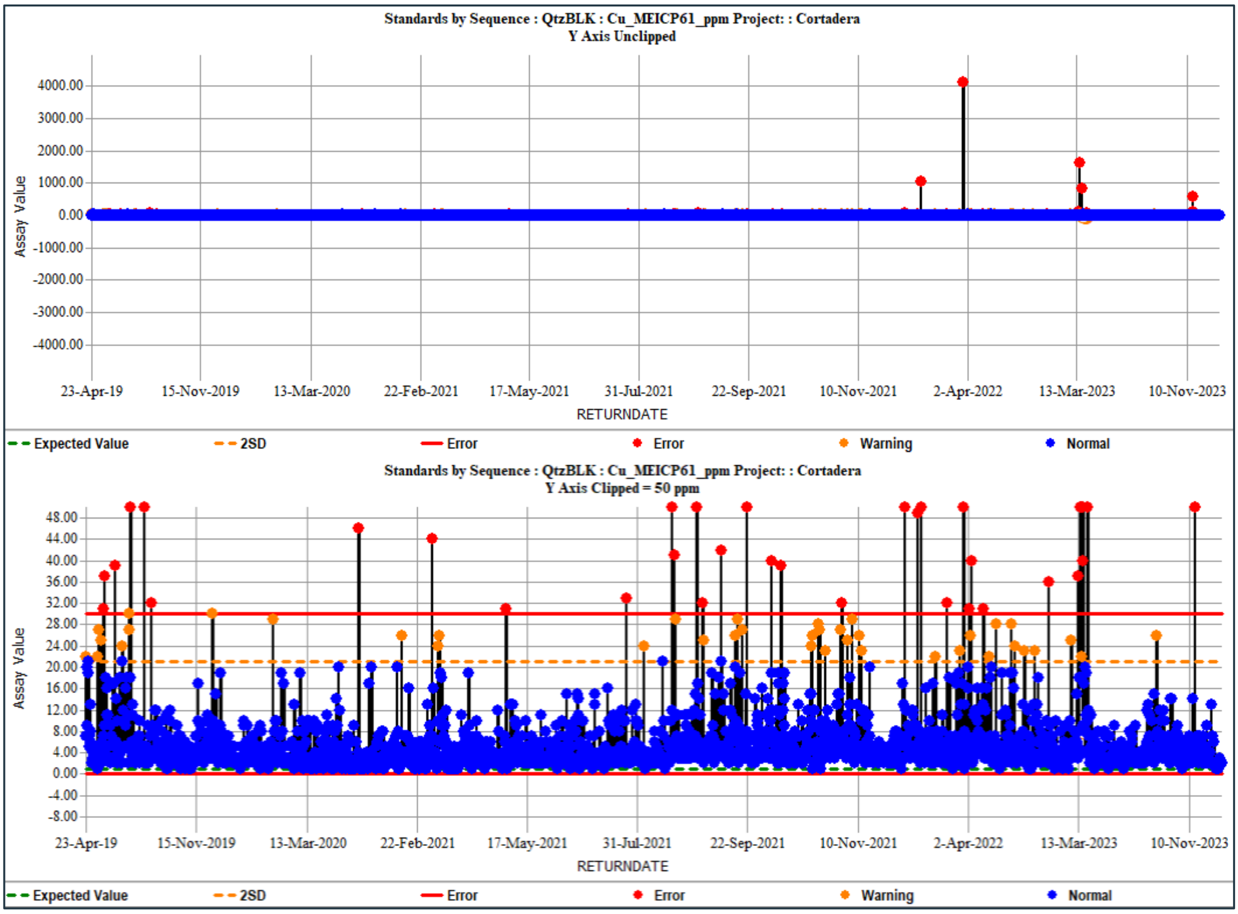


Figure 12.16 : Quartz blank results for Cu ppm for Productora

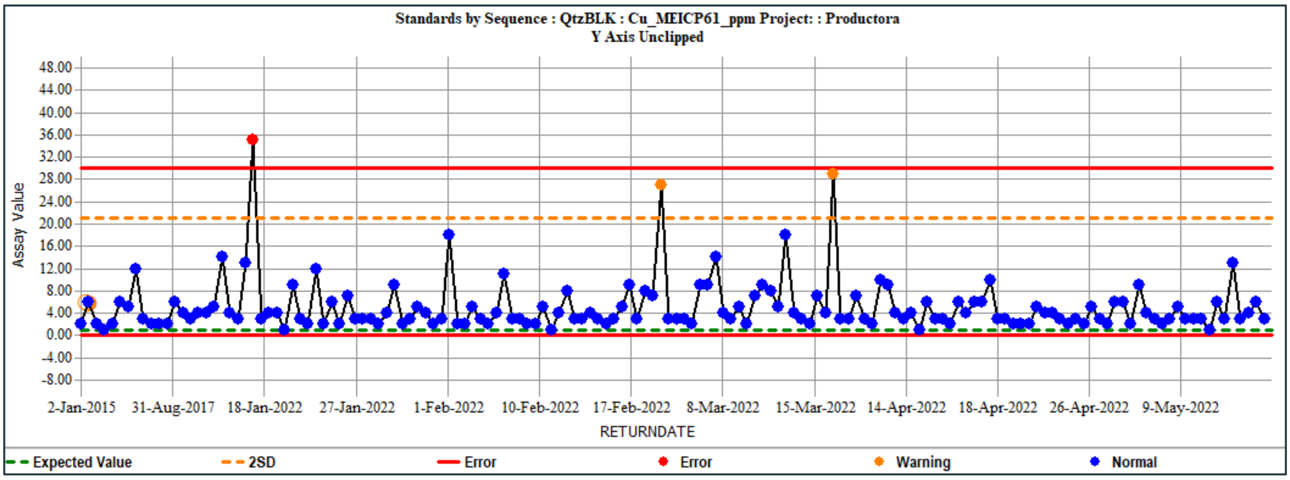
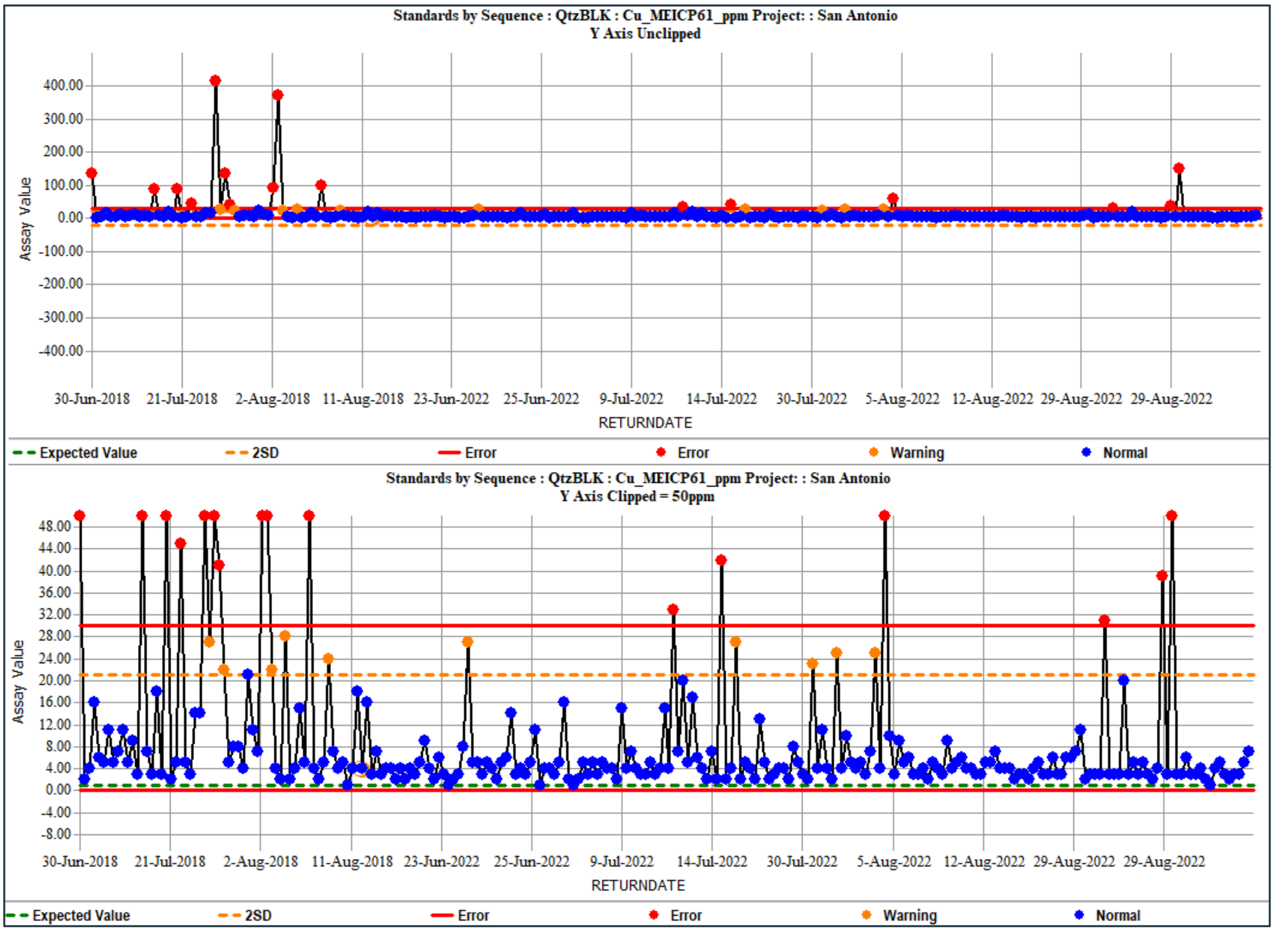


Figure 12.17 : Quartz blank results for Cu ppm for El Fuego San Antonio



12.4 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 12.3).

Table 12.3 : 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 12.18 to Figure 12.20).

Figure 12.18 : Duplicate results for Cu ppm – Cortadera

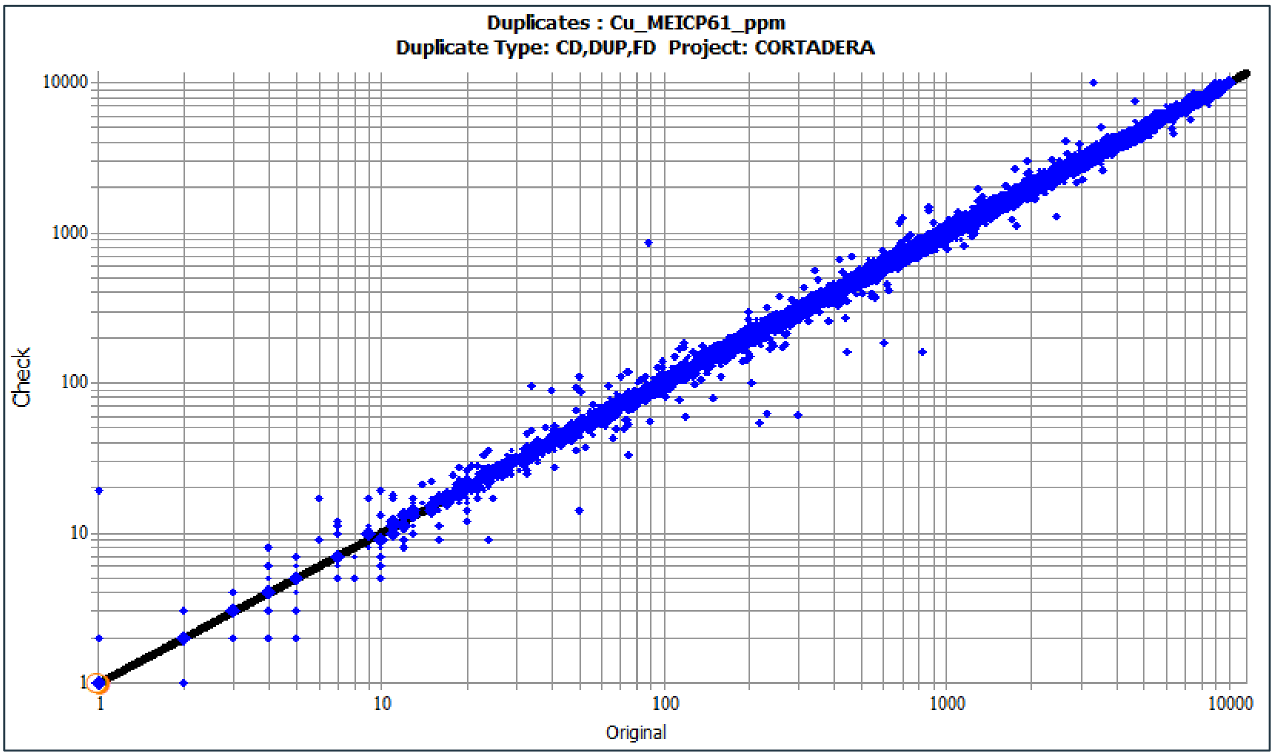


Figure 12.19 : Duplicate results for Cu ppm – Productora

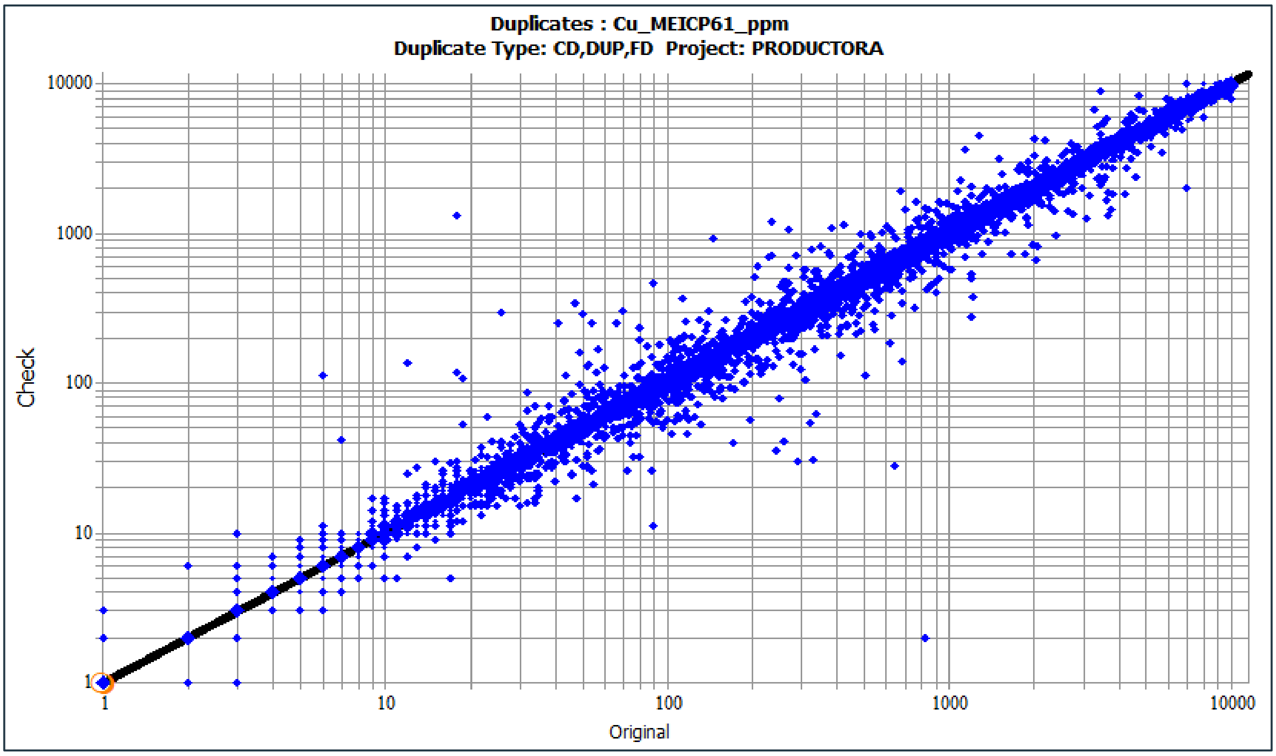
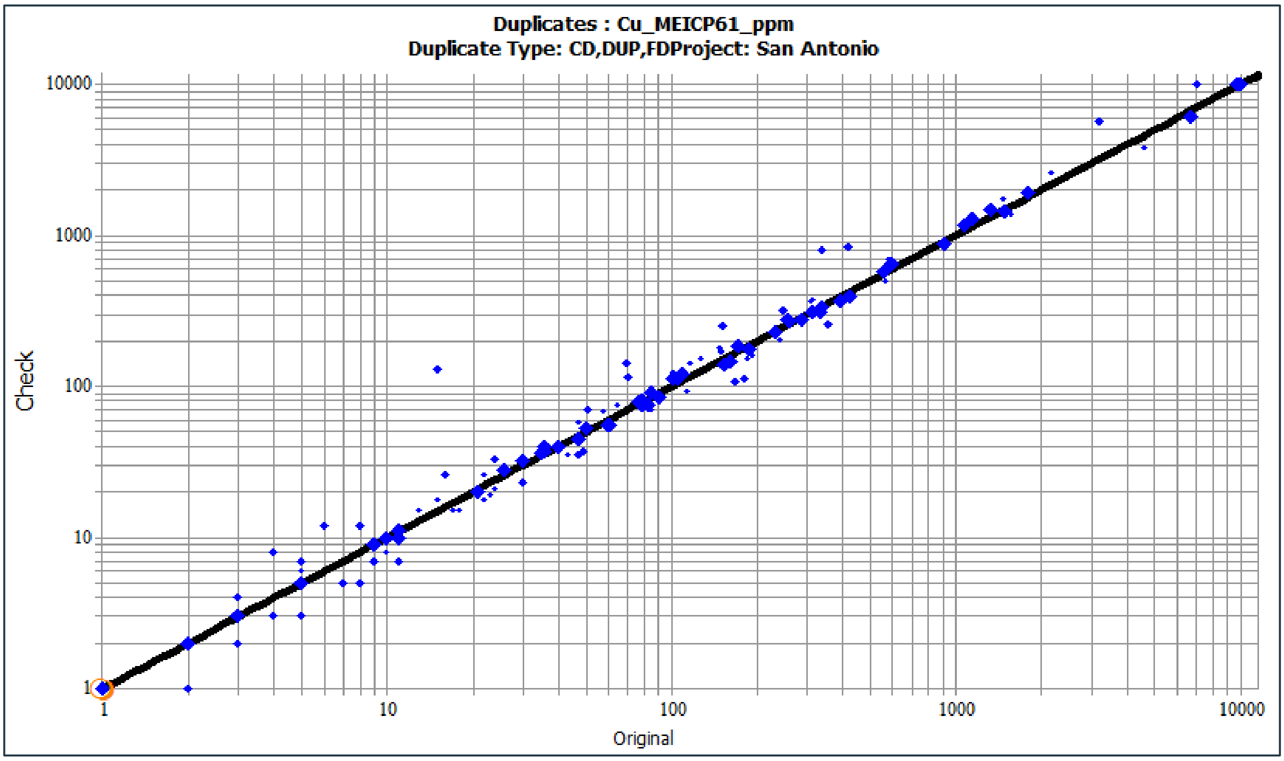


Figure 12.20 : Duplicate results for Cu ppm- EL Fuego San Antonio



12.5 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.

12.6 Independent Qualified Person Review and Verifications

12.6.1 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.

12.6.2 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.

Subsequent verification of the Productora database was completed by the QP in 2021, during review of the Productora Mineral Resource Estimate.

12.6.3 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.

12.6.4 2024 Costa Fuego MRE Independent Review

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.

12.7 Verifications by Wood

Chapter 13 – Mineral Processing and Metallurgical Testing

As part of data verification exercise, the metallurgical samples and procedures selected for the analytical testing the 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 suitable for the assessment of the mineral processing and metallurgical testwork.

Chapter 17 – Recovery Methods
As part of data verification exercise, the 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 recovery methods described in the PEA.

Chapter 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.

Chapter 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.

Chapter 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.

Chapter 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 the technical 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.

12.8 Verifications by ABGM

Chapter 15
Mineral Reserves are not estimated in this Preliminary Economic Assessment.

Chapter 16 Mining Methods
No third-party audits were conducted on the Costa Fuego data, block models or Mineral Resource Estimates as the internal data verification and quality assurance programs were deemed adequate by the Qualified Person for the Mineral Resource Estimation. 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.

12.9 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 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.

12.10 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.

12.11 Comments on Section 12

The QP is satisfied with the exploration, sampling, security, and QA/QC procedures employed by HCH for Costa Fuego and that their results are sufficient to produce data suitable for the purposes described in this technical 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 PEA.

ABGM is of the opinion that the data pertaining to the mine design and schedule is sufficient to support this PEA.

13 Mineral Processing and Metallurgical Testing

This section has been taken directly from the technical report titled, "Costa Fuego Copper Project: NI 43-101 Technical Report & Preliminary Economic Assessment (PEA)” dated August 14, 2023 with an effective date of June 28, 2023. The Mineral Resource estimate update, as described in this Report, does not materially affect the mineral resource inventory that formed the basis of the 2023 PEA, the mineralization encountered in the additional drill holes was similar to that encountered and tested to support the 2023 PEA, Further, no new scientific or technical information has been developed that would materially affect the outcome of the 2023 PEA and, therefore, the information, results and conclusions of the 2023 PEA are considered current and have been restated into this Report.

13.1 Introduction

This section is a summary of the metallurgical testwork results achieved on the copper/gold/molybdenum sulphide production feed at Productora, the shallow copper/gold/silver/molybdenum sulphide production feed at Cortadera, the deeper (underground) copper/gold/silver/molybdenum production feed at Cortadera, as well as the copper oxide production feed from both Productora and Cortadera.

New testwork reported here was performed to confirm that the Cortadera production feed would respond in a favourable technical and economic fashion using the recovery methodologies developed in the 2016 study. Although ongoing testwork is aimed at proving that improved flowsheets (sulphide and oxide) are applicable across all production feed types, this work has not yet reached its conclusion.

Historical testwork was conducted at SGS Laboratories and ALS Laboratories, Perth, Australia (comminution and flotation), ALS Santiago in Chile (oxide testwork and comminution testwork), Outotec in Perth (sulphide concentrate thickening and filtration) as well as HydroGeoSense (oxide testwork) in the United States of America.

Recent flotation work was carried out at Auralia Metallurgy in Perth with comminution work carried out at both Auralia and ALS in Perth. Recent oxide testwork was carried out at the Independent Metallurgical Operations Pty Ltd (IMO) Laboratory in Perth.

Average anticipated recoveries for sulphide and oxide material, broken down by mine area, are shown below in Table 13.1 and Table 13.2 respectively.

Table 13.1 : Average[2] Recoveries Applied to Sulphide Material for Each Mine Area

Deposit	Average Recovery to Concentrate (%)				# Samples
	Copper	Gold	Silver	Molybdenum	
Productora	87	56	-	52	19
Alice	91	51	-	67	5
Cortadera Open Pit	77	44	27	50	19
Cortadera Block Cave	90	58	38	69	25
San Antonio	93	70	65	50	1
Average	**87**	**56**	**37**	**58**	

[2] Average for 'Recovery to Concentrate' weighted by proportion of copper metal production feed.

Table 13.2 : Average[3] Recoveries Applied to Oxide Material for Each Mine Area

Deposit	Copper Recovery (%)	# Samples Bottle Roll	# Samples Column	% of Production Feed
Productora	56	22	5	80
Alice	46	3	0	8
Cortadera Open Pit	50	4	0	12
Average	55			

[3] Average for 'Copper Recovery %' weighted by proportion of copper metal production feed.

13.2 Copper Oxide – Productora

13.2.1 Background

Oxide material overlays the transition and sulphide material along the full strike length at Productora and for a substantial amount of the strike length at Cortadera. Productora currently has the dominant share of oxide classified material at the Project. Testing of the Productora oxide production feed was undertaken to establish its amenability to copper recovery by sulphuric acid-based heap leaching. Some tests were also performed on Alice pit material, which is a short distance west of the Northern Oxide zone at Productora.

13.2.2 Amenability to Sulphuric Acid Leaching

13.2.2.1 Introduction

Successful heap leaching of copper oxides relies on achieving economic levels of copper recovery at an acceptable operating cost.

Another key aspect is ensuring that the crushed and agglomerated production feed parameters are in the ranges suitable for successful heap construction and operation.

13.2.2.2 Preliminary Heap Leach Characterisation – Production Feed Samples

Table 13.3 lists the samples selected for oxide copper recovery testwork. For all samples except CC7, D1 and AL2, the acid soluble copper level is greater than 38%. Sample CC7 is more typical of a transition (partially oxidised and mineralogically altered) production feed and Sample D1 is more typical of a fresh (sulphide mineralisation only) production feed.

Table 13.3 : Oxide Samples for Potential Testing

MET Sample	Domain	Avg Depth (m)	Head Cu%	Head Acid Soluble Cu%	Acid Soluble Cu Proportion
NO1	Nth Ox	10	0.46	0.40	87.0%
NO2	Nth Ox	33	0.52	0.43	82.7%
NO3	Nth Ox	33	0.49	0.33	67.3%
D4	Nth Ox	21	0.37	0.35	94.6%
SM1	Sth Mantos	52	0.65	0.30	46.2%
SM2	Sth Mantos	18	0.44	0.17	38.6%
CC1	CCHEN sth	47	0.56	0.37	66.1%
CC2	CCHEN sth	91	0.65	0.25	38.5%
CC2a	CCHEN sth	101	0.74	0.60	81.1%
CC3	CCHEN sth	60	0.47	0.24	51.1%
CC4	CCHEN sth	112	0.60	0.31	51.7%
CC7	CCHEN sth	98	0.80	0.18	22.5%
CC8	CCHEN sth	91	0.58	0.22	37.9%
D1	CCHEN Sth	110	0.25	0.02	8.0%
SI1	Santa In	10	0.40	0.24	60.0%
SI2	Santa In	37	0.59	0.26	44.1%
SI4	Santa In	10	0.44	0.26	59.1%
D3	Santa In	48	0.17	0.10	58.8%
Al1	Alice		1.12	0.93	83.0%
Al2	Alice		0.48	0.16	33.3%
Add_Br_1	East Oxide		1.22	0.98	80.3%
Add_Br_2	East Oxide		0.44	0.26	59.1%

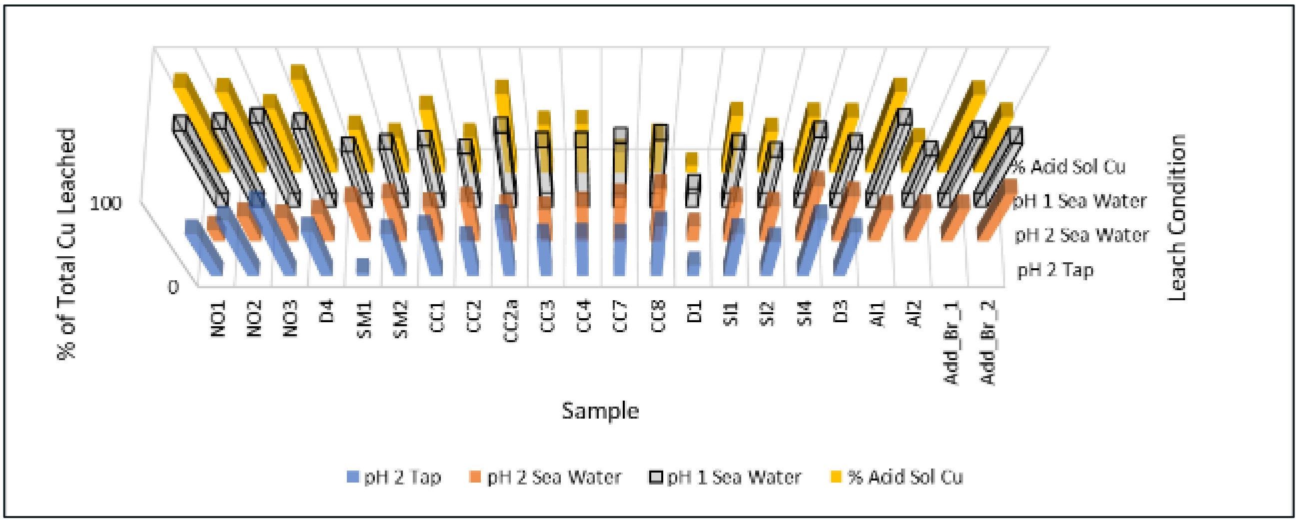
The samples encompass a broad range of oxidation levels and have been selected to provide a good representation of the Productora oxide types.

13.2.2.3 Bottle Roll Tests

The results of the bottle roll tests are summarised in Figure 13.1 where they are shown in relation to the percent acid soluble copper values from Table 13.3.

Bottle roll tests were also conducted on samples selected for 1 m columns in order to establish leaching parameters for the column tests.

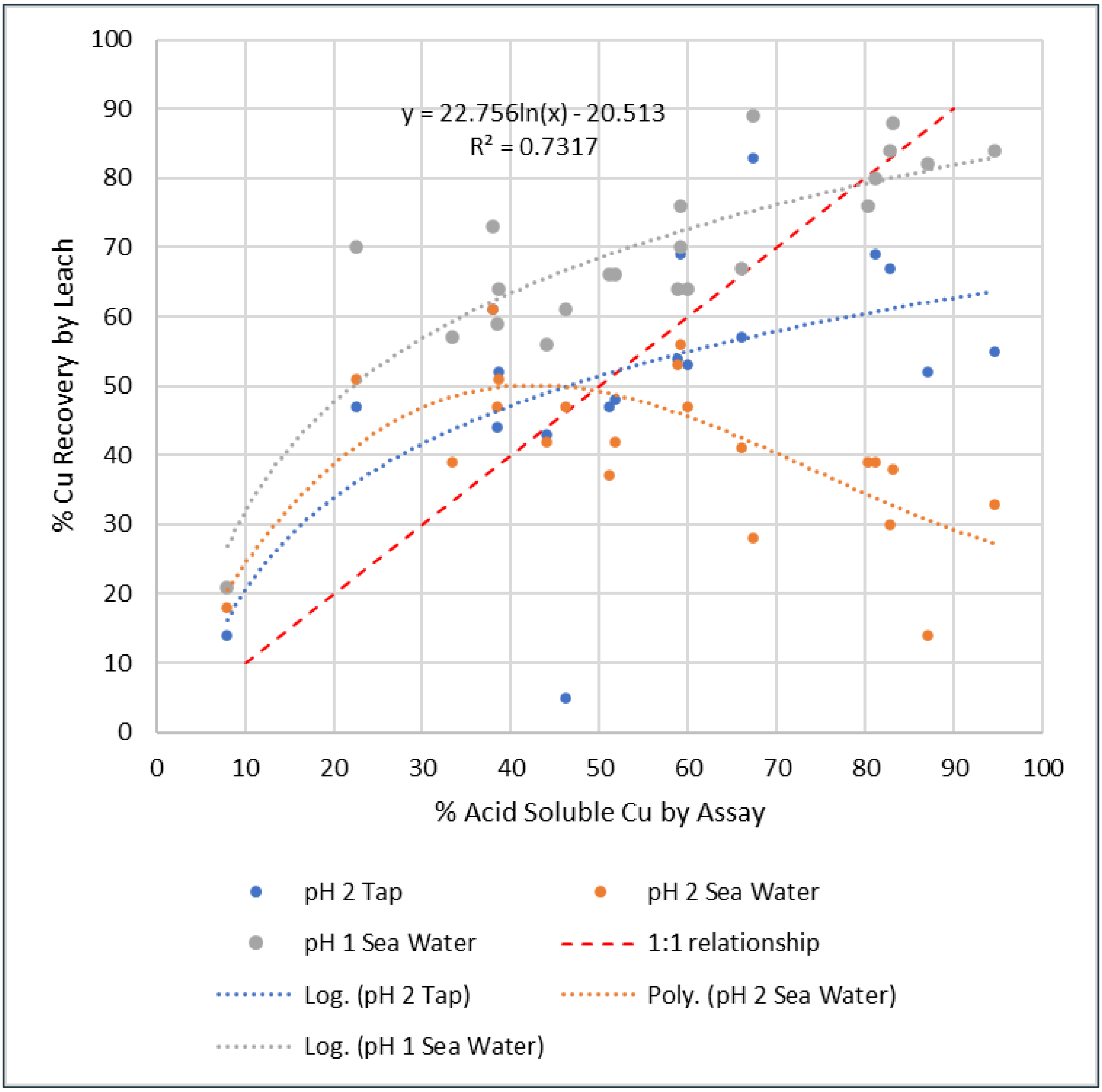
Figure 13.1 : Relative Copper Recoveries by Bottle Roll Leach Condition



The results show that the pH 2 seawater results were the poorest and the pH 1 seawater results were the best. The pH 2 tap water results were between these and had some inconsistent outcomes, as can be seen for sample SM1.

The data was re-plotted to evaluate the relationship between the acid soluble copper percentage and the leaching results (Figure 13.2). Note that if the geological measure of percent acid soluble copper was a perfect indication of leaching recovery, the data would follow the 1:1 line.

Figure 13.2 : Predictability of Copper Leach Recovery



The pH 1 seawater results are clearly above the 1:1 line for the majority of cases and give the best overall leaching results, as demonstrated previously. The equation on the graph is for prediction of copper leach recovery (y) from the % Acid Soluble Cu by Assay (x), a geological parameter. If pH 1 conditions are to be approximated in the heap leach, then this equation can be used as the basis for heap leach recovery prediction in mine planning. However, a recovery reduction factor may need to be considered to limit acid consumption to viable economic levels.

The pH 2 tap water results generally follow the pH 1 seawater trend, but at a significantly lower recovery level. The pH 2 seawater results are unusual as they show that leaching is reasonable for production feed showing intermediate levels of oxidation, but that the recovery decreases as the oxidation level increases. This is an unexpected result and is a warning against allowing acid leach conditions to fall below optimal (i.e. allowing the pH to rise too high).

Acid consumptions in the bottle roll tests were not indicative of operational values and were designed to accelerate copper dissolution to provide a set of leaching reference recoveries for comparison with column testing results. The average acid consumption across all bottle roll tests was 70 kg/t, far higher than expected in operations.

13.2.3 Crushing Testwork

A summary of the results of the crushing/comminution testing is shown in Table 13.4.

Table 13.4 : Crushing and Unconfined Compressive Strength (UCS) Test Results	
Specific gravity	2.62
Bond CWi (18 samples)	
Minimum	1.3 kWh/t
Maximum	29.7 kWh/t
Average	7.9 kWh/t
Standard deviation	6.9 kWh/t
UCS (6 samples)	17.9 MPa (6.1 to 32.4)

These results indicate a moderately hard production feed that would require three stages of crushing to achieve a target crush size P_{80} in the range 25 mm to 12.5 mm. The unconfined compressive strength (UCS) values are very low and suited to the use of a twin roll sizer.

13.2.4 Heap Agglomeration, Stacking and Hydrodynamic Testing

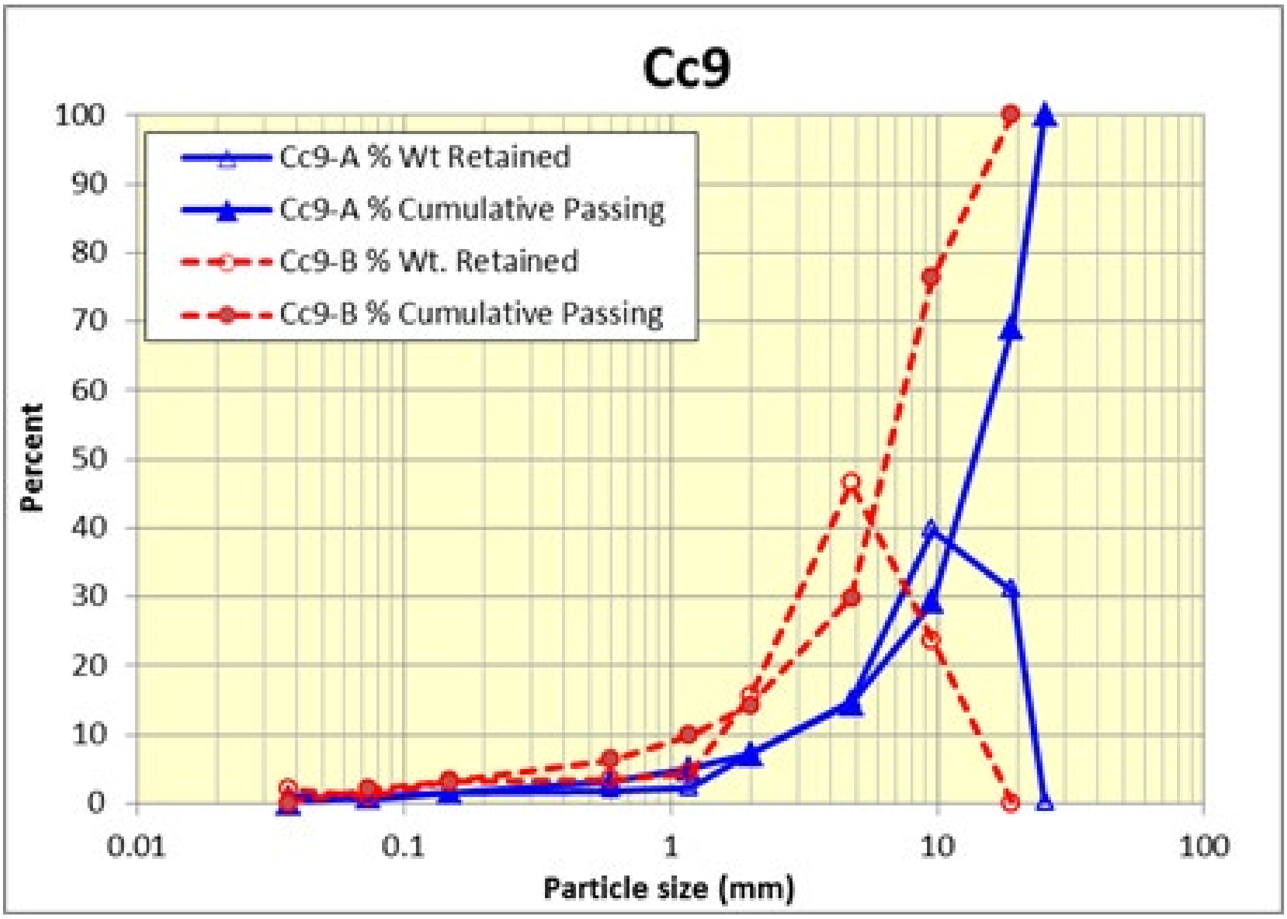
13.2.4.1 Testwork Summary

This work was carried out by HydroGeoSense Laboratory, specialists in heap leach production feed characterisation and heap leach system design.

The test program involves the assessment of a number of physical production feed properties, and these were assessed at two crush sizes, 25 mm and 12.7 mm P_{80} (determined after preliminary work). The samples were also assessed at a variety of agglomeration effectiveness levels from L1, the least agglomerated, (similar to mixing achieved at a conveyor transfer) to L5, effectively perfect agglomeration (rarely achieved in operations but a good reference). These levels are determined by comparison of agglomerated sample appearance and measured outcomes with standards that have been developed by the laboratory. Agglomeration levels achieved with the Productora samples ranged from L1 to L3. The behaviour of samples in stacking tests was determined at the two crush sizes and two agglomeration levels for each sample.

The samples for testing were crushed to minus 25.4 mm (Sample A) and minus 12.7 mm (Sample B). Size distributions were then determined for each crushed sample. It must be noted that crushed samples are almost devoid of sub 100 µm material and this is due to the use of relatively solid core. Typical examples of the crushed product size distributions are shown in Figure 13.3. Blasting and handling would generate more sub 100 µm material in operations (compared to crushed core) and this needs to be considered when interpreting the results. The lack of fines in test samples compared to operational practice would mean that the test results are conservative.

Figure 13.3 : Particle Size Distribution for Crushed Main Pit 9 (Cc9)



The agglomeration levels (and associated moisture contents) of the test samples are shown in Table 13.5. An "A" designation is a -25.4 mm sample, and a "B" designation is -12.7 mm.

Table 13.5 : Agglomeration Level and Agglomeration Moisture Content

Test ID	Agglomeration Level	Moisture Content (%)
Cc9-A-L1	L1	2.4
Cc9-A-L2	L2	4.0
Cc9-B-L1	L1	3.9
Cc9-B-L2	L2	5.2
No4-A-L3	L3	3.5
No4-B-L3	L3	5.8
Sm3-A-L1	L1	4.8
Sm3-A-L2	L2	5.5
Sm3-B-L1	L1	5.8
Sm3-B-L2	L2	6.5

A positive outcome to a Stacking Test (ST) indicates that the sample is a candidate for percolation heap leaching and further testing can be pursued. Negative results indicate categorically the production feed as prepared would not support percolation leaching.

The results from the ST are summarised in terms of the following:

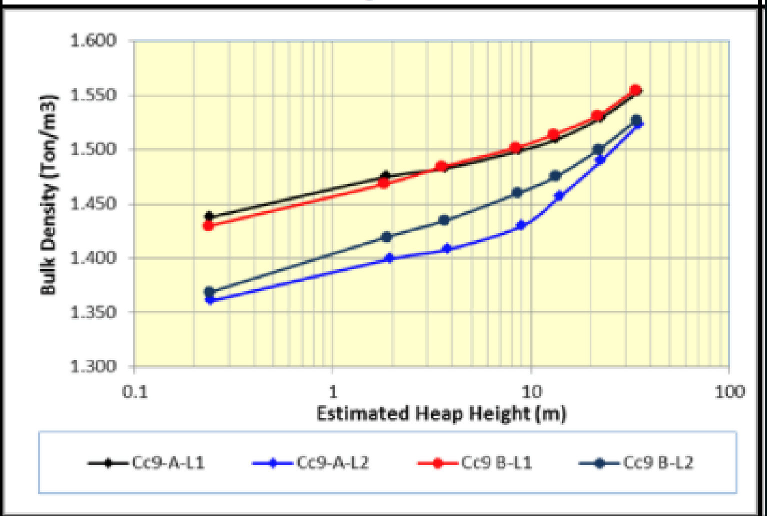
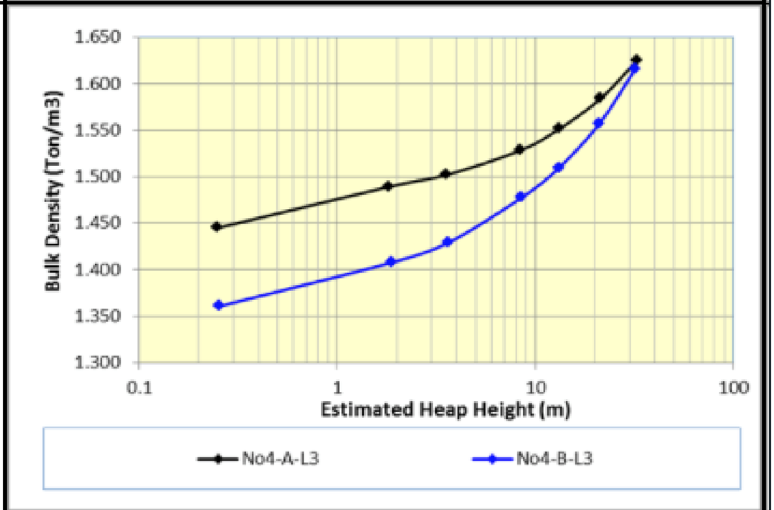
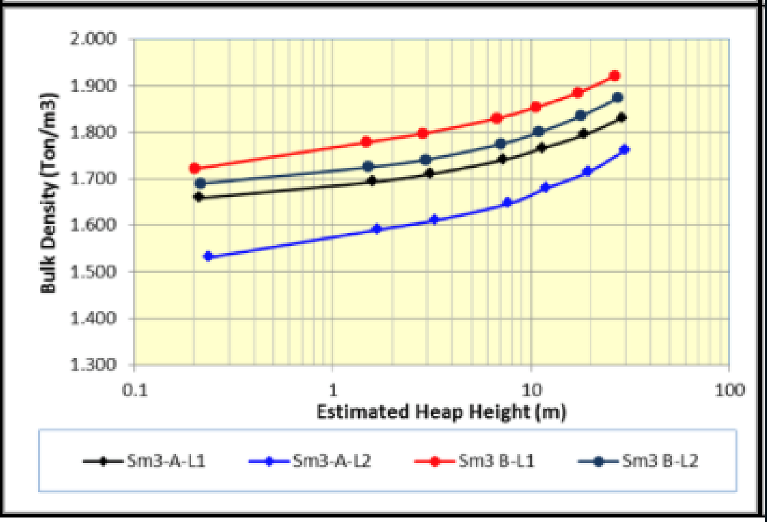
- The density profile
- The conductivity profile
- The partitioning between micro- and macro-porosity.

13.2.4.2 Stacking Density Profile Testing Results

Figure 13.4 shows the stacked production feed density variation with heap height for the samples listed above. In general, as the heap height increases, the density of the stacked material increases in line with expectations. Heap heights up to 32 m were simulated in the experiments.

There are significant differences between the A and B samples, with the finer samples giving higher densities than the equivalent coarse samples in all cases.

Figure 13.4 : Density vs. Potential Heap Height Profiles

<table>
<tr><th>Composite</th><th>Density Profile</th></tr>
<tr><td>Cc9</td><td>
</td></tr>
<tr><td>No4</td><td>
</td></tr>
<tr><td>Sm3</td><td>
</td></tr>
</table>

13.2.4.3 Hydraulic Conductivity Profile

As depicted in the following figures (Figure 13.5 to Figure 13.7), the hydraulic conductivity of all the samples remains above the minimum value of 0.167 cm/s. This figure represents a leach solution application rate which is 100 times greater than the target mid-point application rate of 6 L/h/m^2, even at the maximum heap height tested during this investigation (~36 m). Conductivity is therefore not an issue.

Figure 13.5 : Hydraulic Conductivity vs. Potential Heap Height Profiles

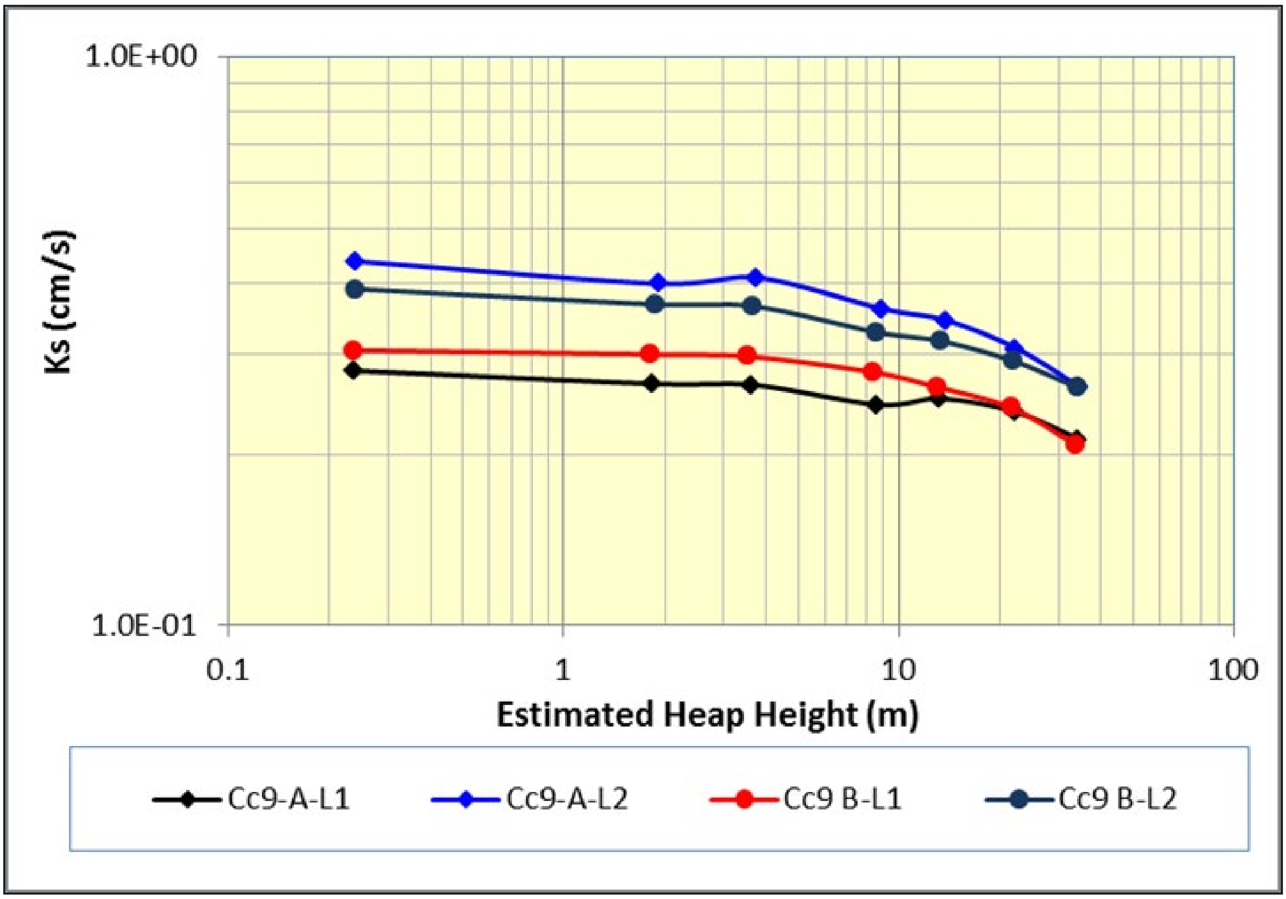


Figure 13.6 : Hydraulic Conductivity vs. Potential Heap Height Profiles

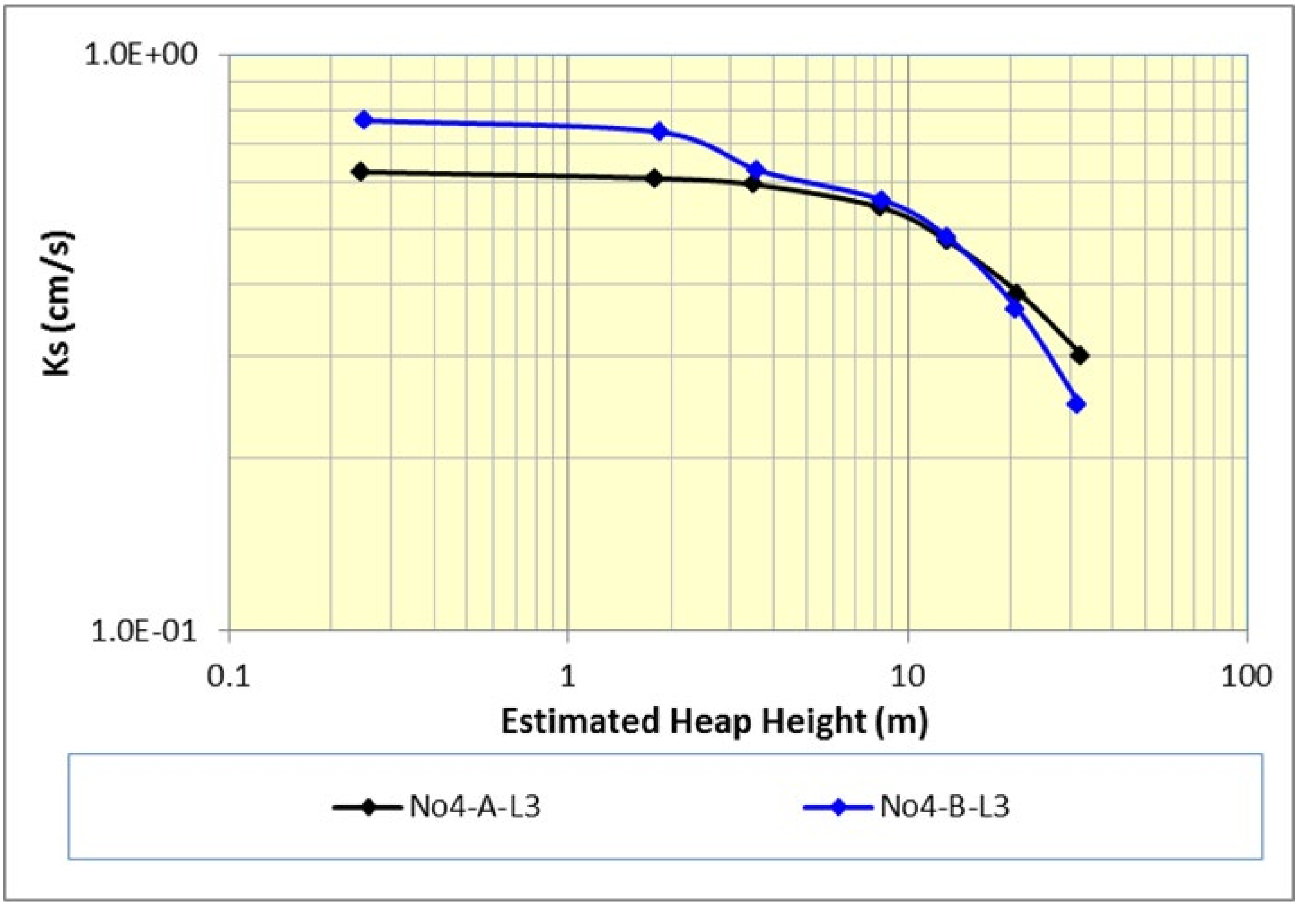
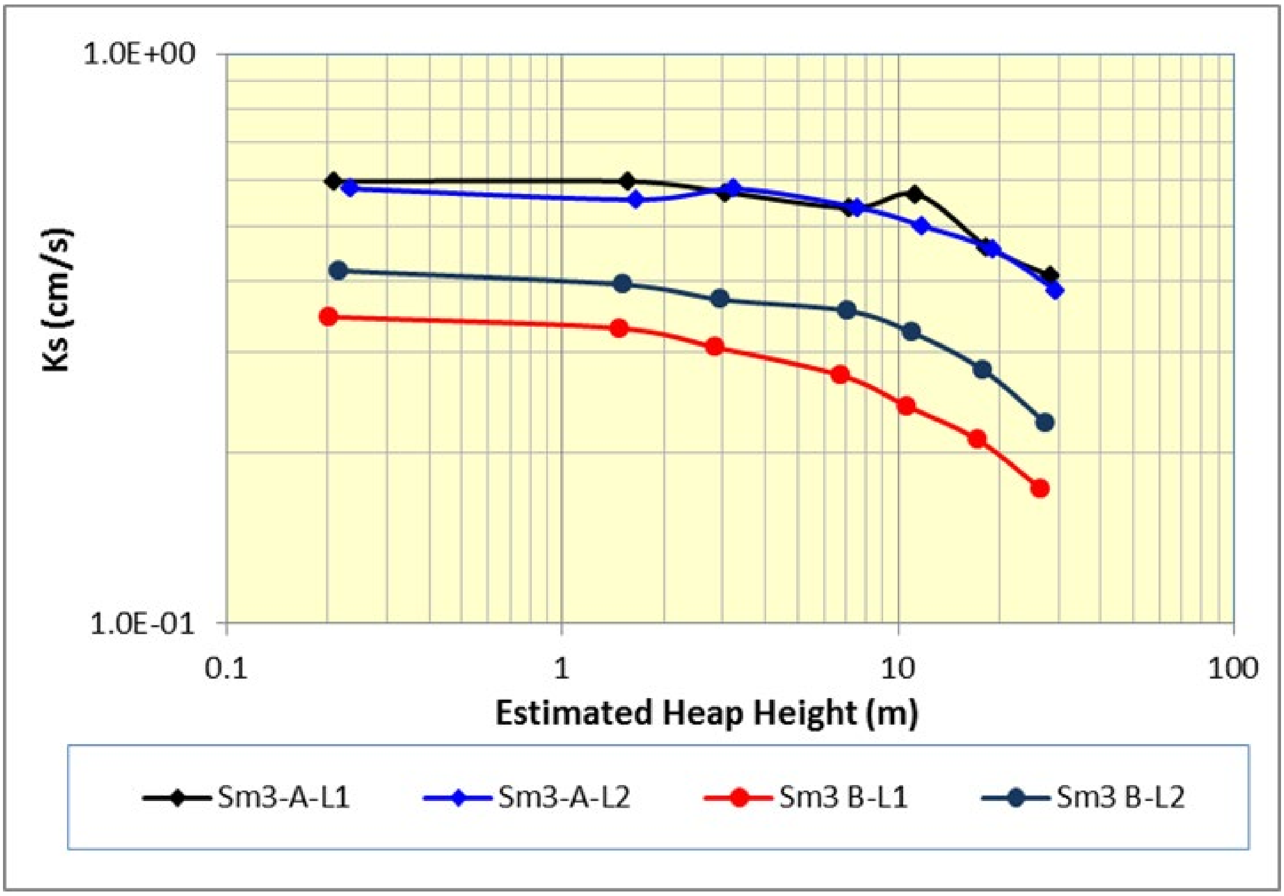


Figure 13.7 : Hydraulic Conductivity vs. Potential Heap Height Profiles

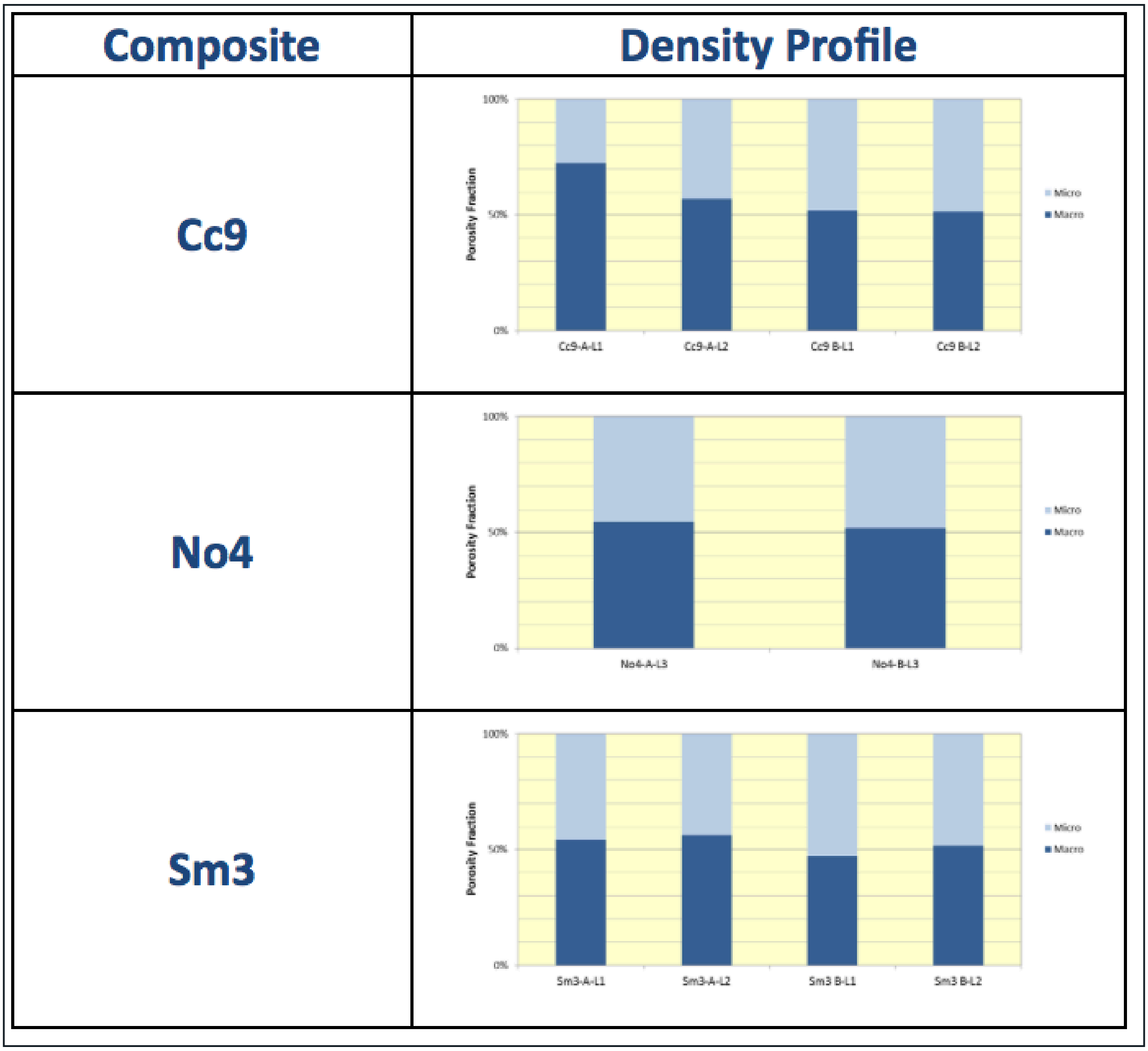


13.2.4.4 Porosity Size Class Partitioning

Partitioning of porosity into micro and macro categories has also been determined (Figure 13.8).

The resulting porous structure produces a total porosity larger than 30% for all of these samples, which is partitioned between macro- and micro-porosity of about 50:50. The work indicates that this ratio allows bulk solution movement and intimate contact between solution and production feed.

Figure 13.8 : Porosity Partitioning



One Hydrodynamic Column Test (HCT) was conducted for each of the B samples (crushed to 12.7 mm) to determine the hydrodynamic response of the bottom portion of a 16 m lift. Table 13.6 summarises the conditions for each of the samples. Importantly, synthetic raffinate solution was prepared for each sample and used during the process of agglomeration.

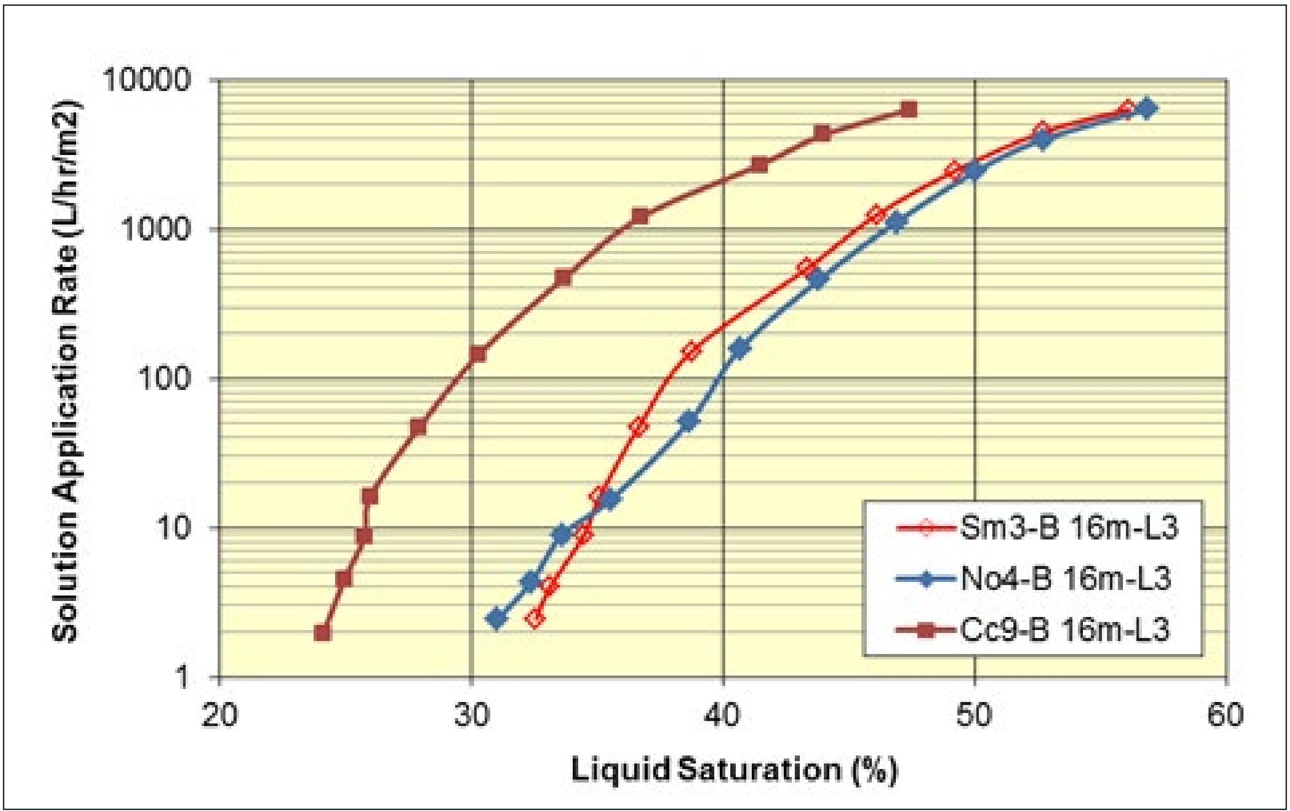
Table 13.6 : Hydrodynamic Columns Testing Conditions						
Sample ID	SG (t/m^3)	Grav. Moisture Content (%)	Initial Liquid Saturation (%)	Dry Bulk Density (t/m^3)	Porosity Partitioning Macro (%)	Porosity Partitioning Micro (%)
Sm3-B 16m (L3)	3.644	7.4	26.2	1.824	47.6	52.4
No4-B 16m (L3)	3.186	6.3	17.9	1.527	55.9	44.1
Cc9-B 16m (L3)	2.734	5.3	16.8	1.485	55.3	44.7

The Macro:Micro porosity partitioning of these samples at agglomeration levels of L3 results in nearly optimal ratios (50:50) in all cases.

13.2.4.5 Saturation vs. Irrigation (Application) Rate

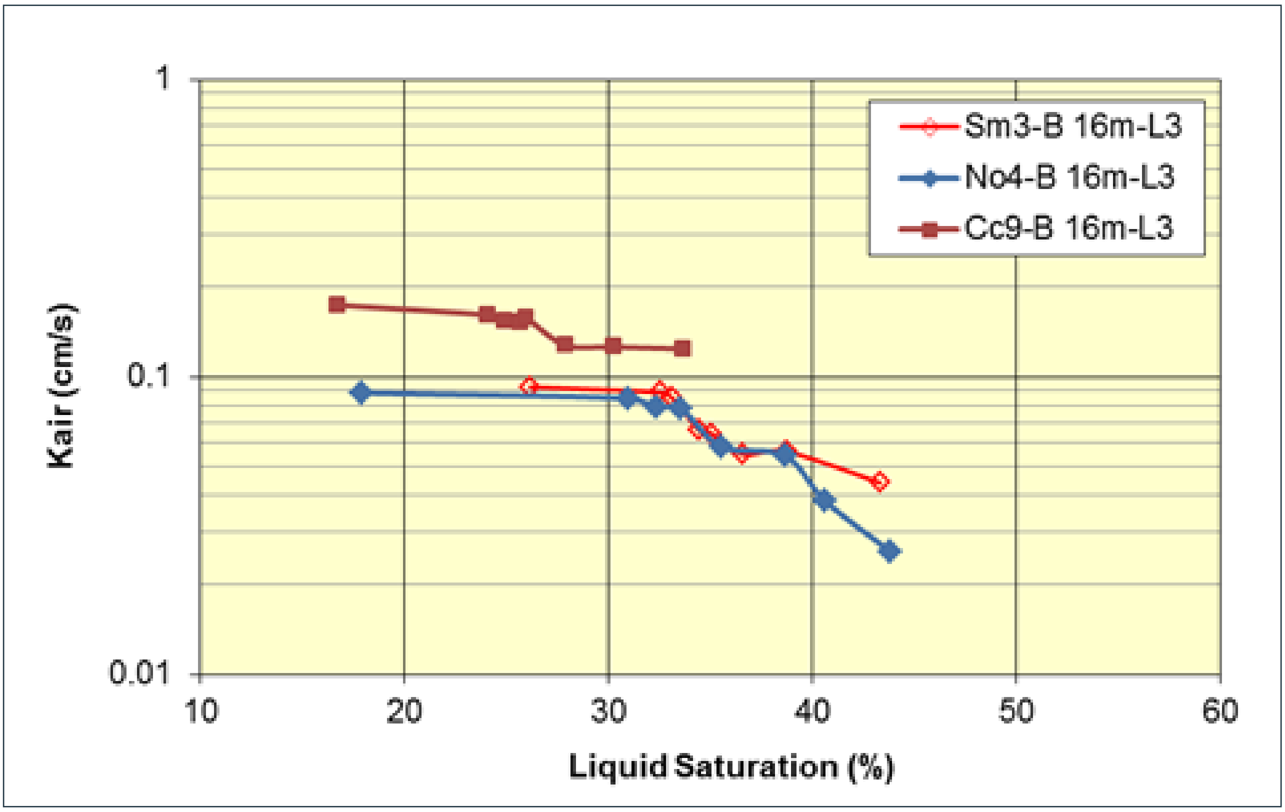
Figure 13.9 shows the degree of saturation associated with various solution application rates imposed during the tests and Figure 13.10 shows the gas conductivity as a function of liquid saturation for each of the samples.

Figure 13.9 : Liquid Saturation at Variable Application Rates



This relation demonstrates that for irrigation rates under 10 $L/h/m^2$, the stacks would be far from saturated, and this is a positive outcome.

Figure 13.10 : Gas Conductivity



The gas conductivity rates at the low operational saturation levels are also acceptable as they would allow free passage of oxygen to the full heap height.

13.2.5 HydroGeoSense Testwork Summary

Although crushing the samples from a top size of 25.4 mm to a -12.7 mm has a noticeable effect on the physical and hydraulic properties of the Productora composite samples, the samples with a top size of 12.7 mm are sufficiently robust to potentially support multi-lift percolation leaching. This data confirms that all tested samples agglomerated to a Level 3 (L3) would efficiently support percolation leaching.

It is estimated that the maximum heap height supported by the samples crushed to a top size of -12.7 mm is approximately 40 m (which could translate to four 10 m lifts or five 8 m lifts).

All samples tested have liquid conductivity values approximately 100 times the typical application rate, have sufficient porosity (>30%) and good ratio between micro- and macro-porosity.

The excellent hydrodynamic performance of the -12.7 mm crushed samples is consistent with the minimal fines contents of these samples. An increase in fines content due to blasting and materials handling is unlikely to reduce the tested values to anything that is of concern for design.

The operational degree of saturation under standard solution application rates (4 L/h/m^2 < q < 15 L/h/m^2) is below 40%. These operational saturation values indicate that all these samples would be able to support the percolation process at the heap up to 16 m without difficulty.

Almost no slumping was noted for the duration of the hydraulic conductivity tests (HCTs), indicating a robust porous structure for all samples.

13.2.6 Column Testwork (1 m) Results

13.2.6.1 General

Column testwork was conducted by ALS Metalurgia Santiago (Chile). These initial column leach tests are indicative and small scale. The column diameter used was 80 mm and the column height was 1 m. The production feed was crushed to -12.5 mm and about 7 to 10 kg of production feed was used in each test. The irrigation rate used was 10 L/m^2/h and acid strength was either 5 or 10 g/L H_2SO_4.

Based on the bottle roll test results and preliminary agglomeration trials, agglomeration and leach solution acid concentrations were selected as shown in Table 13.7. All production feed types showed high acid consumption potential under highly acidic leach conditions. They also all showed some degree of reduced copper recovery under more moderate acidic conditions. The Northern Oxide samples show a very strong copper recovery dependence on acid, and this is reflected in the conditions selected.

Table 13.7 : Production feed Agglomeration Acid and Column Leach Solution Acid

Samples	Acid Addition to Agglomerate (kg/t)	Acid in Leach Solution (g/L)
Cc7	10	5
Cc8	5	5
Si4	5	5
No3	25	10
Sm2	10	5

Table 13.8 shows a comparison of bottle roll results to the completed 1 m column results. The high-level summarised results are reasonably consistent with the bottle roll tests and indicate that the agglomeration and solution acidities used were adequate. For Cc7, Cc8 and No3 samples, the acid soluble copper recovery (as determined by sequential copper analysis of head samples) has exceeded 100%, as would be expected, as these are oxide/transition samples.

Table 13.8 : Bottle Rolls vs. Column Leach Results

Sample	Domain	%Cu (head)	Economic Point from Bottle Roll Tests		1m Column Day 50 Results		
			Total Cu Recovery (%)	Acid Consumption (kg/t)	Total Cu Recovery (%)	Acid Sol Cu Recovery (%)	Acid Cons (kg/t)
No 3	North Ox	0.54	76	50	80	120	54
Sm2	Sth Mantos	0.45	34	49	48	124	24
Cc7	CCHEN Sth	0.82	46	56	49	220	16
Cc8	CCHEN Sth	0.57	56	37	60	157	7
Si4	Santa Innes	0.39	70	32	60	101	7

The following sections summarise the performance of each of the five individual column tests. For each test, two graphs are provided. The first graph for each sample shows copper recovery and the total acid consumption in kg of acid per tonne of production feed (kg/t) vs. leach time (days). The second graph shows copper recovery, net acid consumption (kg/t) and acid consumption based on kg of acid consumed per kg of copper extracted (kg/kg) vs. solution application (kL/t).

13.2.6.2 Satellite Pits

13.2.6.3 Northern Oxide (No3)

No3 (Northern Oxide 3) leach performance (Figure 13.11 : and Figure 13.12 :) reflects the slow leaching characteristics that would be expected from a mineralogy that is dominated by copper silicates. Despite this sample exhibiting lower acid soluble copper content and an elevated insoluble copper content, its performance was very similar to other samples based on bottle roll testing. Total copper recovery of 80% was achieved using a solution application of 5.5 kL/t. A 120% recovery of the acid soluble copper was achieved, which would indicate that for this particular sample, there were probably some secondary sulphides present that were leached.

The production feed shows steady but high acid consumption characteristics, and it is the acid consumption that would dictate the maximum economic extraction of copper achievable. Leaching was terminated at Day 30 as acid consumption had exceeded the economic limit of 25 kg/kg copper produced.

From the bottle rolls, it was predicted that the economic copper recovery would be 76% at 50 kg/t of acid consumed. Interpretation of the column tests result indicated although the maximum copper recovery was 80%, the economic maximum recovery based on acid consumption was 76% total copper recovery at 50 kg/t of acid consumed and solution application of 4.5 kL/t.

The variability in acid consumption (kg/kg) is a function of fluctuations in leach solution feed rate and the effect of the residence time of the leach solution within the column. This variation is quite acceptable and usual.

Figure 13.11 : Northern Oxide 3 – 1 m Column Results

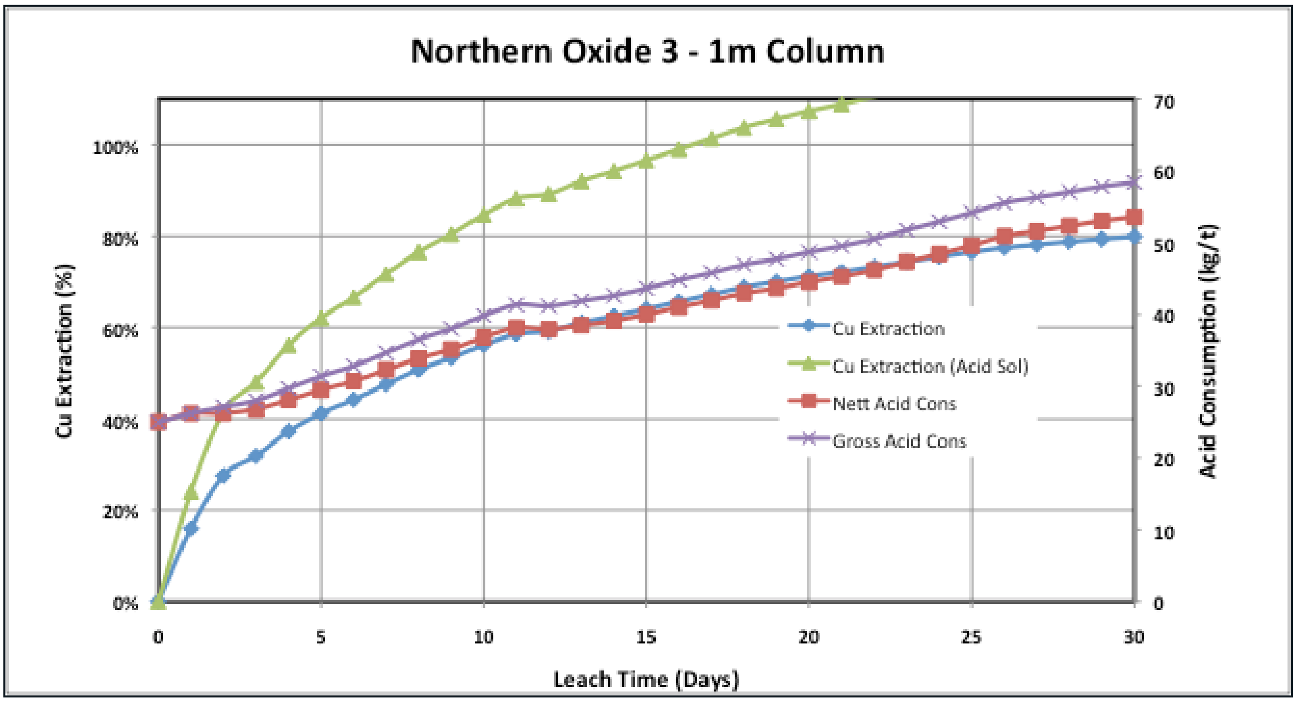
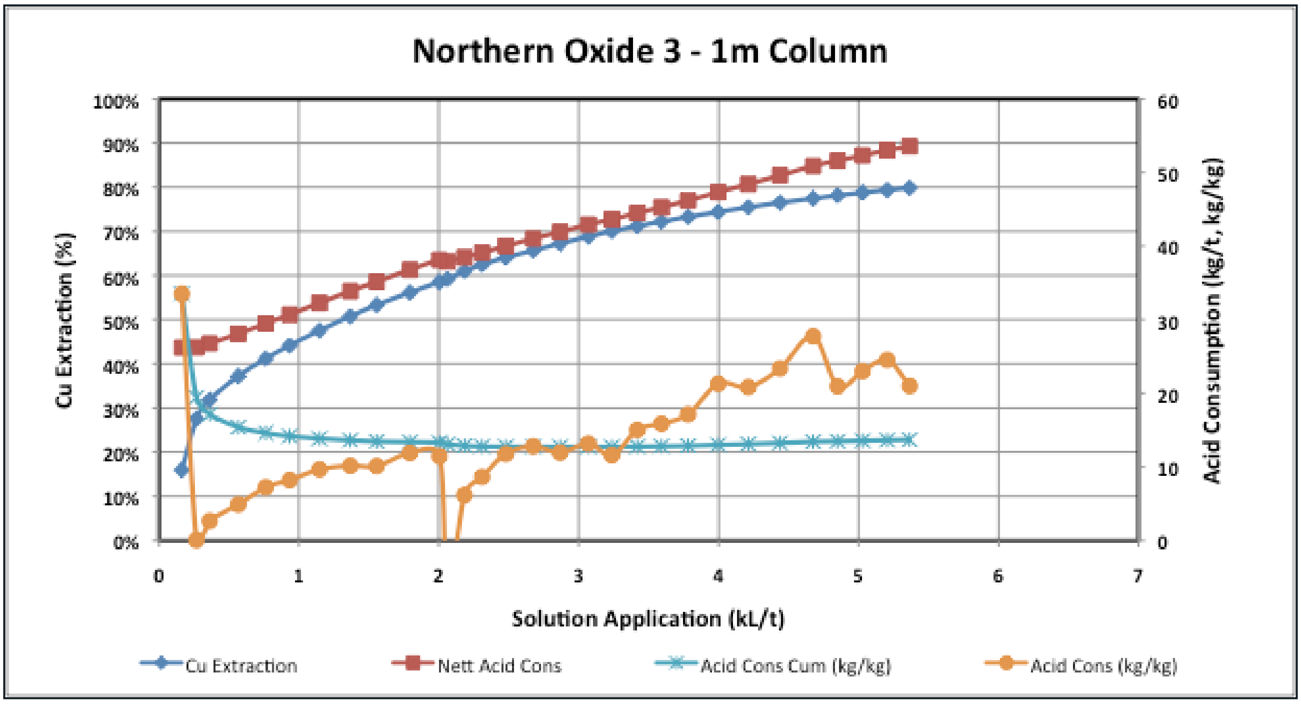


Figure 13.12 : Northern Oxide 3 – 1 m Column Results



13.2.6.4 Southern Satellite (Sm2)

Sm2 (Southern Satellite 2) column test was terminated at Day 50 and a solution application of 6.5 kL/t production feed, as the acid consumption had exceeded the economic limit (Figure 13.13 and Figure 13.14). As would be expected from a combination of copper silicate minerals and secondary sulphides (chalcocite/digenite), the leaching kinetics are slow. Acid soluble copper recovery was 120% with an acid soluble plus CN soluble copper recovery of 67%, indicating a significant proportion of the chalcocite and digenite are yet to leach.

From the bottle rolls, it was predicted that the economic copper recovery would be 51% at 14 kg/t of acid consumed. The column tests results were 49% total copper recovery at 25 kg/t of acid consumed. The acid consumption characteristics of the sample were higher than in the bottle roll testing and varied significantly from other Southern Satellite samples tested by the bottle roll test.

Figure 13.13 : Southern Satellite 2 – 1 m Column Results

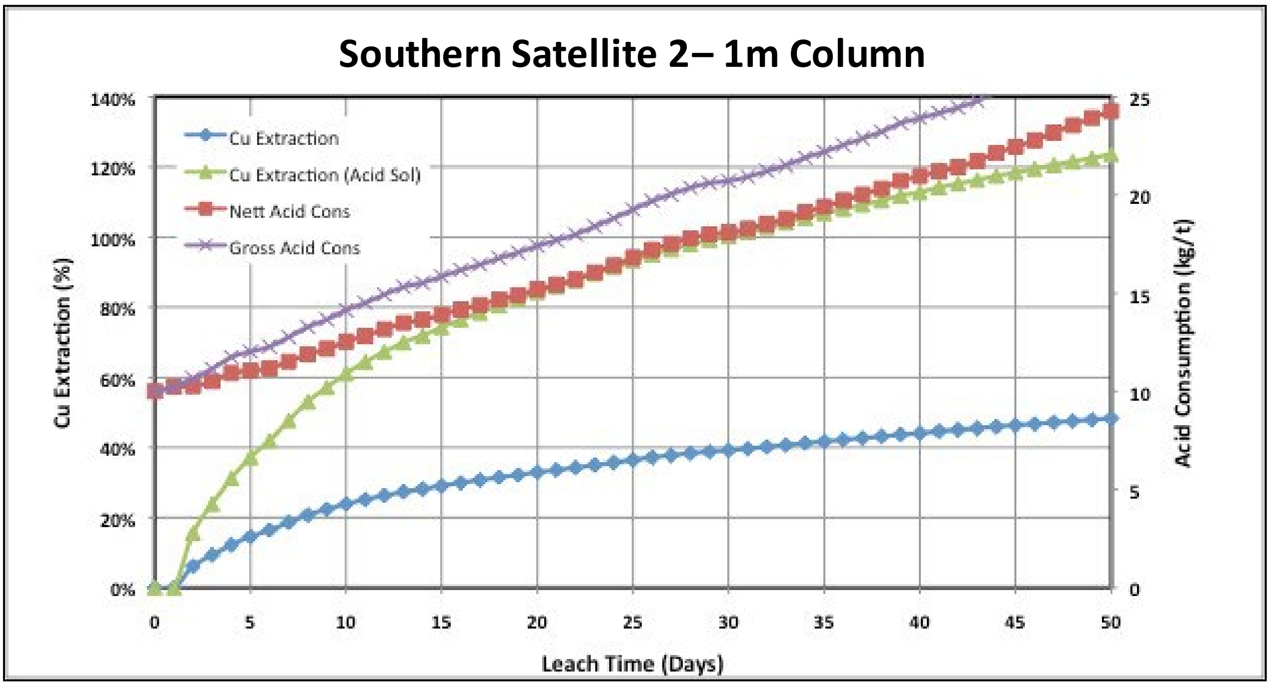
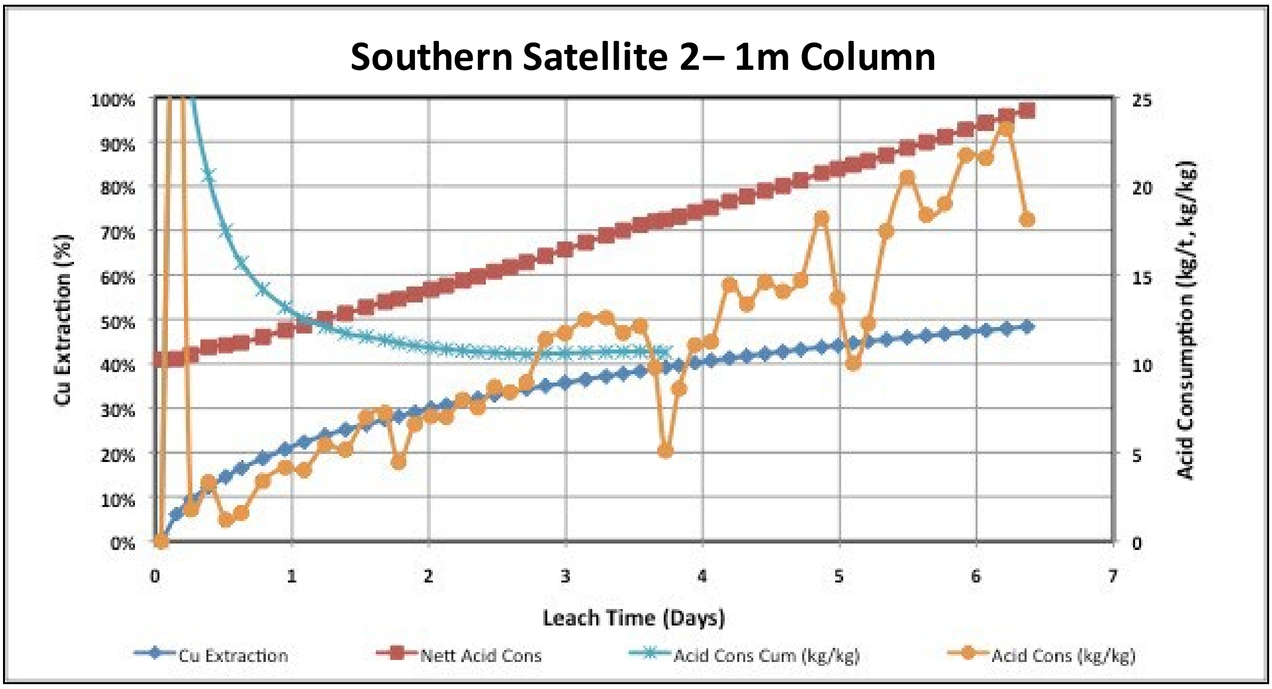


Figure 13.14 : Southern Satellite 2 – 1 m Column Results



13.2.6.5 Main Pit (Cc7 and Cc8)

Both samples, Cc7 and Cc8, exhibit slow copper leaching kinetics (Figure 13.15 to Figure 13.18) symptomatic of the copper mineralogy present, slow leaching copper silicates and secondary sulphides (chalcocite and digenite). The total copper recovery is low for both (49% and 60%) but consistent with the low content of acid soluble copper (23% and 38% of total copper). Acid soluble copper extraction was 160 and 220%, reflecting the influence of leaching of the secondary copper sulphides. The acid soluble plus CN soluble copper recoveries were very similar at 76 and 77%. Both columns were terminated at 40 days and solution application of 7 kL/t of production feed due to increasing acid consumption (in kg/kg copper leached which was approaching the economic limit) and the slow leaching rates indicating that ultimate copper recovery (based on bottle roll tests) was being approached.

From the bottle rolls, it was predicted that the economic total copper recovery would be 51% and 61% at an acid consumption of 12 and 4 kg/t for Cc7 and Cc8, respectively. The column tests results were 49% and 60% total copper recovery at 16 and 7 kg/t of acid consumed, respectively.

For both samples, acid consumption trends higher with time and solution application. This is consistent with bottle roll results and the host rock mineralogy.

Figure 13.15 : Main Pit 7 (Cc7) 1 m Column Results

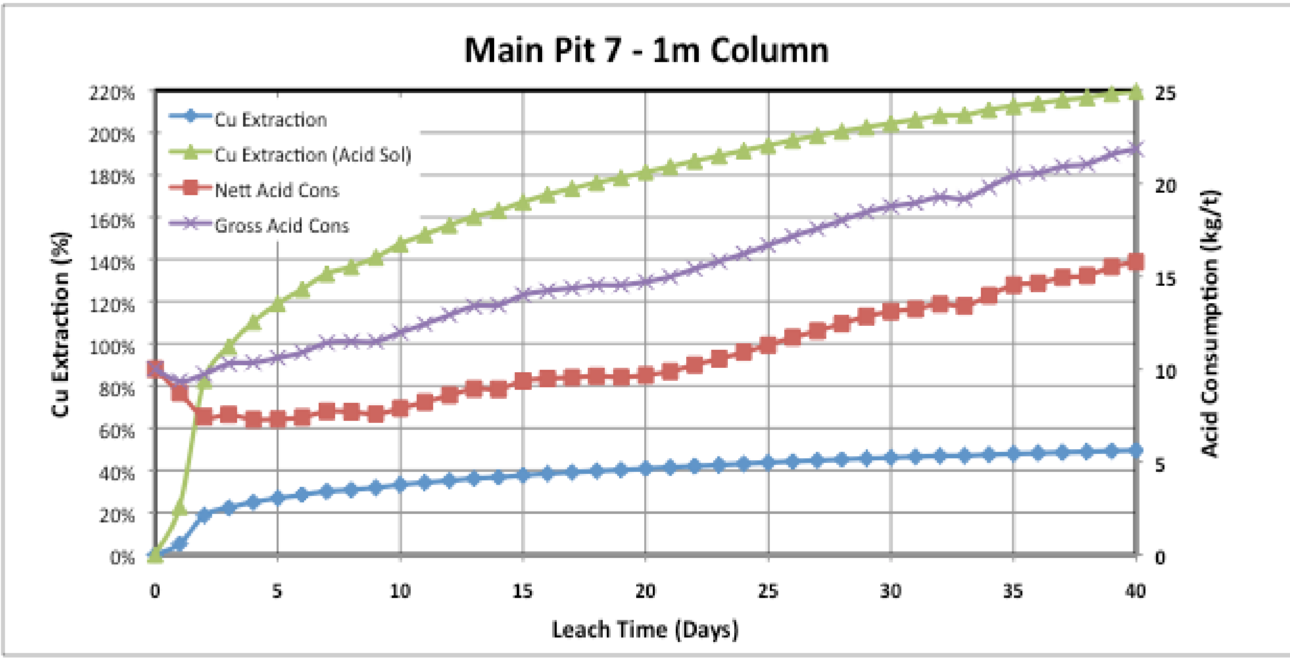


Figure 13.16 : Main Pit 7 (Cc7) 1 m Column Results

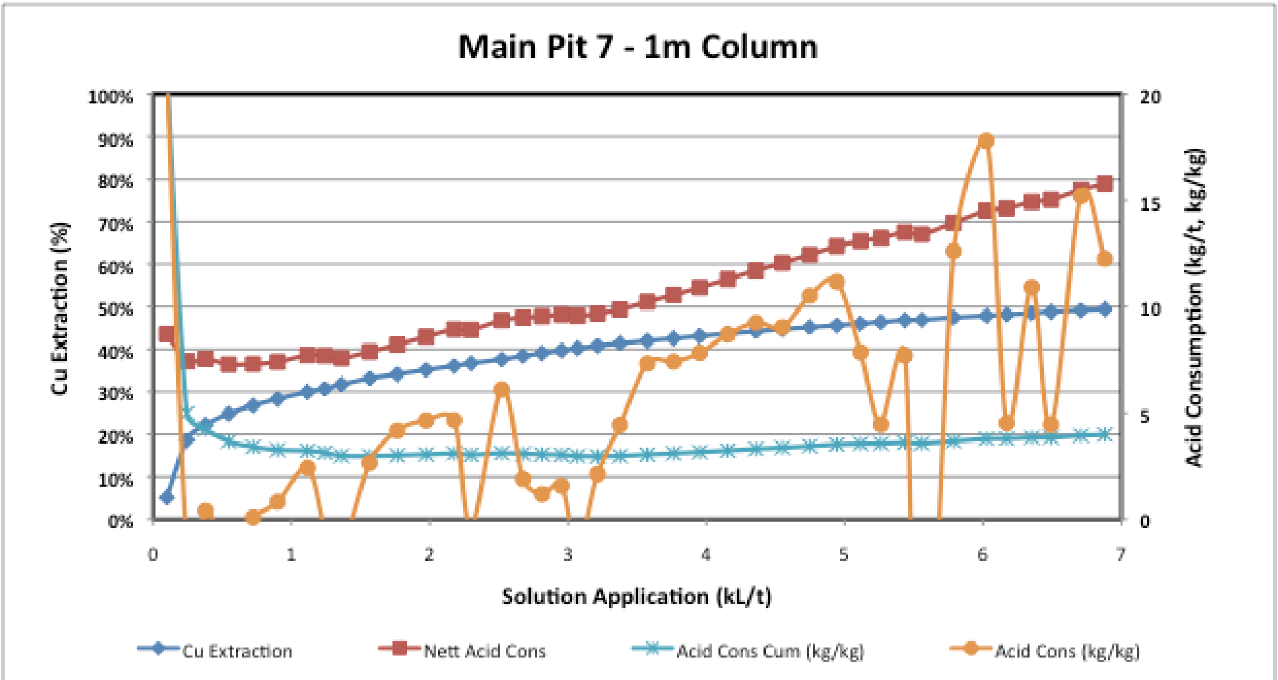


Figure 13.17 : Main Pit 8 (Cc8) 1 m Column Results

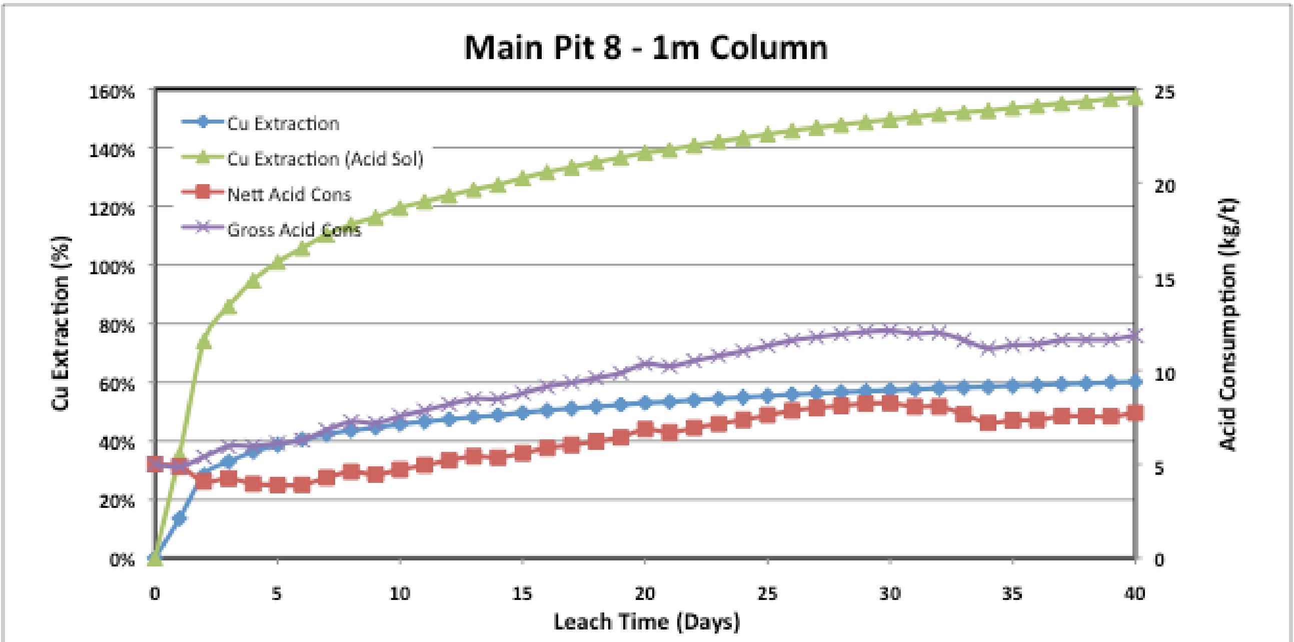
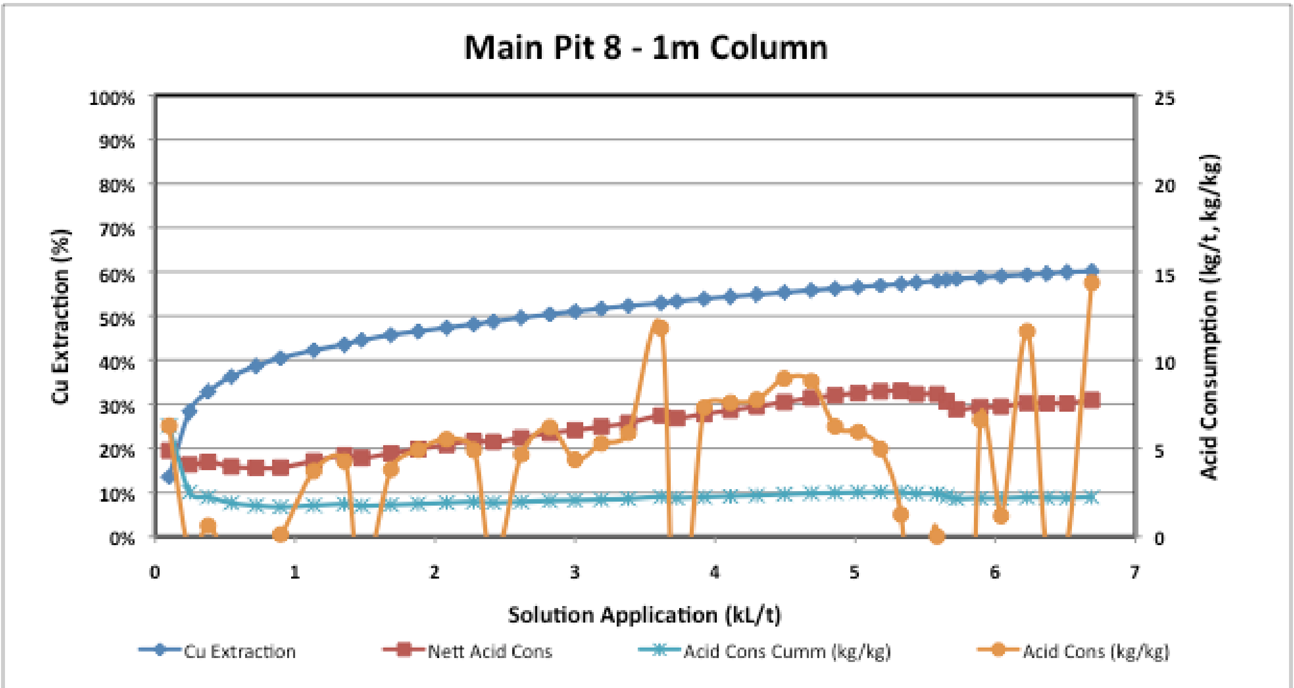


Figure 13.18 : Main Pit 8 (Cc8) 1 m Column Results



13.2.6.6 Santa Innes (Si4)

Santa Innes (Si4) column test was terminated at Day 30 and a solution application of 5.3 kL/t production feed, as the acid consumption had exceeded the economic limit (Figure 13.19 and Figure 13.20). As would be expected from a combination of copper silicate minerals and secondary sulphides (chalcocite/digenite), the leaching kinetics were very slow. Acid soluble copper recovery was 120% with an acid soluble plus CN soluble copper recovery of 67%, indicating a significant proportion of the chalcocite and digenite are yet to leach.

From the bottle rolls, it was predicted that the economic total copper recovery would be 64% at an acid consumption of 4 kg/t. The column tests results were 49% total copper recovery at 25 kg/t of acid consumed. The acid consumption characteristics of the sample were higher than in the bottle roll testing and varied significantly from other Southern Mantos sample tested by the bottle roll.

Figure 13.19 : Santa Innes 4 – 1 m Column Results

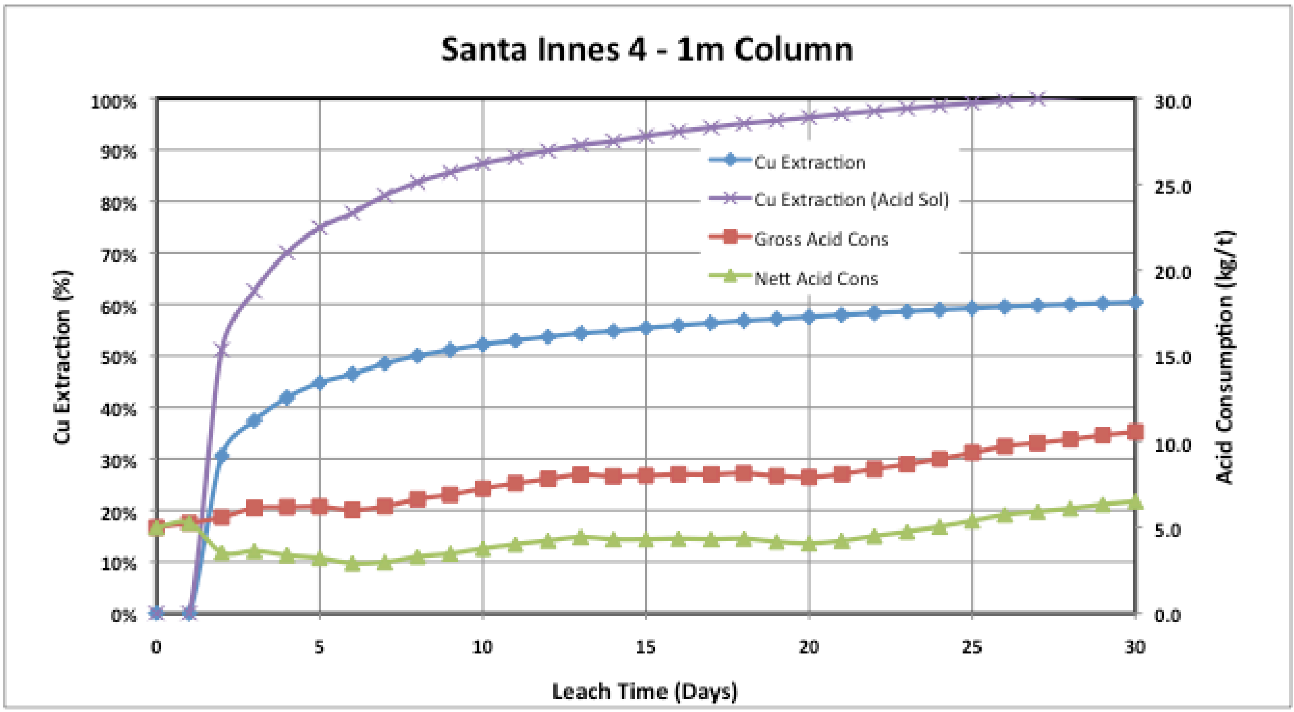
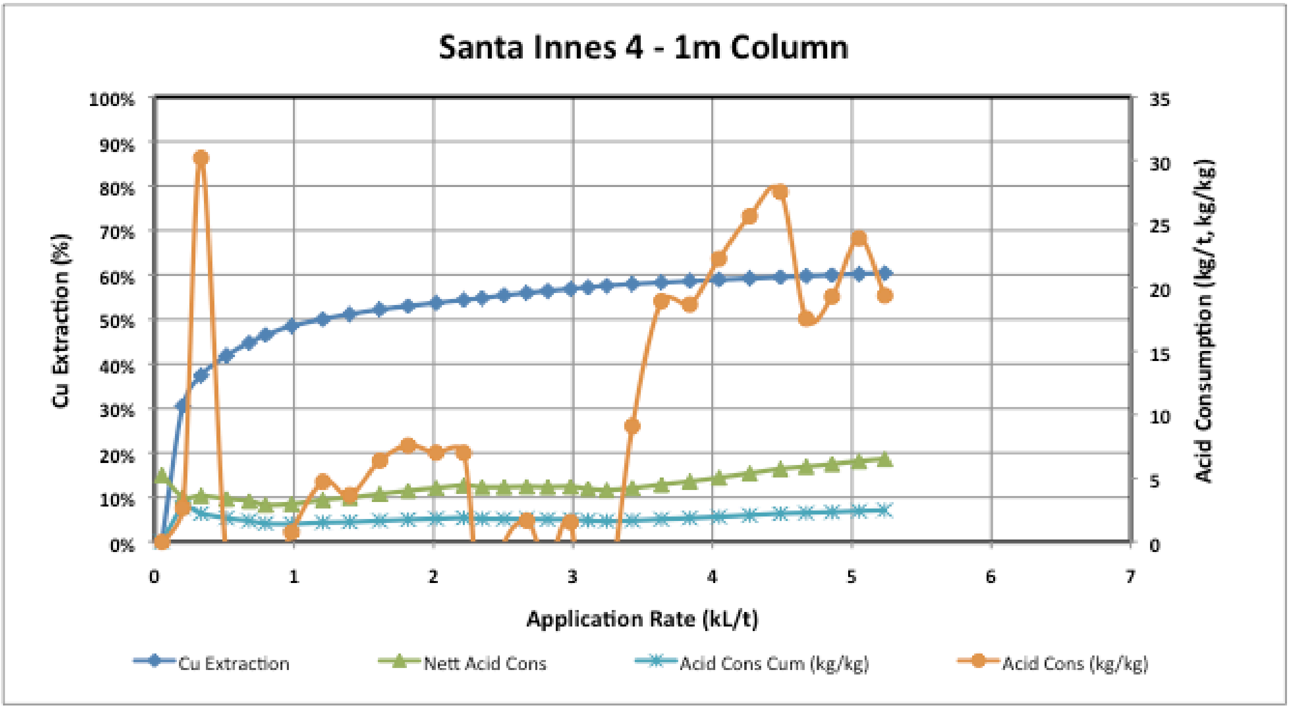


Figure 13.20 : Santa Innes 4 – 1 m Column Results



13.2.6.7 Summary of 1 m Column Testwork

All 1 m column tests showed similar general characteristics with slow copper leach kinetics symptomatic of the oxide and secondary sulphide minerals present. In addition, they show relatively high acid consumptions, which is symptomatic of the host rock mineralogy. The acid consumption increases with time almost independent of copper recovery, and certainly as the copper recovery approached terminal extraction, acid consumption continues at a reasonably constant rate.

These general characteristics mean that the copper extractions targeted are dictated by the resultant economic acid consumption. This feature results in a leaching strategy that dictates that the leach cycle time is kept to a minimum to minimise acid consumption. Consequently, it is necessary to consider commercial scale heaps with a limited heap height, as the higher the heap, the more extended the leach time.

The stacking and hydrodynamic testing support heap heights from a geotechnical, hydrological and permeability perspective of up to 16 m with application rates in the order of 5-10 L/hr/m^2. They also show a potential to allow for over-stacking of up to five 8 m lifts.

For scaling up from a 1 m column, a 6 m heap height requiring 5.4 kL/t of solution and assuming a design application rate of 10 L/hr/m^2, the leaching would take 265 days as compared to the 1m column taking 45 days. For scaling up from a 1 m column, an 8 m heap height requiring 5.4 kL/t of solution and assuming a design application rate of 10 L/hr/m^2, the leaching would take 353 days as compared to the 1m column taking 45 days. Therefore, an 8 m heap would require an additional 90 days of leaching with the additional acid consumption.

Stacking production feed at less than 6 m is not considered economically practical at this stage, however, there is precedence for operations in Chile with specific production feed types whereby leaching has been conducted with heaps as low as 3 m to 4 m.

For this study, without additional testwork on 6 m or 8 m columns available, 6 m heap heights are considered to be acceptable.

13.2.7 Conclusions

Major conclusions of the copper oxide testwork include the following:

- The Productora oxide production feed contains various oxide copper minerals and often has remnant sulphides present.
- A major oxide mineral in the Productora production feed is the slow leaching copper bearing silicate, chrysocolla.
- There is significant benefit in utilising seawater at appropriate pH, as bottle roll testing demonstrated that the copper recovery exceeded expectations based on acid soluble copper assays alone. This suggests that the chlorides in seawater promote leaching of secondary sulphide copper minerals.
- Generally, the host rock presents an environment of reasonably high acid consumption, with a number of samples requiring column leaching to be terminated due to excessive acid consumption.
- The bottle roll tests have provided, in most cases, good guidance for conditions for column leaching and an indication of likely copper recovery and acid consumption characteristics.
- The 1 m columns provide good guidance for data required for the scale-up and determination of the Process Design Criteria (PDC).
- The stacking and hydrodynamic column testing confirm that the agglomerated production feed tested would support percolation heap leaching at the stack heights and application rates provided in the PDC. The tests also indicate that the production feed would have the necessary geotechnical stability to allow for multiple lift stacking of the heaps.

13.3 Copper Oxide - Cortadera

Oxide production feed has been identified at Cortadera, but in smaller quantities than that present at Productora. A program of testing is planned for Cortadera oxide production feed, but it would not be as comprehensive as for Productora given the smaller production feed volumes.

Cortadera oxide production feed would be mined separately from other Cortadera open pit production feed. The run-of-mine (ROM) oxide production feed would be stockpiled until a sufficient batch has accumulated to send to Productora on the overland conveyor. A batch for transfer would be fed to the Cortadera primary crusher and carried by the overland conveyor to Productora. Conveyor transfers would be made for oxide production feed such that it reports to a holding pad from which it can be fed to the Productora heap leach preparation circuit.

For this Report, it is assumed that the Cortadera oxide production feed would respond to heap leaching in a similar manner to the Productora oxide production feeds. An initial bottle roll test has been conducted however further testing is required.

13.4 Low Grade Leach

13.4.1 Introduction

In 2022, a conceptual study into the leaching of primary low grade production material was carried out. The aim of the study was to assess the amenability of acid dump leaching via the application of Novaminore technology to low grade ROM material.

Novaminore technology is a polymetallic processing solution that streamlines the treatment of various production feed. It uses a cost-effective physical-chemical attack on low-fracture production feed particles, enabling efficient extraction of valuable minerals. By applying high concentrations of chloride ion and regulated irrigation-rest cycles, the technology promotes greater rock fracturing and directly targets chalcopyrite particles.

The process benefits from saline water by utilising low-quality water sources, such as brackish water and seawater. This reduces dependency on freshwater resources, a significant advantage in the mining industry in Chile where water scarcity can be a major concern. Seawater contains around 20 g/L of chloride ion, and the process requires maintaining a circulating chloride level of between 100 and 130 g/L.

Novaminore can extract copper from both low grade sulphide dump leach and conventional oxide heap leach. It processes diverse production feed types, including oxides and both secondary and primary copper sulphides, allowing mining companies to use their existing SX/EW plants more efficiently. To integrate the technology, some small modifications are needed when defining the oxide plant leaching configuration. These involve incorporating the Pregnant Leach Solution (PLS) solutions from the ROM production feed into the oxide plant process as PLS or ILS solutions, depending on the concentrations achieved. Another modification includes a series-parallel configuration in the solvent extraction stage, allowing for ROM production feed irrigation with a higher copper concentration. These minor modifications can be carried out while the oxide leaching plant is operational.

13.4.2 Testwork

A preliminary working program was prepared to evaluate the behaviour of low-grade primary production feed materials from the Cortadera and Productora deposits to acid-chloride leaching. Samples were obtained from drill core and underground samples, which allowed the development of production feed mixtures, with the purpose of reaching a minimum mass of 4 kg in each case, sufficient for the development of several leaching tests at a granulometry of 100% -½ inch in microcolumns of 4” in diameter and 12” in height. Samples were obtained between November 2021 and January 2022.

Table 13.9 : Sample Identification

Sample Identification	Location	Meters		Weight (kg)	Total Cu Grade (%)
		From (m)	To (m)		
		238.9	239.8	2.4	
		240.8	241.6	2.2	
CRP0045D	Cortadera	242.3	243.0	2.0	0.18
		249.2	250.3	2.3	
		252.2	252.0	2.1	
CRD0080	Cortadera	211.1	212.1	2.2	
		213.1	214.1	3	0.16
		214.8	215.5	2.8	
		215.8	216.0	1.0	
FJOD-01	Cortadera	111.0	112.0	2.9	0.30
CRP0061D	Cortadera	205.5	206.0	1.5	*
MET015	Productora	297.2	299.2	6.6	0.21
MET017	Productora	247.0	248.4	1.6	*
MET011	Productora	213.5	214.0	0.9	*
MET008B	Productora	112.0	113.0	3.1	0.59
		117.0	118.0	3.2	
PSUS-01	Productora	Underground Samples		10.0	0.26
PSUS-02	Productora	Underground Samples		10.7	< 0.005
PSUS-03	Productora	Underground Samples		10.4	0.04

Samples were then combined based on preliminary chemical analysis and leaching tests defined. The following conditions were considered for the tests:
a) Curing Stage:

- Sulphuric acid: 20 kg/t
- Sodium chloride: 30 kg/t
- Maximum humidity: 10%.

b) Irrigation Stage:
After an initial resting time of 7 days, the first irrigation period of the micro columns was carried out under the following conditions:

- Irrigation rate: 10 L/hm^2Initial irrigation time: 24 h
- Production feed bed temperature: 30°C.

The irrigation is carried out with a synthetic raffinate solution, with industrial characteristics, according to the following content:

- Copper: 0.5 g/L
- Acid: 10 g/L

- Total iron: 10 g/L
- Ferrous iron: 5 g/L
- Chloride: 120 g/L.

13.4.3 Test Results

Test results were completed after 140 processing days and 15 irrigation cycles (each one followed by a 7-day resting period). Once complete, the micro columns were unloaded and dried, and tails analysed by copper, to determine the calculated head grade and other metallurgical parameters (see Table 13.10 and Table 13.11).

Kinetics of copper extraction and net acid consumption of acid are shown in Table 13.10 and Table 13.11 respectively.

Table 13.10 : Tail Grades and Calculated Head Grades

Mcol	Sample ID	Calculated Cu Head Grade				Tail Grade			
		Cu_T %	CuS_H %	CuS_{CN} %	Cu_{Ins} %	Cu_T %	CuS_H %	CuS_{CN} %	Cu_{Ins} %
1	CRP0045D	0.19	0.00	0.01	0.18	0.12	0.01	0.02	0.10
2	FJOD-01/CRP0061D/PSUS-01	0.25	0.03	0.05	0.18	0.08	0.00	0.01	0.07
3	CRD0080	0.19	0.01	0.01	0.17	0.06	0.00	0.01	0.05
4	PSUS-01	0.25	0.05	0.07	0.12	0.04	0.00	0.01	0.03
5	MET015	0.22	0.01	0.02	0.19	0.04	0.00	0.01	0.03
6	MET008B	0.64	0.02	0.04	0.57	0.21	0.01	0.03	0.17
7	MET017/MET011/PSUS-1	0.31	0.04	0.05	0.23	0.06	0.00	0.01	0.04

Table 13.11 : Copper Extraction and Net Acid Consumption

Mcol	Sample ID	Cu Extraction		Acid Consumption	
		Analysed Head Base %	Calculated Head Base %	Gross kg/t	Net kg/t
1	CRP0045D	32.0	31.4	53.6	52.7
2	FJOD-01/CRP0061D/PSUS-01	68.5	67.3	44.4	41.7
3	CRD0080	74.6	66.0	39.5	37.6
4	PSUS-01	80.2	83.7	31.7	28.5
5	MET015	83.1	80.0	46.8	44.1
6	MET008B	72.0	66.6	44.3	37.7
7	MET017/MET011/PSUS-1	86.6	81.0	41.0	37.1

Figure 13.21 : Copper Extraction (Total Cu Base)

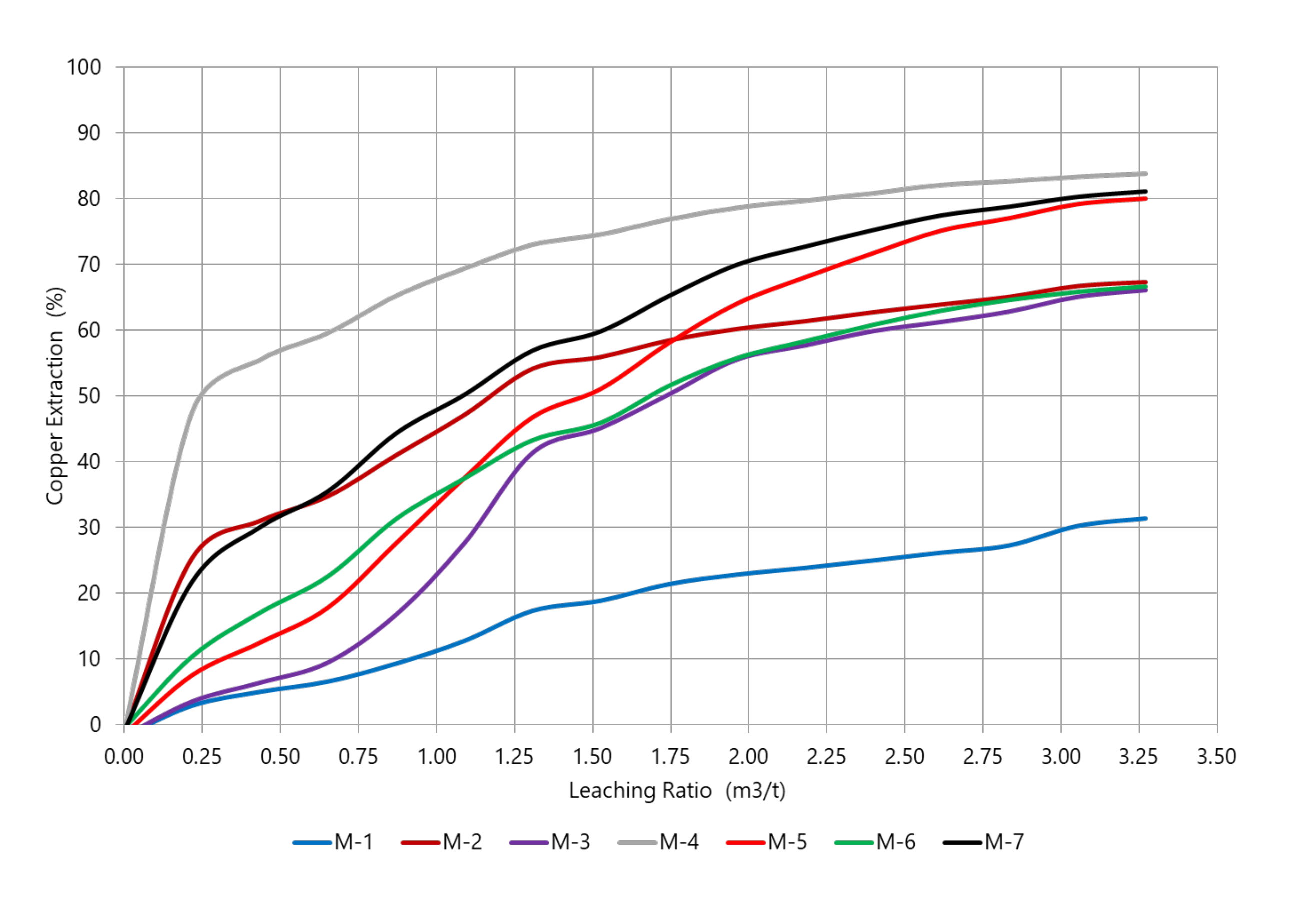
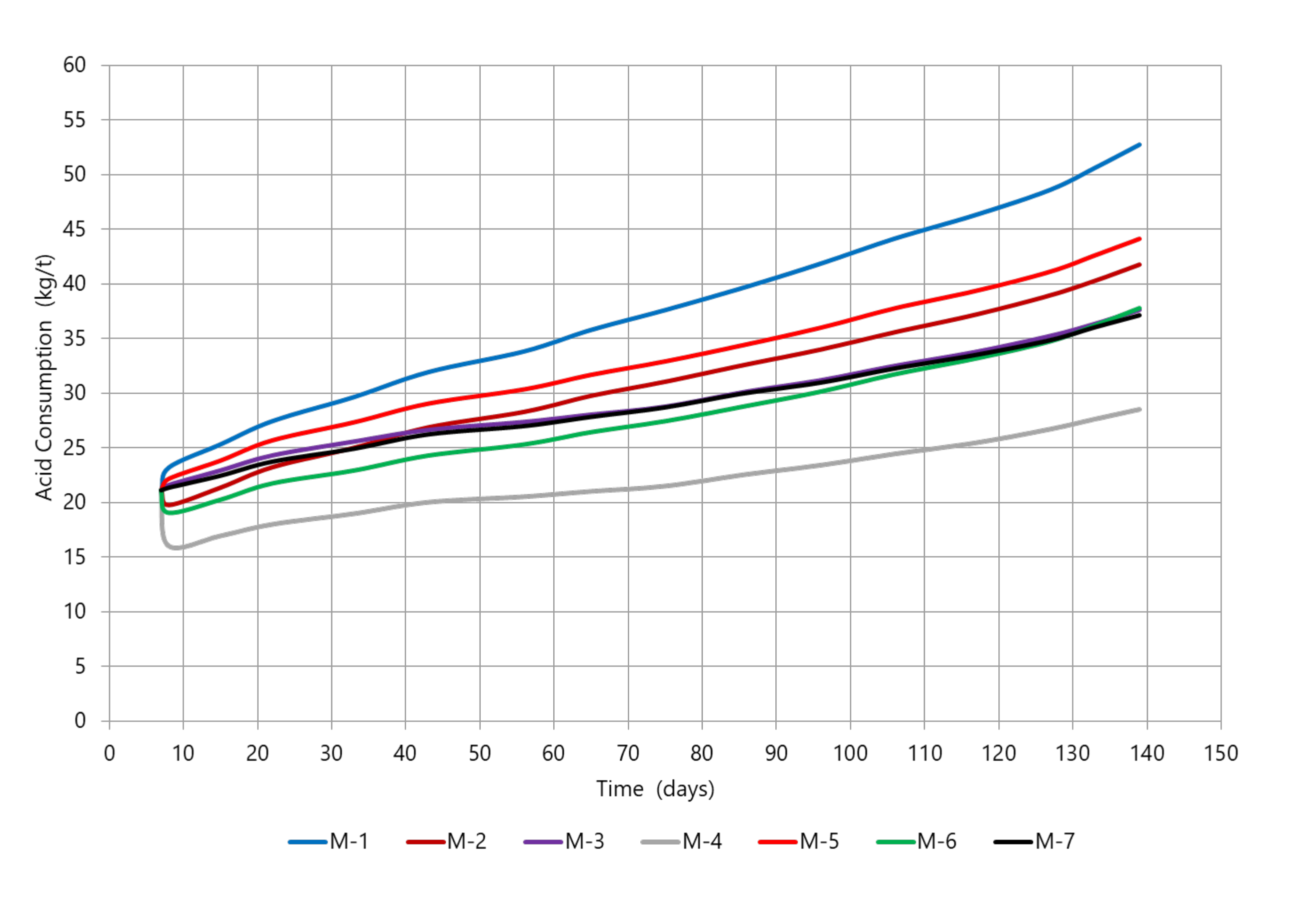


Figure 13.22 : Net Acid Consumption



13.4.4 Discussion of Results

Although the production feed head grades were very low (except sample M-7), which naturally limits the ability to extract copper using a raffinate solution that already contains copper, the results achieved by most of the experiments are considered very good. Except for one test (M-1), the remainder reported copper extraction between 65% and 85%, which is a success, taking into account that in each test sample primary sulphides were present.

Regarding acid consumption, it should be noted that the 30 cm columns used exacerbate the reactivity of the acid with the gangue. However, acid consumptions (except for test M-1) were between 30 and 45 kg/t, which is usual for this type of test with primary production feed, particularly because the specific surfaces exposed to the action of acid-chloride solutions is less than in the case of an oxidised production feed, which presents greater fracturing, greater degradation and a larger exposed surface.

Using the Novaminore technology in a ROM granulometry leaching operation (dump leach) is certainly feasible, but is not expected to achieve the excellent results shown in the small scale experiments reported here.

There are several feasible routes to improve the fracturing conditions of the oxides in such a way that the acid chloride solutions can access the interior of the particles. It should be noted that the key difference between sulphides and oxides mineral species is that the oxides present a high level of fracturing and porosity to allow access for the leaching agent to the copper bearing minerals.

The degree of fracturing (i.e. the particle size distribution) strongly influences the effectiveness of the low grade leach. The test work has been conducted on finely crushed material and the results must be downgraded to reflect a large particle (blasted ore) low grade dump leach operation. the dump leach Cu recovery has been estimated to be 30% to 40%.

The acid consumptions achieved in the preliminary leaching tests do not represent the reality of an industrial level operation. In columns of about 4 m or more, it has been found that the acid consumption achieved using this saline leach method, compared to a micro column, is a third of what was measured on this occasion, which is around 10 to 15 kg/t. However, in the case of ROM granulometry leaching, these consumptions are even further reduced due to the lower reactivity and less exposed surface when compared to the production feed crushed at ½ inch. A reasonable estimate for PEA purposes allows inferring a consumption of 50 to 60% of what is likely to be achieved in heap leaching with crushed production feeds (10 to 15 kg/t stated above), to between 5 and 10 kg/t of ROM feed.

13.5 Copper Sulphide Mineralisation – Comminution, Copper Flotation and Molybdenum Recovery

13.5.1 Metallurgical Testwork Program (Productora, 2013)

For the flotation testwork, there are no longer any divisions within the Productora mineralisation, as described throughout the comminution discussion. All samples are “Productora” open pit samples and are assessed as a group. Some tests were conducted on samples that are now outside of the Productora open pit, and they have been excluded from the assessments below.

During this program, the following testwork was completed on new samples from Productora:

Comminution calibration results:

- Four samples were sent to SGS Laboratories for SAG mill comminution (SMC) and/or JK Drop Weight testing, as well as normal crushing and Bond work index testwork.
- Flotation testwork was conducted on 14 selected samples.
- No samples were selected or testwork performed on any oxide samples.

13.5.2 Sulphide Production Feed Comminution Testwork (Productora, 2014-15)

Comminution testwork was performed on the Project flowsheet development (FD) and variability (V) samples at the ALS Metallurgy Santiago and Perth laboratories supervised by Mintrex Pty Ltd (Mintrex). Comminution testwork was conducted to determine the variability of comminution parameters throughout Productora and allow parameters for comminution circuit design to be derived. Tests conducted included the following:

- Unconfined compressive strength (UCS). UCS testing results indicate the strength of a rock in terms of resisting compressive forces prior to breakage and assist with primary crusher selection.

- Bond Crushing Work Index (CWi). CWi test results allow determination of crushing energy requirements at the secondary and tertiary stages and for pebble crushing.
- Bond Rod Mill Work Index (RWi). RWi test results allow determination of power requirements for a rod milling grinding process.
- Bond Ball Mill Work Index (BWi). BWi test results allow determination of power requirements for a ball milling grinding process.
- Abrasion Index (Ai). This test is used to determine the abrasiveness of the production feed and is used to calculate grinding media consumption as well as wear rates for crusher and mill liners.
- JK Drop Weight tests (JKDWi). Determination of production feed competence for semi-autogenous grind (SAG) mill design.
- SAG mill comminution (SMC) tests. Determination of production feed competence and crushability parameters for crushing and grinding design.

This testwork provides the Preliminary Economic Analysis (PEA) comminution data set for Productora.

13.5.3 Copper Sulphide Flotation Testwork (Productora, 2014-2016)

A large program of flotation testwork was conducted on the Productora and Alice deposits. Testwork was based on a standardised flowsheet and reagent scheme and performed in seawater. The scheme used RTD2086 (xanthate ester supplied by Tall Bennett) at a natural pH (~pH 8) with two stages of cleaning preceded by a 25 µm regrind. Natural pH was preferred as pH adjustments in seawater require large amounts of reagent which can have negative effects on flotation performance.

The target grade for the final concentrate was 25% Cu. To achieve this, optimisation testwork was conducted on a number of samples to improve copper grade and recovery. Optimisation testwork included the following variables:

- Primary grind size
- Reagent dosage
- Flotation time
- Alternate reagents
- Diesel addition for increased molybdenum flotation
- Seawater vs. tap water.

When optimised conditions were tested, it was identified that samples from different zones of the deposits gave a different flotation response. Some samples were also later identified as containing transitional sulphide and oxide material (designated as containing greater than 10% acid soluble copper). These samples gave reduced performance with the standardised reagent scheme.

This set of flotation results provides the basis for recovery of Productora sulphides in the PEA.

13.5.4 Molybdenum Testwork (Productora, 2014-2015)

Development of molybdenum flotation at laboratory scale is problematic due to the low molybdenum content of copper production feeds. Only limited aspects of molybdenum flotation could be confirmed, which is normal for many projects with low grades of molybdenum.

Molybdenum, as the mineral molybdenite (MoS_2) floats with the copper minerals in the main flotation circuit. It is then necessary to depress the majority of the material in the copper concentrate and only float the small amount of contained molybdenum. Often molybdenum flotation has unusual requirements, such as conducting all flotation in nitrogen or using "air" recycling systems that allow all oxygen to be consumed so that the flotation is essentially occurring in nitrogen. It is also normal to employ column flotation, column froth washing and multiple cleaning stages to arrive at a saleable molybdenum concentrate.

A targeted molybdenum testwork program was conducted to determine if a saleable molybdenum concentrate (~50% Mo) could be produced. Molybdenum recoveries for each flotation sample were also determined and a relationship with feed grades was established. However, molybdenum flotation was not fully optimised in either the copper circuit or the molybdenum circuit.

The initial testwork was conducted on the FD1 composite. Bulk copper floats and multi-stage molybdenum cleaners were conducted on RC chip sample (PRP0812) with a higher molybdenum head grade (0.56% Mo).

13.5.5 Historical Sulphide Process Feed Flotation Results

The early flotation work resulted in a flowsheet framework that was the basis for the 2016 Study.

The work demonstrated the effectiveness of RTD2086 (a xanthate ester) as the copper collector of choice for this Project. This collector is highly selective for recovery of chalcopyrite and against recovery of pyrite. In addition, grind sizes in the range 106 to 150 µm were established for rougher flotation and regrinding to about 25 µm P_{80} was established for the copper cleaner stage.

The "Productora flowsheet" was the outcome of this work and it was the starting point for testwork on the newly discovered Cortadera mineralisation.

13.5.6 Cortadera Sulphide Deposits – Testwork

The metallurgical testwork performed since the 2016 Study has mainly focused on testing the new Cortadera Province. The Cortadera sulphides have been partitioned by depth into Cortadera Open Pit (Cortadera OP) and the Cortadera Underground (Cortadera UG). Limited additional testwork has been conducted on Productora and Alice deposits. Only one test has been performed to date on the small, high grade, San Antonio deposit.

As the intention is to process all Costa Fuego sulphide production feed through a single concentrator located in the vicinity of the Productora deposit, all sulphide testwork would be reported on a comparative basis.

13.6 Costa Fuego Deposits – Comminution

13.6.1 Comminution Testwork Program

Mineralised samples from the Cortadera deposits were tested to determine their comminution properties so that the common Costa Fuego milling circuit could be developed. The results of all available comminution tests have been compiled and analysed to provide a comparison of the three main deposits (Productora, Cortadera OP and Cortadera UG) and the much smaller Alice deposit. The comparative comminution properties are summarised in Table 13.12.

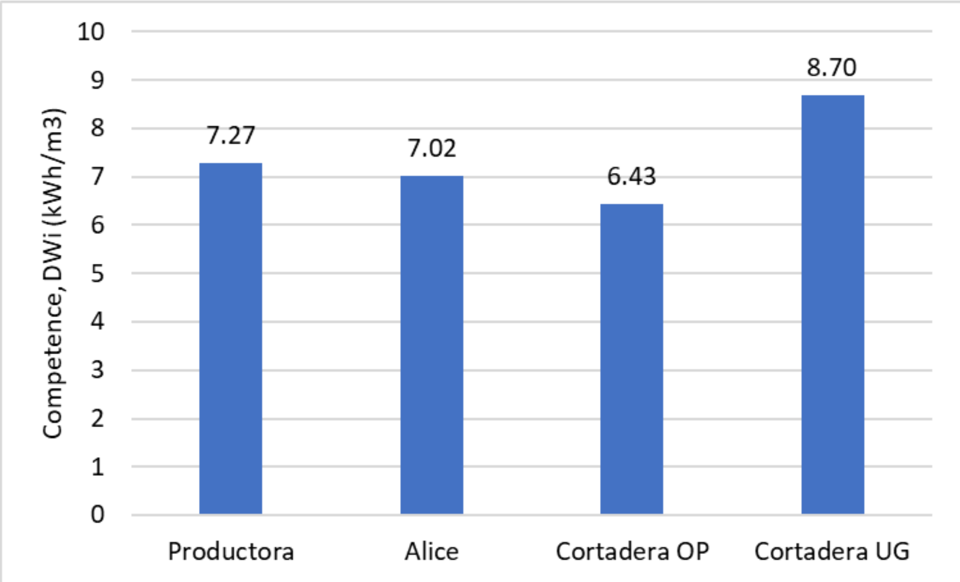
Table 13.12 : Comminution Properties for Costa Fuego Deposits

		Productora	Alice	Cortadera OP	Cortadera UG
Samples	n	36	2 (3)	23	22
DWi	Max	12.1	7.50	9.00	10.7
	80th	8.28		8.14	9.46
	Average	7.27	7.02	6.43	8.70
	Minimum	4.51		2.31	7.20
Mia	Max	31.0	21.6	24.3	26.7
	80th	23.2		22.3	25.2
	Average	20.9	20.5	18.8	23.2
	Minimum	14.5		8.90	20.3
Mic	Max	13.4	8.40	9.90	11.1
	80th	9.22		8.80	10.3
	Average	8.14	7.90	7.20	9.33
	Minimum	5.20		2.70	7.90
Mih	Max	25.8	16.3	19.1	21.5
	80th	17.8		17.0	19.9
	Average	15.7	15.3	13.9	18.1
	Minimum	10.0		5.30	15.2
BWI	Max	26.5	18.9	15.6	19.3
	80th	21.9		14.6	18.1
	Average	19.6	17.4	13.6	16.4
	Minimum	14.8	15.5	9.31	13.8
Ai	Max	0.79	0.329	0.21	0.30
	80th	0.44		0.17	0.22
	Average	0.35	0.268	0.12	0.17
	Minimum	0.11	0.201	0.02	0.06
SG	Max	3.01	2.70	3.13	3.11
	80th	2.70		2.73	2.81
	Average	2.68	2.69	2.67	2.79
	Minimum	2.50	2.66	2.50	2.66

The averages of each of these properties are compared from Figure 13.23 to Figure 13.30, and there is an accompanying short discussion about the implications of each comparison.

In Figure 13.23 the competence (Drop Weight Index or DWi) is compared across the deposits. Competence describes how difficult it is to fracture large rocks and is important for designing SAG mills.

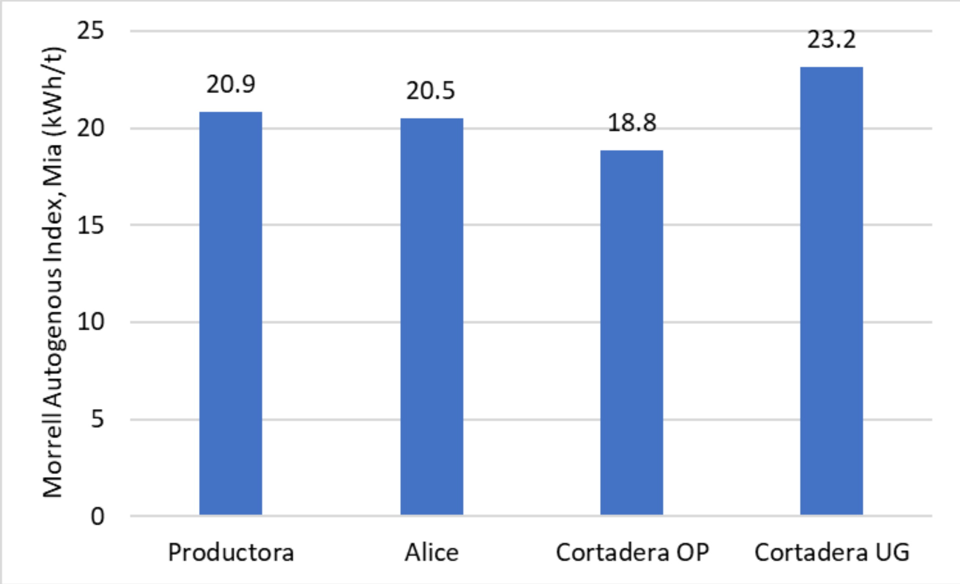
Figure 13.23 : Relative Competence of Costa Fuego Deposits



All Costa Fuego deposits have high competence, with the highest being Cortadera UG. Note that the relative differences between DWi values are closely correlated to the SAG mill specific energy requirements. Therefore, Cortadera UG would have the highest specific energy requirement, and this naturally gives it the lowest SAG mill throughput rate.

In Figure 13.24, the Morrell Mia factor is compared across the deposits. Mia is used in equations that predict the 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.

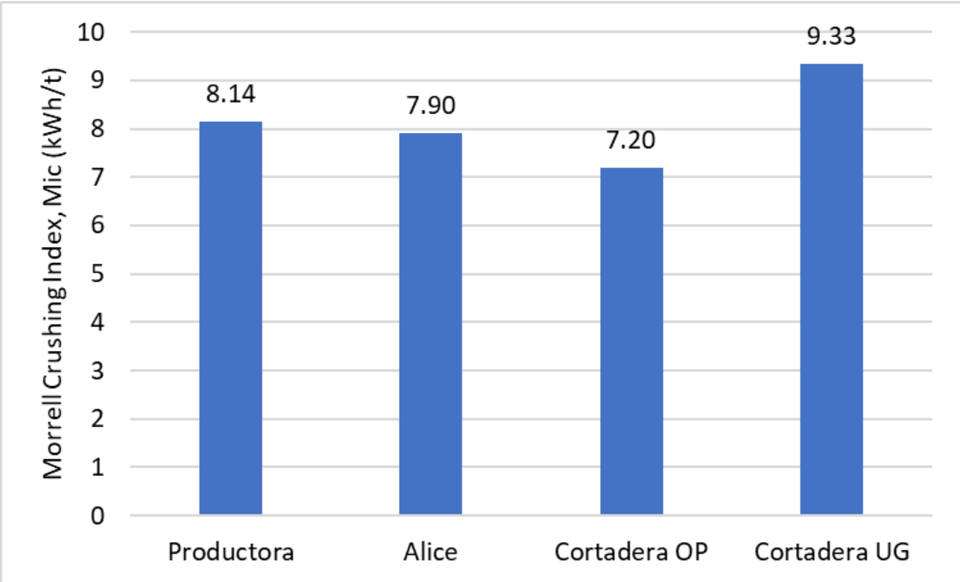
Figure 13.24 : Relative Mia Values of Costa Fuego Deposits



All Costa Fuego deposits have high Mia values, with the highest being Cortadera UG.

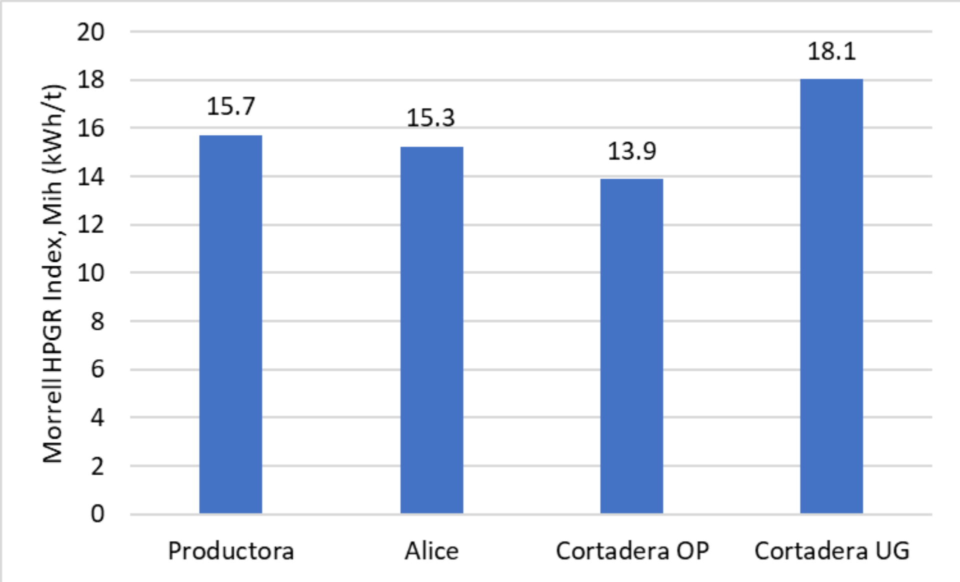
In Figure 13.25 the Morrell Mic factor is compared across the deposits. Mic is used in equations that predict the conventional crushing energy requirements such as cone, jaw and gyratory crushers. Like Mia, it is closely related to the DWi and follows a similar pattern.

Figure 13.25 : Relative Mic Values of Costa Fuego Deposits



In Figure 13.26 the Morrell Mih factor is compared across the deposits. Mih is used in equations that predict the high-pressure grinding rolls (HPGR) grinding energy requirements for the typical single pass HPGR applications treating 50 or 60 mm topsize. Like Mia, it is closely related to the DWi and follows a similar pattern.

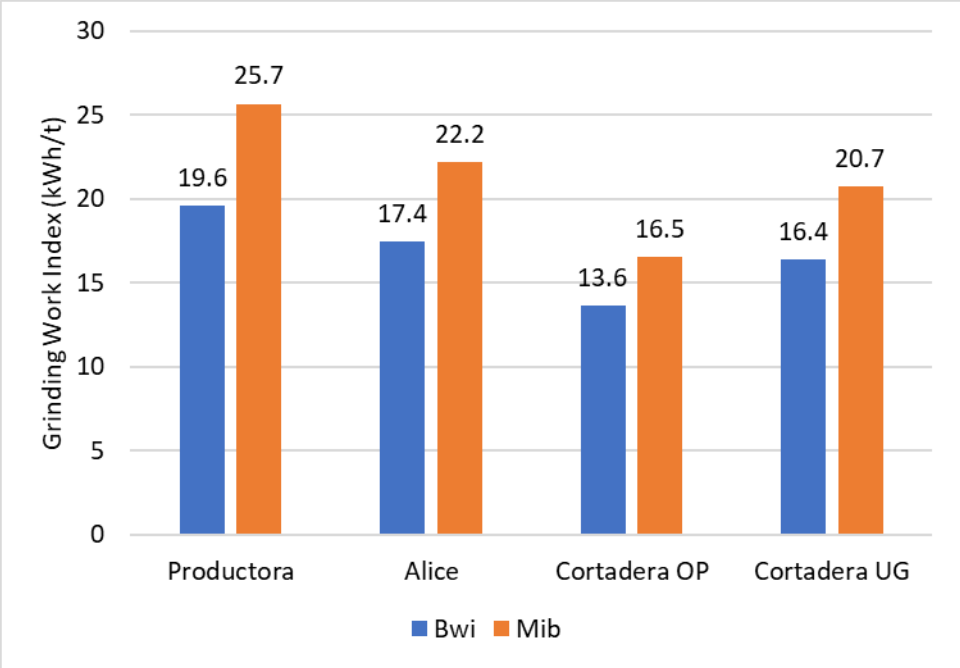
Figure 13.26 : Relative Mih Values of Costa Fuego Deposits



The Mih values for Costa Fuego production feeds are relatively high and, again, Cortadera UG would require the most energy which translates to the lowest throughput rate. Note that Mih is not utilised in the current design calculations as HPGR design is not under consideration.

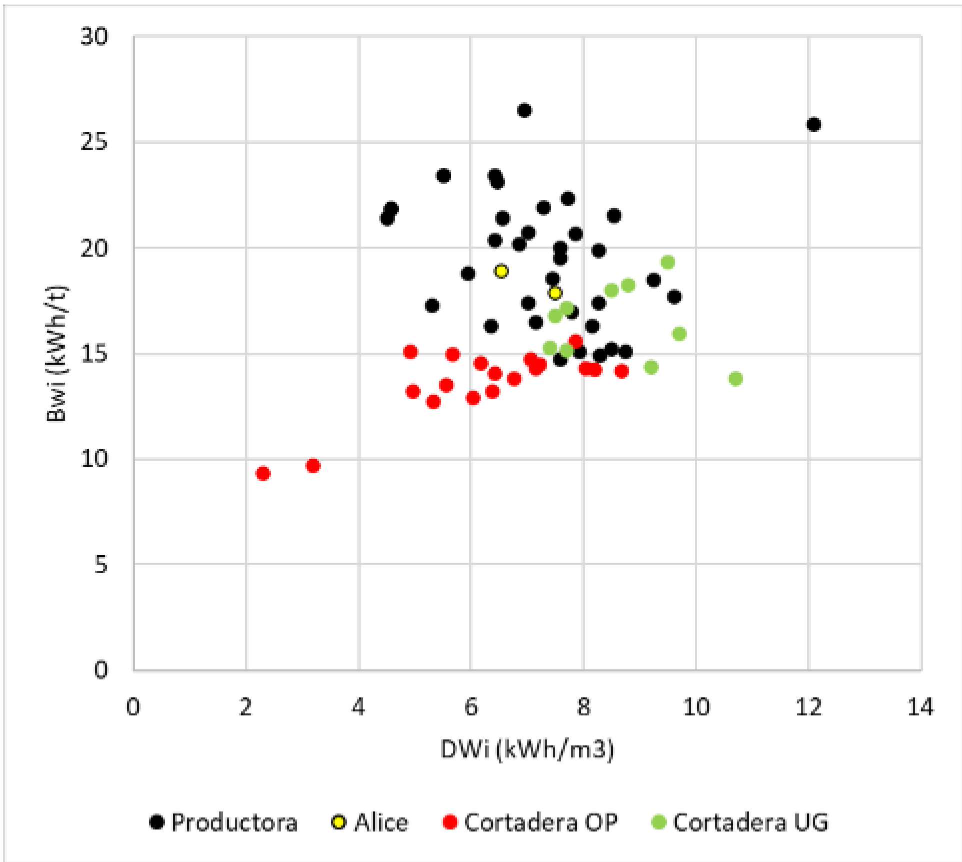
In Figure 13.27the grinding work indices are compared across the deposits. BWi is used in equations that predict ball mill grinding energy from about 2 mm topsize down to about 50 µm (using the Bond method) and Mib is used across the range from 0.75 mm down to about 50 µm in the Morrell Method. The Mib, the Morrell ball milling index, is calculated from the same Bond BWi Test data.

Figure 13.27 : Relative BWi and Mib Values of Costa Fuego Deposits



The grinding work indices are consistent with each other but follow a different pattern to the competence (DWi) related factors. Given the two different tests and the two distinct breakage mechanisms involved in those tests, the difference in ranking is expected. The relationship between the two parameters, differentiated by deposit, are shown in Figure 13.28.

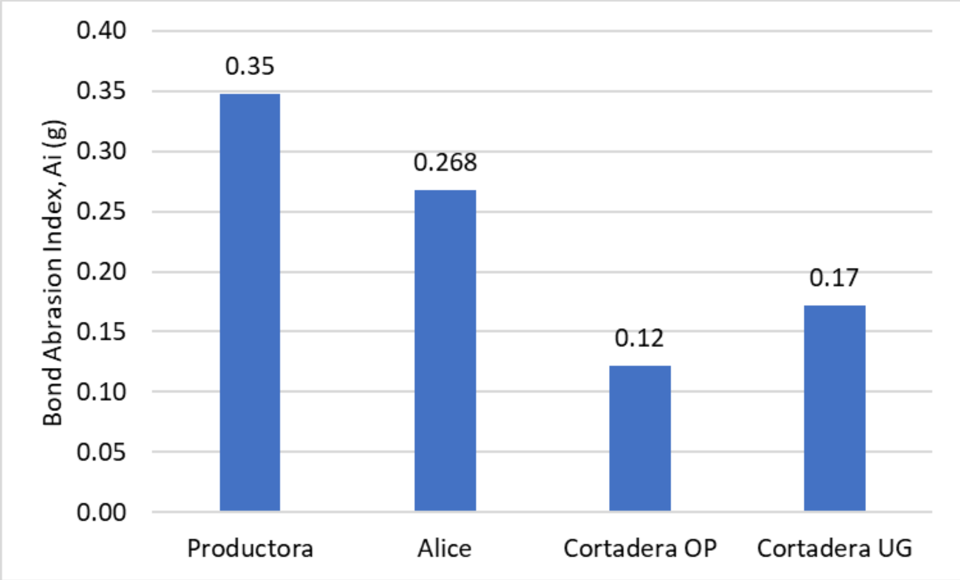
Figure 13.28 : Relationship between BWi and DWi for the Costa Fuego Deposits



The Cortadera OP relationship between BWi and DWi appears to be proportional and the Cortadera UG samples generally continue this trend to harder values. Conversely, the Productora relationship is highly scattered and could be construed to be a negative relationship. The two results for the Alice deposit (geographically adjacent to Productora) lie within the Productora scatter. From this graph, the three main deposits have distinctly different sets of properties, and these differences must be recognised when considering plant throughput calculations.

In Figure 13.29 Bond Abrasion indices (Ai) are compared across the deposits. Ai 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, if high enough, must be taken into account when determining annual utilisation of crushers and mills.

Figure 13.29 : Relative Ai Values of Costa Fuego Deposits

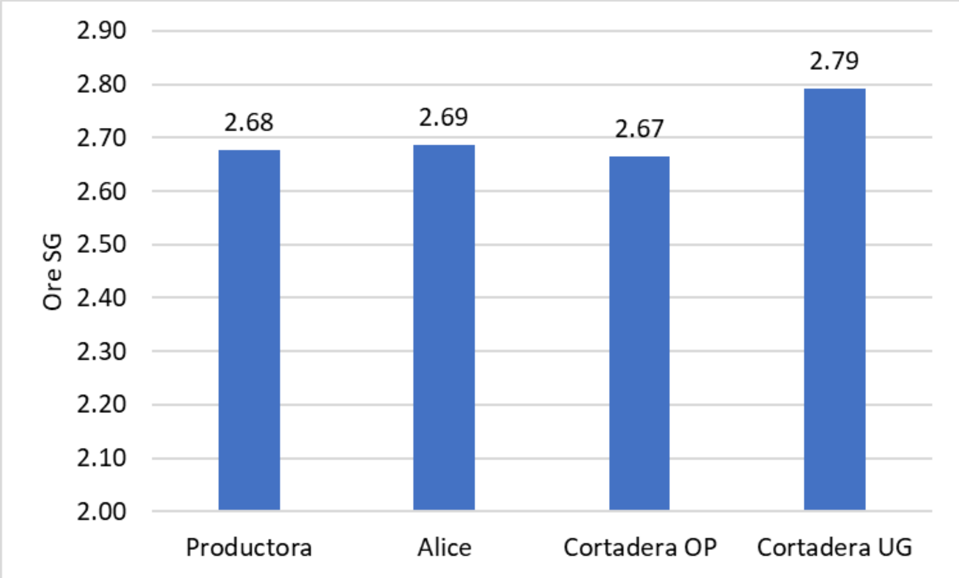


All the average Ai results are low to moderate values, meaning that abrasion related costs would be relatively low. Although there are a small number of Productora Ai values in the "Abrasive" range (above 0.5) that contribute to the high Productora value, all of these were performed very early in the life of the Productora deposit and are inconsistent with the later Productora variability Ai testing. The five Ai values from 2014 average 0.63 while the 17 Productora Variability Ai test results average only 0.28, almost identical to Alice.

Both Cortadera deposits have low abrasiveness and would have lower operating costs for milling and crushing consumables.

The last factor to be considered is the SG, as shown in Figure 13.30. SG can be important in comminution considerations. However, the differences between the Costa Fuego deposits are minor and would have little impact on the calculations or the operations.

Figure 13.30 : Relative SGs of Costa Fuego Deposits



In general, the comminution properties are similar across Costa Fuego, but the 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 production feed at design rates across the life of the project.

13.6.2 Comminution Testing Implications

The 80th percentile comminution results have been used to predict specific power consumption values for SABC (SAG, ball mill and pebble crushing circuit) operation for each of the Costa Fuego deposits. These are summarised in Table 13.13.

Table 13.13 : Design (80th Percentile) Specific Energy by Equipment for Costa Fuego Deposits

Specific Energy (kWh/t)	Productora	Alice	Cortadera OP	Cortadera UG
Primary Crusher	0.10	0.09	0.12	0.10
SAG Mill	8.74	8.44	9.41	10.5
Pebble Crusher	0.24	0.22	0.23	0.27
Ball Mill	13.4	12.4	9.1	11.8
Total	22.5	21.2	18.8	22.6

The maximum total energy consumption for the deposits is 23 kWh/t and, for a 20 Mt/a plant this equates to a power demand of 57.5 MW for comminution alone. When transmission efficiencies are considered, this rises to a combined motor output power for the mills and crushers of about 60 MW.

Although the overfall power requirements are similar for Productora and Cortadera UG, where the power is used in the circuit would be different in each case. To illustrate this, the percentage of power utilised in each plant area is shown in Table 13.14.

Table 13.14 : Design Power Distributions for the Costa Fuego Deposits				
Specific Energy (kWh/t)	**Productora**	**Alice**	**Cortadera OP**	**Cortadera UG**
Primary Crusher	0.5	0.4	0.6	0.4
SAG Mill	39	40	50	46
Pebble Crusher	1.1	1.0	1.2	1.2
Ball Mill	60	59	48	52
Total	100	100	100	100

When treating Productora (or Alice) alone, the SAG mill power would be relatively low while the ball mill power would be at maximum achieving the 125 µm flotation feed P_{80}. When treating Cortadera UG the SAG mill would be operating at maximum power (it would be installed with less power than ball milling) and the ball mill must have its power deliberately reduced if the 125 µm P_{80} target is to be maintained.

On this basis, the SAG and ball mills would be oversized for most of the deposits, but the SAG mill size would be determined by Cortadera UG power requirements while the ball mill size would be determined by the Productora power requirements. Mill selections would be addressed in Section 23.

13.7 Costa Fuego Deposits - Flotation of Sulphides

13.7.1 Introduction

This test program was designed to establish optimal flotation conditions for recovery of copper and molybdenum into saleable products, describe how recovery varies with the production feed grade and provide flotation circuit design data. It also generated samples of concentrate and tailings for settling, pumping and filtration testwork.

Other testing has been successfully conducted but is not relevant to this Report. This includes the selective flotation of pyrite from copper scavenger tails for recovery of cobalt, sulphur and misplaced copper and the non-selective flotation of all sulphides into a single concentrate. These results would not form part of the PEA.

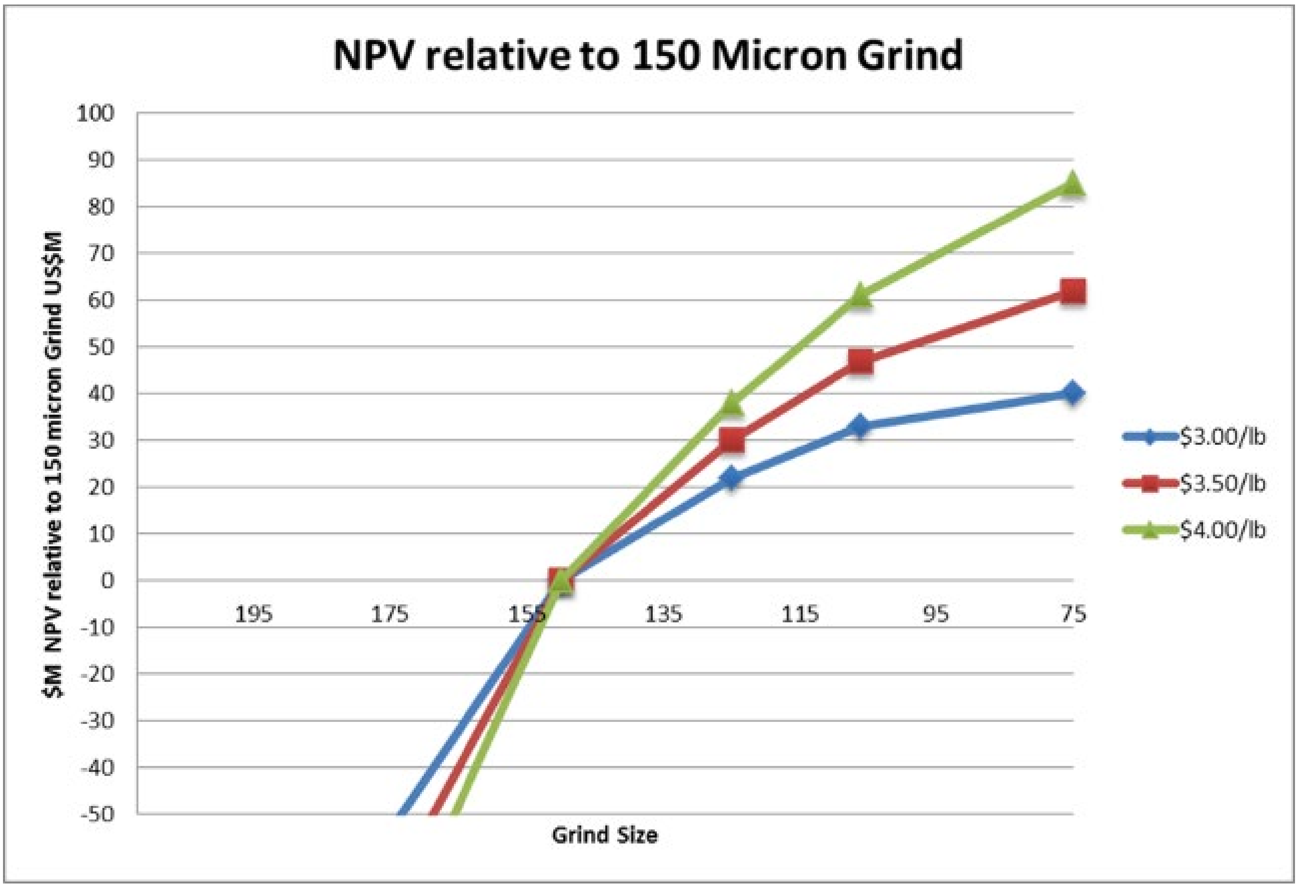
Key aspects of the Productora copper and molybdenum testwork would be discussed before progressing to discuss more recent testwork on the Cortadera production feed. Finally, copper and molybdenum flotation behaviours of all the Costa Fuego deposits would be compared.

13.7.2 Grind Size Optimisation - Productora

The optimal grind size was estimated using a net present value (NPV) based approach incorporating all known benefits and costs attributed to each grind P_{80}. To the greatest extent possible, all test results were normalised before determining costs and benefits.

The results in Figure 13.31 show an increasing relative NPV benefit with finer grind sizes. At the time of the 2016 study, the copper price was in the region of \$3/lb and it was appropriate to adopt a conservative grind size of 125 µm for the Project. At recent (2022) higher copper prices, it may be justified to grind even finer.

Figure 13.31 : Grind Optimisation Analysis Results



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}. However, in a plant when using a hydrocyclone classifier for ball milling, the high copper mineral SG results in a finer P_{80} for copper than the overall ground product. Consequently, the laboratory optimal P_{80} (as determined by the NPV analysis) 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 would report to plant flotation feed at a coarser P_{80}. This occurs because cyclone cut point is inversely proportional to the particle SG.

Consequently, the overall grind size in the plant is set to be one standard sieve size coarser than the optimal P_{80} for Cu recovery. Practically, this means that a 106 µm P_{80} grind size for copper translates to approximately a 125 µm P_{80} grind target in the plant.

Note that this P_{80} theory is not exact and may be optimistic. The target of grinding is to liberate valuable mineral and liberation occurs when gangue particles are broken, not high SG sulphides. This means that grind size is only truly optimised in operations, and it also means that the design must ensure that sufficient ball mill power is installed to allow that optimisation to occur within at least ±10 µm of the nominal (optimal) grind size.

13.7.3 Standard Productora Flotation Flowsheet

A standardised flowsheet and reagent scheme had been determined in previous programs. The scheme uses RTD2086 (xanthate ester supplied by Tall Bennett) at a natural pH (~pH 8) with two stages of cleaning and a nominal 25 µm regrind of combined rougher concentrate. The standardised scheme is presented in Table 13.15.

Table 13.15 : Productora Standardised Flotation Scheme

Operation	Conditioning time (minutes)	pH	Collector RTD2086 (g/t)	Frother MIBC (drops)	Flotation time (minutes)
Mill		Natural pH		As required	
Cu Rgh Con 1	1		10		1
Cu Rgh Con 2	1		5		2
Cu Rgh Con 3	1		5		3
Cu Rgh Con 4	1		5		7
Combine 4xRougher Cons; Regrind to P80 25µm					
Cu Cleaner Con 1	1		5		1
Cu Cleaner Con 2					2
Cu Cleaner Con 3	1		5		3
Cu Cleaner Con 4					5
Cu Cleaner Scav Con	1		2		7
Combine 4x Cleaner Cons					
Cu Re-Cleaner Con 1	1		5		1
Cu Re-Cleaner Con 2					2
Cu Re-Cleaner Con 3					3
Cu Re-Cleaner Con 4					6

Optimisation testwork was conducted on a number of samples but was mainly focused on the larger mass FD-1 composite sample. Primary grind size optimisation test series were conducted on all the Productora Flotation Development samples (designated FD-X, where X is the sample number) and Alice variability samples. Variability samples were also prepared and tested for Productora (all variability samples are designated V-X).

Optimisation variables included reagent dosage, alternate reagents, diesel addition for improved molybdenum recovery, water type (sea and tap) and flotation time. Once optimised conditions were being tested on the variability samples, it was clear that samples from different zones were giving different flotation results. Samples identified as containing transitional sulphide and oxide material (designated as containing greater than 10% acid soluble copper) gave reduced recovery and grade performance with the standardised scheme compared to fresh production feed samples.

Grouping of the samples was as follows:

- Productora Fresh (also in-pit fresh) – FD-1, FD-2, FD-6, FD-13, FD-14, V-2, V-7, V-8, V-10, V-11, V-12
- Productora Transition (>10% Acid Soluble Cu) – FD-3, FD-4, FD-5, V-1, V-9

- Alice – V-13, V-14, V-18.

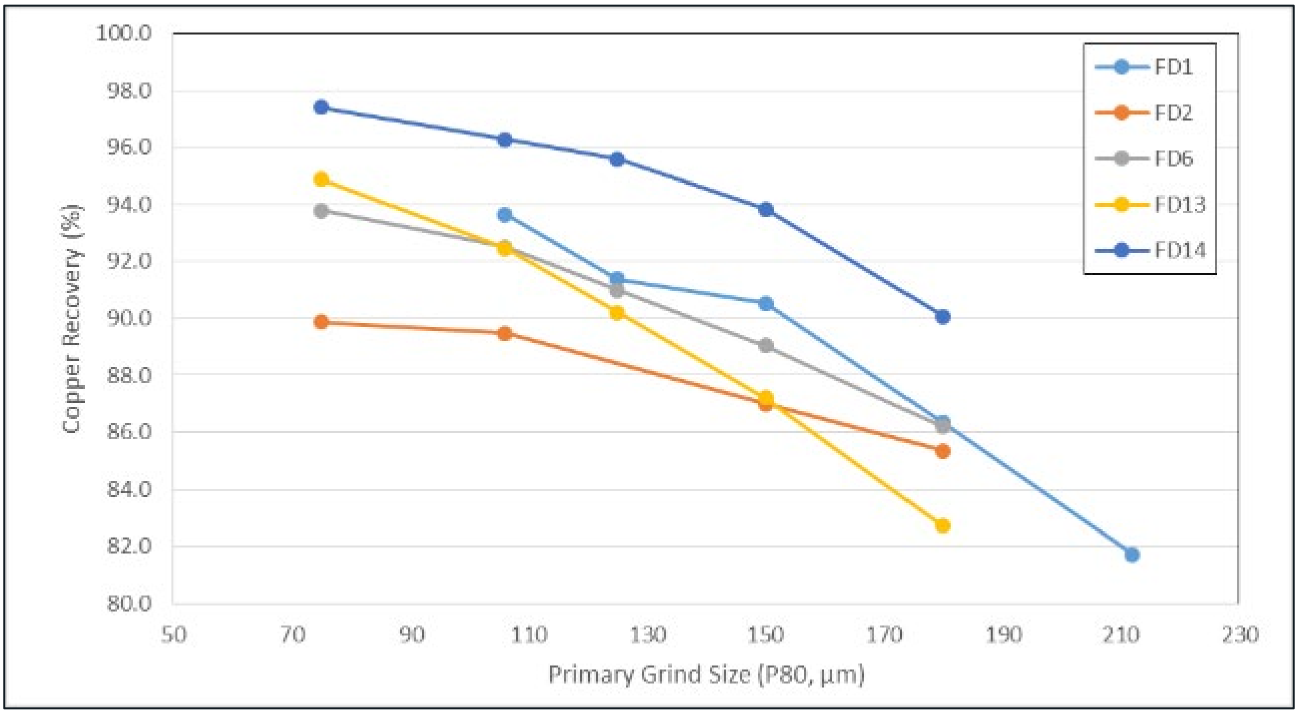
Apart from Alice, which is a distinct nearby deposit, all other sample designations (Central, Habanero, CChen, etc.) now fall under the one classification label as "Productora".

The standard Productora scheme has been modified in a number of instances to improve the recovery of copper and to enhance flotation of accessory minerals like molybdenite. These variations are especially advantageous when treating samples of transition production feed (supergene) and may also provide some benefits with the fresh samples.

13.7.4 Productora In-Pit Fresh Samples - Flotation Testwork Results

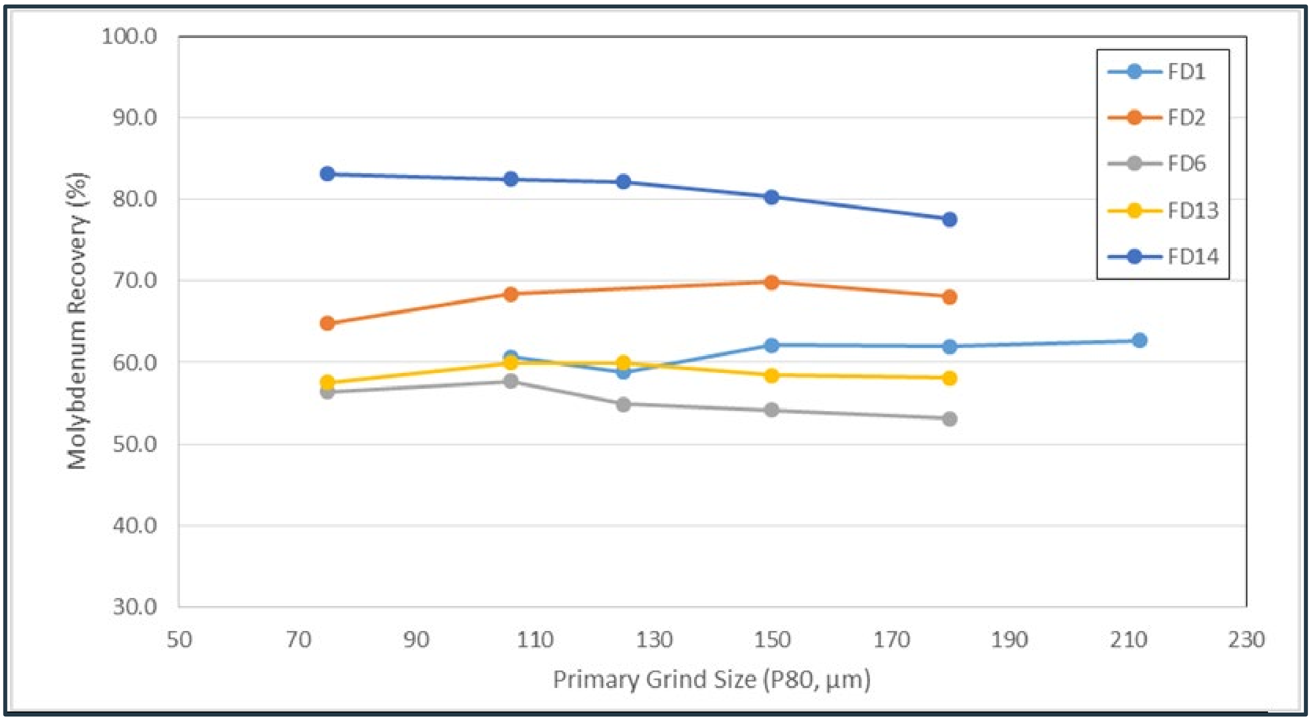
Primary grind sizes ranging from P_{80} =75 µm to 212 µm were tested on each of the FD samples. Grind size vs. rougher float copper and molybdenum rougher recovery are shown in Figure 13.32 and Figure 13.33.

Figure 13.32 : Productora Fresh – Primary Grind Size Copper Rougher Recoveries



Copper recovery from Productora shows significant sensitivity to grind P_{80} while molybdenum recovery shows almost no sensitivity. Copper rougher recoveries varied from 83% to 90% at 180 µm (FD-13 and FD-14, respectively) and 90% to 96% at 106 µm (FD-2 and FD-14, respectively). The recovery difference between floating at 180 µm and 106 µm for individual samples varies between 5% (FD-2) and 12% (FD-13).

Figure 13.33 : Productora Fresh – Central – Primary Grind Size Molybdenum Results



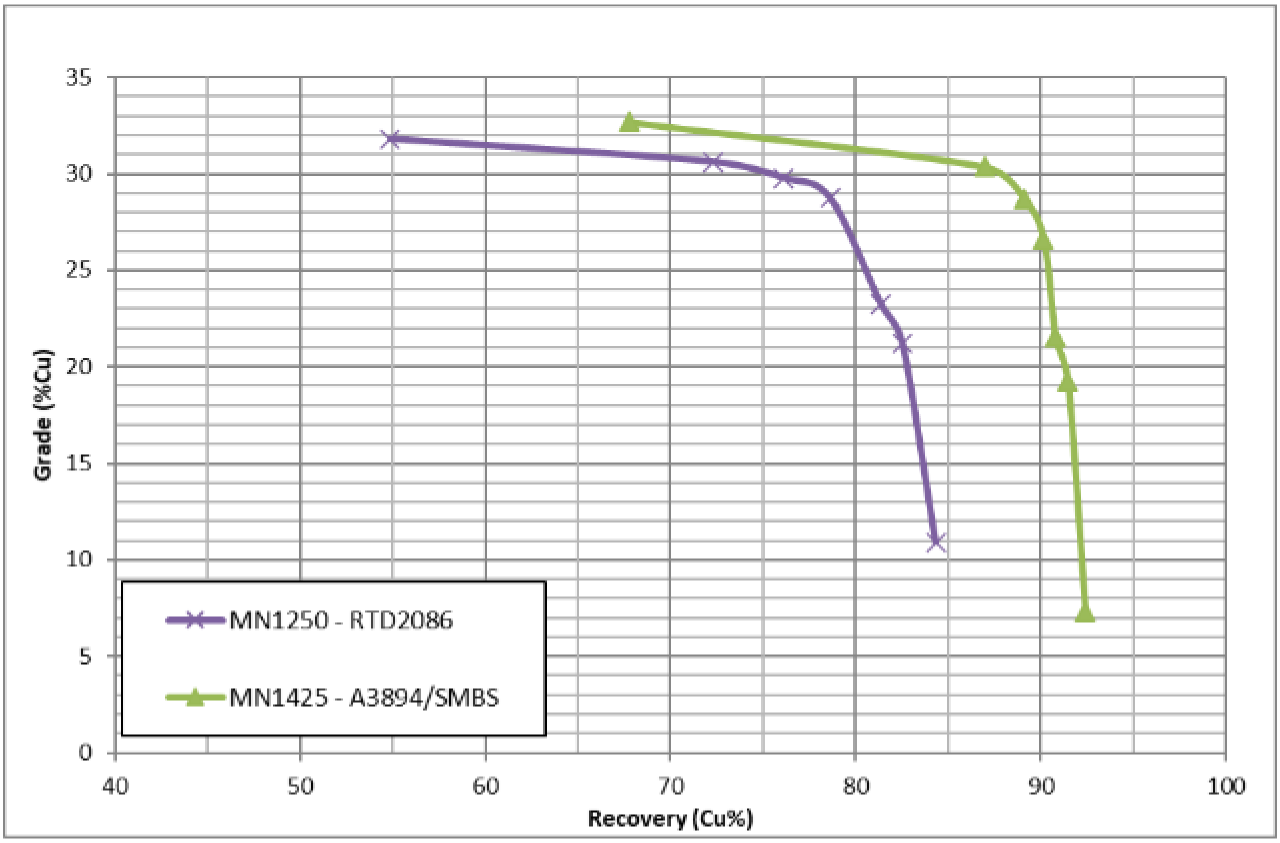
For the purposes of optimisation, there is no trend for molybdenum recovery with grind size with any of the samples.

Various reagents and depressants were tested to determine if the selected scheme (Table 13.15) was optimal when treating all potential feed samples. The optimisation led to some changes in addition rates of the xanthate ester (Tall Bennett RTD2086) and some flotation time changes, but the xanthate ester proved to be superior to all alternative approaches. RTD2086 is highly effective at floating chalcopyrite and has high selectivity against pyrite.

Concentrate cleaning was investigated and two cleaning stages were found to be optimal, as was the 25 µm regrind P_{80}.

With weathered (but not fully oxidised) production feeds, RTD2086 was unable to recover secondary sulphides. However, it was possible to increase copper recovery using the Cytec thiocarbamate collector, A3849, used in combination with sodium metabisulphite (SMBS) to depress pyrite. This benefit is demonstrated in Figure 13.34 with tests on sample FD-2.

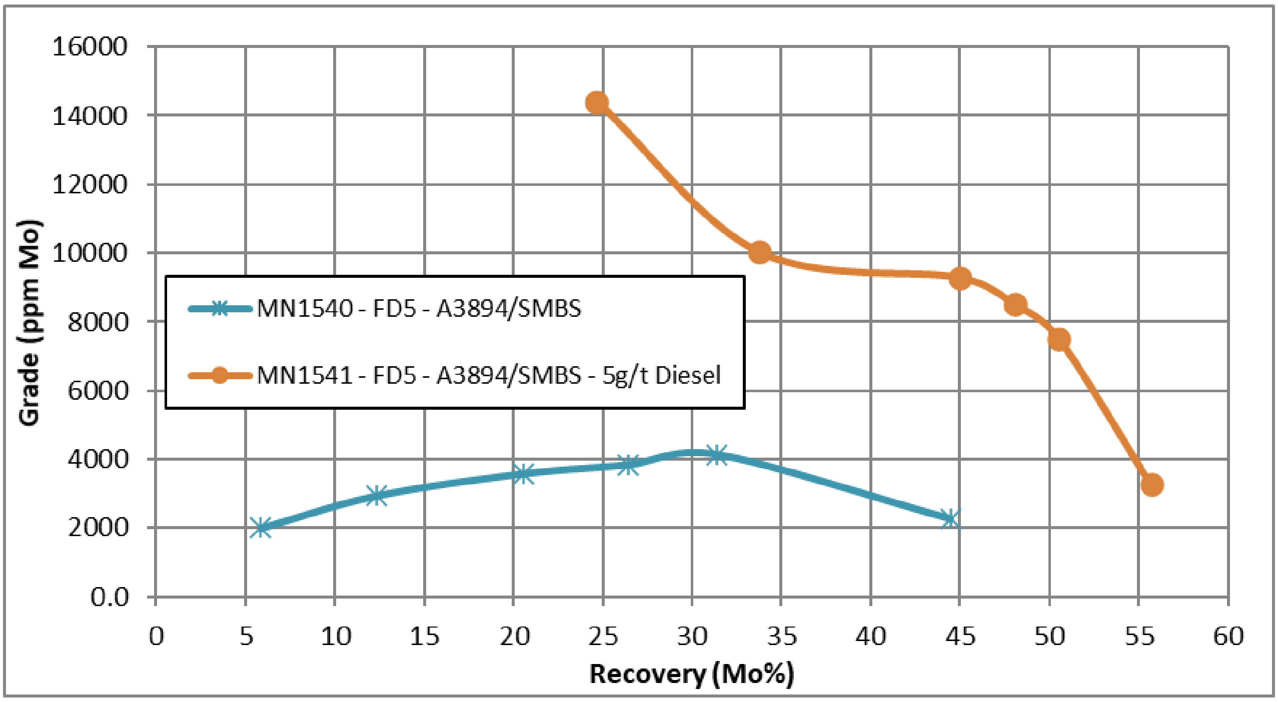
Figure 13.34 : Productora – Central – FD2 RTD2086 vs. A3894/SMBS Copper Grade Recovery



These results suggest that the alternate reagents Cytec A3849 and SMBS must have storage and feeding facilities provided in the design so that copper recoveries can be maximised when treating altered and partially oxidised production feeds.

The molybdenum recoveries achieved by using A3894 in combination with SMBS were found to be lower than expected. Comparative testing showed improved molybdenum grade and recovery when 5 g/t of diesel was added to the copper flotation, as demonstrated in Figure 13.35.

Figure 13.35 : Comparison of Molybdenum Recovery (%) vs. Molybdenum Grade (ppm) for Productora Central Sample FD5 A3894/SMBS vs. A3894/SMBS - 5 g/t Diesel Molybdenum Grade Recovery



This work shows that diesel provides a method of regaining lost molybdenum recovery with altered and partly weathered production feeds. Provision should be made in the reagent area for feeding diesel to rougher flotation.

13.7.4.1 Quality of Water Testwork

All the Project flotation testwork was conducted using seawater. This is due to insufficient availability of fresh water at the Productora site. Supply of fresh water would require a reverse osmosis (RO) plant and this would add considerable CAPEX and OPEX to the Project.

Tests were performed to demonstrate the differences between flotation in seawater and flotation in tap water. Some minor differences were shown but the small benefit of using tap water does not justify the expense of treating the seawater in an RO plant.

Seawater is now used for copper flotation at numerous sites globally, including many in Chile. The Project testwork showed similar responses compared to these other sites.

13.7.4.2 Cleaner and Locked Cycle Testwork

For each of the samples, locked cycle copper recovery at 25% Cu in concentrate (target grade) was either interpolated from the results or extrapolated (less certain) when the maximum grade achieved was less than 25% Cu. Molybdenum and gold recoveries were also determined at a 25% Cu grade. Final results for copper, molybdenum and gold are shown in Table 13.16, Table 13.17 and Table 13.18 respectively.

In the absence of an actual locked cycle test (LCT), equivalent locked cycle results were determined by averaging rougher and recleaner recoveries based on the assumption that 50% of the copper "lost" to middlings between roughers and cleaners would be recovered into the final concentrate in a locked cycle test. The other 50% of the copper would be rejected to tailings. In a locked cycle test all materials are eventually distributed between the concentrate and tailings by design. Locked cycle test performance is calculated on this steady-state basis, even though some stable middlings streams remain at the conclusion of each test.

Table 13.16 : Productora Rougher, Cleaner and Locked Cycle Copper Recovery Results

Sample	Head Grade		Rougher		Recleaner (actual)		Recleaner @ 25%Cu	Locked Cycle (actual)		Equiv. Locked Cycle @ 25%Cu	
	% Cu	%ASCu	% Cu	% dist.	% Cu	% dist.	% dist.	% Cu	% dist.	% Cu	% dist.
SDI	0.48		8.9	93.3	22.8	90.9	86.0			25	89.7
SDG	0.57		9.4	95.6	20.8	93.8	86.0			25	90.8
FD-1	0.44	5%	9.3	91.4	28.0	78.0	82.0	28.2	88.8	25	88.8
FD-2	0.59	9%	7.2	93.2	26.6	90.2	90.5	27.2	91.0	25	91.0
FD-3	0.54	10%	5.7	87.5	23.5	84.1	83.2	30.7	82.3	25	82.3
FD-4	0.72	16%	5.8	85.5	26.9	81.6	82.0	28.8		25	83.8
FD-5	0.24	26%	5.3	75.2						25	63.0
FD-6	0.22	7%	6.3	91.0	25.8	80.4	81.0	28.1	85.1	25	85.1
FD-13	0.89	6%	11.6	94.5	25.4	90.4	90.5	27.4	91.7	25	91.7
FD-14	0.85	4%	10.5	95.6	25.6	89.3	89.5			25	92.5
V-1	0.18	25%	3.4	67.7			57.4			25	62.5
V-2	0.26	3%	9.0	88.2	26.9	78.7	80.5			25	84.4
V-7	0.34	3%	8.0	91.9	25.1	90.0	90.0	29.7	87.5	25	87.5
V-8	0.70	3%	12.1	96.4	24.0	93.0	91.5			25	93.9
V-9	0.74	32%	9.8	76.2	25.0	71.3	72.0			25	74.1
V-10	0.49	7%	9.0	92.4	28.5	85.1	86.0			25	89.2
V-11	1.01	3%	8.1	98.3	20.2	94.7		31.7	94.3	25	94.3
V-12	0.11	5%	6.0	94.6			87.7			25	91.2
FC1	0.56	14%	0.56	80.0	25.5	76.3	76.3			25	78.2
FC2	0.62	16%	0.62	81.0	23.5	72.0	71.0			25	76.0
FC3	0.59	24%	0.59	61.7	25.9	59.9	60.0			25	60.9
FC4	0.56	8%	0.56	87.4	24.7	81.9	81.5			25	84.5
FC5	0.64	8%	0.64	87.4	25.8	79.2	80.0			25	83.7
FC6	0.37	15%	0.37	93.6	24.4	82.1	82.0			25	87.8

The FD and V series of tests were completed for the 2016 Study while the FC series tests were completed in 2022. No Productora locked cycle tests were completed in 2022.

Table 13.17 : Productora Rougher, Cleaner and Locked Cycle Molybdenum Recovery Results

Sample	Head grade		Mo Recovery (%, actual)			Equiv. Locked Cycle @ 25%Cu in Cons	
	g/t Mo	%ASCu	Rougher	Recleaner	Locked Cycle	% Cu in Con	% Mo Recovery
SDI	170		62	63		25	63
SDG	400		78	74		25	74
FD-1	160	5%	70	56	39	25	39
FD-2	180	9%	72	60	59	25	59
FD-3	120	10%	70	55	25	25	25
FD-4	140	16%	58	22	36	25	36
FD-5	160	26%	59	40		25	50
FD-6	140	7%	68	40	47	25	47
FD-13	120	6%	72	60	60	25	60
FD-14	440	4%	83	70		25	77
V-1	40	25%	41	30		25	36
V-2	40	3%	42	20		25	31
V-7	40	3%	43	43	30	25	30
V-8	160	3%	63	56		25	60
V-9	60	32%	41	40		25	41
V-10	120	7%	57	46		25	52
V-11	120	3%	68	50	48	25	48
V-12	80	5%	55	29		25	42
FC1	52	14%	50	47		25	49
FC2	520	16%	87	85		25	86
FC3	103	24%	71	70		25	71
FC4	270	8%	89	81		25	85
FC5	1 070	8%	77	70		25	74
FC6	284	15%	90	88		25	89

Table 13.18 : Productora Rougher, Cleaner and Locked Cycle Gold Recovery Results

Sample	Head grade		Au Recovery (%, actual)			Equiv. Locked Cycle @ 25%Cu in Cons	
	g/t Au	%ASCu	Rougher	Recleaner	Locked Cycle	% Cu in Con	% Au Recovery
SDI	0.09		79	63		25	71
SDG	0.12		78	50		25	64
FD-1	0.17	5%	70	50	58	25	58
FD-2	0.17	9%	86	63	65	25	65
FD-3	0.08	10%	73	52	42	25	42
FD-4	0.14	16%	63	44	53	25	53
FD-5	0.14	26%	66	54		25	60
FD-6	0.12	7%	82	63	57	25	57

Table 13.18 : Productora Rougher, Cleaner and Locked Cycle Gold Recovery Results							
Sample	Head grade		Au Recovery (%, actual)			Equiv. Locked Cycle @ 25%Cu in Cons	
	g/t Au	%ASCu	Rougher	Recleaner	Locked Cycle	% Cu in Con	% Au Recovery
FD-13	0.18	6%	86	83	73	25	73
FD-14	0.21	4%	95	88		25	92
V-1	0.05	25%	55	31		25	43
V-2	0.06	3%	62	33		25	48
V-7	0.10	3%	59	43	57	25	57
V-8	0.17	3%	83	60		25	72
V-9	0.11	32%	70	60		25	65
V-10	0.15	7%	77	51		25	64
V-11	0.25	3%	93	60	64	25	64
V-12	0.05	5%	36	13		25	25
FC1	0.11	14%	48	43		25	46
FC2	0.11	16%	63	52		25	58
FC3	0.14	24%	47	45		25	46
FC4	0.11	8%	68	56		25	62
FC5	0.10	8%	57	44		25	51
FC6	0.08	15%	64	58		25	61

13.7.5 Alice - Flotation Testwork Results

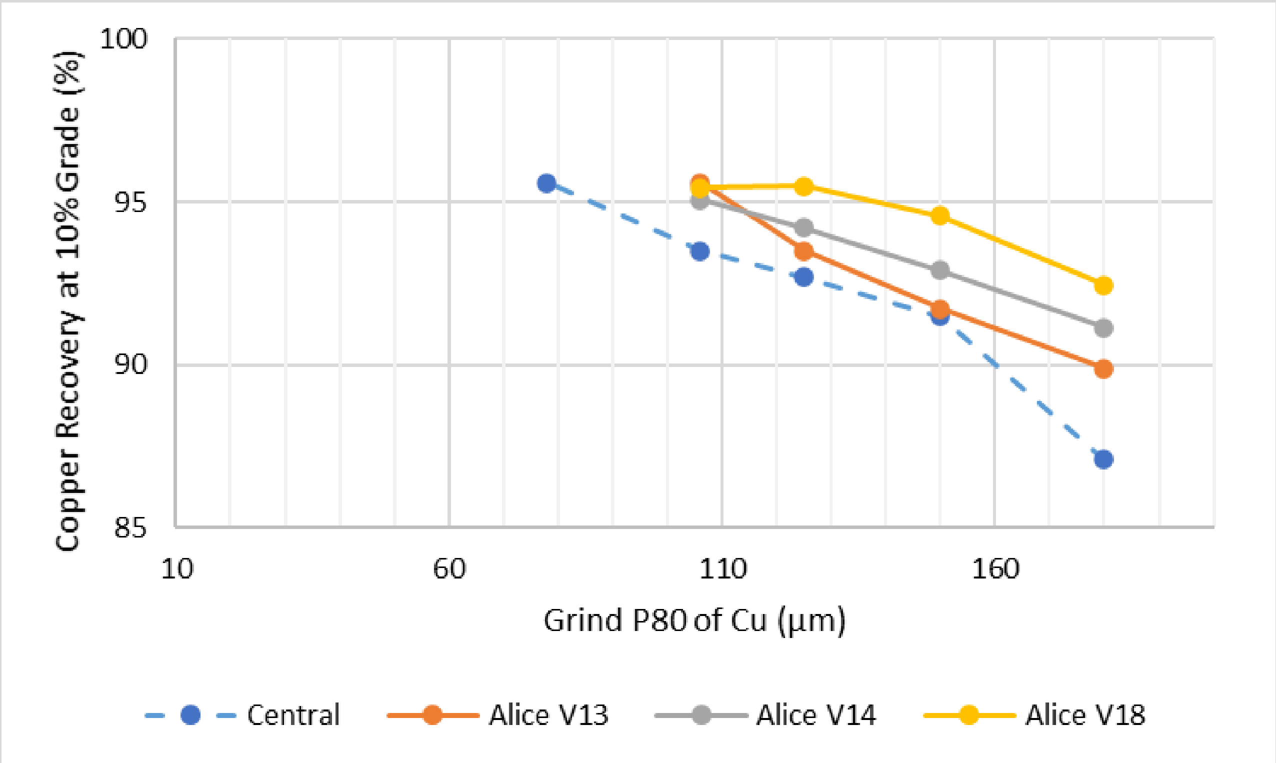
For the PFS, three samples were prepared from the Alice deposit (V-13 and V14 from PXP0001 and V-18 from MET024) for comminution and flotation testwork. Standard flotation tests were conducted on these samples including rougher grind series, two stage cleaners and locked cycles (V-13 and V-18 only). Additional samples from Alice were tested in 2022 and were tested in rougher, cleaner and recleaner open circuit tests. No locked cycle tests were performed on Alice samples in 2022.

Mineralogically, the Alice production feed is slightly different to Productora. The dominant sulphide minerals are still chalcopyrite and pyrite, but the non-sulphides differ. In the Productora deposit, K-feldspars are generally the dominant mineral present, followed by quartz, chlorites and micas. In the Alice samples, there is little K-feldspar. Instead, albite (Na-feldspar) and other plagioclase feldspars and higher levels of calcite are present.

13.7.5.1 Primary Grind Series Testwork

Copper rougher recovery vs. grind size (normalised to 10% Cu rougher grade) is shown in Figure 13.36.

Figure 13.36 : Alice – Primary Grind Size Copper Results – Normalised to 10% Cu in Rougher Concentrate



Though V-18 copper head grade was low (0.24% Cu), copper recovery was very high with only 0.01% Cu remaining in the tailings (typically 0.04 to 0.05%). All show superior recoveries to Productora samples suggesting that the chalcopyrite in Alice has a higher degree of liberation than in Productora.

There was little change in molybdenum recovery with grind size, showing that (like Productora) molybdenite liberation is unaffected by grind in the 106 to 200 µm range.

13.7.5.2 Cleaner and Locked Cycle Testwork

Both single and two stage cleaner floats were to be conducted for each of the samples. For each sample, tests were conducted using the standard RTD2086 scheme at a primary grind of P_{80}=180 µm and regrind of P_{80}=25 µm. No other optimisation was conducted.

Locked cycle tests were conducted on V-13 and V-18, with standard conditions (cleaner scavenger con recirculated to regrind of next cycle, recleaner tail recirculated to cleaner feed) at P_{80}=180 µm. An additional test was conducted on V-13 at P_{80}=125 µm.

From the results, locked cycle recoveries at 25% Cu were estimated for modelling. For the V-14 sample (in the absence of a LCT), the recovery was based on the average of rougher and recleaner recoveries. As done for Productora, molybdenum and gold recoveries were also estimated at a 25% Cu concentrate grade. Two recent (2022) tests on new Alice samples were included in the results. Recoveries for copper, molybdenum and gold are summarised in Table 13.19, Table 13.20 and Table 13.21.

Table 13.19 : Alice Rougher, Cleaner and Locked Cycle Copper Results											
Sample	Overall Copper Results (P_{80}=180 µm)										
	Head grade	Rougher		Recleaner		Recleaner @ 25%Cu		Locked Cycle		Equiv. Locked Cycle @ 25%Cu	
	%Cu	%Cu	% dist.	%Cu	% dist.	%Cu	% dist.	%Cu	% dist.	%Cu	% dist.
V-13	0.50	8.5	92.8	25.4	83.2	25.0	83.5	28.7	91.9	25.0	91.9
V-14	0.32	9.6	91.3	22.9	88.2	25.0	87.5	-	-	25.0	89.4
V-18	0.24	7.7	93.8	28.0	90.2	25.0	90.5	25.1	91.5	25.0	91.5
AF1	0.50	12.7	92.5	25.4	88.5	25.0	88.5			25.0	90.5
AF2	0.53	13.3	92.8	25.3	88.5	25.0	88.5			25.0	90.7

Table 13.20 : Alice Rougher, Cleaner and Locked Cycle Molybdenum Results					
Sample	Overall Molybdenum Results (P_{80}=180 µm)				
	Head grade	Rougher	Recleaner @ 25%Cu	Locked Cycle	Equiv. Locked Cycle @ 25%Cu
	ppm Mo	% dist.	% dist.	% dist.	% dist.
V-13	60	67	52	62.3	62
V-14	40	65	61	-	68
V-18	20	36	34	36.9	37
AF1	47	77	73	-	75
AF2	60	87	83	-	85

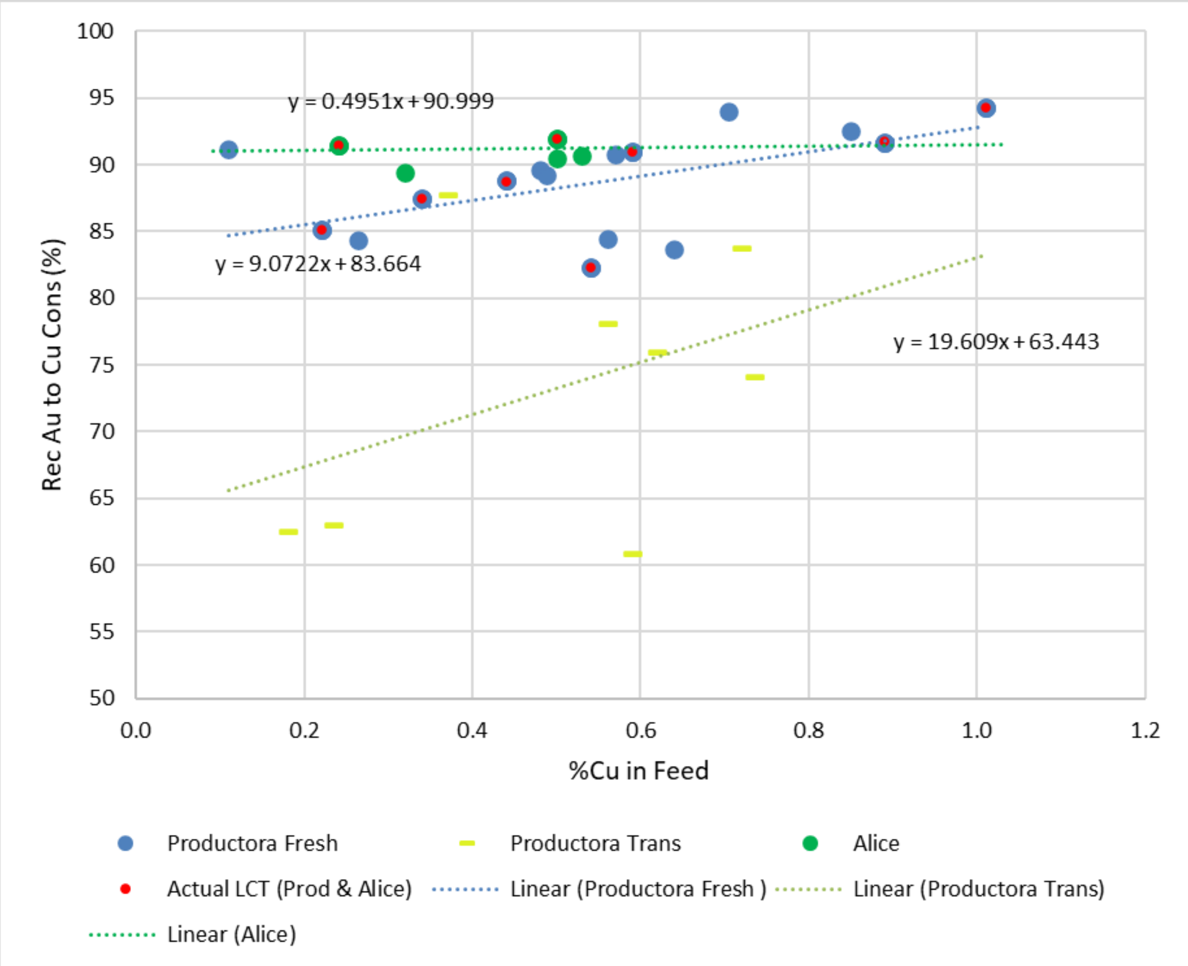
Table 13.21 : Alice Rougher, Cleaner and Locked Cycle Gold Results					
Sample	Overall Gold Results (P80=180µm)				
	Head grade	Rougher	Recleaner @ 25%Cu	Locked Cycle	Equiv. Locked Cycle @ 25%Cu
	ppm Au	% dist.	% dist.	% dist.	% dist.
V-13	0.08	62	52	56	56
V-14	0.04	63	37	-	50
V-18	0.01	31	24	45	45
AF1	0.02	85	74	-	79
AF2	0.04	60	53	-	57

13.7.5.3 Recovery Model Development

Using the equivalent locked cycle results for both Productora and Alice, recovery to a 25% copper concentrate was plotted against head grade to derive a predictive relationship for use in mine planning and revenue calculation.

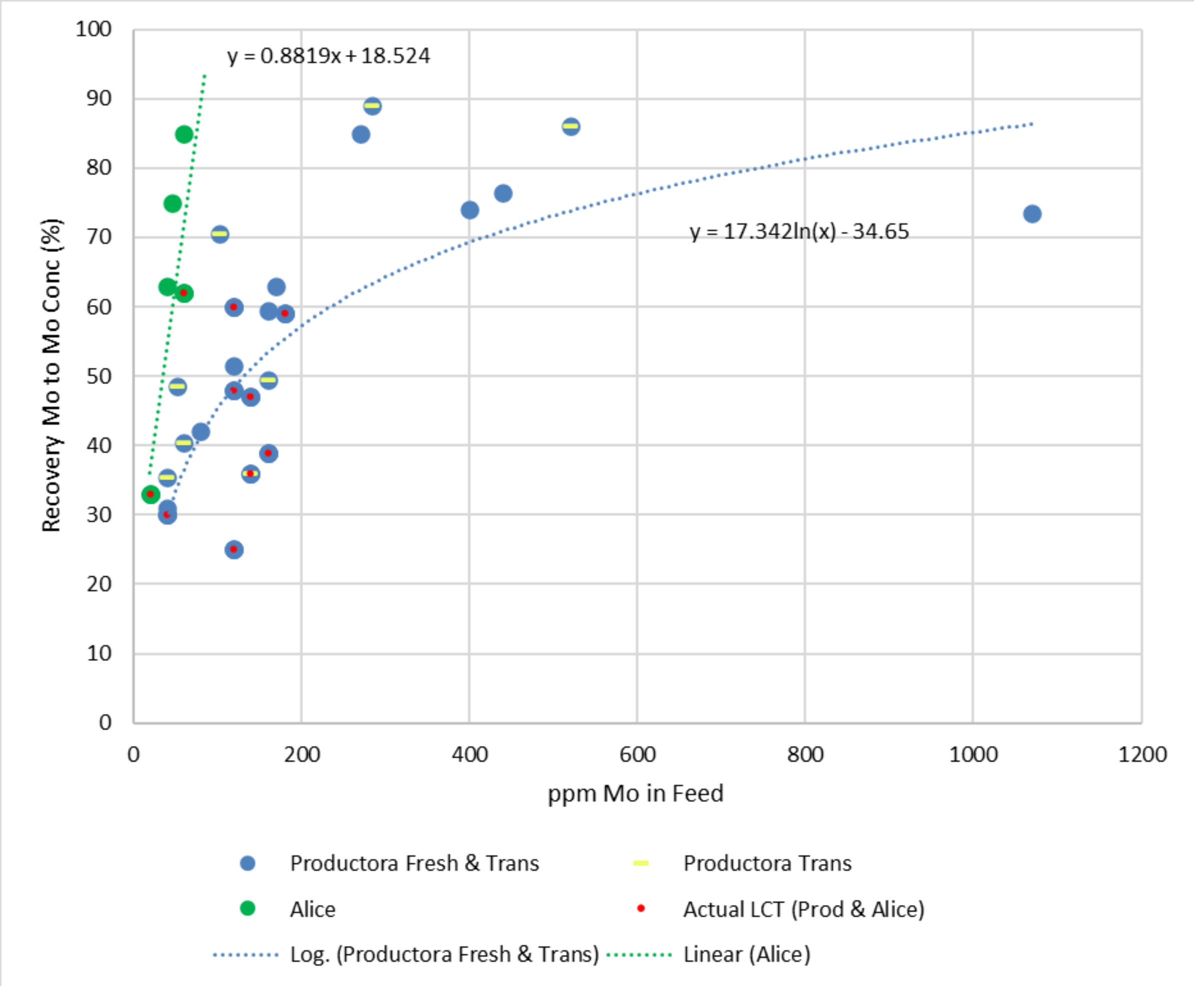
For copper, head grade vs. recovery is shown in Figure 13.37. As expected, there is a spectrum of performance from high recoveries for fresh Productora samples to lower recoveries for various levels of oxidation and alteration in Productora transition samples. A nominal division has been applied between fresh and transition samples, based on transition being any sample with >10% acid soluble copper. As is clear from the graph, there is overlap between the fresh and transition types. All Alice samples are classed as fresh.

Figure 13.37 : Productora and Alice Copper Recoveries vs. Copper in Feed



Linear fits were applied to the Productora fresh and transition results and the Alice results. These linear fits are deliberately biased toward the LCT results. This bias is achieved by including two repeats of the LCT results in the fitting data sets (i.e. each LCT is represented by three sets of the same results) compared to a single result for each open circuit cleaning test (which only provides an "equivalent" LCT recovery value). Locked cycle test results are marked with red dots in Figure 13.37 This same biasing was applied to molybdenum and gold model development.

Figure 13.38 : Productora and Alice Molybdenum Recoveries in Copper Concentrate vs. Molybdenum in Feed

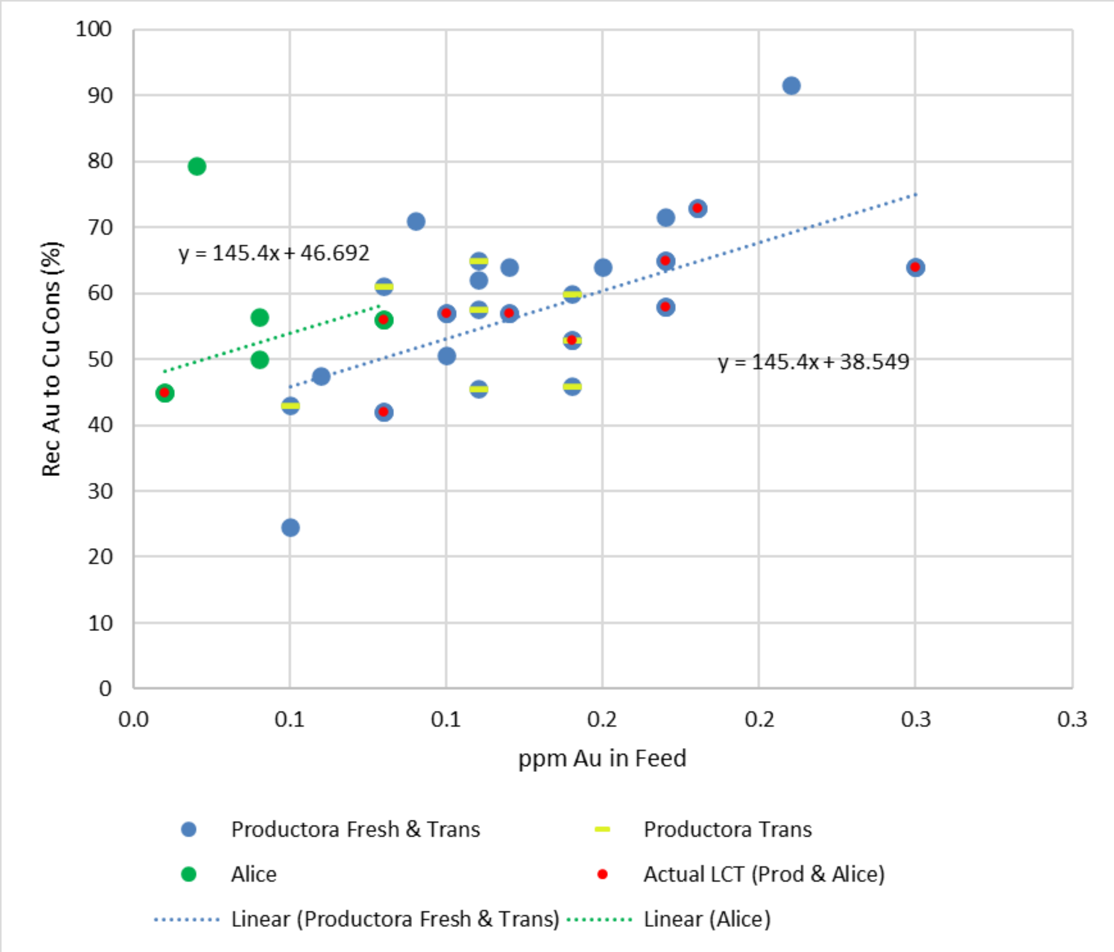


For molybdenum, there is a significant difference between the Alice behaviour and Productora behaviour. As was seen for copper, it appears that the Alice molybdenite is also more responsive to flotation than the Productora molybdenite. In addition, Productora molybdenum flotation is not affected by oxidation or alteration of the transition samples. Quite the opposite may be true, with alteration enhancing Productora molybdenum recovery compared to the fresh samples.

Both the Alice and Productora predictive curves have been capped at 90% recovery. It is also necessary to place a lower limit on the logarithmic Productora recovery calculation to prevent negative outcomes.

For molybdenum revenue calculations further allowance must be made for losses in the molybdenum cleaning circuit. These predictions only provide a recovery to copper concentrate (where % Mo is at maximum 2% and usually about 0.5%) but the saleable product needs to assay 50% Mo.

Figure 13.39 : Productora and Alice Gold Recoveries in Copper Concentrate vs Gold in Feed



Gold recovery showed a strong dependence on feed grade. However, there was no difference between recovery from fresh samples compared to transition samples. The Productora fit line is generated by Excel without intervention. The Alice data, however, has a single significant outlier result compared to the four other results. To reflect the behaviour of the four regular points (including two which are locked cycle results) the intercept value in Excel was manually set so as to match the slope of the Alice line with the Productora line. As can be seen, both equations commence with 145.4x, where x is the % Cu in feed and 145.4 is the slope in terms of (%Rec)/(g/t Au in feed).

Both equations require a cap at 90% recovery.

Gold would be sold in the copper concentrate so gold recovery to copper concentrate is the basis of gold revenue calculations.

All the predictive equations are summarised in a comparative table after the Cortadera testwork section refer to Table 13.45.

13.7.6 Productora Copper Concentrate Comprehensive Analysis

Comprehensive concentrate assays were conducted on final concentrate from the FD-6, FD-13 and V13 locked cycle tests. Assays were conducted to determine diluent elements and potential level of penalty elements in the Productora production feed. Comprehensive assay results are shown in Table 13.22.

Table 13.22 : Comprehensive Productora Concentrate Analysis

Element	Productora MN1537 FD-6 Final Con	Productora MN1478 FD-13 Final Con	Alice MN1536 V13 Final Con
Al_2O_3 (%)	1.22	2.41	1.17
As (%)	0.02	<0.01	0.02
Au (ppm)	4.77	5.23	2.96
Ba (%)	<0.01	<0.01	<0.01
Be (ppm)	<20	<20	<20
Bi (%)	0.003	<0.002	0.005
CaO (%)	0.09	0.10	0.3
Cd (ppm)	<20	<20	<20
Cl (ppm)	360	350	160
Co (%)	0.2	0.063	0.009
Cr (%)	0.01	0.02	<0.01
Cu (%)	28.1	27.6	29.4
F (ppm)	80	140	90
Fe (%)	27.6	26.2	27.9
Hg (ppm)	2	0.7	1.1
K (%)	0.38	0.68	0.10
MgO (%)	0.06	0.39	0.20
Mn (%)	0.01	<0.01	<0.01
Ni (%)	0.02	0.02	<0.01
P (%)	<0.01	0.01	<0.01
Pb (%)	0.42	<0.01	<0.01
S (%)	33.5	30.1	32.2
Sb (%)	0.04	0.02	<0.01
Se (ppm)	80	75	160
SiO_2 (%)	3.8	8.76	3.63
Sn (%)	<0.01	<0.01	<0.01
Sr (%)	<0.001	<0.001	<0.001
Te (ppm)	4.4	2.6	0.8
Th (ppm)	4	8	2
Ti (%)	0.05	0.1	0.22
U (ppm)	14	38	<2
V (%)	<0.001	<0.001	<0.001
Zn (%)	0.18	0.02	<0.01
Zr (%)	0.014	0.023	0.009

The analyses show that the concentrate is free of impurities according to penalty trigger limits.

13.7.6.1 Molybdenum in Copper Concentrate

The molybdenum content is not reported above in Table 13.22 as it is misleading. The concentrates have not been subject to molybdenum removal for the purposes of making a saleable molybdenum concentrate. The molybdenum contents for the three concentrates were 13 434, 4185 and 3300 ppm respectively. Molybdenum flotation testwork (reported later in this Report) showed that between 80 and 90% of the molybdenum could be removed from the copper concentrate into a saleable grade molybdenum concentrate (>50% Mo). Note that the molybdenum testing was deliberately performed using a high-grade molybdenum sample, so that it was possible to conduct some meaningful molybdenum upgrading flotation work on the bulk copper concentrate. The copper flotation component of the upgrading test was conducted in open circuit, as were all subsequent molybdenum flotation tests. Therefore, the molybdenum recoveries from copper concentrate would be conservative. The copper recleaner concentrate, equivalent to the concentrates in Table 13.22, was 22 to 23% Cu and it was nominally 1.87% (18 700 ppm) Mo.

Molybdenum removal in Test MN1494 resulted in a molybdenum concentrate grading 49.8% Mo and 1.58% Cu. It contained 87% of the molybdenum that floated with copper using the Productora flowsheet. The molybdenum in flotation tailings was reduced to ~1700 ppm and the grade of the copper concentrate increased from 22.3% Cu to 23.0% Cu. Based on these actual results a 90% recovery has been applied to the three tests above to allow estimated molybdenum contents of final copper concentrate to be calculated. The results are in Table 13.23.

Table 13.23 : Effect of Removing Molybdenum from Copper Concentrate

	MN1494 (Actual)	MN1537 (Calc)	MN1478 (calc)	MN1536 (calc)
% Cu – Cu Recleaner	22.3	27.6	26.2	27.9
ppm Mo Cu Recleaner	17400	13434	4185	3300
Grade of Mo Concentrate (%Mo)	49.8	50	50	50
Mo Recovery to Mo Concentrate (%)	86.7	90	90	90
%Cu in Mo Concentrate	1.58	1.5	1.5	1.5
Cu Dist to Mo Cons (%)	0.21	0.131	0.043	0.032
Mass Mo Cons (Wt%)	3.02	2.42	0.75	0.59
Mass Cu Cons (Wt%)	96.97	97.58	99.25	99.41
%Cu Cu Con	22.9	28.2	26.4	28.1
ppm Mo Cu Conc	2386	1377	422	332

The final molybdenum grade in copper concentrate is reduced to about 10% of the molybdenum grade in the copper Recleaner concentrate. Due to the removal of the molybdenum concentrate, the copper grade in concentrate increases by between 0.2 and 0.6% Cu.

13.7.6.2 Copper Sulphide Flotation Testwork Summary

A program of flotation testwork was conducted on the Productora and Alice deposits. Testwork was based on a standardised flowsheet and reagent scheme which had been determined during previous testwork programs. The scheme used RTD2086 (xanthate ester supplied by Tall Bennett) at a natural pH of seawater with two stages of cleaning and a 25 µm regrind.

Target grade for the final concentrate was 25% Cu. To achieve this, optimisation testwork was conducted on a number of composites to improve copper grade and recovery. Optimisation testwork included the following:

- Primary grind size tests
- Reagent dosage
- Flotation time
- Alternate reagents
- Diesel addition for increased molybdenum flotation
- Seawater vs. tap water.

When optimised conditions were tested, it was identified that samples from different zones were reporting different flotation responses. Some samples were also later identified as containing transitional sulphide and oxide material (designated as containing greater than 10% acid soluble copper). These samples gave sub-optimal results with the standardised scheme.

A primary grind size of P_{80}=125 µm was selected for all Productora samples to target a similar sulphide P_{80} compared to a plant grind achieving P_{80}=150 µm. For the Alice deposit, a primary grind size of P_{80}=125 µm was also chosen to benefit from improved copper recovery. This compares to the previous recommendation of 180 µm for Alice.

For modelling purposes, the samples are separated into only two designations, Productora and Alice, based on deposit name. Earlier reports and the comminution work in this report refer to sub-areas within the Productora deposit (Central, Habanero, etc.), but these divisions are now obsolete. In addition, some of the Productora testwork was conducted on mineralisation from outside the PEA mining pit and consequently this information has been excluded from the analysis.

The flowsheet optimisation findings include:

Grinding for both Productora and Alice is to be to 125 µm for copper, which is equivalent to 150 µm for the whole production feed.

Alternative copper collectors must be available for treating specific parts of the Productora deposit. These include the collector CC4403 for production feed in the southern margins of the deposit and the collector A3894 (in combination with SMBS and diesel) for transition feeds.

Oxidised feeds with >50% acid soluble copper should not form part of flotation feed and should be directed to heap leach.

The relationships between copper, molybdenum and gold recoveries and head grades were all developed graphically to give the mathematical recovery calculations in Table 13.24.

Table 13.24 : Productora and Alice Recovery Prediction Equations

Deposit	Rec Cu	Rec Mo	Rec Au	Rec Ag	Comments
Productora & Alice – Oxide	Productora = 56% Fixed Alice = 46% Fixed	-	-	-	-
Productora – Fresh	Rec = 9.072 x CuF% + 83.66 Cap 95%	RecMo = 17.342 x ln (MoFppm) - 34.65 Cap 95%	RecAu = 145.4 x AuFppm + 38.549 Cap 80%	40% Fixed	-
Productora - Transitional	RecCu = 19.609 x CuF% + 63.443 Cap 90%			40% Fixed	Mo and Au recoveries are both unaffected by the oxidation and alteration in the transition zone
Alice – Fresh	RecCu = 0.4951 x CuF% + 91.0	RecMo = 0.882 x MoFppm + 18.52 Cap 95%	RecAu = 145.4 x AuFppm + 46.692 Cap 80%	40% Fixed	Alice samples are all fresh and have performed better than Productora Fresh in all cases.
Cortadera OP – Oxide	50% Fixed	-	-	-	-
Cortadera OP – Fresh	Rec=17.016 x LN(CuF%) + 96.378 Max = 90%, Min=18%	50% Fixed	Rec=104.74 x AuF + 29.42	27% Fixed	-
Cortadera OP – Trans	Rec=17.016 x LN(CuF%) + 86.378 Max = 80%, Min=8%	50% Fixed	Rec=104.74 x AuF + 29.42	27% Fixed	Cu Recovery reduced by 10% from Fresh
Cortadera UG	Rec=8.615 x LN(CuF%) + 96.122 Max=95% at CuF>0.88%	50% Fixed	Rec=30.368 x AuF + 51.637	38% Fixed	-

13.8 Cortadera Flotation Testwork

13.8.1 Introduction

As with the comminution program, Cortadera flotation testwork was segmented into the Open Pit accessible mineralisation and the Underground accessible mineralisation.

13.8.2 Flotation Schemes for the Open Pit and Underground Resources

Flotation testwork on the Cortadera open pit deposit (OP) commenced with the preparation of a single flowsheet development composite (OPFDC). The resulting "Cortadera OP Flowsheet" was then tested across a set of OP variability samples.

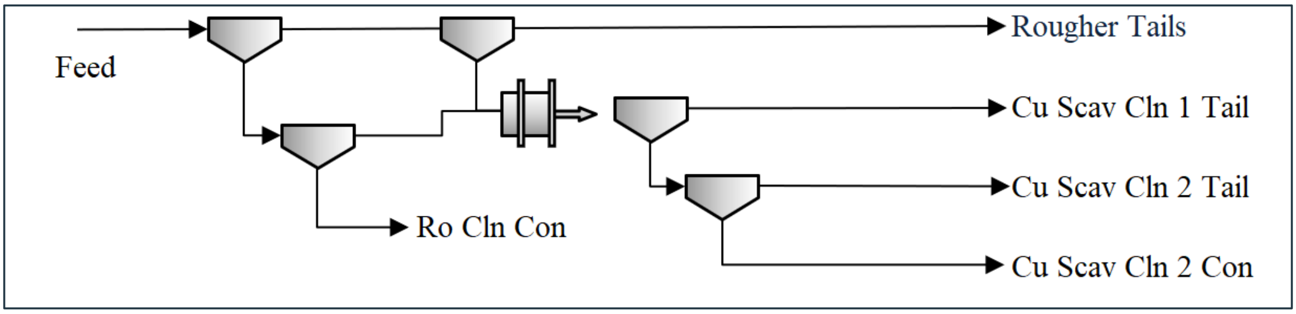
Initially, the Productora standard flotation scheme was applied to the OPFDC with reasonable, but not optimal success. The Productora flowsheet was changed to optimise for OP production feed and the standard OP flotation scheme in Table 13.25 was developed.

Table 13.25 : Cortadera OP Standardised Flotation Scheme

Operation	Conditioning time (minutes)	pH	Collector RTD2086 (g/t)	Frother MIBC (drops)	Flotation time (minutes)
Mill P_{80} 106 µm					
Cu Rgh Con 1	1		5		2
Cu Rgh Clnr Con 1	1				1
Cu Scav Con 1	1	Natural pH	3	As required	2
Cu Scav Con 2					3
Combine 2xScav Cons; Regrind to P_{80} 25µm					
Cu Scav Cln 1 Con	1		5		4
Cu Scav Cln 2 Con	1				3

The schematic OP standard flowsheet is in Figure 13.40. The final copper concentrate is the combination of the Ro Clnr Con and the Cu Scav Cln 2 Con.

Figure 13.40 : Cortadera Open Pit Standard Flowsheet



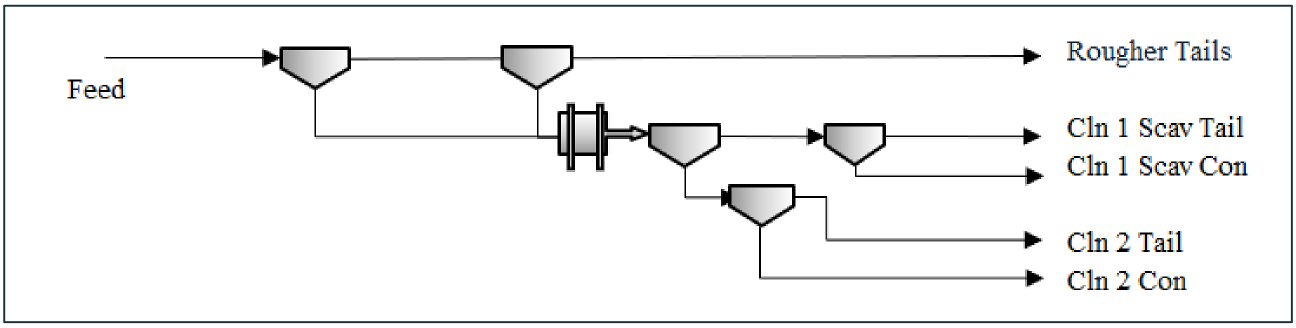
The rougher concentrate upgrades to >25% Cu with a single cleaner float and no regrind. The Ro Clnr Tail joins the scavenger concentrates for regrinding and is then cleaned twice.

The Cortadera underground (UG) flowsheet is slightly different again as shown in Table 13.26.

Table 13.26 : Cortadera Underground Standardised Flotation Scheme					
Operation	**Conditioning time (minutes)**	**pH**	**Collector RTD2086 (g/t)**	**Frother MIBC (drops)**	**Flotation time (minutes)**
Mill P_{80} 106 µm		Natural pH		As required	
Cu Rgh Con	1		5		2
Cu Scav Con 1	1		3		2
Cu Scav Con 2					3
Combine Rgh and 2xScav Cons; Regrind to P_{80} 25µm					
Cu Clnr Con 1	1		7		6
Cu Scav Cln 1 Con	1		3		4
Cu Cln 2 Con					4.5

The schematic UG standard flowsheet is in Figure 13.41. The final copper concentrate is only the Cln 2 Con.

Figure 13.41 : Cortadera UG Standard Flowsheet



The rougher concentrate does not achieve 25% Cu without regrinding, so it is combined with the scavenger concentrates. As well as the two cleaner stages the first cleaner tail is scavenged.

The two Cortadera schemes are compared with the Productora scheme in Table 13.27.

Table 13.27 : Comparison of the Three Standard Flotation Schemes			
	Productora & Alice	**Cortadera OP**	**Cortadera UG**
Flotation Feed F80 (µm)	125	106	106
Rougher/Scav Collector (g/t)	25	8	8
Cleaners Collector (g/t)	17	5	10
Total Collector (g/t)	42	13	18
Rougher conc regrind	Yes	No	Yes
Middlings only regrind	No	Yes	No
Cleaning & Cln Scav stages	4	1	2
Recleaning stages	4	1	1

Although these schemes have differences, they can all be configured with one set of flotation cells by having adequate regrind power and flexible concentrate destinations.

13.8.3 Cortadera Flotation Results

Cortadera has lower copper recoveries, lower gold recoveries and more pyrite than samples from the Productora deposit. It was found that when slightly too high copper recoveries were targeted when floating Cortadera samples, the pyrite would strongly accompany the final 3 to 5% of the copper and sometimes prevent attainment of 25% Cu in concentrate. It was also found that the pyrite co-recovery with copper could be avoided by using lower addition rates of RTD2086 collector. The selectivity of the collector against pyrite flotation is also responsible for the higher rougher grades in the Cortadera results.

LCT was conducted for Cortadera, but only on the flowsheet development sample, not the variability samples. Cortadera is more reliant upon variability testing than Productora.

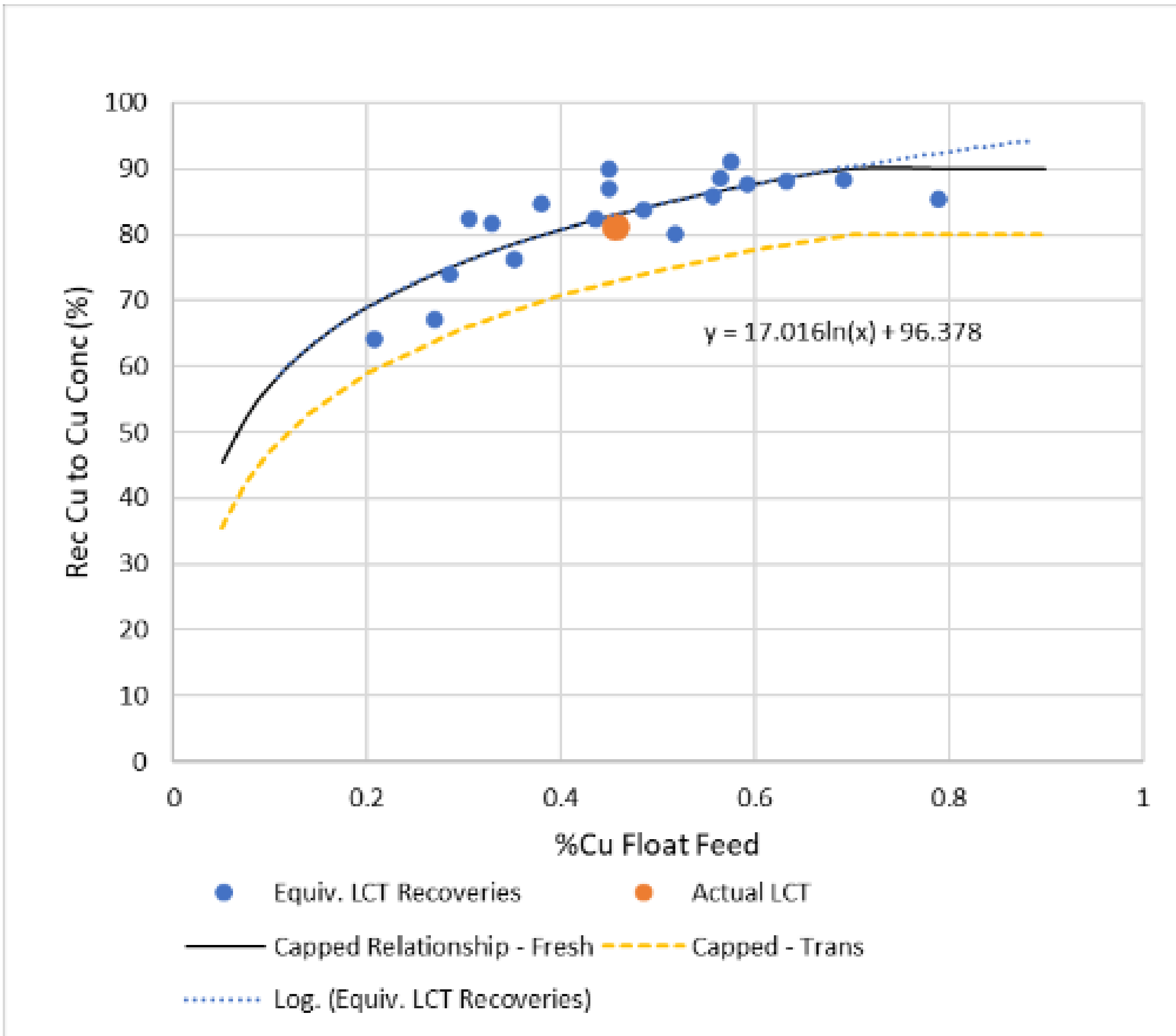
The locked cycle results and the variability results have been analysed together to arrive at models of recovery vs. head grade for the value elements. Differences in flotation response were evident between the OP and UG samples and between fresh and transition OP samples.

13.8.3.1 Copper Recovery

For the OP samples, the equivalent locked cycle test recovery (standardised to 25% Cu in concentrate) has been estimated from open circuit test results by analysis of the copper held in middlings streams. In this case (compared to Productora) the middling streams include copper that is deliberately misplaced to a pyrite concentrate extracted from the copper scavenger tails. If pyrite flotation was not being attempted, then more of this copper would have been targeted for copper scavenger recovery. One possible method for returning middlings copper from the pyrite concentrate back to the copper circuit is to regrind the pyrite rougher concentrate and selectively float the copper from that to make a medium grade concentrate. Because of this intentional misplacement of copper with pyrite cons, it has been assumed that 80% of the middlings copper would be recovered to the final copper concentrate, rather than the 50% assumed for Productora.

The plot of equivalent copper recovery is in Figure 13.42. The orange point is the LCT and it was not included in the model fitting. The LCT result is consistent with the calculated equivalent results for the open circuit tests. The relationship is capped at 90% and the minimum recovery value is 18%.

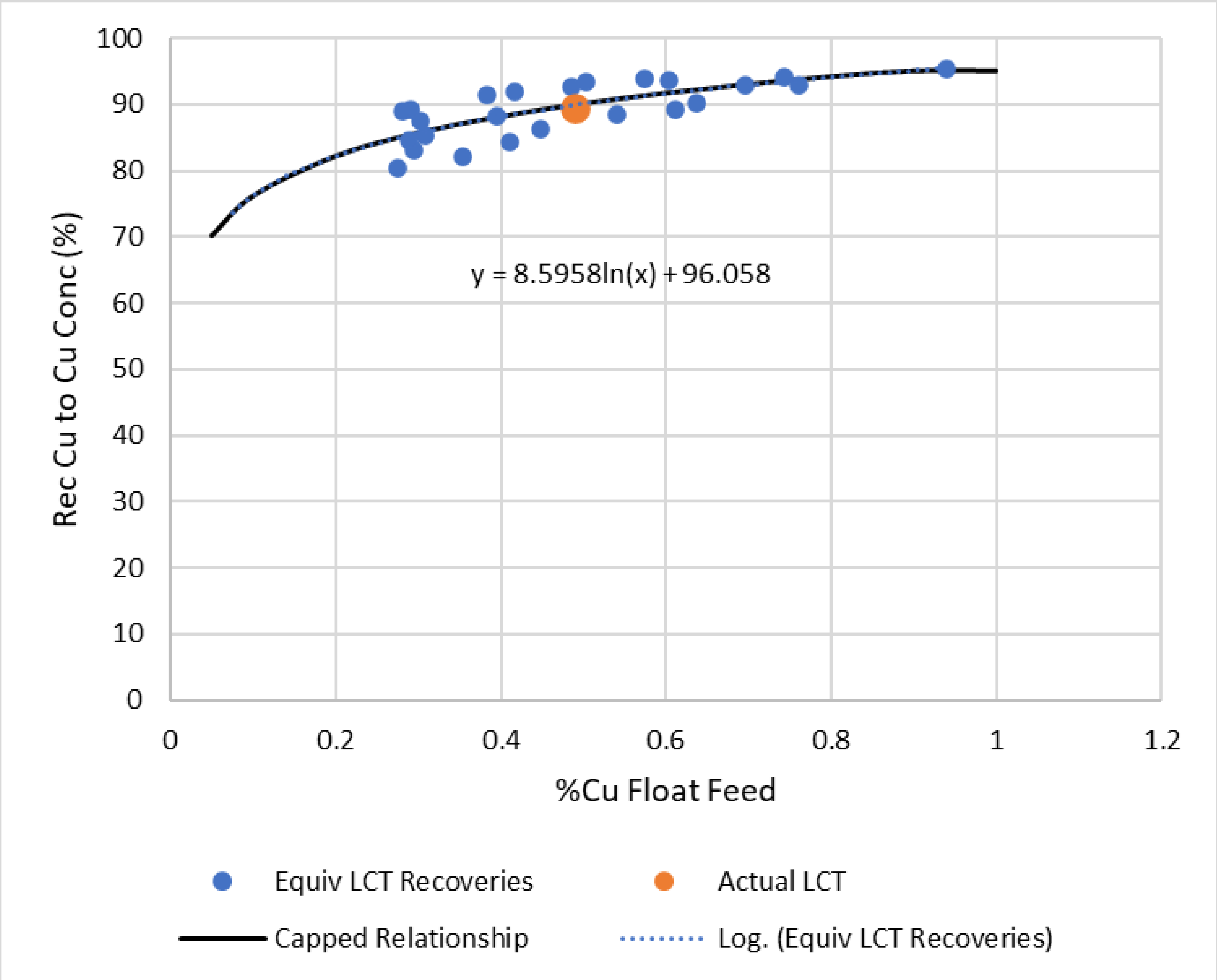
Figure 13.42 : Cortadera Open Pit Samples Copper Recovery



Two of these samples are probably transition (ASCu ~20%) but there is insufficient information to develop a separate transition relationship. In place of a fitted relationship, transition copper recovery is simply fresh copper recovery minus 10% and is shown as the yellow line above. Transition intervals would be based on transition allocation within the geological and mining databases, currently made on a boundary basis rather than determined for core intervals or for individual mining blocks. Transition recovery is capped at 80% and the lower limit is 8%.

The plot of equivalent copper recovery for Cortadera underground is in Figure 13.43. The orange point is the LCT and it was not included in the model fitting. Again, the LCT results is consistent with the model based on 80% of middlings copper reporting to the copper concentrate and only 20% reporting to the plant tailings. The relationship is capped at 95%. Note that underground production feed is too deep to be influenced by transition alteration and all samples are fresh.

Figure 13.43 : Cortadera Underground Samples Copper Recovery

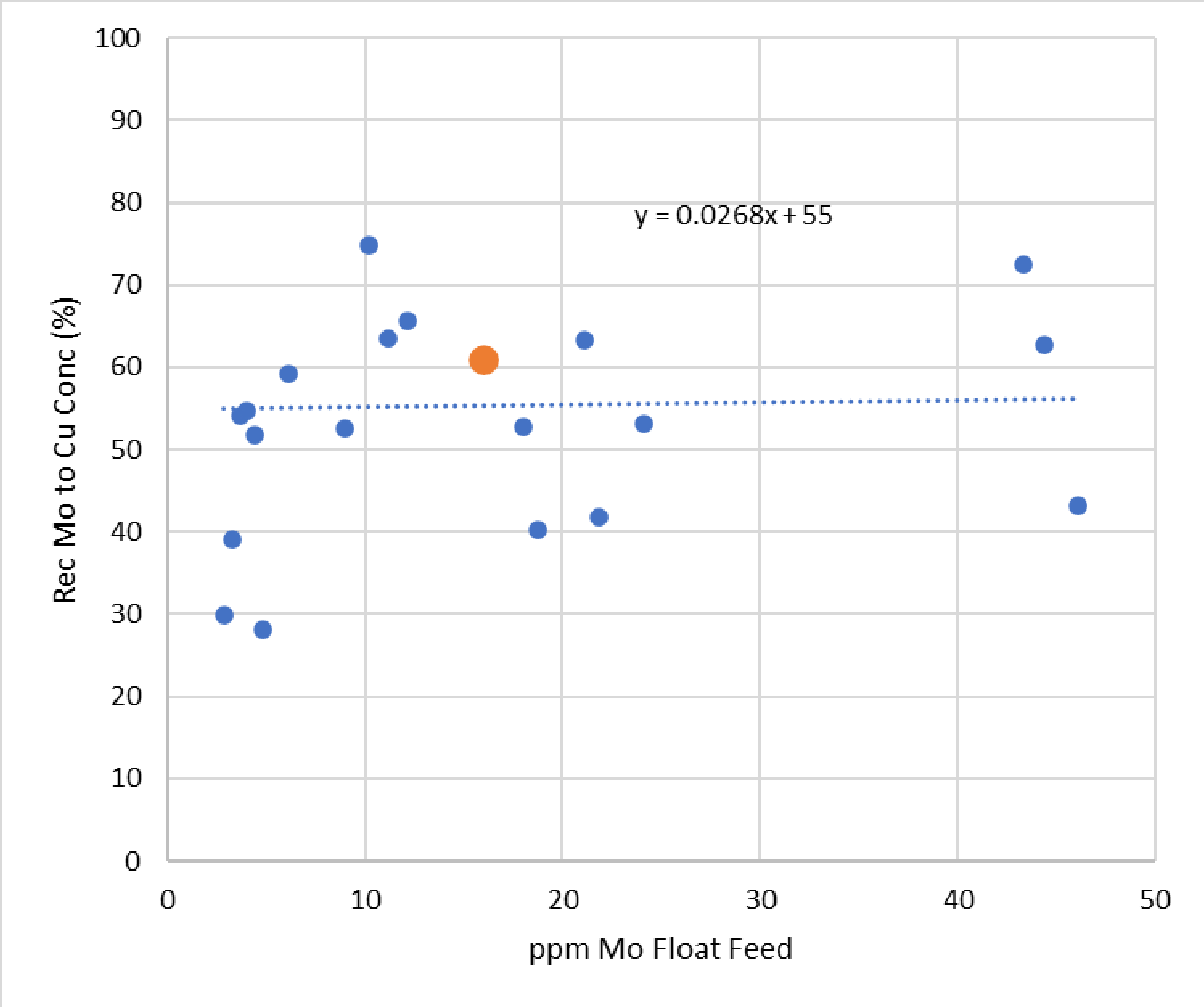


The UG production feed follows the expected trend of increasing recovery with higher feed grade. The recovery relationship gives higher values than those calculated for the OP samples.

13.8.3.2 Molybdenum Recovery

The plot of equivalent molybdenum recovery for Cortadera OP is in Figure 13.44. The orange point is the LCT. Note that UG production feed is too deep to be influenced by transition alteration and all samples are fresh.

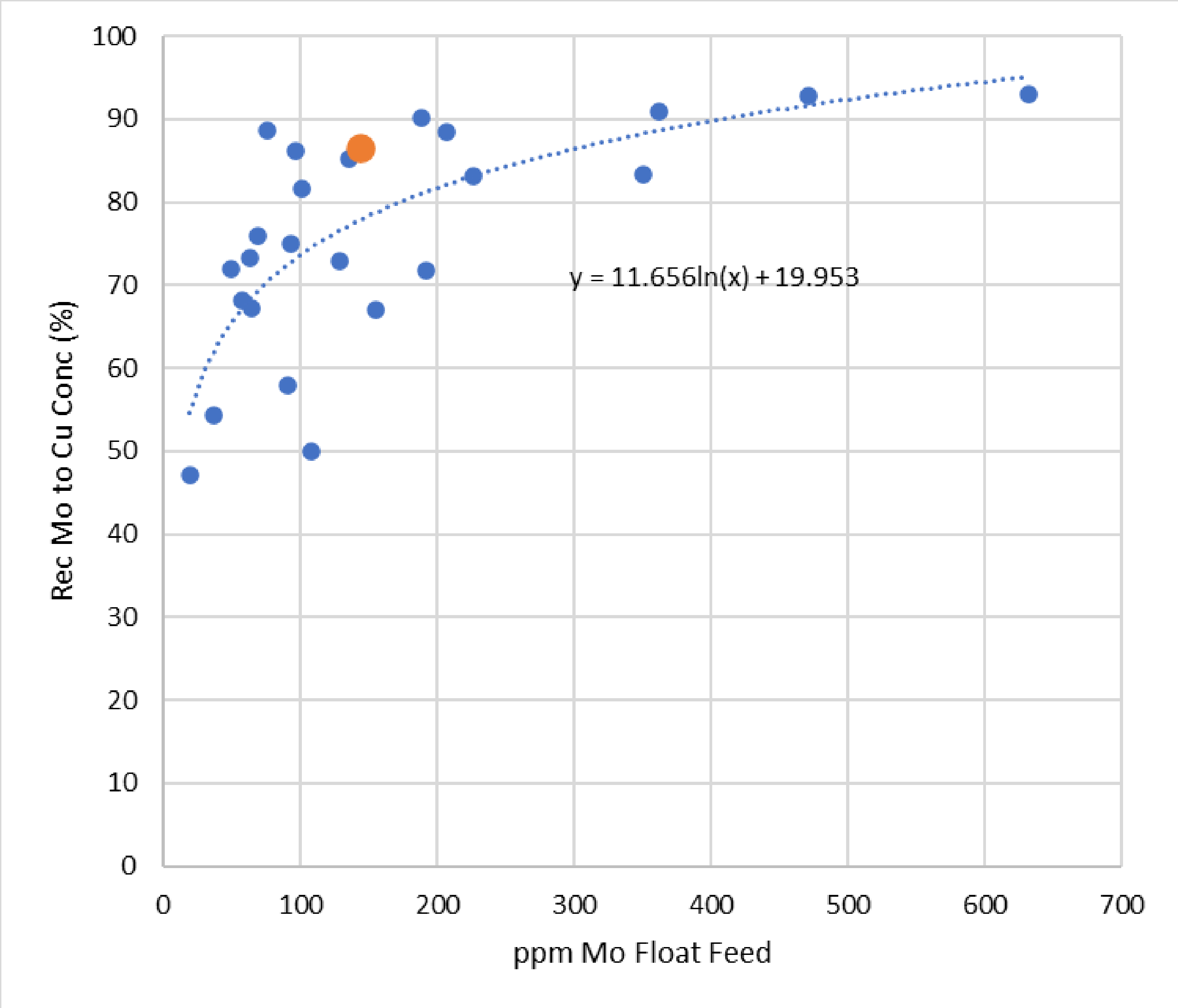
Figure 13.44 : Cortadera Open Pit Samples Molybdenum Recovery



Molybdenum recovery is highly scattered and the relationship has been manipulated to give a constant 55% recovery at all feed grades. The prediction is for molybdenum recovered to copper concentrate. To take this to molybdenum recovered to a 50% Mo concentrate it needs to be multiplied by 0.9, giving a constant 50% Mo recovery. Note that the head grades for OP samples are low and OP production feeds is only a minor source of molybdenum. The fixed 50% recovery applies to both fresh and transition production feeds.

The plot of equivalent molybdenum recovery for Cortadera underground is in Figure 13.45. The orange point is the LCT and it was not included in the model fitting. Although this relationship exists, the economics have been performed on a fixed 50% recovery of molybdenum to molybdenum concentrate, identical to the OP samples. This is a conservative assumption and Wood approves of the use of the predictive equation, reduced by 10%, for prediction of molybdenum recovery to molybdenum concentrate.

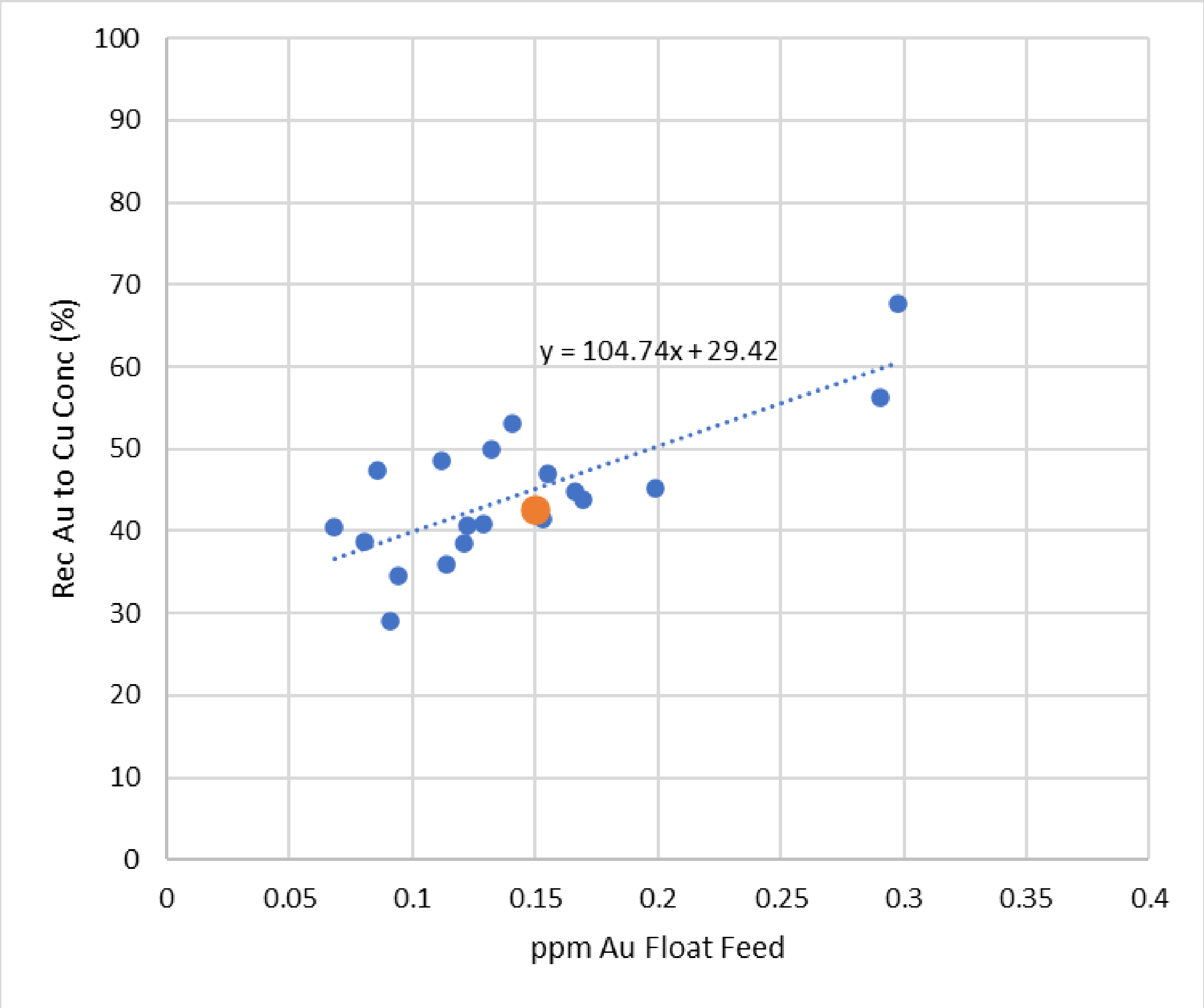
Figure 13.45 : Cortadera Underground Samples Molybdenum Recovery



13.8.3.3 Gold Recovery

The plot of equivalent gold recovery for Cortadera OP is in Figure 13.46. The orange point is the LCT and it was not included in the model fitting.

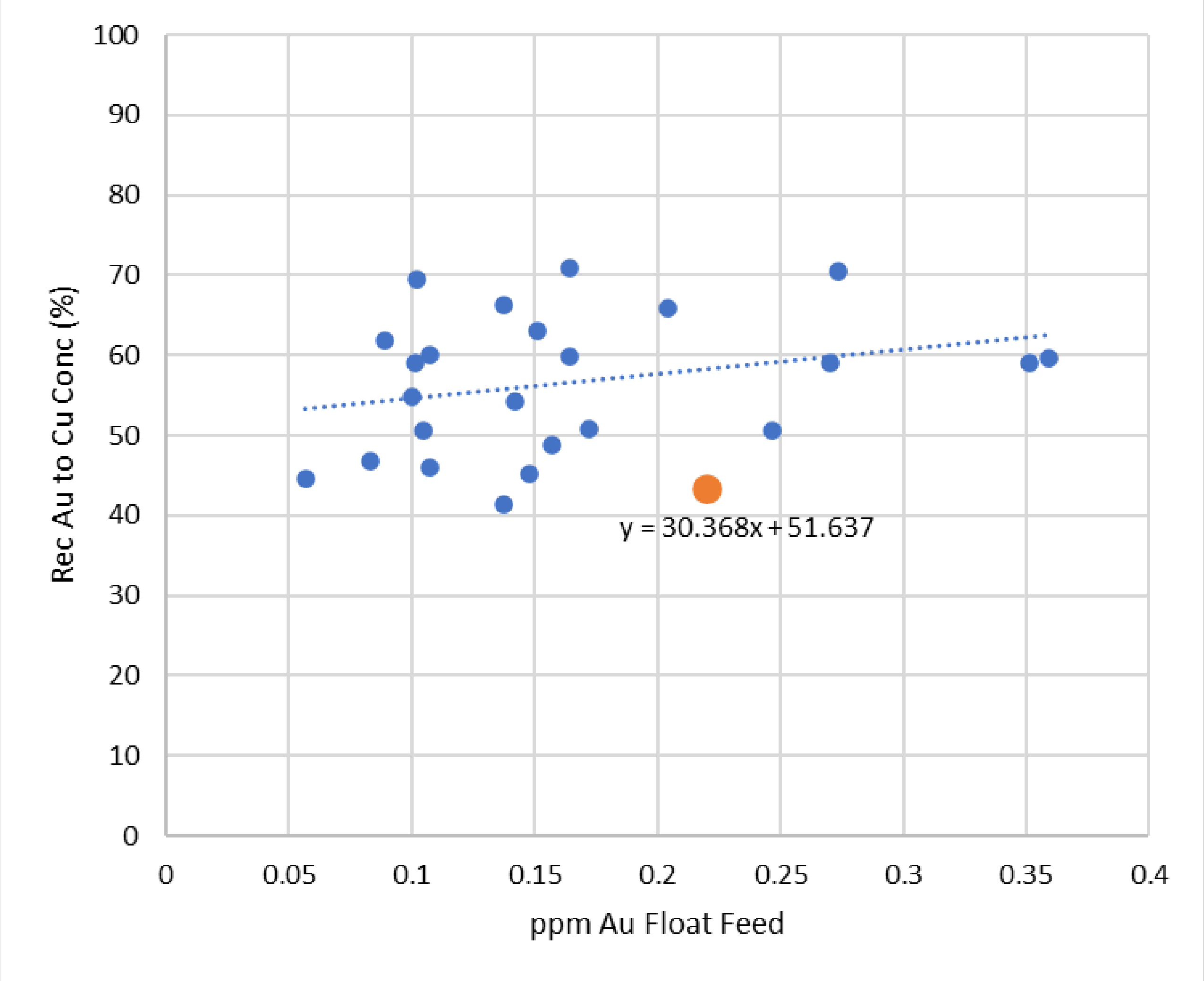
Figure 13.46 : Cortadera Open Pit Samples Gold Recovery



A positive relationship does exist between gold in feed and recovery of gold to copper concentrates. Note that Au would be sold in the copper concentrate so no further recovery discounting is needed.

The plot of equivalent gold recovery for Cortadera UG is in Figure 13.47. The orange point is the LCT result and it was not included in the model fitting. The relationship is highly scattered.

Figure 13.47 : Cortadera Underground Samples Gold Recovery

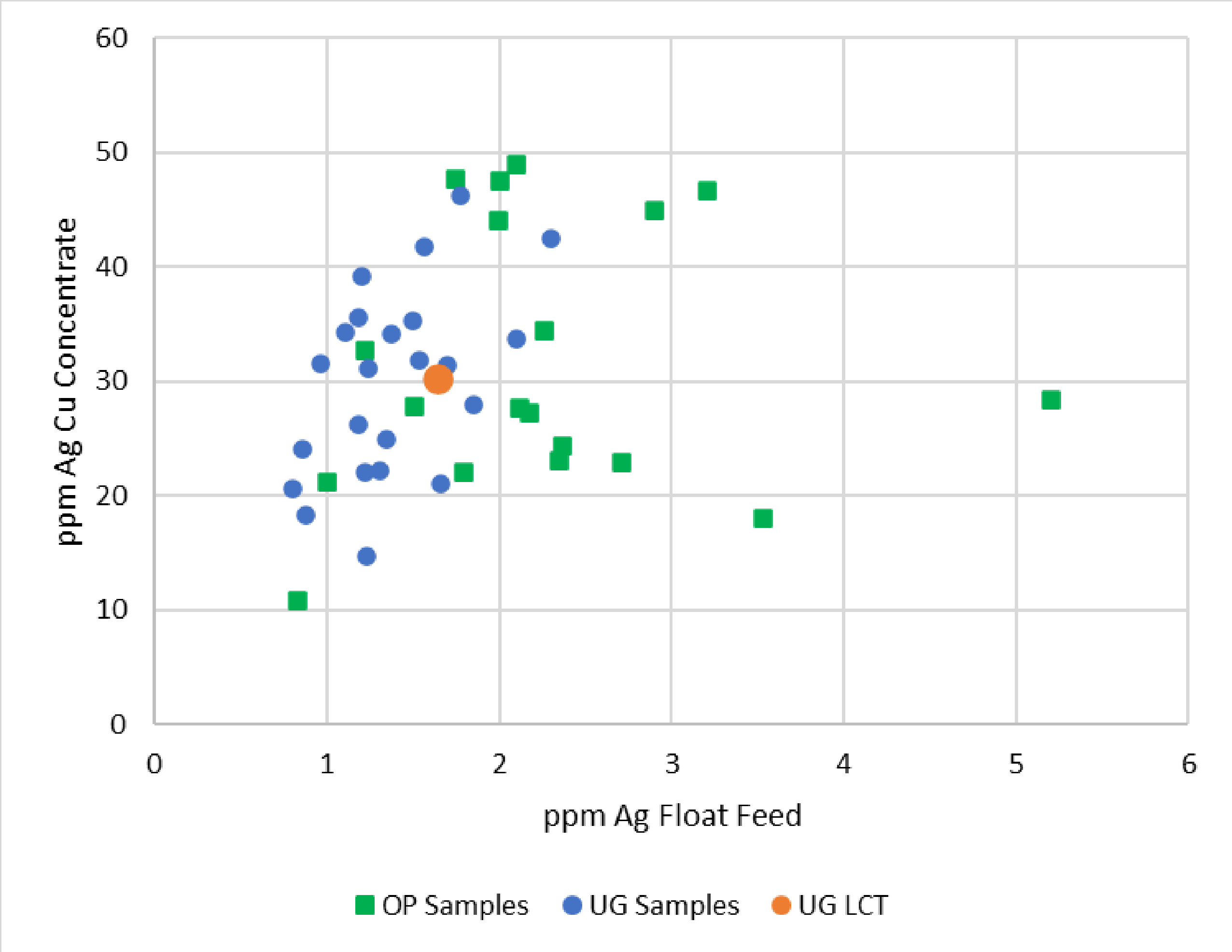


The recovery of gold from UG production feed could be set at a constant 58% given the scatter in the results. Interestingly, the LCT result was well below the trend line of the data, suggesting that there may be sone risk for gold loss during the regrind and cleaning flotation stages, which are more influential when conducting a LCT.

13.8.3.4 Silver Recovery

Silver is present at about 1 to 4 ppm in the Cortadera OP samples and at only 1 to 2 ppm in the UG samples. Silver is recovered in flotation and becomes payable at 30 g/t in copper concentrate. The grades of silver in the copper concentrates for all Cortadera testwork are plotted in Figure 13.48. The LCT result (UG only – awaiting silver assays on OP LCT) is 30 g/t and the average of all grades is 30.5 g/t.

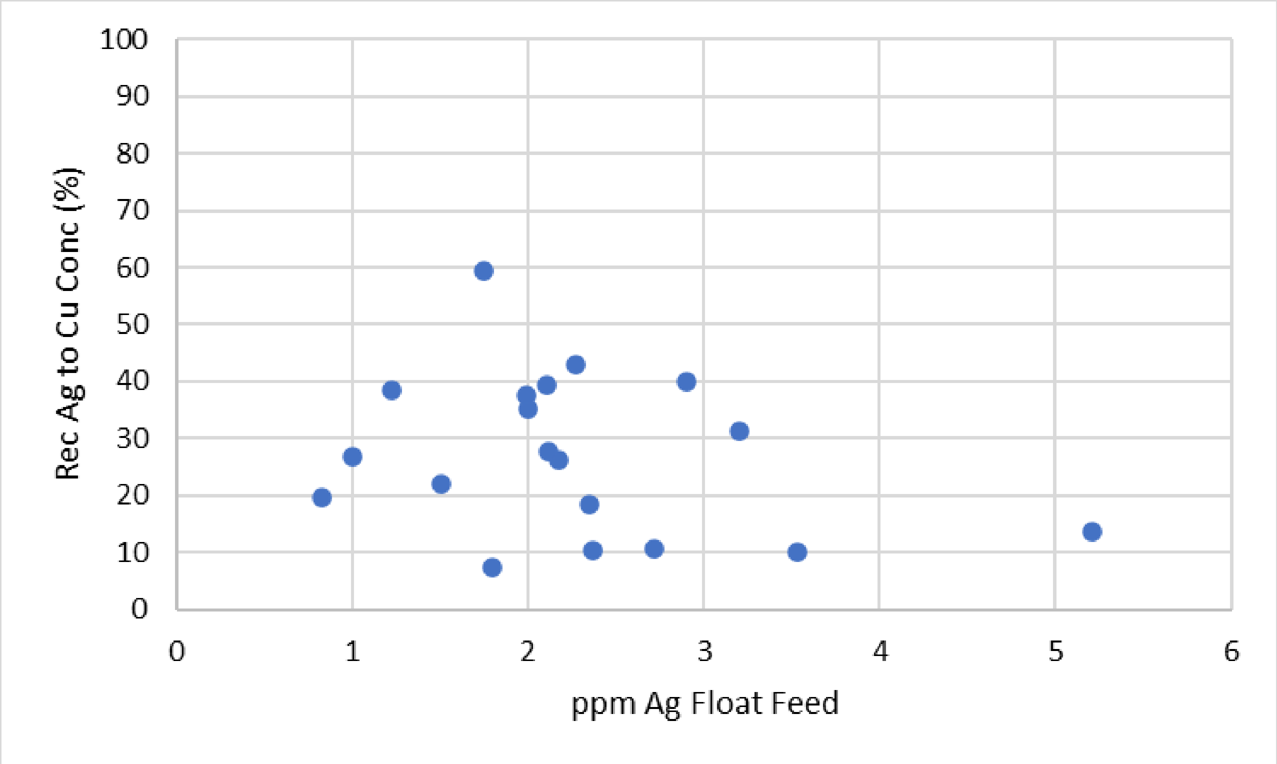
Figure 13.48 : Cortadera Silver Grades in Copper Concentrates



Although many samples are below 30 g/t, most are above 20 g/t and Wood believes that payability for silver must be considered in the economic calculations for Cortadera concentrates. Note that Productora was analysed sparsely for silver, both in the geological and metallurgical work. The few result-based indications that are available only achieve 20 g/t in Productora concentrates. However, it is likely that Productora would behave in a similar fashion to Cortadera and silver payability remains likely following an improved silver focus in future work.

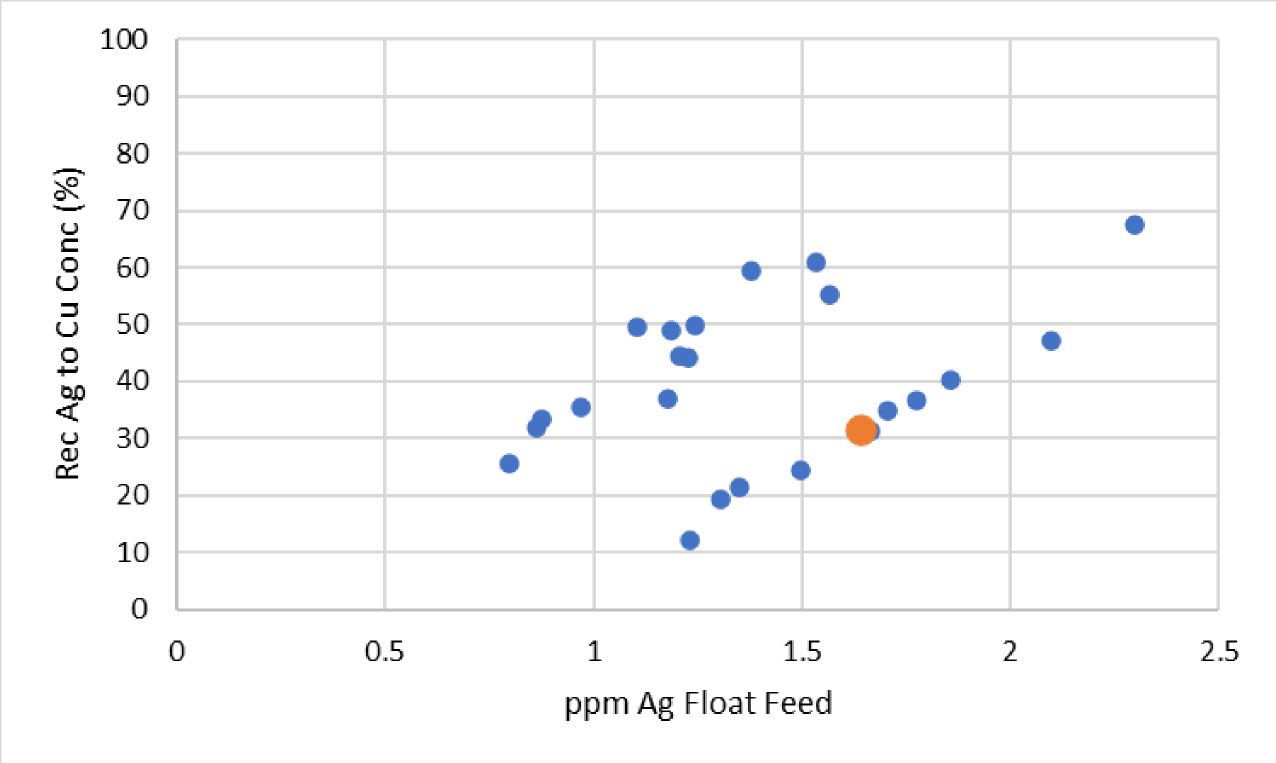
Silver recovery for the OP samples is shown in Figure 13.49. These results are extremely poorly defined because the sensitivity of measurement of silver in flotation tails is insufficient. Most flotation tailings results are at the limit of detection of the silver assay method, which has been 1 ppm. This is problematic because the head assay and calculated head assays are also in the vicinity of 1 ppm. When both feed assay and tailings assays are below detection limits the reported recoveries would be meaningless.

Figure 13.49 : Cortadera Open Pit Silver Recoveries in Copper Concentrates – Poor Data Definition



The situation is even worse for the UG recovery calculation as seen in Figure 13.50. The recoveries are split into two groups because the tailings grades are all either 1.0 or 0.5 ppm.

Figure 13.50 : Cortadera Underground Silver Recoveries in Cu Concentrates – Poor Data Definition



A conservative assumption for silver recovery, in the absence of reliable assays, would be to assume 27% recovery (fixed) for OP and 38% recovery (fixed) for UG samples (simple data averages from the above graphs). However, it is highly likely that silver recovery would be determined as higher than 50% once a sensitive silver assay method is implemented.

13.8.4 Molybdenum Concentrate Flotation Testwork Program

13.8.4.1 Introduction

A testwork program was conducted on Productora samples during the 2016 study to determine if a saleable molybdenum concentrate of 50% Mo could be produced. Molybdenum recoveries for each flotation composite were also determined and a relationship determined. Molybdenum flotation was successful, but not fully optimised in either the copper circuit or the molybdenum circuit.

In all the samples tested mineralogically, molybdenite (MoS_2) had little to no association with the dominant sulphides, pyrite and chalcopyrite.

13.8.4.2 Initial Molybdenum Recovery

Molybdenum floats readily with the chalcopyrite and the majority follows the chalcopyrite into the recleaner copper concentrate. This is only the first molybdenum recovery stage, as the intention is to produce a saleable molybdenum concentrate rather than sell molybdenum as a copper concentrate by-product.

In the previous section, the batch and locked cycle flotation tests conducted and discussed on the Productora and Alice samples were optimised to achieve the best copper recovery at a 25% Cu grade. The associated molybdenum recovery in this testwork is shown in Figure 13.38 and predictive curves were fitted to the data. RTD2086 collector (xanthate ester) is generally good for molybdenite recovery, but under some circumstances other reagents have proven beneficial. For some tests, diesel was added to the primary mill to improve molybdenum recovery. For flotation using the A3894/SMBS scheme, the addition of diesel gave a significant increase in molybdenum recovery.

The next steps on the path to making saleable molybdenum concentrates include regrinding and suppression of chalcopyrite flotation while recovering molybdenite to cleaner and recleaner molybdenum concentrate.

It is never possible to recover all the molybdenum in a copper concentrate to a molybdenum concentrate, as some would always remain with the final copper concentrate. The aim of the molybdenum concentration work was to maximise the recovery to the molybdenum concentrate while achieving 50% Mo in the concentrate. Ultimately the molybdenum recovery across molybdenum upgrading has been determined as described below.

The overall molybdenum recovery to product from plant feed is a combination of the recovery in the copper flotation stage combined with the recovery in the molybdenum upgrade stage.

13.8.4.3 Molybdenum Cleaner Testwork

Flotation testwork was conducted to determine whether molybdenum could be easily separated from the bulk copper/molybdenum concentrate and if a greater than 50% Mo concentrate could be achieved.

The testwork was conducted in two phases. The first phase used the FD-1 composite sample with only a few tests being completed. These tests gave an initially positive response that a good molybdenum concentrate could be achieved (32% Mo after one stage of cleaning). However, with the low molybdenum head grade (160 ppm) and limited sample, molybdenum testwork was not able to be further optimised.

For the second phase of the testwork, a composite was produced from PRP0812 (RC chips, preparation as explained previously) with a high molybdenum head grade (0.056% Mo, 560 ppm) to produce higher mass pulls through the cleaners.

To produce sufficient copper concentrate for the molybdenum testwork, bulk copper cleaner floats on 350 kg of production feed were conducted. The concentrates (about 7 kg) were combined, blended and split into 1 kg charges. These charges were then tested individually to achieve copper and molybdenum separation.

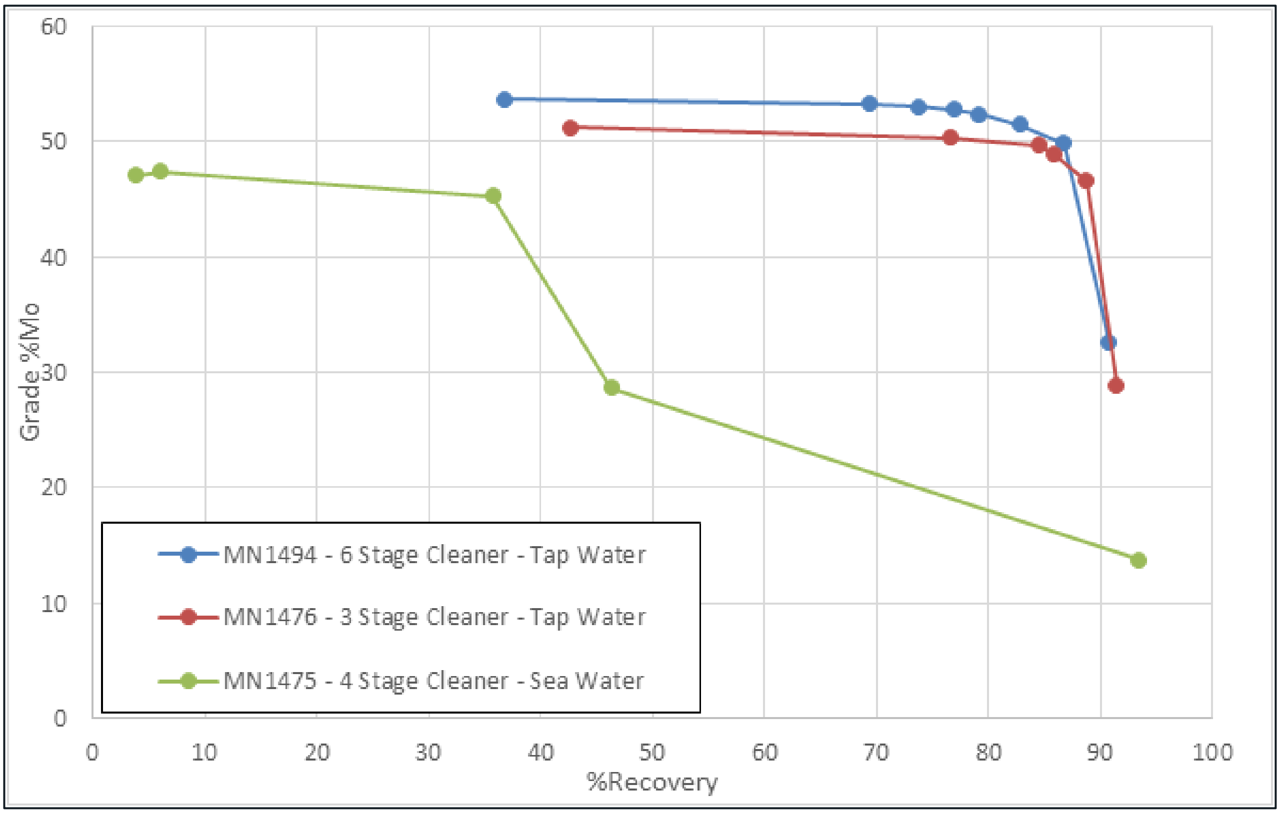
All tests conducted used NaHS for copper depression and kerosene added to aid molybdenum recovery. The NaHS was used to target a slurry potential. However, the slurry tended to buffer and excess NaHS was required. This resulted in depression of both copper and molybdenum.

The initial tests were all conducted in seawater. The froth during these floats was very poor and quite unstable. Final tests switched to using Perth tap water and a significant change was seen in the flotation performance. Froth was heavily loaded with molybdenite and additional cleaners were able to be performed without depression of the molybdenum.

With the use of tap water, a +50% molybdenum concentrate was easily achieved with little recovery loss, even in open circuit. This can be further seen in Figure 13.51 and the final froth is shown in Figure 13.52. This work showed that the use of seawater is not recommended in the molybdenum circuit. A thickening/wash stage using desalinated (RO) water is required prior to the copper/molybdenum separation and then the remainder of the cleaning separations should be conducted in RO water. Further testwork is required to determine the total dissolved salts (TDS) level for RO water that would still give a good molybdenum separation and cleaning. This washing stage should be designed such that it would leave the copper concentrate with a chloride level of <500 ppm.

Testwork confirmed that air can be used in the molybdenum circuit for flotation rather than nitrogen.

Figure 13.51 : Molybdenum Cleaner Flotation – Seawater vs. Tap Water

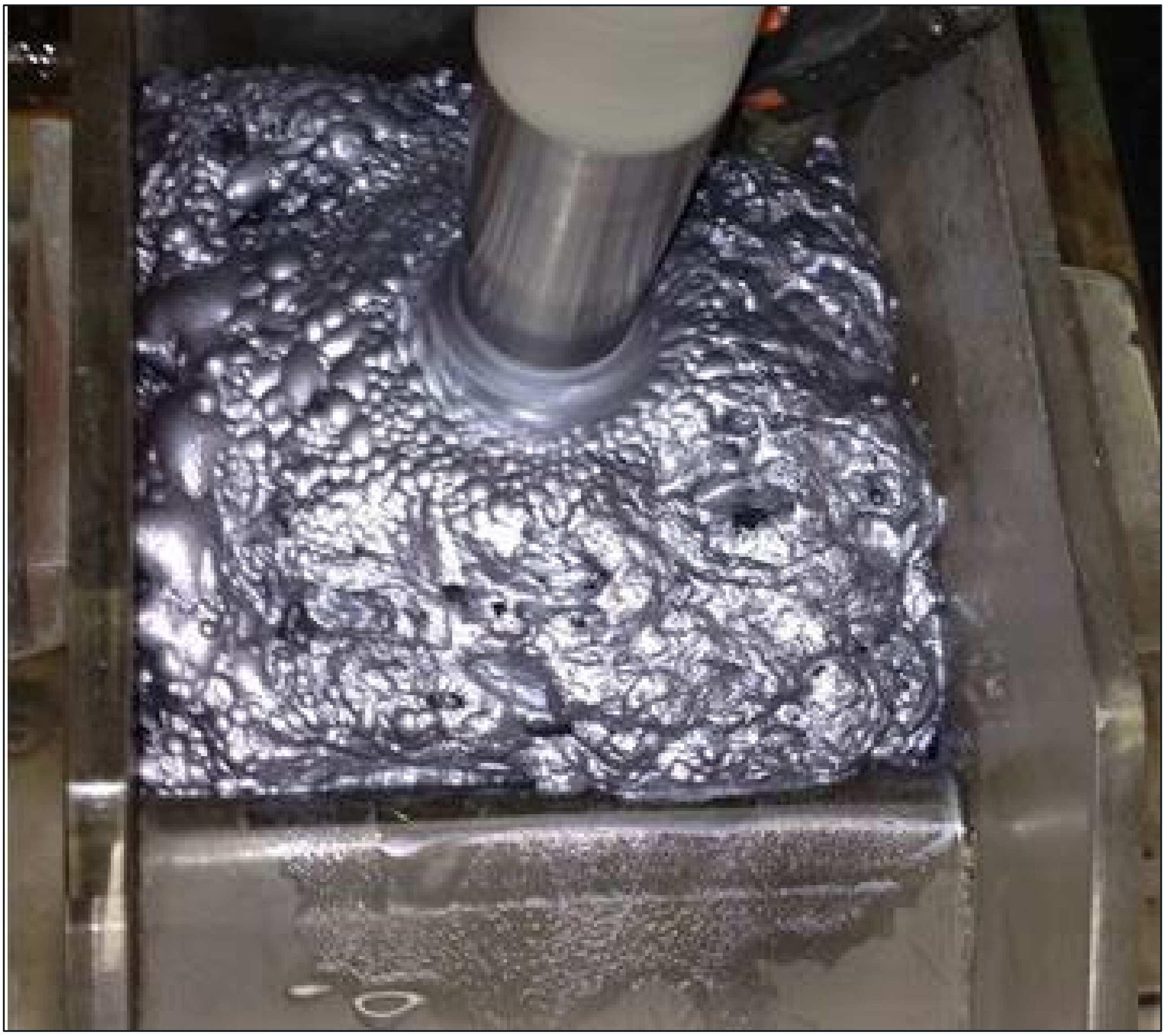


Further tests were conducted with up to six stages of molybdenum cleaning. In the bulk cleaner test (355 kg copper circuit feed), MN1494, molybdenum concentrates with up to 53% Mo were produced. At 50% Mo in concentrate the open circuit recovery of molybdenum from the copper concentrate in the bulk test was 86.7%. This suggests that in closed circuit testing or operations very high recoveries (>90%) of saleable molybdenum concentrate should be readily achieved.

To predict the molybdenum recovered from plant feed to the molybdenum concentrate, the molybdenum recovery to copper concentrate predicted by equations in Figure 13.38 is reduced by 10%.

The main gangue diluting the concentrate was mica (muscovite). Mica is easily entrained and could even be actively floating in the molybdenum reagent environment. The use of column cells and froth washing on the final cleaners is recommended to help reduce gangue entrainment.

Figure 13.52 : Molybdenum Cleaner – Tap Water Test



13.9 Cortadera Open Pit Mineralogical Summary

In LCT 82 to 85% of the copper in Cortadera OP production feed is recovered to a 27.5% Cu copper concentrate. The misplaced copper is mostly in the rougher tails (up to 10%) and with the pyrite concentrate (8%). The copper in rougher tails is mostly unrecoverable locked chalcopyrite. The copper in the pyrite concentrate is about 50% recoverable and is a mix of chalcopyrite and chalcocite. Some of this can be recovered ahead of pyrite flotation by using a stronger copper collector in the copper scavenger stage. This could be mixed with the xanthate ester that is doing an excellent job of selectively recovering chalcopyrite.

A targeted improvement program for copper flotation from the Cortadera OP production feed has the potential to increase copper recovery to copper concentrate by between 2 and 6%.

About 4% of the OP copper concentrate is pyrite contaminant, but this contamination level still allows a grade higher than 27% Cu to be produced. Should the pyrite contamination in copper concentrate reach levels where it is not possible to achieve 25% Cu, then it would become necessary to reduce or eliminate the directly cleaned (without regrinding) copper rougher concentrate step from the flowsheet. A total rougher concentrate regrind is identical to the approach used for UG production feed. Therefore, it is essential to install sufficient copper concentrate regrind power to treat all the rougher and scavenger concentrate likely to be produced from OP production feed but have the flexibility to use (or not use) the power as appropriate.

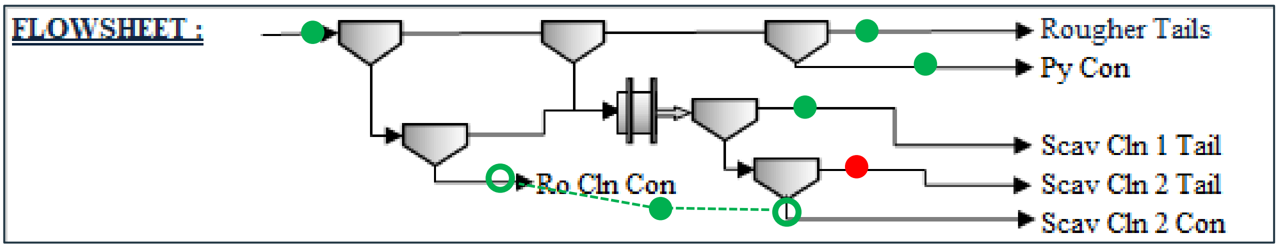
Chalcopyrite, molybdenite and pyrite in middlings streams are well liberated and mostly amenable to recovery to the correct concentrate, given the appropriate flotation conditions.

The low copper and sulphide grades of the Rougher Tailings streams are presenting accounting issues at the accuracy level of the testwork reported here. If this problem persists, then recovery predictions (and the resulting revenue stream calculations) would continue to carry unnecessary uncertainty. If the same analysis techniques are used in operations, this would make metallurgical accounting unreliable. It is recommended that more sensitive tailings analysis methods be investigated and adopted for the ongoing test program.

13.10 Open Pit Flowsheet

The Open Pit Master Composite (MC) flowsheet in Figure 13.53 generated the QEMScan samples for the streams marked with green dots. The copper concentrate sample tested by QEMScan is the combination of the Rougher Cleaner Concentrate with the re-ground Scavenger Cleaner 2 Concentrate, as shown. The Scav Clnr 2 Tail (red) was not submitted for QEMScan, thereby excluding 8.3% of the copper in the feed from QEMScan analysis. Almost all the copper in this stream would be recovered to the final copper concentrate and its form is of lesser interest compared to the other streams.

Figure 13.53 : Cortadera Open Pit Production Feed Flotation Test Flowsheet, Test AM099-80



The key investigation areas for the OP MC sample test are to understand the nature of:

- Pyrite in copper concentrate
- Molybdenum in copper concentrate

- Copper in pyrite concentrate
- Copper in rougher tails
- Copper, pyrite and molybdenum in copper Scavenger Cleaner Tails (SCT).

13.11 Mineralogical Analysis

13.11.1 Introduction

Mineralogical examination was conducted at ALS Metallurgy (Perth) on the feed samples used for flotation testwork. Mineralogy included x-ray diffraction (XRD) and QEMScan (Quantitative Evaluation of Minerals by Scanning Electron Microscopy). Samples examined included FD-1 to FD-14 (excluding FD-7), V-1 to V-6, V-11 to V-14 and V-18. All samples were either composites from various metallurgical drill holes (FD-1 to FD-4) or composites from sections of a specific metallurgical drill hole (FD-5, V-1 and V-2) or specific sections of a specific metallurgical drill hole (FD-7 to FD-14 and V-3 to V-6 and V-11 to V-15 and V-18).

Mineralogical examination on FD-1, FD-2, FD-3 and FD-4 composite samples (previously referred to as Composite 1, 2, 3 and 4) was conducted as part of an earlier testwork program. However, a summary of results from these composites has been included here.

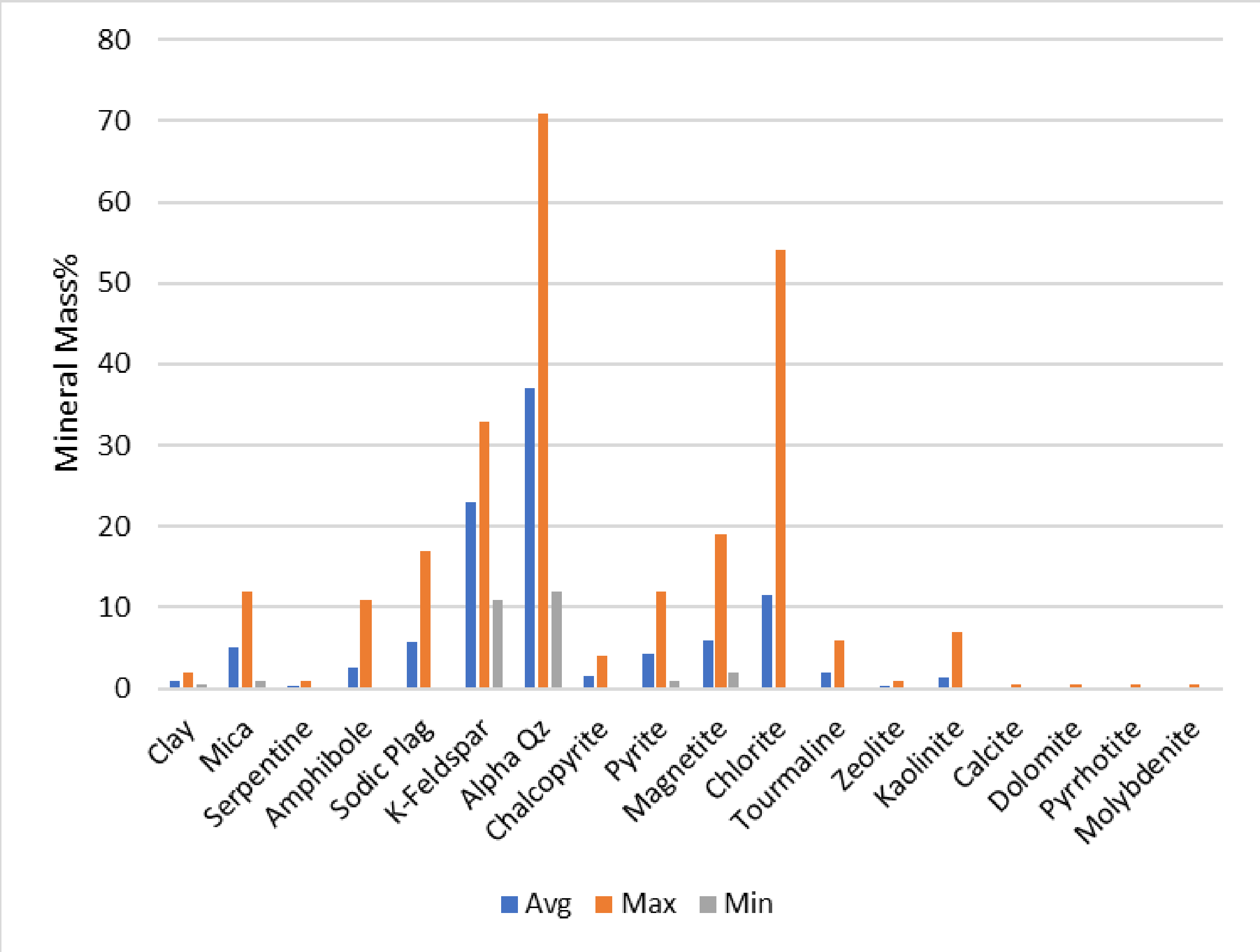
13.11.2 Mineralogy by X-ray Diffraction (XRD)

XRD examination was conducted on each of the Flotation Development samples (FD-1 to FD-14, excluding FD-7), the initial Variability samples (V-1 to V-6), later MET023 variability samples (V11 and V12, previously Habanero) and the three Alice samples (V-13, V-14 and V-18).

XRD relies on the diffraction (deviation) of x-rays as they pass through a sample. Each mineral (provided it is not amorphous) has its own distinct diffraction pattern based on its crystal structure. Amorphous material generally contributes to a continuum of diffracted x-rays which make the method semi-quantitative at best.

The average, maximum and minimum compositions of flotation test samples from the two deposits (Productora and Alice) are shown in Figure 13.54 and Figure 13.55.

Figure 13.54 : XRD Analysis of 21 Productora Flotation Feed Samples



Quartz is the dominant mineral in Productora followed by potassium feldspar and chlorite. 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.

As the span between Max and Min values demonstrate, the mineral contents are variable across the samples.

The same mineral list is used to graph the Alice results in Figure 13.55.

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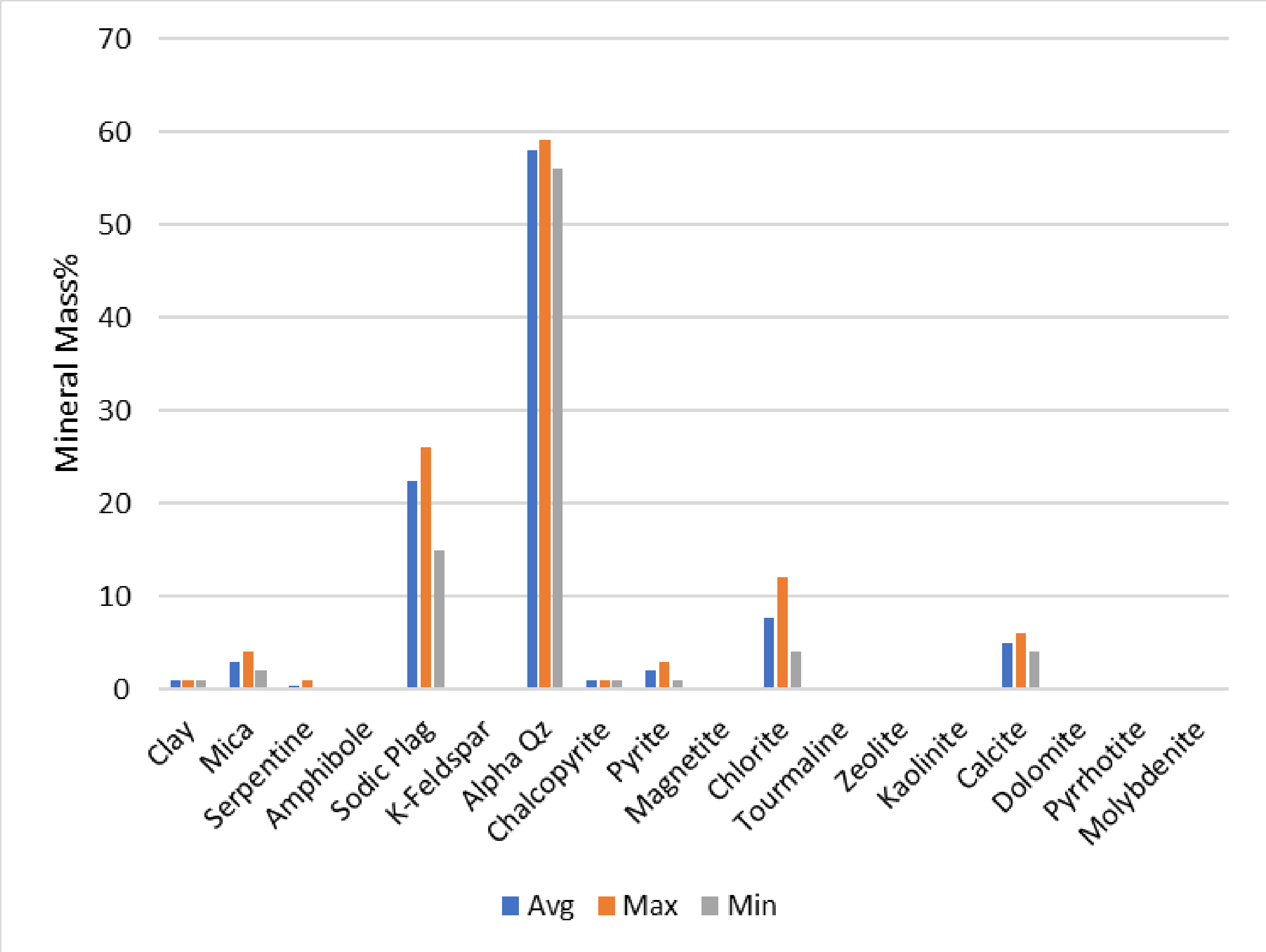
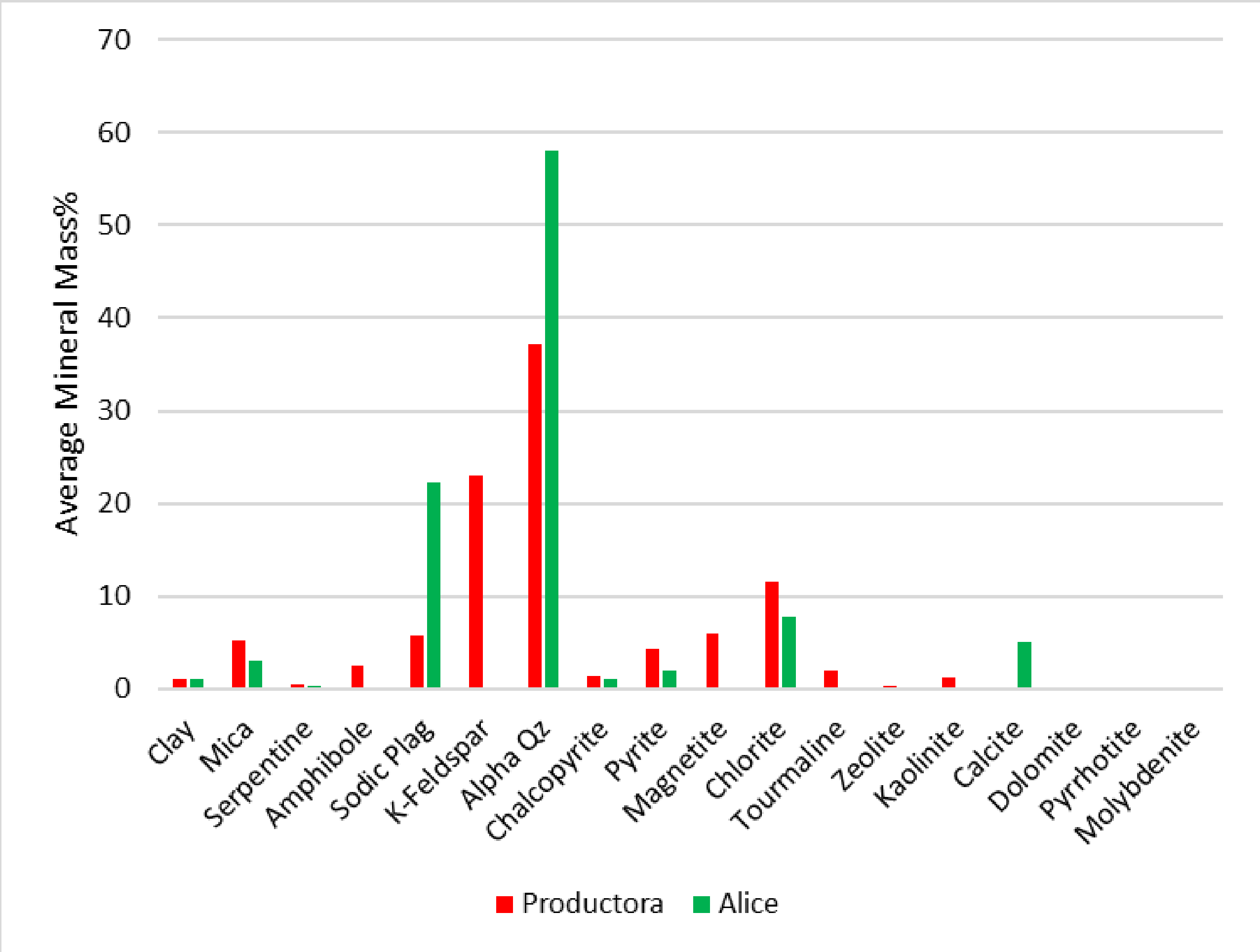
Figure 13.55 : XRD Analysis of Three Alice Flotation Feed Samples



Alice is geologically different to Productora and is dominated by quartz and sodium feldspar (sodic plagioclase). There is also calcite in Alice. Again, the only two sulphides are chalcopyrite and pyrite. The span of compositions is narrow, but there are only three samples.

Productora and Alice average XRD results are compared in Figure 13.56.

Figure 13.56 : Comparison of Average XRD Analyses for Productora and Alice



Noticeable differences are the absence of K-feldspar in Alice, higher pyrite levels in Productora, absence of calcite in Productora and a lack of amphibole, tourmaline, and kaolinite in Alice. The presence of clay and mica in both samples is likely to lead to entrainment of these platy minerals in flotation.

13.11.3 Mineralogy by QEMScan

QEMScan is a more precise method for determining mineralogical composition than XRD. Its main limitation is that it is a 2-dimensional analysis of polished particle cross sections, rather than a 3D analysis of a granular material (XRD). The main benefits are that QEMScan always provides a mineral allocation when measuring and QEMScan can provide information about each particle’s composition.

QEMScan focuses an electron microscope beam to a user-determined pixel size (as small as one square micron) to determine the composition of the mineral illuminated by the beam. The electron microscope gathers information from that pixel and then moves to the next pixel to repeat the analysis. The reflected electron brightness (back-scattered electrons, BSE) of the pixel is proportional to the average atomic number of the mineral and x-rays are backscattered from the pixel and detected by specialised equipment (energy dispersive, EDS detectors) to gather elemental information about the mineral at the pixel. Brightness and a rapidly collected x-ray signature (taken over milliseconds) are used to assign the pixel to a mineral already in the

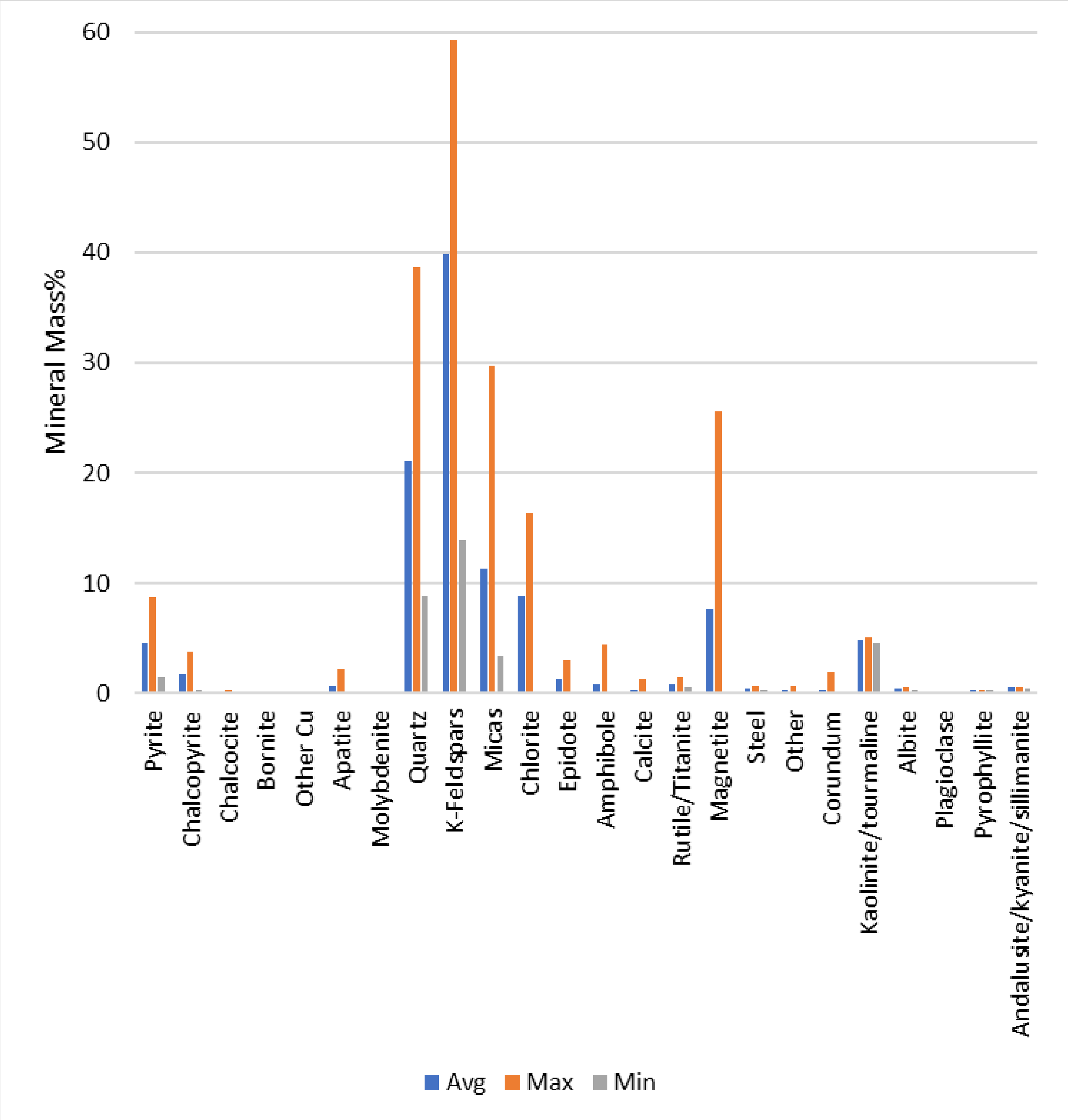
QEMScan database. It is important to note that there is no actual chemical analysis occurring. Chemical composition of each mineral in the QEMScan database is fixed, but can be changed based on actual mineral composition measurement using techniques such as laser ablation.

The data gathered by QEMScan is then arranged into mineral phases and mineral phases are arranged into particles. It is then possible to analyse properties of the particles in the sample, such as the liberation of the mineral from all the other minerals, or the sizes of the mineral phases and the particles. It is also possible to generate informative particle pictures showing the mineral grain texture, which can be critical for choosing mineral separation pathways.

The simplest output of QEMScan is the mineral composition of the sample, and these values have been averaged for the Productora and Alice test samples. All the same samples analysed by XRD were separately analysed by QEMScan.

The average composition (together with max and min values) of the 21 Productora samples are in Figure 13.57.

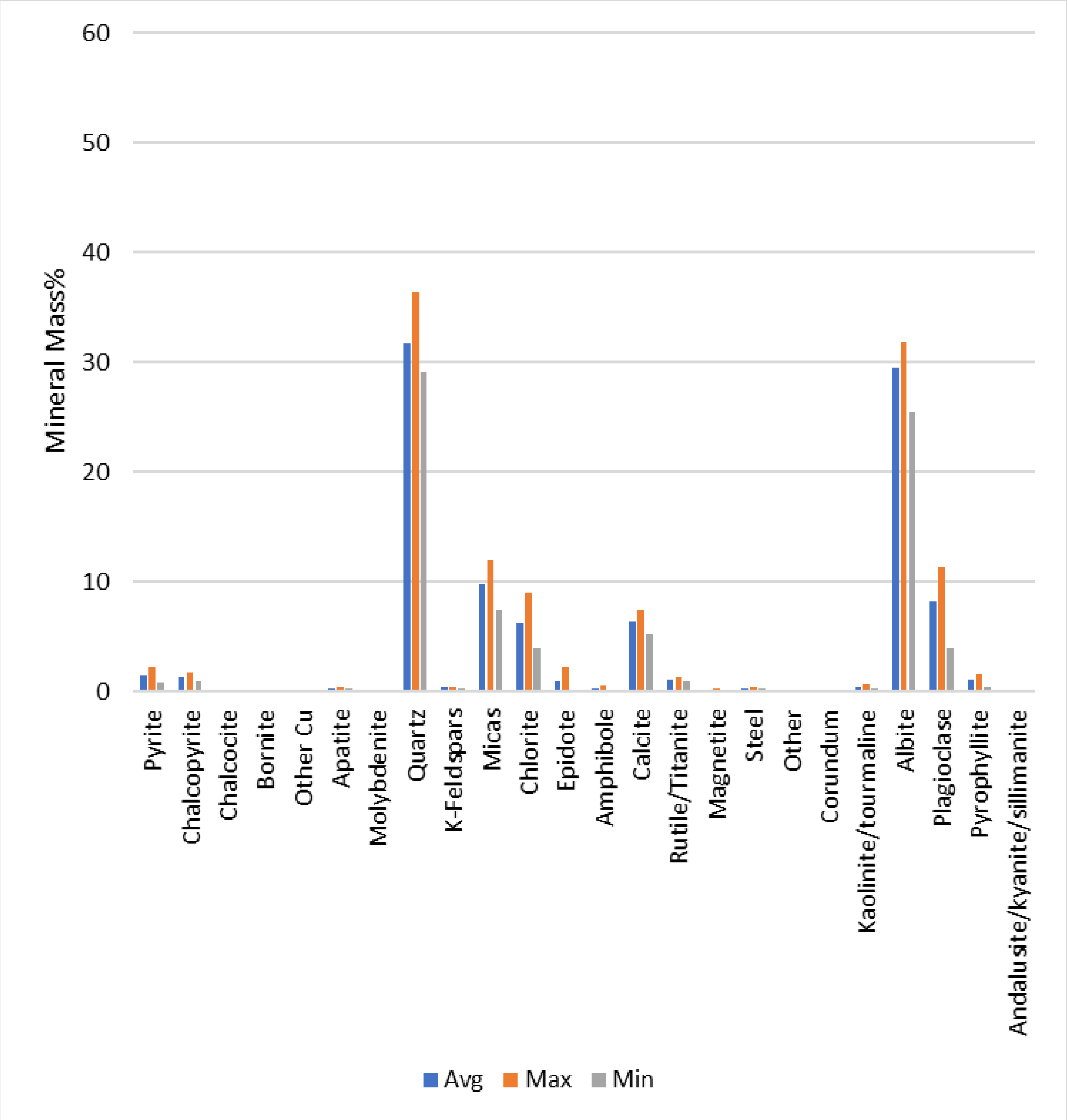
Figure 13.57 : QEMScan Analysis of 21 Productora Flotation Feed Samples



Unlike the XRD result, quartz is not dominant by this measure and the K-feldspar is dominant. Quartz is second followed by mica, chlorite, magnetite and kaolinite. Pyrite and chalcopyrite are again the dominant sulphides, but there is also some detection of minor quantities of other copper sulphide minerals. Again, the compositions span wide ranges of values.

The average composition (together with max and min values) of the three Alice samples are in Figure 13.58. The minerals are in the same order as the previous graph.

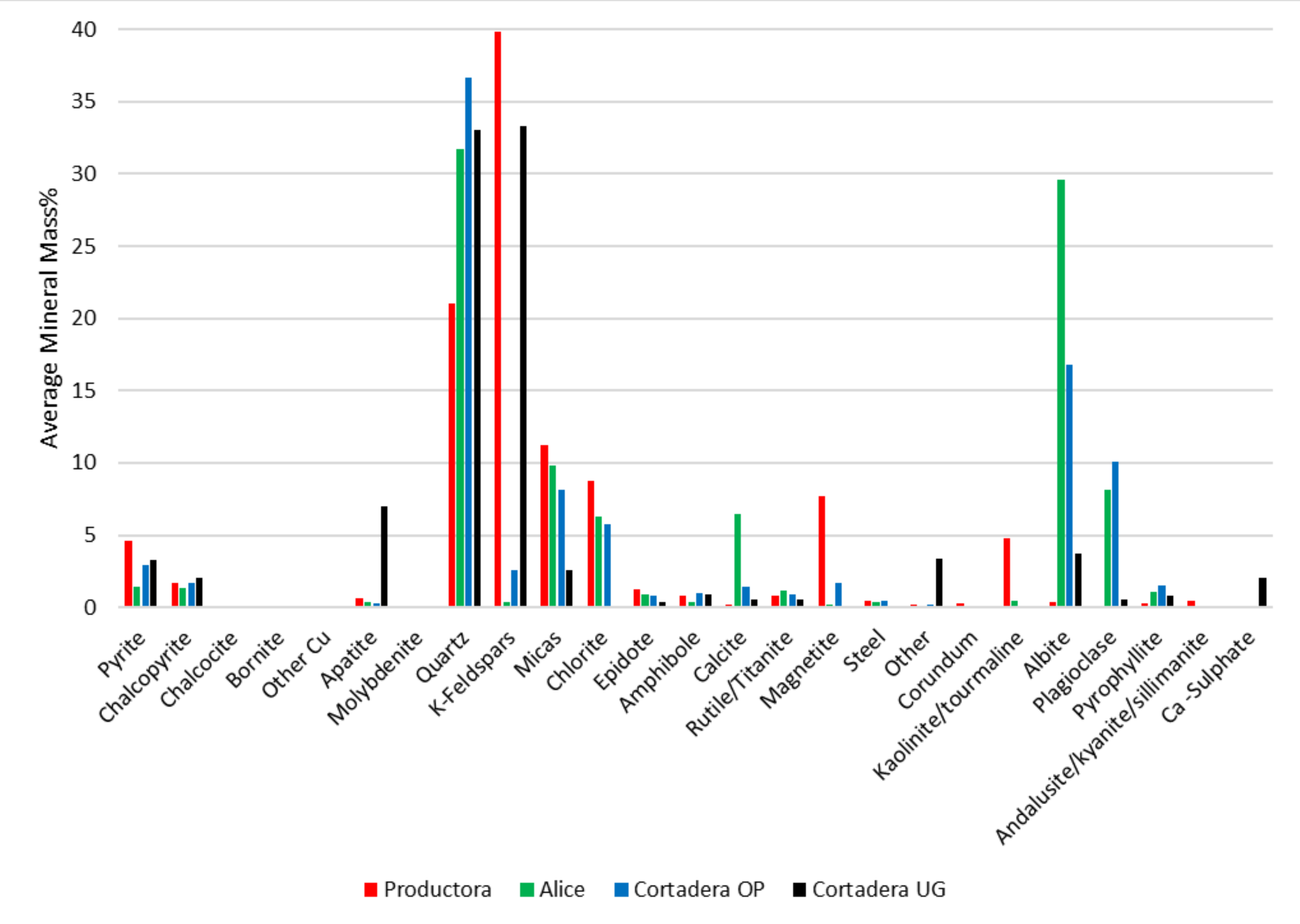
Figure 13.58 : QEMScan Analysis of Three Alice Flotation Feed Samples



The Alice samples are dominated by quartz with albite the second most abundant mineral. As the XRD showed, there is no k-feldspar and the dominant feldspar is plagioclase (sodium feldspar). Calcite is confirmed and pyrite and chalcopyrite are the major sulphides present. No other significant copper sulphide minerals were detected in Alice.

Only the two Cortadera master composite samples (as examples of Cortadera flotation feeds) were analysed by QEMScan. As the Master Composites (MC) were prepared from all variability samples, they represent the average of the tested samples from each of the parts of Cortadera. These have been included in Figure 13.59 to compare mineralogies across the main Costa Fuego deposits.

Figure 13.59 : Comparison of Average QEMScan Analyses for Productora, Alice and Cortadera Deposits



The only minerals present in similar magnitudes across all four deposits are pyrite, chalcopyrite, quartz and micas. All other minerals are only present in one, two or three of the deposits.

The main minerals (>2% on average in any deposit) are compared in Table 13.28.

Table 13.28 : Main Mineral Presence / Absence in Each of the Deposits

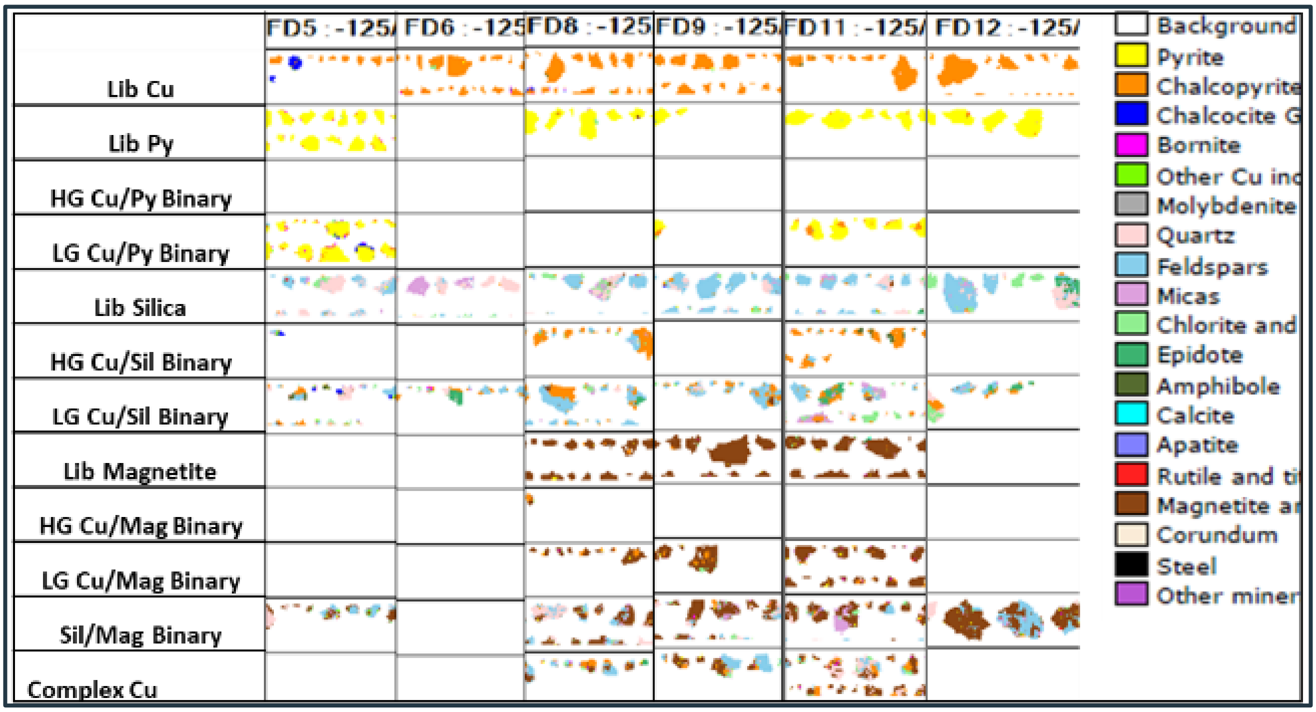
	Productora	Alice	Cortadera OP	Cortadera UG
Pyrite	Y	Y	Y	Y
Chalcopyrite	Y	Y	Y	Y
Apatite				Y

Table 13.28 : Main Mineral Presence / Absence in Each of the Deposits

	Productora	Alice	Cortadera OP	Cortadera UG
Quartz	Y	Y	Y	Y
Feldspars	Y			Y
Micas	Y	Y	Y	Y
Chlorite	Y	Y	Y	
Calcite		Y	Y	
Magnetite	Y		Y	
Kaolinite/tourmaline	Y			
Albite		Y	Y	
Plagioclase		Y	Y	
Ca -Sulphate				Y

An example of the additional mineral insights that can be produced using QEMScan is the comparison of the textures of the deposits and the liberation status of chalcopyrite in the two deposits. QEMScan images have been captured from various pages of the ALS report appendices to compare copper mineral liberation across the Productora fresh samples in Figure 13.60.

Figure 13.60 : Productora Fresh Samples QEMScan Locking Image Extracts, 125 µm

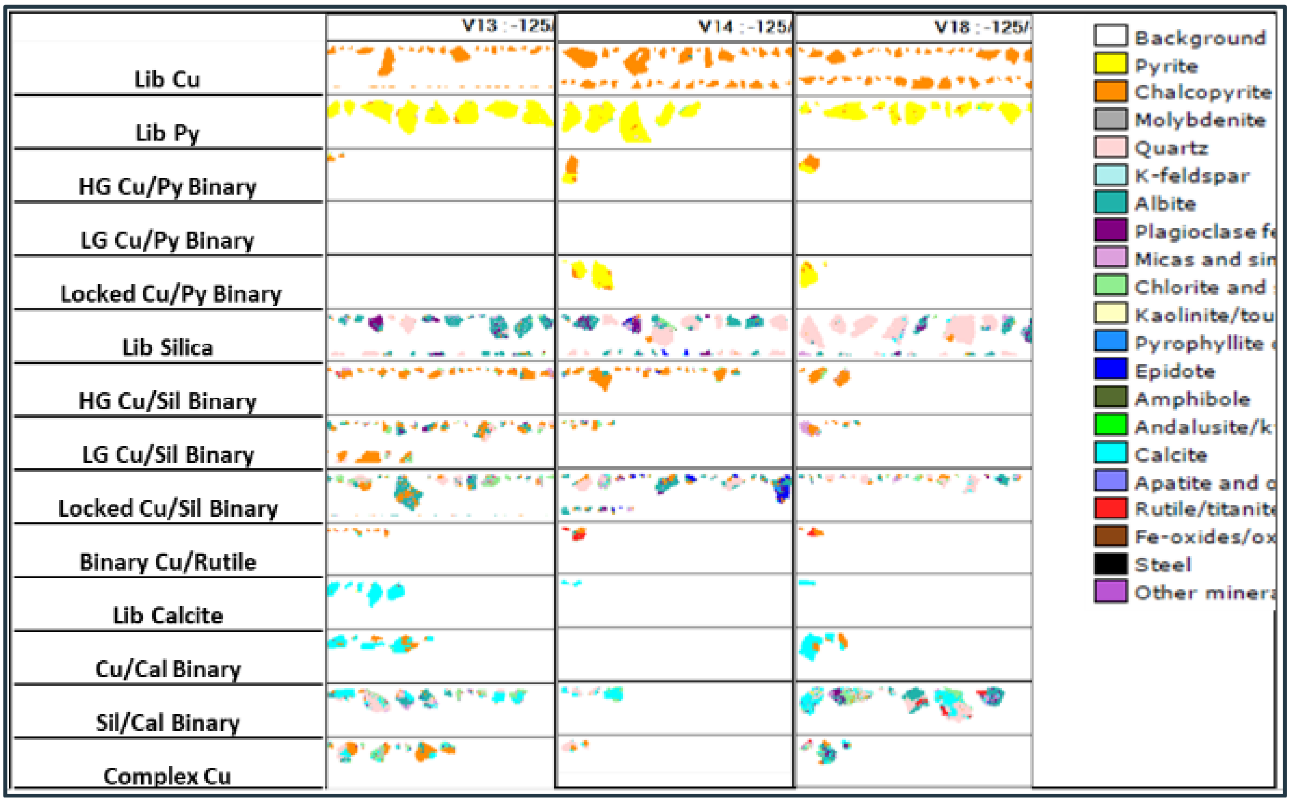


The variability of the Productora deposit is clear with some samples having associations between copper minerals and pyrite and others not. The same comment applies to copper in composite with silicates and copper with magnetite. The textures are relatively similar across all the samples, and this means that parameters such as primary and regrind size would not have to be varied during production. While the association between copper minerals and silicates involves relatively coarse chalcopyrite phases, the same is not true for all copper/pyrite cases, especially sample FD5 where the copper phases in pyrite are relatively fine.

On average for Productora, 70% of the contained copper was in the liberated copper particle class.

Copper mineral liberation in the three Alice samples is compared in Figure 13.61.

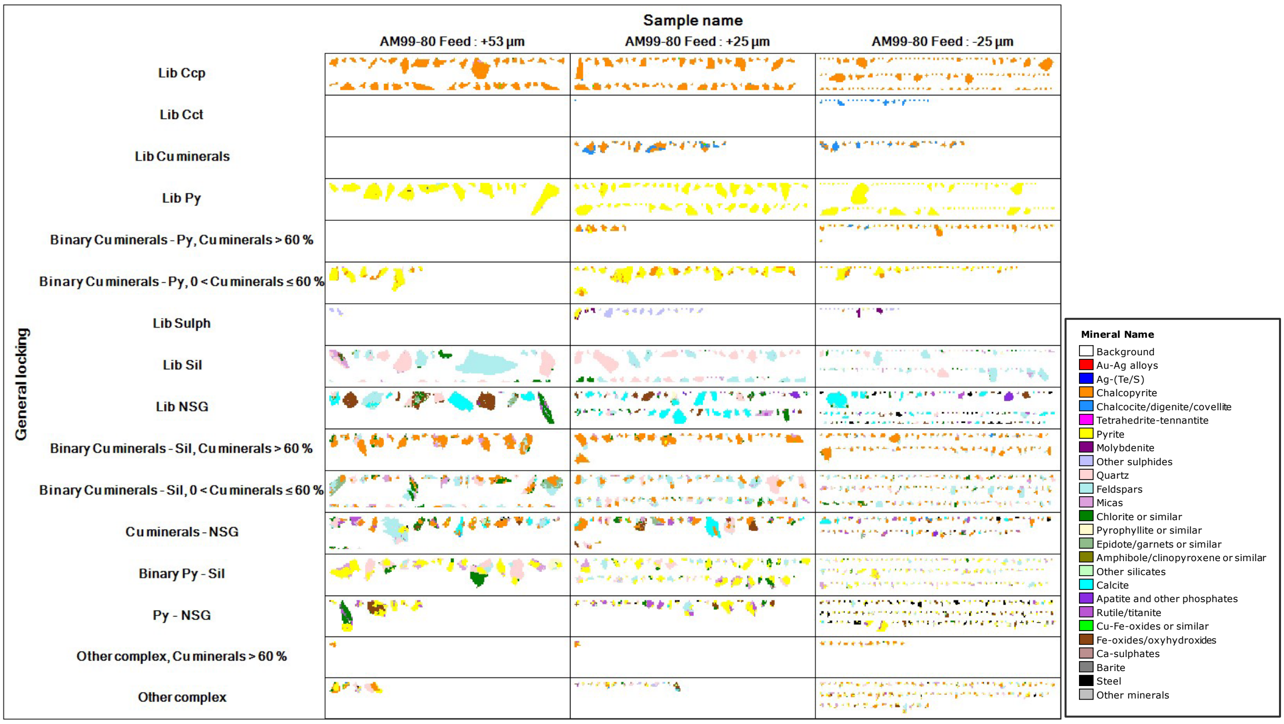
Figure 13.61 : Alice Samples QEMScan Locking Image Extracts, 125 µm



The Alice samples are much more consistent than Productora. These samples contain calcite, which is absent from Productora, and both albite and plagioclase are added to the mineral suite. Iron oxides are absent. Chalcopyrite is well liberated in Alice with 77% in the liberated copper class (10% more than Productora).

Copper mineral liberation in the Cortadera OP composite (for three particle sizes) is shown in Figure 13.62.

Figure 13.62 : Cortadera Open Pit Composite QEMScan Locking Image

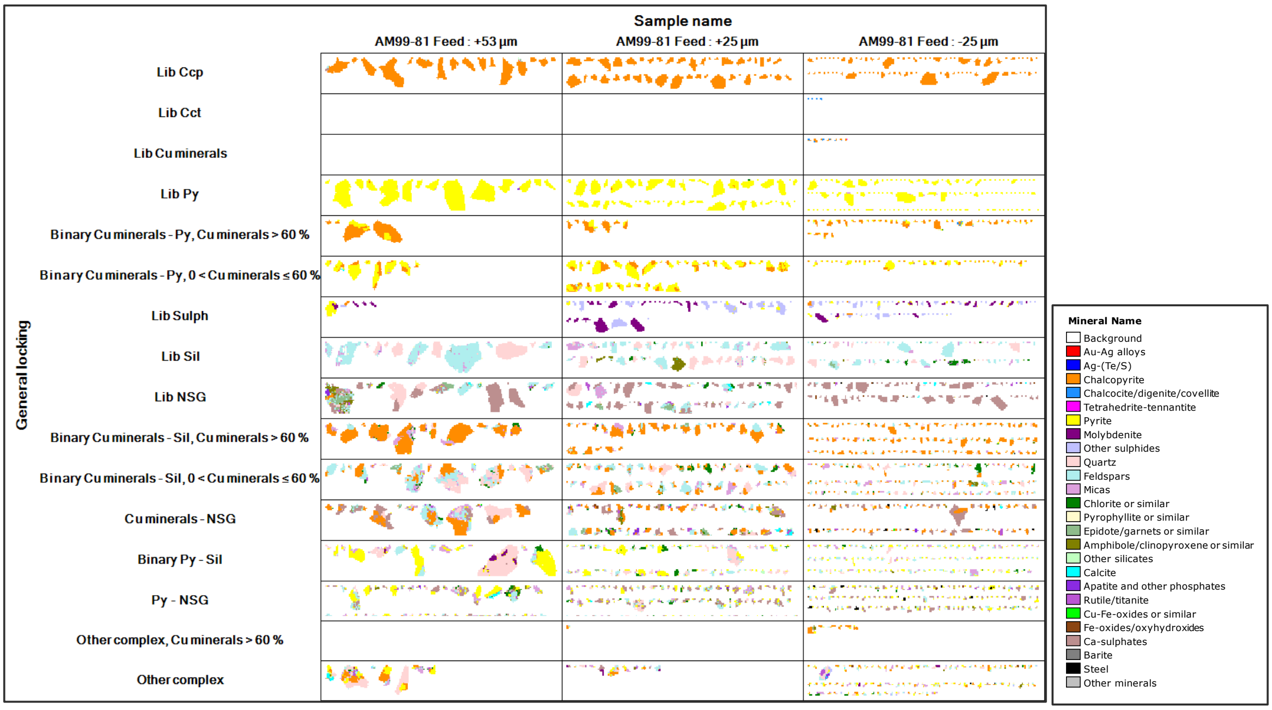


Interesting aspects of the Cortadera OP deposit are the presence of chalcocite and calcite, the absence of magnetite and the significant amount of copper composites with pyrite and silicates. Anhydrite (calcium sulphate) has been identified at Cortadera but only accounted for 0.1% of the OP composite.

Although the associations with pyrite look significant in this image, it is the silica and non-sulphide gangue (NSG) associations that account for 18% of the copper overall and 43% of the copper in the +53 µm fraction. Associations with pyrite only account for 1.4% of the copper in the sample. For the overall composite, 79% of the copper is fully liberated but in the +53 µm range only 55% is liberated. The textures of the more complex particles show the importance of regrinding to about 25 µm in the flowsheets. Note that chalcopyrite accounts for 98% of the copper in the composite so Ccp (chalcopyrite) data is essentially copper data.

Copper mineral liberation in the Cortadera UG composite (across three particle sizes) is shown in Figure 13.63.

Figure 13.63 : Cortadera Underground Composite QEMScan Locking Image



Thera are almost no secondary copper minerals in Cortadera UG with chalcopyrite carrying 99.7% of the copper. The composite has 7.3% anhydrite (calcium sulphate) and this distinguishes it from all other deposits, including Cortadera OP immediately above it. It also makes sulphur accounting difficult from chemical analysis alone as there is both sulphide and sulphate sulphur.

The chalcopyrite is 80% liberated (almost identical to the OP value) and 17% of the chalcopyrite is locked with silicates and NSG (again almost identical to the OP composite). The copper locked with pyrite was 2.6% and this is double the value for the OP. This is probably responsible for the need to regrind all of the UG rougher concentrate, rather than being able to recover >25% Cu concentrate copper without needing to regrind before cleaning.

Another significant aspect of the UG production feed is the high level of molybdenite detected. The liberation of molybdenite in feed is excellent at 92%.

13.11.4 Other Issues informed by QEMScan - Cortadera

To provide a greater understanding of mineral separation behaviour, QEMScan is best used when some separation processes have been applied to generate samples. In this program, concentrate, middlings and tailings samples from definitive open circuit testing on Cortadera master composites were sized and submitted for QEMScan analysis.

13.12 Pyrite in Open Pit Copper Concentrate

The OP copper concentrate sample contains 3.5% pyrite by QEMScan. This is similar to the LCT results where there is calculated to be between 4 and 5% pyrite in OP copper concentrate.

The average pyrite and chalcopyrite grain sizes are similar at 17 and 16 µm. As the sizes are similar, entrainment of pyrite when floating copper minerals is only a minor contributor to concentrate contamination.

61% of the pyrite in copper concentrate is well liberated and 25% is 90% liberated. The liberation data is consistent in the two particle sizes measured. The liberation grid for pyrite in copper concentrate (Figure 13.64) shows a strong association between pyrite and chalcopyrite

Figure 13.64 : Open Pit Production Feed – Pyrite Liberation in Copper Concentrate

Most of the pyrite particles in the three middlings classes (and some in the liberated class) have a single small attachment of chalcopyrite. These attachments are sufficient to allow these particles to be recovered in copper flotation. Liberated pyrite recovery in copper flotation is minimal compared to the total pyrite content of the sample. It is almost certain that the "well liberated" class of pyrite particles would mostly have a small attachment of chalcopyrite in the third dimension that is invisible to QEMScan.

The locking analysis shows that 70% of the pyrite in copper concentrate is classed as liberated and 24% is in binary particles with copper minerals. Only 2% of the pyrite is locked with silicates and NSG and 1.6% is in complex particles.

The analysis shows that pyrite is reporting to copper concentrate by true flotation (not entrainment) and has a strong association and locking with chalcopyrite. To prevent pyrite floating it is necessary to regrind to a finer size so that chalcopyrite/pyrite associations are broken. Extra concentrate regrinding is necessary if it is not possible to achieve 25% Cu in concentrate. As 70% of the concentrate has not been subjected to regrinding after rougher flotation, this would involve directing some or all rougher concentrate to regrinding.

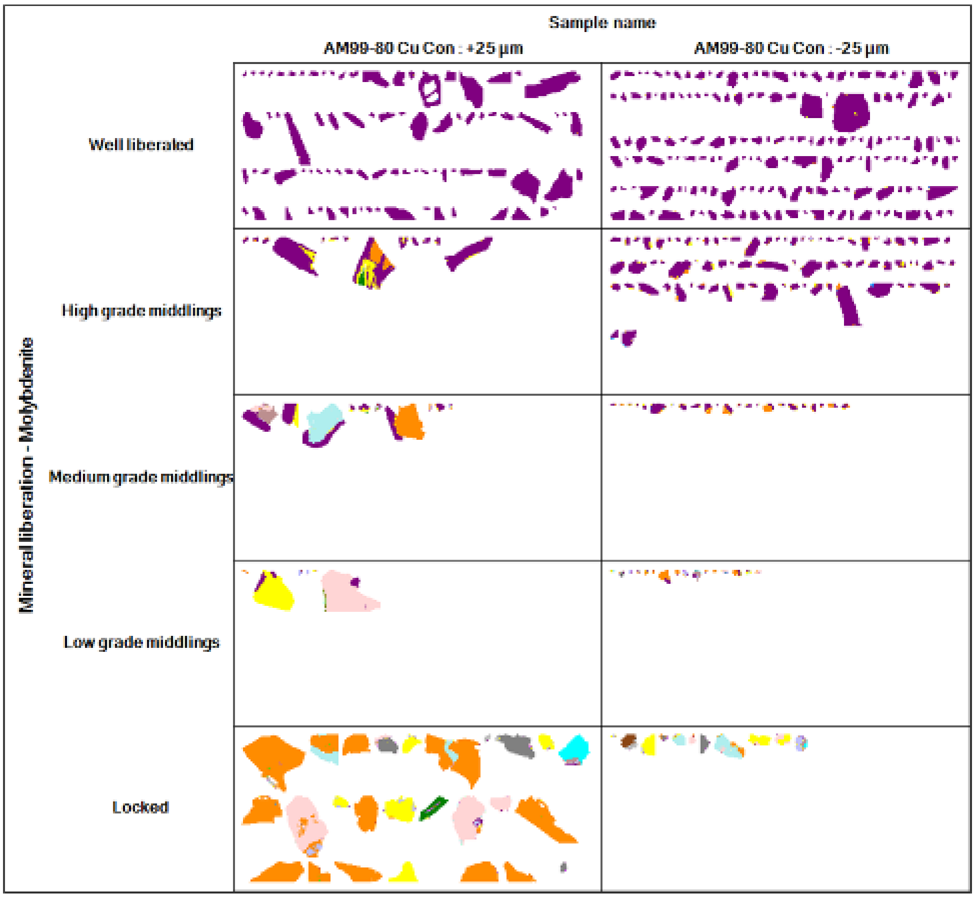
13.13 Molybdenum in Copper Concentrate

The copper concentrate contains 0.24% molybdenite (2400 ppm)

The average molybdenite grain size in copper concentrate is 15 µm, close to the 18 µm concentrate particle size and close to the 17 µm chalcopyrite size.

Molybdenite in copper concentrate is shown in Figure 13.65. 88% of the molybdenite is in the well liberated class and another 10% is in the high-grade middlings class. The particles in the lower liberation classes are of little relevance.

Figure 13.65 : Liberation Grid for Molybdenite in Copper Concentrate



The platy nature of molybdenite (similar to graphite) is evident in the elongated particles in the well liberated and high grade middlings classes. It is also possible that the molybdenite in the +25 µm medium grade middlings is not attached to the other minerals but is stuck or wrapped on the surface of the grains.

94% of the molybdenite is in the liberated molybdenum class in the locking analysis. A further 3% is in binary particles with pyrite and 1.3% with chalcopyrite. There is little other association.

Only 61% of the molybdenite is recovered to the copper concentrate in the LCT with the main losses in Pyrite Concentrate (20%) and Ro Tail (19%).

31% of the molybdenite in pyrite concentrate is classed as well liberated and none of the molybdenite in rougher tails is well liberated. In pyrite concentrate, 60% of the molybdenite is in the lowest three liberation classes. In Ro Tails 98% is in the lowest two liberation classes.

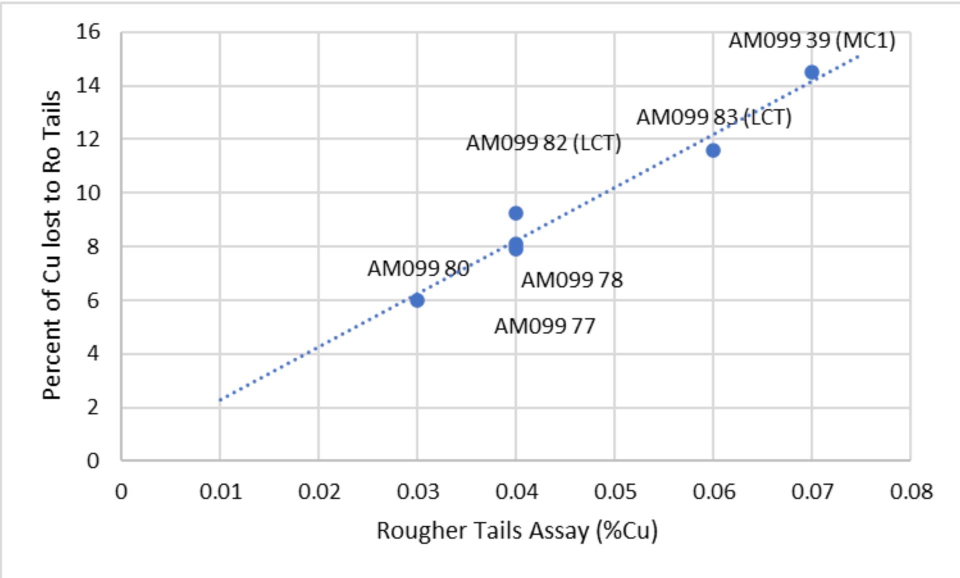
Given the low head grade of molybdenite in the OP sample (16 ppm), the poor recovery to copper concentrate is understandable. All but a small proportion of the molybdenite not in copper concentrate is considered unfloatable.

13.14 Copper in Rougher Tails

In LCT 83, 12% of the copper was lost to rougher tails, while in open circuit test AM099-80 (QEMScan analysed here) only 6% was lost. The magnitude and variability of these losses is, in part, due to the difficulty in assaying low copper grades and partly due to the presence of non-sulphide copper in the production feed.

The dependence of the copper losses to tails on % Cu in tails is shown in Figure 13.66 Currently the assay can only be 0.02, 0.03, 0.04, etc., and not between these values due to the sensitivity of the assay method. Each increase in tailings assay results in a 2% increase in copper lost to rougher tails. Note that the high loss for test AM099-39 was due to high oxide copper in the MC1 feed. This was the reason MC2 was prepared and excluded oxidised copper samples. All other sample points were generated using MC2.

Figure 13.66 : Cu Rougher Tail Losses – Dependency on Assay



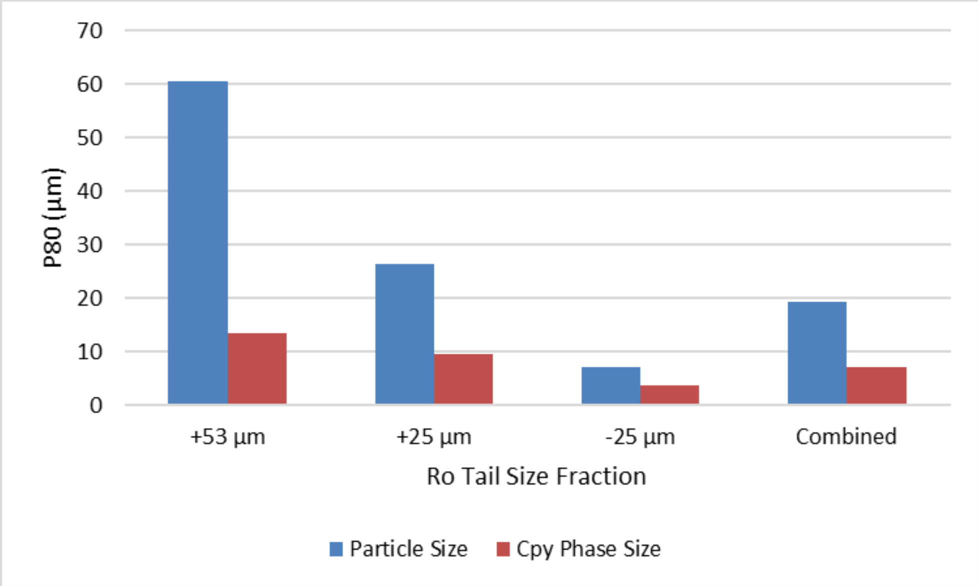
For test 80, the QEMScan data and the assay data are in strong agreement, resulting in a 6% calculated tails copper loss.

It is likely that the loss of copper to Ro Tails is less than 10% and not 12% or more, as LCT 83 seems to suggest.

Almost 95% of the copper lost to tails is as chalcopyrite and less than 1% is lost as chalcocite or other copper sulphides.

As can be seen in Figure 13.67, the chalcopyrite phases in Ro Tails are much smaller than the particle sizes. The chalcopyrite phases are locked within larger particles, and most are impossible to recover by flotation.

Figure 13.67 : Relative Particle Sizes and Chalcopyrite Phase Sizes in Rougher Tail



- Almost 60% of the lost copper is in the +53 µm fraction and about 30% is in the -25 µm fraction.
- The distribution of lost copper amongst liberation and size classes is shown in Table 13.29.
- Coarse (+53 µm) lost chalcopyrite is in the locked and low-grade middlings classes and is unavailable for recovery without grinding the flotation feed to a much finer and uneconomical size.
- Approximately 30% of the lost copper is in the fine (-25 µm) fraction and a little more than half is in the "well liberated" class and (possibly) available for recovery.

Table 13.29 : Liberation of Copper Minerals in Rougher Tail

Liberation Classes (based on mineral surface area %)	AM99-80 Ro Tail			
	+53 µm	+25 µm	-25 µm	Combined
	Combined Cu minerals (mass% in fraction)			
Well liberated	0.00	0.92	15.8	19.5
High grade middlings	0.00	0.05	2.78	3.34
Medium grade middlings	3.07	2.29	3.02	8.40
Low grade middlings	17.2	3.71	4.01	24.2
Locked	38.5	6.80	1.92	44.5
TOTAL	58.7	13.8	27.5	100.0

It must be remembered that copper discarded to Ro Tail has already had two chances of flotation in this test flowsheet. Therefore, the copper in the pyrite concentrate, and the reasons why it has reported there, are also of interest.

13.15 Copper in Pyrite Concentrate

8% of the copper in the LCT reported to pyrite concentrate, a concerning amount.

77% of this misplaced copper is in chalcopyrite and 21% is in chalcocite or other minerals. This is a high amount of non-chalcopyrite minerals and is a result of the high selectivity of the copper collector (xanthate ester) for chalcopyrite and against almost all other copper sulphides.

Chalcopyrite has an average size of 8 µm, suggesting it is in the pyrite concentrate by entrainment rather than true flotation. Chalcocite has an average grain size of 17 µm, suggesting it is there by true flotation, having been collected by PAX (a universal sulphide mineral collector).

The combined copper minerals are only 41% liberated in the pyrite concentrate. Many of these copper minerals would have pyrite associations.

Chalcocite is most strongly associated with chalcopyrite (27%) within the pyrite concentrate. There is a minor association with pyrite (3.6%).

Only 36% of the chalcopyrite in the pyrite concentrate is in the liberated chalcopyrite class and 45% is associated with silicate minerals. Only 13% is associated with other copper minerals or with pyrite.

54% of the chalcocite in the pyrite concentrate is classed as liberated. A further 22% is locked with liberated copper minerals and 12% is with other sulphides including pyrite. Locking with silicates accounts for much of the remaining material.

The liberation table for the copper minerals in pyrite concentrate is in Table 13.30. The majority source of copper in the pyrite concentrate is in the well liberated -25 µm fraction. Next highest is the low-grade middlings +25 µm fraction.

Table 13.30 : Liberation of Copper Minerals in Pyrite Concentrate

Liberation Classes (based on mineral surface area %)	AM99-80 Py Con		
	+25 µm	-25 µm	Combined
	Combined Cu minerals (mass% in fraction)		
Well liberated	14.9	23.7	40.6
High grade middlings	4.67	4.25	9.13
Medium grade middlings	10.8	3.42	13.9
Low grade middlings	21.6	2.77	23.2
Locked	12.6	1.25	13.2
TOTAL	64.6	35.4	100.0

The grid for combined copper minerals in pyrite concentrate is shown in Figure 13.68. The particle image boxes correspond to the blue boxes in Table 13.30.

The well liberated fine (-25 µm) copper minerals are very fine and appear to be in the concentrate by entrainment.

The coarse low-grade middlings require significant regrinding (possibly to 10 µm P_{80}) to release the locked copper minerals, which are mostly chalcopyrite.

Figure 13.68 : Copper Mineral Liberation in Pyrite Concentrate

Opportunities exist to recover some of the 8% of copper misplaced in the pyrite concentrate. Some of it can be recovered by selective flotation at the current regrind size but some needs new flotation chemistry to recover non-chalcopyrite copper sulphide minerals, while rejecting pyrite and NSG.

Further flotation investigation is recommended.

13.15.1 Copper, Pyrite and Molybdenum in Scavenger Cleaner Tail (SCT)

Copper SCT is an open circuit test stream that is retreated in the proposed flowsheet. It is mixture of fine values and poorly liberated coarser values.

The stream contains 7% chalcopyrite, 9% pyrite, 0.03% molybdenite, 21% quartz, 28% feldspar, 26% other silicates and 4.5% oxides and other minerals.

SCT is very fine with an average grain size of 6 µm and sizes of 4 µm, 5 µm and 3 µm respectively for chalcopyrite, pyrite and molybdenite.

Much of the valuable mineralisation is well liberated with 73% of the chalcopyrite, 71% of the pyrite and 78% of the molybdenite being classed as “well liberated”.

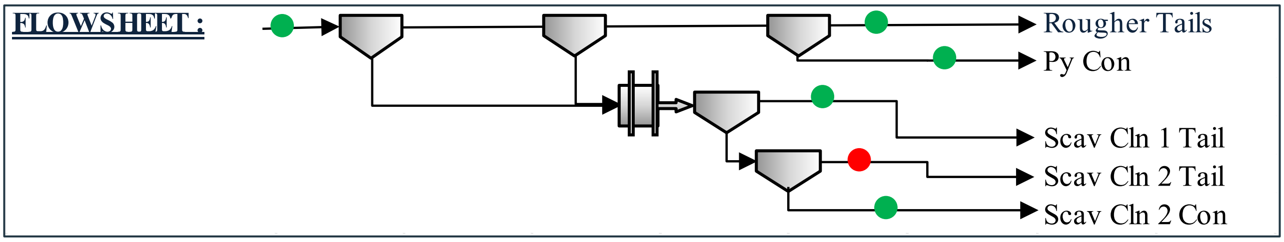
The high levels of liberation means that sulphides in this stream can be recovered strongly in pyrite flotation.

The fine sulphide and gangue particle sizes are the result of regrinding copper concentrate and indicate that the current regrind size being used is appropriate.

13.16 Underground Flowsheet

The UG deposit flowsheet is shown in Figure 13.69 and the only difference with the OP flowsheet is that all of the copper rougher concentrate is reground ahead of the first cleaning stage. The QEMScan samples are shown as green dots. Again, the Scav Clnr 2 tail was not submitted for QEMScan and this excludes 12.9% of the copper in the feed from QEMScan analysis. The Scav Clnr 2 Con is 56% passing 25 µm, much finer than the combined open pit concentrate sample.

Figure 13.69 : Cortadera Underground Production Feed Flotation Test Flowsheet, Test AM099-81



Similar questions are investigated for the UG deposit, but only in a comparative fashion to contrast with OP testwork.

- Pyrite in copper concentrate
- Molybdenum in copper concentrate
- Copper in pyrite concentrate
- Copper in rougher tails
- Copper, pyrite and molybdenum in SCT.

The comparative grades and distributions are shown in Table 13.31.

Table 13.31 : Cortadera Open Pit vs. Underground: Grades, Correct Placements (Green) and Misplacements (Red)

	Cortadera OP Deposit		Cortadera UG Deposit	
	Grade	Recovery %	Grade	Recovery %
Cu in Cu Concentrate (%)	27.5	82	25.7	90
Mo in Cu Cons (ppm)	732	63	7263	86.5
Pyrite in Cu Cons (%)	2.3	2.3	8.5	7.2
Cu in Pyrite Cons (%)	1.1	8.0	0.65	3.9
Cu in Rougher Tails (%)	0.05	9.3	0.03	6.5
Cu in SCT (%)	1.2	5.5	16	10
Mo in SCT (ppm)	100	6.8	365	3.1
Pyrite in SCT (%)	9.0	4.0	16	6.3

Apart from pyrite content in copper concentrate, all other comparisons in the table above are in favour of the UG deposit. A comparison of pyrite liberation in copper concentrate is shown in Table 13.32.

Table 13.32 : Cortadera Open Pit vs. Underground: Pyrite Misplaced to Copper Concentrate – Rejection Opportunity

	OP	UG
Pyrite in Cu Cons (Grade %)	2.3	8.5
Pyrite in Cu Cons (Dist. %)	2.3	7.2
Well liberated	61	71
High grade middlings	25	19
Medium grade middlings	8.5	6.1
Low grade middlings	3.6	2.4
Locked	2.8	1.5
Potentially Rejectable from Con	2.0	6.5

Well liberated and high-grade middlings pyrite are considered "potentially" rejectable from the copper concentrate by improved selectivity. Pyrite in the three lesser liberation classes is only rejectable with an undesirable increase in copper losses. Note that the copper concentrates measured by QEMScan are partially reground (OP) and totally reground (UG) respectively, and there is no opportunity currently contemplated in the flowsheet for improving pyrite liberation from copper with additional regrinding stages.

The two feed composites are materially different with respect to liberation of pyrite, with almost four times as much pyrite in the UG copper concentrate as in the OP copper concentrate. However, a similar proportion of the pyrite in each concentrate is potentially rejectable. To reject liberated pyrite, it is recommended that froth washing be used in the copper cleaner flotation cells.

The aim is to maximise molybdenum recovery into the copper concentrate so that it is available for recovery into a molybdenum concentrate. The molybdenum recovery comparison is given in Table 13.33.

Table 13.33 : Cortadera Open Pit vs. Underground: Molybdenum in Copper Concentrate – HG Molybdenum Concentrate Potential		
	OP	**UG**
Mo in Cu Cons (Grade ppm)	732	7263
Mo in Cu Cons (Dist. %)	39	87
Well liberated	88	87
High grade middlings	10	11
Medium grade middlings	1.7	1.4
Low grade middlings	0.26	0.45
Locked	0.11	0.22
Stage Rec Potential to make HG Mo cons	98	98

The first major difference is that the concentrate grade of molybdenum in the OP concentrate is an order of magnitude lower than in the UG. This is due to the head grade in the OP deposit (16 ppm) also being an order of magnitude lower than the UG (140 ppm). Despite this grade difference, the liberation status of molybdenum in the two concentrates is identical and effectively fully liberated.

Given the molybdenum is effectively fully liberated, the ability to make a final molybdenum concentrate would not depend upon the molybdenum grade of the plant feed or the copper concentrate. More importantly, there should be no influence on molybdenum cleaning due to the source of the feed. A high grade and saleable molybdenum concentrate should be possible under all circumstances when feeding the plant from Cortadera.

The copper that has misreported to pyrite concentrate is shown in Table 13.34.

Table 13.34 : Cortadera Open Pit vs. Underground: Copper in Pyrite Concentrate – Copper Recovery Opportunity		
	OP	**UG**
Cu in Py Cons (Grade %)	1.1	0.65
Cu in Py Cons (Dist. %)	8.8	3.9
Well liberated	41	28
High grade middlings	9.1	5.6
Medium grade middlings	14	9.5
Low grade middlings	23	30
Locked	13	27
Cu Potentially Recoverable from Py Con	4.4	1.3

Almost twice as much copper has been lost to the OP pyrite concentrate compared to the UG pyrite concentrate, and the misreporting OP copper is twice as liberated. This translates to three times as much potentially recoverable copper being available from the OP as from UG.

This behaviour is not about liberation but is about copper mineralogy. A large proportion of the copper in pyrite concentrate is there because it is chalcocite and is not able to be recovered by the chalcopyrite-selective xanthate ester used in copper flotation. This is clear in Figure 13.68 where many of the liberated phases are chalcocite (blue). This is avoidable behaviour with only slight modifications to the collector regime.

It is recommended that in addition to the xanthate ester, a less selective copper collector be utilised in small quantities in the copper flotation scavenger stage, especially when treating feed from the Cortadera OP deposit.

Due to the presence of transition mineralisation, chalcocite is more prevalent in the OP. For the OP, 1.7% of the copper in feed is attributable to chalcocite, while for UG only 0.23% is attributable to chalcocite.

The misplacement of molybdenum to pyrite concentrate is shown in Table 13.35

Table 13.35 : Cortadera Open Pit vs. Underground: Molybdenum in Pyrite Concentrate – Molybdenum Recovery Opportunity

	OP	UG
Mo in Py Cons (Grade %)	104	171
Mo in Py Cons (Dist. %)	16	3.5
Well liberated	31	25
High grade middlings	10	5.8
Medium grade middlings	13	20
Low grade middlings	24	31
Locked	23	19
Mo Potentially Recoverable from Py Con	6.5	1.1

Only a minor amount of molybdenum is misplaced to pyrite concentrate for both deposits (the 16% value for OP is for a very low head grade and is expected). Like the molybdenum in copper concentrate, the liberation characteristics of molybdenum are similar in both cases despite the major difference in head grade. No recommendations are made with respect to reducing misplacement of molybdenum to the pyrite concentrate. However, the use of a less selective copper collector in copper scavenging may recover additional molybdenum with the chalcocite.

The remaining analysis tables are for tails (RT and SCT). It must be recognised that any values that have reached the RT have had two opportunities (copper float and Py float) to be recovered by flotation.

The analysis for loss of copper to rougher tail is shown in Table 13.36.

Table 13.36 : Cortadera Open Pit vs. Underground: Copper in RT – Copper Recovery Opportunity		
	OP	**UG**
Cu in RT (Grade %)	0.04	0.03
Cu in RT Cons (Dist. %)	9.3	6.5
Well liberated	20	21
High grade middlings	3.3	3.0
Medium grade middlings	2.9	2.9
Low grade middlings	23	23
Locked	51	51
Cu Potentially Recoverable from Rougher Tail	2.1	1.5

While more copper is lost to tails from the OP deposit, the level of loss is a strong function of the % Cu grade of RT, which is also close to the detection limit of the technique. Within the measurement accuracy there may be little to no difference between these results.

The liberation distributions of the lost copper are almost identical, so it is concluded that between 1 and 2% additional copper recovery may be possible. However, to access this additional recovery it may be necessary to grind the flotation feed even finer and/or extend flotation time and increase collector additions.

The SCT is an open-circuit test middlings stream that is not present in LCT except as a "leftover" recycle stream at the end. The grade and recovery in these streams are taken from LCT cycle 6 as the open circuit stream variability can be high. The analysis of copper present in the SCT is shown in Table 13.37.

Table 13.37 : Cortadera Open Pit vs. Underground: Copper in SCT– Copper Recovery Opportunity		
	OP	**UG**
Cu in SCT (Grade %)	2.6	1.3
Cu in SCT Cons (Dist. %)	8.6	3.9
Well liberated	74	73
High grade middlings	9.3	11
Medium grade middlings	9.4	9.2
Low grade middlings	5.9	5.7
Locked	1.9	1.8
Cu Potentially Recoverable from Scav Cleaner Tail	7.2	3.3

The copper liberation patterns for copper in the SCT are effectively identical. About 80% of copper in these streams is potentially recoverable to copper concentrate.

The analysis of molybdenum in the SCT is in Table 13.38.

Table 13.38 : Cortadera Open Pit vs. Underground: Molybdenum in SCT – Molybdenum Recovery Opportunity		
	OP	**UG**
Mo in SCT (Grade ppm)	188	185
Mo in SCT Cons (Dist. %)	10.5	2.0
Well liberated	78	90
High grade middlings	14	5.7
Medium grade middlings	7.0	0.57
Low grade middlings	0.14	2.6
Locked	1.2	1.6
Mo Potentially Recoverable from Scav Cleaner Tail	9.7	1.9

The SCT streams have similar grades of molybdenum despite the major difference in feed grade between the two deposits. As the majority of the molybdenum in the SCT is liberated, this suggests that it is simply floating a little slower than the copper minerals. Most of the molybdenum would eventually report to copper concentrate and boost overall recovery. The relatively high potential recovery of molybdenum from the OP deposit is linked to the low feed grade and the low overall recoveries that this low feed grade results in. Note that the low feed grade of molybdenum in OP production feed means that the molybdenum losses are less of a problem compared to the higher grade and recovery UG deposit.

Combining the potential gains indicates if there is greater potential recovery for the metal values. The summary of combined potential copper recovery is shown in Table 13.39.

Table 13.39 : Cortadera Open Pit vs. Underground: Copper Opportunity		
	OP	**UG**
Cu in Cu Conc (Open Cct)	74	91
Cu from Py Con	4.4	1.3
Cu from SCT	7.2	3.3
Cu from RT	2.1	1.5
Potential Cu Rec	87.7	97.1
Actual Cu Rec (LCT)	82	90

The open circuit recovery starting point for this calculation has been determined from cycle 1 of each of the respective LCTs. As indicated, the actual recoveries are from the LCTs and represent average performance of the final three cycles.

For the OP deposit, the difference between the actual recovery of 82% and the potential recovery is 5.7%. This is mainly from the pyrite concentrate and the SCT and a large proportion is due to the presence of chalcocite, which floats poorly with the copper collector. Additional flotation tests, and possibly an additional LCT, would be performed on the OP MC2 sample to determine how much of this potential additional recovery can be retrieved in practice.

For UG production feed, almost all of the copper (97%) is potentially recoverable. This appears to be an impossibly high value, especially when the open circuit recovery (91%) is effectively identical to the 90% LCT recovery. There may be potential for a slight increase in recovery due to the presence of chalcocite.

One aspect applicable to both deposits is the recovery benefits of using froth washing. Recovery is normalised to 25% Cu by mathematically moving along a grade/recovery curve from a high or low copper grade (greater or less than 25% Cu). Typically, froth washing can raise the copper grade by removing free silica and pyrite, but it does this with minimal to no loss of copper recovery. This means that with froth washing a new and better grade/recovery curve can be drawn that results in a higher recovery at 25% Cu. This is difficult to test in the laboratory but has previously been confirmed in pilot operation and is also standard plant practice when cleaning or recleaning in column or similar cells.

The summary of combined potential molybdenum recovery is shown in Table 13.40.

Table 13.40 : Cortadera Open Pit vs. Underground: Molybdenum Opportunity

	OP	UG
Mo in Cu Conc (Open Cct)	20	85
Mo from Py Con	6.5	1.1
Mo from SCT	9.7	1.9
Potential Mo Rec	36.2	88.8
Actual Mo Rec (LCT)	39	86.5

For molybdenum, the potential and actual recoveries are almost identical. This suggests that there is little likelihood of strongly improving on the current molybdenum performance in the LCTs.

Measures to improve copper recovery such as using less selective copper collectors and applying froth washing are likely to provide some benefit for molybdenum recovery, but this is not expected to be large. It must also be remembered that molybdenum recovery to copper concentrate only provides feed to a molybdenum float circuit and any gains in copper flotation are not guaranteed to flow through to ultimate molybdenum recovery to a saleable concentrate.

13.17 QEMScan Summary

QEMScan analysis has revealed the following important features of the Costa Fuego Deposits

The Costa Fuego deposits have a range of distinct mineralogies, as shown in Figure 13.59 and Table 13.28. It is not clear what the effect of the varying gangue mineralogy values has on copper recovery, but some small differences in copper sulphide mineralogy do have an effect.

An allowance is to be made for regrinding of all copper rougher concentrate from commencement of operations as this would give maximum flexibility as the liberation of chalcopyrite changes across deposits.

A large proportion of copper lost to pyrite concentrates from OP samples is in the form of chalcocite. Use of a less selective copper collector is recommended during the copper scavenger flotation stage as this would recover the chalcocite as well as the chalcopyrite.

A large proportion of the pyrite in the final copper concentrate is liberated and can be removed from the concentrate using froth washing during cleaning.

Extended flotation times, the use of froth washing, and extended copper scavenger flotation may recover some of the material that is classed as "potentially recoverable" from rougher tail.

13.18 Productora Flotation Products Dewatering Testwork

13.18.1.1 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 and tailings samples were prepared from various flotation tests. The concentrate was from bulk flotation conducted for molybdenum testwork. The sample used for the molybdenum testwork was PRP0812 and was supplied as RC chip. The concentrate tested was at P_{80}=29.9 µm, though it contained a high level of fines (due to being from RC chip), and copper grade was 23% Cu. The concentrate had the copper/molybdenum separation conducted before dispatch to Outotec for thickening and filtration testwork. Due to low flotation mass pulls and high mass requirement for the thickening and filtration testwork, a high production feed mass (300 to 500 kg) is required to produce sufficient concentrate mass. The molybdenum bulk flotation testwork was the only pathway to produce sufficient concentrate. This sample is not ideal (due to being RC chip) and further testwork should be conducted at an appropriate time on bulk concentrate produced from drill core.

Samples for tailings testwork were composited from the rougher tailings from 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. The compositing of both tailings samples is shown in Table 13.41.

Dried rougher tailings from the 1 kg batch flotation tests, using the RTD 2086 collector, were selected for compositing, to make up 15 kg of dry tailings for each composite. Due to sample limitations, a number of P_{80}=180 µm samples were ground to 150 µm, and a number of P_{80}=125 µm samples were ground to 106 µm. The dry tailings composites were slurried in seawater. Furthermore, to simulate cleaner tailings, approximately 4% of the dry sample mass was ground to ≈25 µm, then added back to each composite.

Table 13.41 : Tailings Sample Make up				
FD composite	**Tailings Sample**			
	106 µm		150 µm	
	Mass (g)	Distribution (%)	Mass (g)	Distribution (%)
1	5 170	30.0	6 155	35.7
2	836	4.9	1 547	9.0
6	1 752	10.2	1 744	10.1
8	844	4.9	889	5.2

Table 13.41 : Tailings Sample Make up				
FD composite	**Tailings Sample**			
9	851	4.9	850	4.9
10	855	5.0	887	5.1
11	1 675	9.7	1 739	10.1
12	1 806	10.5	904	5.2
13	1 679	9.8	1 662	9.6
14	1 738	10.1	867	5.0
Total	**17 206**	**100**	**17 244**	**100**

13.18.1.2 Copper Concentrate and Tailings Thickening Testwork Results

Testing showed that the flotation tailings could achieve the desired underflow density of 70% solids (w/w) at 0.5 t/m^{2}h and 10 g/t of flocculant. Under these conditions, the overflow solids content was less than 150 mg/L. For the copper concentrate, underflow density of 65% solids (w/w) was achievable at 0.25 t/m^{2}h and 10 g/t of flocculant. Overflow solids under these conditions was less than 150 mg/L.

From the testwork, the thickeners recommended for the Project were the Outotec high-rate thickener with Outotec Vane feedwell. Other recommendations are given in Table 13.42.

Table 13.42 : Copper Concentrate and Tailings Thickening Results		
Process Stream	**Flotation Tails**	**Copper Concentrate**
Solids feed rate (tph)	1 724	30
Solids loading (t/m^{2}h)	0.5	0.25
Feed slurry density (% w/w solids)	26	18.3
Slurry pH	7-8	7-8
Flocculant dosage (g/t)	10	10
Underflow density (% w/w solids)	69.6-72.6	66.9-69.9
Overflow clarity (ppm)	<150	<150
Required thickener diameter (m)	1 x 67 or 2 x 47	13

It should be noted that the concentrate used was not ideal, as RC chip was used to produce the sample and concentrate. The concentrate contained a high level of fines (from the RC chip) and this may have influenced the concentrate thickening. More thickening testwork is recommended to be conducted on concentrate generated from diamond drill samples during the PFS.

13.18.1.3 Copper Concentrate Filtration Testwork Results

The same concentrate sample that was used for thickening testwork was also used for filtration testwork. The thickened product from the thickening testwork (set at 65% solids) was used as the starting product for the filtration testwork. Initially, testwork focused on membrane filter press technology. Pressure filtration technology was then tested as an alternative, aimed at achieving reduce wash times. A summary of the major outcomes from the Outotec report are described below.

13.18.1.4 Membrane Filter Press (MFP)

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 98.8%.
- 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 6000 ppm).

13.18.1.5 Pressure Filtration (PF)

Cycle time with washing is significantly less for pressure filtration technology than it is for membrane filter press technology.

- 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 (correspondence with ALS). 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 shows 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 6000 ppm).

Filtration testwork was conducted in seawater. Testwork shows that the molybdenum flotation circuit may require a reduced TDS water and as a result, the copper concentrate slurry feeding filtration would have a lower chloride level than tested.

For the molybdenum testwork, only tap water was tested. Further molybdenum flotation testwork is required to define a maximum water TDS. This level would then need to be used for further filtration testwork.

It was noted that the sample contained more than the usual level of fines and that this made filtration and the subsequent wash stage difficult. This high level of fines was due to the sample coming from RC chips. It is therefore recommended that further filtration testwork should be conducted from concentrate produced from diamond drill samples.

13.18.1.6 Copper Tailings Rheology Testwork Results

The tailing samples produced for testwork at KP (at P_{80}=106 and 150 µm) was also used for rheology testwork at ALS Perth and Newpark. Results from the Newpark testwork are shown in Table 13.43.

Table 13.43 : Flotation Tailings Rheology Results

FD tailings (P_{80}=106 µm)		FD tailings (P_{80}=150 µm)	
Solids (%)	**Yield (Pa)**	**Solids (%)**	**Yield (Pa)**
74	31.02	74	22.56
65	2.268	65	1.62
57	0.594	57	0.216
50	0	50	0
45	0	45	0

Significant comments from the Newpark report include:

- The rheograms show that the samples exhibit shear thinning behaviour.
- No significant settling was observed during the testing period for each rheogram.
- The percentage solids required to get yield stress readings of 10, 30 and 100 Pa is very high.

It is concluded that centrifugal pumping can be used for both concentrate and tailings.

13.19 Comments on Section 13

13.19.1 Comminution

The comminution work showed that the samples tested displayed medium to medium-high competency and would require moderate to high grinding energies. The Ai results indicated average abrasiveness, which would result in relatively moderate ball and liner consumptions. The Alice material is similar to Productora samples. Cortadera production feed is softer with respect to ball milling than the Productora production feeds but is similar with respect to SAG milling.

Based on Morrell's design calculations the design power requirements in Table 13.44 have been estimated for each of the deposits.

Table 13.44 : Design (80th Percentile) Specific Energy by Equipment for Costa Fuego Deposits

Specific Energy (kWh/t)	Productora	Alice	Cortadera OP	Cortadera UG
Primary Crusher	0.10	0.09	0.12	0.10
SAG Mill	8.74	8.44	9.41	10.5
Pebble Crusher	0.24	0.22	0.23	0.27
Ball Mill	13.4	12.4	9.1	11.8
Total	22.5	21.2	18.8	22.6

13.19.2 Copper Sulphide Flotation

Productora and Alice testwork was based on a standardised flowsheet and reagent scheme which had been established in previous testwork programs. The scheme used RTD2086 (xanthate ester supplied by Tall Bennett) at a natural pH of seawater with two stages of cleaning and a 25 µm regrind. Natural seawater pH was preferred as attempting to control pH in seawater is always more difficult than in fresh water due to buffering by the salt content.

Target (and reference) copper grade for the final concentrate was 25% Cu. Copper recovery was to be maximised, as was rejection of the main gangue species, pyrite and NSG. The Cortadera flotation test program commenced by applying the Productora float scheme, but excessive pyrite floated in the copper circuit. The scheme was optimised by varying:

- Primary grind size
- Reagent dosage
- Flotation time
- Alternate sulphide collectors
- Diesel addition for increased molybdenum flotation
- Seawater vs. tap water.

A primary grind size of P_{80} = 125 µm was selected for all Productora samples to target a similar sulphide P_{80}. To achieve the same sulphide size in plant grinding a P_{80} of 150 µm is required. The 2022 Productora flotation work was conducted at 106 µm P_{80} (125 µm in plant) to maximise copper recovery. For the Alice deposit, a primary grind P_{80} of 180 µm was chosen for the 2016 study, but this was changed to 125 µm P_{80} and finally

106 µm was used in the 2022 flotation work. For this analysis, the results at 125 µm and 106 µm have been plotted without differentiation and this means that some of the recoveries (at 125 µm) would be conservative.

For Cortadera flowsheet development the collector addition rate and flotation times were manipulated to improve selectivity against pyrite. It was also evident with the OP master composite that it was possible to make a final copper concentrate out of the rougher flotation stage concentrate without conducting regrinding. This has the benefit of resulting in a coarser concentrate, which assists dewatering processes.

For Cortadera UG it was again necessary to modify reagent additions and flotation times. It was also necessary to reinstate regrinding of the full copper rougher plus scavenger concentrate to achieve 25% Cu in concentrate.

Predictive recovery equations were developed for copper, molybdenum, gold and silver in each of the deposits using the results of multiple well performed tests from across the program.

Alice typically performed better than Productora as has been previously described. Tests on blends of Alice and Productora samples showed that the performance of the two deposits is additive, and the equations below apply to deposit treatment alone or in blends.

The predictive equations for recovery of values from all deposits are summarised in Table 13.45.

Table 13.45 : Recovery Prediction Bases – Costa Fuego PEA

Deposit	Rec Cu	Rec Mo	Rec Au	Rec Ag	Comments
Productora & Alice – Oxide	Productora = 56% Fixed Alice = 46% Fixed	-	-	-	-
Productora – Fresh	Rec = 9.072 x CuF% + 83.66 Cap 95%	0.9 x [RecMo = 17.342 x ln(MoFppm) - 34.65] Cap 90%	RecAu = 145.4 x AuFppm + 38.549 Cap 80%	40% Fixed	-
Productora - Transitional	RecCu = 19.609 x CuF% + 63.443 Cap 90%			40% Fixed	Mo and Au recoveries are both unaffected by the oxidation and alteration in the transition zone
Alice – Fresh	RecCu = 0.4951 x CuF% + 91.0	0.9 x [RecMo = 0.882 x MoFppm + 18.52] Cap 90%	RecAu = 145.4 x AuFppm + 46.692 Cap 80%	40% Fixed	Alice samples are all fresh and have performed better than Productora Fresh in all cases
Cortadera OP – Oxide	50% Fixed	-	-	-	-
Cortadera OP – Fresh	Rec=17.016 x LN(CuF%) + 96.378 Max = 90%, Min=18%	50% Fixed	Rec=104.74 x AuF + 29.42	27% Fixed	-

Table 13.45 : Recovery Prediction Bases – Costa Fuego PEA					
Deposit	**Rec Cu**	**Rec Mo**	**Rec Au**	**Rec Ag**	**Comments**
Cortadera OP – Trans	Rec=17.016 x LN(CuF%) + 86.378 Max = 80%, Min=8%	50% Fixed	Rec=104.74 x AuF + 29.42	27% Fixed	Cu Recovery reduced by 10% from Fresh
Cortadera UG	Rec=8.615 x LN(CuF%) + 96.122 Max=95% at CuF>0.88%	0.9 x [RecMo = 11.656 x ln(MoFppm) + 19.953] Cap 90% Min 18%	Rec=30.368 x AuF + 51.637	38% Fixed	-

The flotation schemes for the four deposits are compared in Table 13.46.

Table 13.46 : Flotation Schemes				
		Productora & Alice	**Cortadera OP**	**Cortadera UG**
P_{80} for Cu		125	106	106
Cu Rgh	Condit. (min)	1	1	1
	RTD2086 g/t	10	5	5
	Float Time	1	2	2
Cu Rgh Regrind		Yes, of Scav Con	No	Yes, of Scav Con
Cu Rgh Clnr	Condit.	-	0	-
	RTD2086	-	0	-
	Float Time	-	3	-
Cu Scav	Condit.	3		1
	RTD2086	15		3
	Float Time	13		3
Cu Scav Regrind		With Rougher con	Yes, with Rougher Clnr Tail	With Rougher con
Cu Scav Clnr1	Condit.	2	1	1
	RTD2086	10	5	7
	Float Time	11	4	6
Cu Scav clnr scav	Condit.	1	-	1
	RTD2086	2	-	3
	Float Time	7	-	4
Cu Scav Clnr1	Condit.	1	0	0
	RTD2086	5	0	0
	Float Time	12	3	4.5
Successfully Tested Alternatives				
Additional Cu Collector		A3849 + SMBS	A3849	-
		C4403		
Additional Moly Collector		Diesel	-	-

In general, the main schemes work well with the RTD2086 xanthate ester, but in some circumstances, such as when treating transition production feeds, recoveries are likely to benefit from small additions of A3849 to recover chalcocite and/or diesel to improve molybdenum recovery.

QEMScan analysis confirmed the lack of chalcocite flotation, in tests where it was present in measurable quantities. It also confirmed reasons for misplacement of copper into the pyrite concentrate, the complexity of the copper bearing composites, the reasons it is difficult to recover more copper from the rougher tails and the high potential for additional recovery of values in middlings streams. The well liberated nature of pyrite in the copper concentrate showed that froth washing should be successful in raising concentrate grades, which has the potential to lead to higher recoveries at the target 25% Cu in concentrate.

13.19.3 Molybdenum Flotation

Between 25 and 95% of the molybdenum is recovered in copper rougher flotation with samples from all deposits. Separate molybdenum flotation testing (to produce a saleable molybdenum concentrate) showed that about 10% of the molybdenum is lost in cleaning on the way to 50% Mo. A 10% reduction in recovery is applied to all molybdenum recovery equations (Table 13.45) to allow for cleaning losses.

Bulk flotation on a high-grade molybdenum RC sample (350 kg) achieved the expected rougher and cleaner performance for copper and molybdenum and proved a concentrate of >50% Mo could be generated without an overly specialised flowsheet. Molybdenum flotation can be achieved without specialised gas management (nitrogen for example), with only a few cleaning stages and with depression of copper by sodium sulphide.

In low feed grade cases, the molybdenum tail grade being reported is at the detection limit of the assay technique. All molybdenum flotation tests affected by these problems have significantly higher calculated head molybdenum values compared to assay head molybdenum values. Fortunately, when the molybdenum grade in a sample is in the range where its recovery contributes meaningfully to revenue, the tailings grade is normally above the detection limit and recovery is unaffected.

13.19.4 Gold Recovery

Gold recovery in all deposits is low compared to expectations. Typically, about 70% recovery of gold would be expected for a Chilean copper deposit, but the recoveries have consistently been 40 to 50%. The problem appears to be a combination of dominant chalcopyrite, high pyrite levels and the need to achieve 25% Cu in concentrate.

The chalcopyrite dominance and the 25% Cu concentrate target are two parts of the one problem. Pure chalcopyrite is 34.5% Cu, and this is not significantly far away from the 25% Cu concentrate grade target. Many copper deposits have a mix of copper sulphides, all of which have much higher copper grades than chalcopyrite (bornite 63.3% Cu, chalcocite 79.9% Cu). Even a modest proportion of these minerals in the sample can make it much easier to achieve 25% Cu in concentrate. Typically when these other minerals are present, pyrite (especially pyrite composites with copper sulphides) can be recovered with the concentrate to lower it to 25% Cu and to maximise recovery of all copper sulphides. This additional pyrite recovery may, of itself, raise the associated gold recovery.

Gold recovery has been seen to be related to the proportion of pyrite in the production feed, but not on a consistent basis. If pyrite is present in significant quantities compared to copper sulphides, then the pyrite would typically carry significant amounts of gold, mostly as ultrafine inclusions or possibly within the pyrite lattice. During copper concentrate cleaning, pyrite that has floated in the roughers is lost from the concentrate and this lowers the gold recovery. Achieving 25% Cu in concentrate is then a compromise between gold recovery and concentrate copper grade.

Although not forming part of the PEA, pyrite flotation has been conducted on samples from all deposits and gold distributions amongst products from each deposit are shown in Table 13.47.

Table 13.47 : Flotation Gold Deportment Across the Deposits

	Feed Gold Content (g/t)	Pyrite Mass (% of feed)	Au Deportment (%)		
			Cu Concentrate	Pyrite Rougher Concentrate	Rougher Tails
Productora	0.11	4.6	42	16	23
Alice (2 tests)	0.02, 0.04	1.4	74, 53	8, 8	8, 31
Cortadera OP (LCT)	0.15	3.0	45	24	32
Cortadera UG (LCT)	0.22	3.2	43	8.7	48

This analysis does not show a consistent relationship between recovery of gold and pyrite in feed. The least gold in pyrite occurs in both Alice and Cortadera UG, which are the freshest of the deposits and samples. The results may suggest that gold is easier to float when the production feed is from closer to the surface and has some alteration. Or it may simply suggest that the pyrite formation processes and temperatures were different for each deposit. Also note that the contribution of Alice to gold revenue would be minor due to its very low grade.

For the three deposits with the highest gold contents, there is some consistency between Productora and Cortadera OP, but Cortadera UG follows a different pattern. Notably, all three have about 44% Au recovery in copper concentrate but Cortadera UG has a very high loss of gold to the rougher tails (48%).

Rare phase searching (for pure or alloy gold phase occurrences) during QEMScan analysis of the two sets of Cortadera flotation test samples identified almost no free gold in the OP samples (only one detection). However, 14 detections were made in the UG samples. None were detected in the rougher tails, but with only 0.08 ppm Au in the sample and limited searching for occurrences, this is to be expected. The result confirms that free gold is much more prevalent in the UG sample than the OP and this explains why it is not being recovered with pyrite. Free gold was found to be associated with pyrite, chalcopyrite and silicates and 3 out of the 14 detections were fully liberated free gold particles.

What the analysis does not explain is why the gold is not floating with the copper concentrate and it does not explain the form of the gold in the UG deposit rougher tails.

It can be concluded that Cortadera OP has little free gold and that Productora has gold mainly in sulphides or easily floated free gold. In Alice the gold is mostly with the chalcopyrite and/or is present as easily floated free gold. Cortadera UG definitely has significant free gold, and the form of the gold lost to rougher tails warrants additional investigation.

13.19.5 Silver Recovery

It is not possible to accurately measure silver recovery with the current silver analysis methodology for rougher tails. All rougher tails analyses are below the detection limit of 1 g/t for the method currently used. As the feed grade is only 1 to 3 g/t the errors in calculating recovery are substantial, to the point of being meaningless. The silver grade in copper concentrate for all Cortadera testwork is equal to the payability limit of 30 g/t, so there is no doubt that silver is being recovered and concentrated.

Although silver recoveries have been recommended for use in mine planning, the accuracy of those values is probably ±100% at minimum. It is recommended that a more sensitive method for silver analysis be implemented for future testwork.

14 Mineral Resource Estimates

14.1 Productora

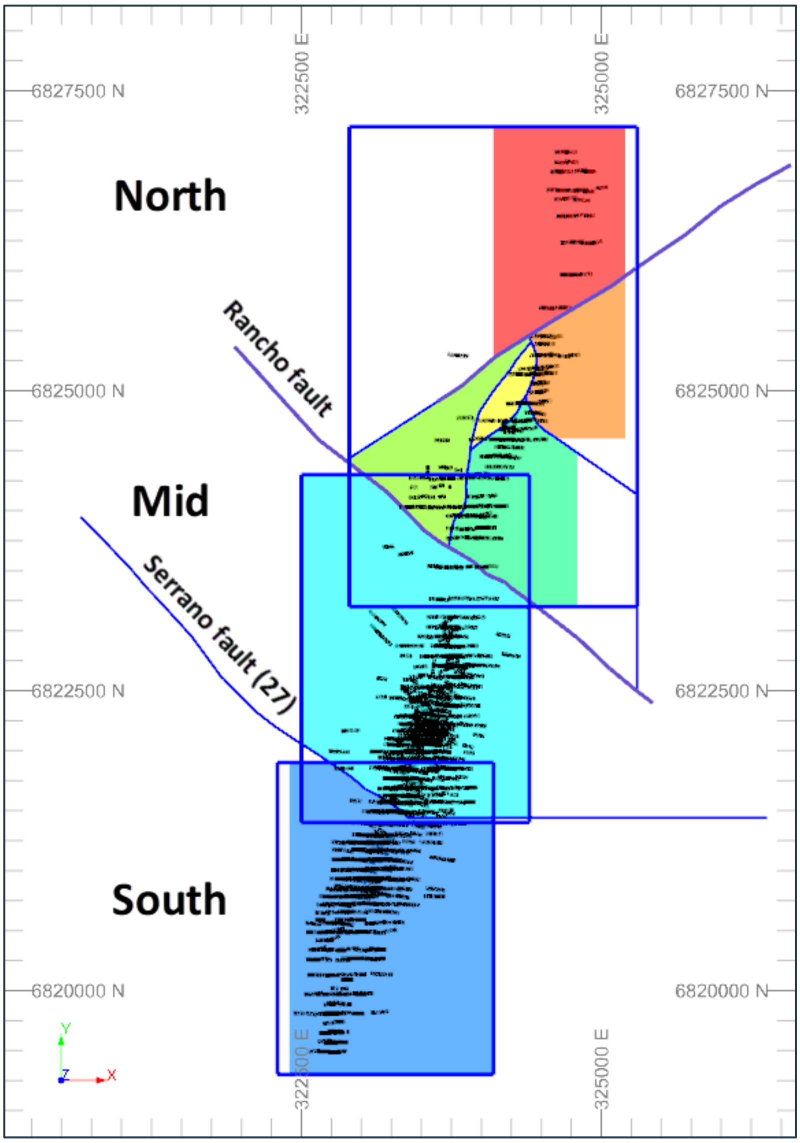
14.1.1 Introduction

The current Productora Mineral Resource was updated in February 2024 for inclusion in the combined Costa Fuego Mineral Resource Release (26 February 2024).

14.1.2 Model Area

The Productora model area is approximately 3 km east-west and 8 km 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 Areas, Drill Hole Traces and Fault Blocks Coloured



14.1.3 Database

14.1.3.1 Database Validation

The Productora database was validated through the following checks:

- Drill collars – comparison of planned vs. actual survey pick up coordinates, checked in 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
- Lithology – reviewed based on surrounding drill holes, with drill core or RC chips reviewed and logging updated where necessary
- Mineralisation intensity - logs completed for each drill hole, and compared with assay results
- 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.

14.1.3.2 Summary of Data Used in Estimate

All drilling data is stored in the HCH acQuire drill hole database.

Data table records consist of:

- Collar – Collar.csv
- Survey – Survey.csv
- Assays – Assay.csv
- Assays - CuSol.csv
- Assays - SIR06A.csv
- Assays - MEMS61_62_Ag.csv
- Geology – Lithology.csv
- Geology – Alteration.csv
- Geology – Weathering.csv
- Geology – Veining.csv
- Geology – Mineralisation.csv
- Geology – Colour.csv
- Geology - Regolith.csv
- Geology - Prod_Dens.csv
- Geometallurgy - Actual_CuSpec.csv
- Geometallurgy - ML_CuSpec.csv

- Geometallurgy - AE_Calc_Min.csv
- Geometallurgy - Geochem_Class.csv
- Geometallurgy - Min_Ratios.csv
- Geotechnical - FracFreq.csv

For the 2022 MRE and 2024 MRE minor errors and overlaps were adjusted, including the removal of duplicate values, the update of incorrect from and/or to values for certain intervals, and the deletion of intervals with 'null' values. All changes were made directly into the AcQuire database.

In addition to the changes above, all instances of commas were replaced with semi-colons in Lithology table 'Comments' field.

Holes drilled for metallurgical sampling were excluded from the database for estimation along with holes which were missing sampling information. The excluded hole list is in Table 14.1.

Table 14.1 : Drill Holes Excluded from the 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 to compile the estimate is listed in Table 14.2. Note that this table only includes drillholes that pass through the limits of the outermost estimation domain at Productora (i.e., excludes some Productora regional exploration drilling).

Table 14.2 : Drill Hole Included in Estimate Database

Drilling Method	Holes	Metres
RC	970	240 924
DD	38	13 158
RC with DD tail	104	44 206

14.1.4 Data Manipulation

Negative-grade values are used in the assay database table to indicate specific events or conditions. To ensure these assays are suitable for use in the mineral resource, edits were made to remove any non-numeric, zero and negative values. These checks were completed for the 2022 MRE and again for the 2024 MRE.

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'.

14.1.5 Geological and Mineralisation Interpretation

Multi-element mineralisation at Productora is developed mostly within a large intrusive hydrothermal breccia-dominated domain that trends in a north-northeasterly direction.

The host-breccia has been modelled from drill hole data over the following dimensions:

- Strike length: 7 900 m
- Depth from surface: 700 m
- Width: 850 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 (e.g. Habanero).

Deep drilling at Productora suggests that lodes thin at depth, with more ductile features 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.

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.

14.1.6 2013 Predictive Modelling

14.1.6.1 Introduction

A 3D predictive mineral system alteration model was completed for Productora in 2013. The predictive model comprises five components derived from a combination of alteration geochemical indices, lithogeochemical indices, and geological and geotechnical logging. Components of this model are detailed in the sections below.

14.1.6.2 3D Alteration Model

A comprehensive set of alteration indices were derived from the ~160 000 drill hole sample database (comprising: 33 element, 4 acid digest, ICP-OES data, plus gold by fire assay) through the application of multi-variant X-Y and ternary analysis of all available geochemistry data (Figure 14.2 and Figure 14.3). An alteration classification was applied to every drill hole sample interval and interpolated utilising Leapfrog to create a comprehensive 3D model of the alteration domains within the Productora Mineral System (Figure 14.4).

The Productora hydrothermal alteration model comprises seven distinct alteration assemblages/domains:

- K-Feldspar Domain: highly potassic proximal alteration spatially associated with tourmaline (± magnetite) breccias hosting the bulk of Productora copper sulphide mineralisation (magmatic-hydrothermal fluid source, probably related to the polyphase Ruta 5 intrusion)
- Sericite Domain: proximal-intermediate moderately potassic alteration, usually just outboard of the K-feldspar/tourmaline breccia domain. The sericite alteration domain also hosts significant copper mineralisation both as tourmaline breccia lodes and finely disseminated Habanero-style mineralisation (magmatic-hydrothermal fluid source)
- Sericite-Albite Domain: intermediate alteration, locally mineralised on discrete structures. The sericite-albite domain represents an important indicator of proximity to the mineralised potassic alteration domains, forming a large alteration halo up to 300 m radius around the outer extents of the mineralised envelope (magmatic-hydrothermal fluid source)
- Albite Domain: distal alteration signal for the Productora mineralising event (magmatic-hydrothermal fluid source)
- Sodic-Calcic Domain: regional sodic-calcic alteration assemblages, most likely associated with circulation/convection of non-magmatic resident formation fluids, with a possible evaporitic brine component. The sodic-calcic alteration appears to have been overprinted by the magmatic-hydrothermal alteration associated with copper mineralisation, and thus likely either predates the main copper mineralising event or represents a second "resident" fluid mixing with the magmatic hydrothermal fluids
- Magnetite-Amphibole Domain: magnetite-amphibole alteration seems to be a hotter, more focussed component of the basinal/resident hydrothermal fluid system. It is closely associated with strata-bound "manto-style replacement" magnetite/pyrite/chalcopyrite mineralisation as well as local semi-massive structurally-controlled magnetite deposits to the west of the main Productora mineral system. Copper mineralisation observed within the magnetite-amphibole alteration domain exhibits much closer morphological and mineralogical affinities with Candalaria-style IOCG mineralisation, with chalcopyrite in magnetite-rich breccias.
- "Kaolinite" Domain: The "kaolinite" alteration domain samples have been so named because they plot close to the kaolinite composition on a K:Al:Fe ternary plot, although these samples are not necessarily kaolinite altered. Rather, they reflect a Si/Al rich assemblage that seems to have been stripped of almost all mobile elements (Na/K/Cu/Fe, etc.). This sample group is completely stripped of all chalcophile elements (with copper at much less than average crustal abundance). The kaolinite classified samples exhibit significant overlap with the acid alteration and intrusive complex.

Figure 14.2 : Al:K:Fe Ternary Plot Demonstrating Alteration Index Classification

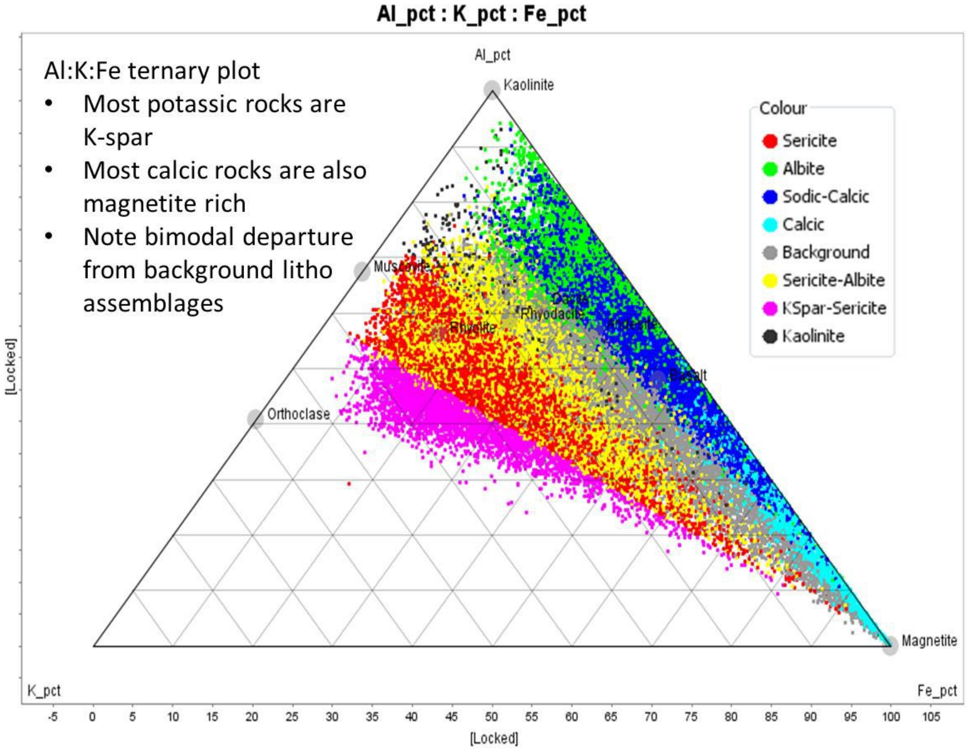


Figure 14.3 : Probability Plots Showing the Correlation of Copper and Molybdenum Grade with Each of the Alteration Domains

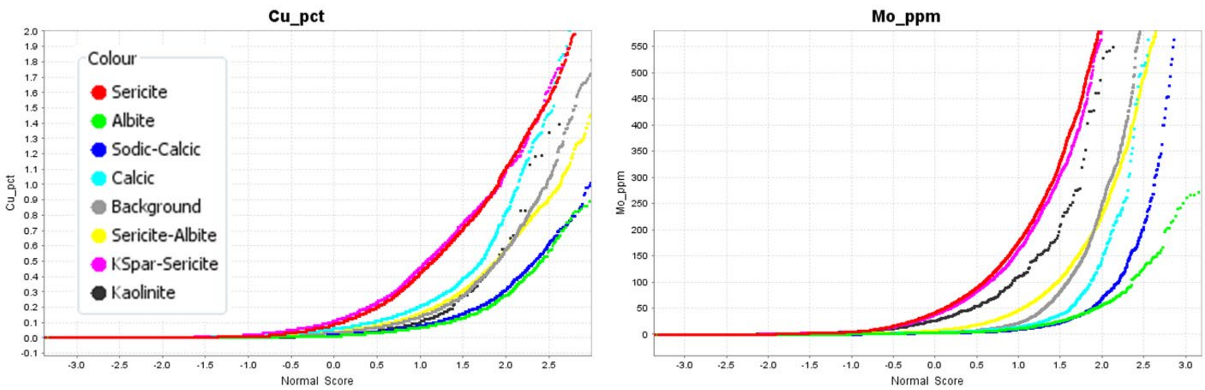
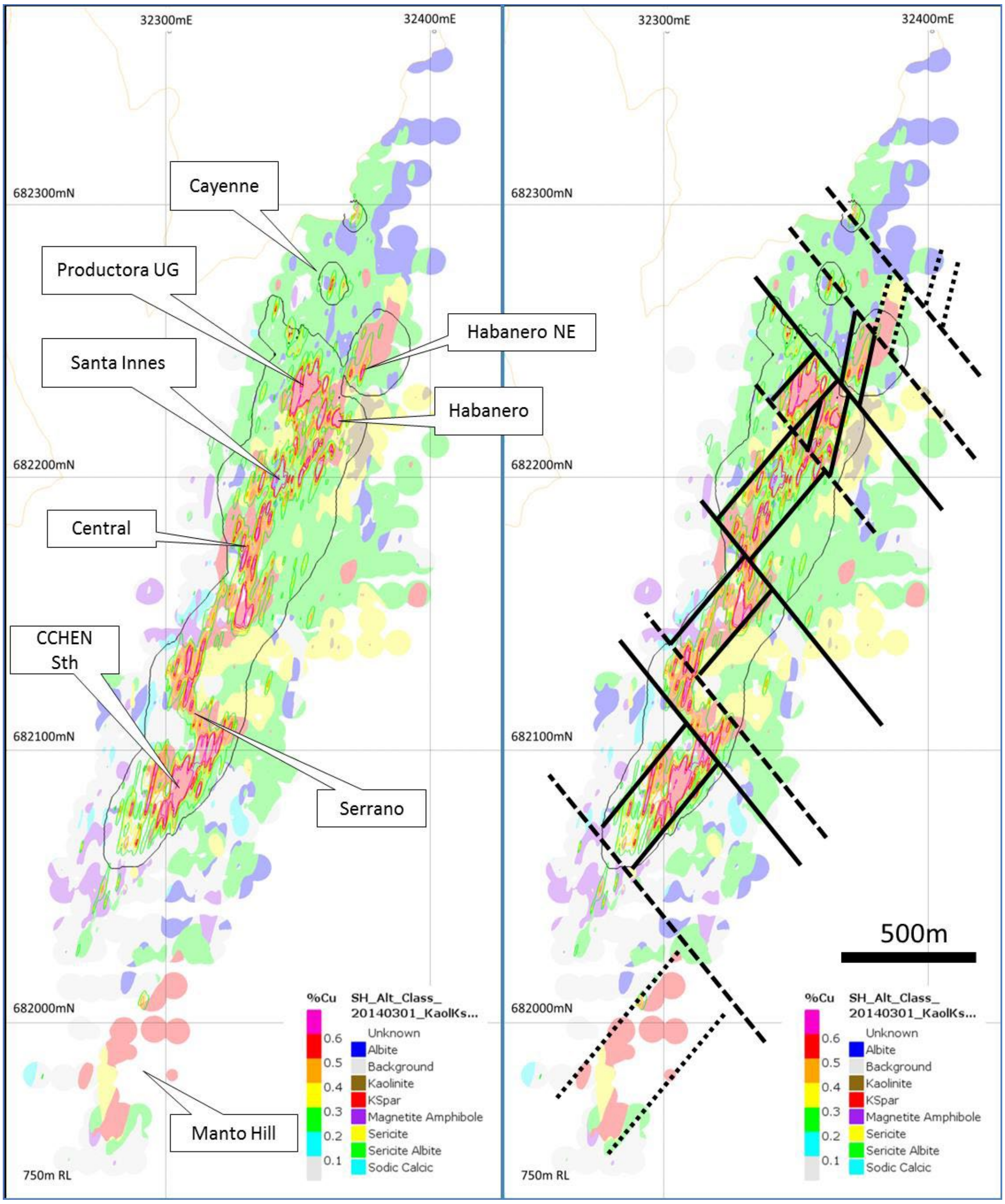


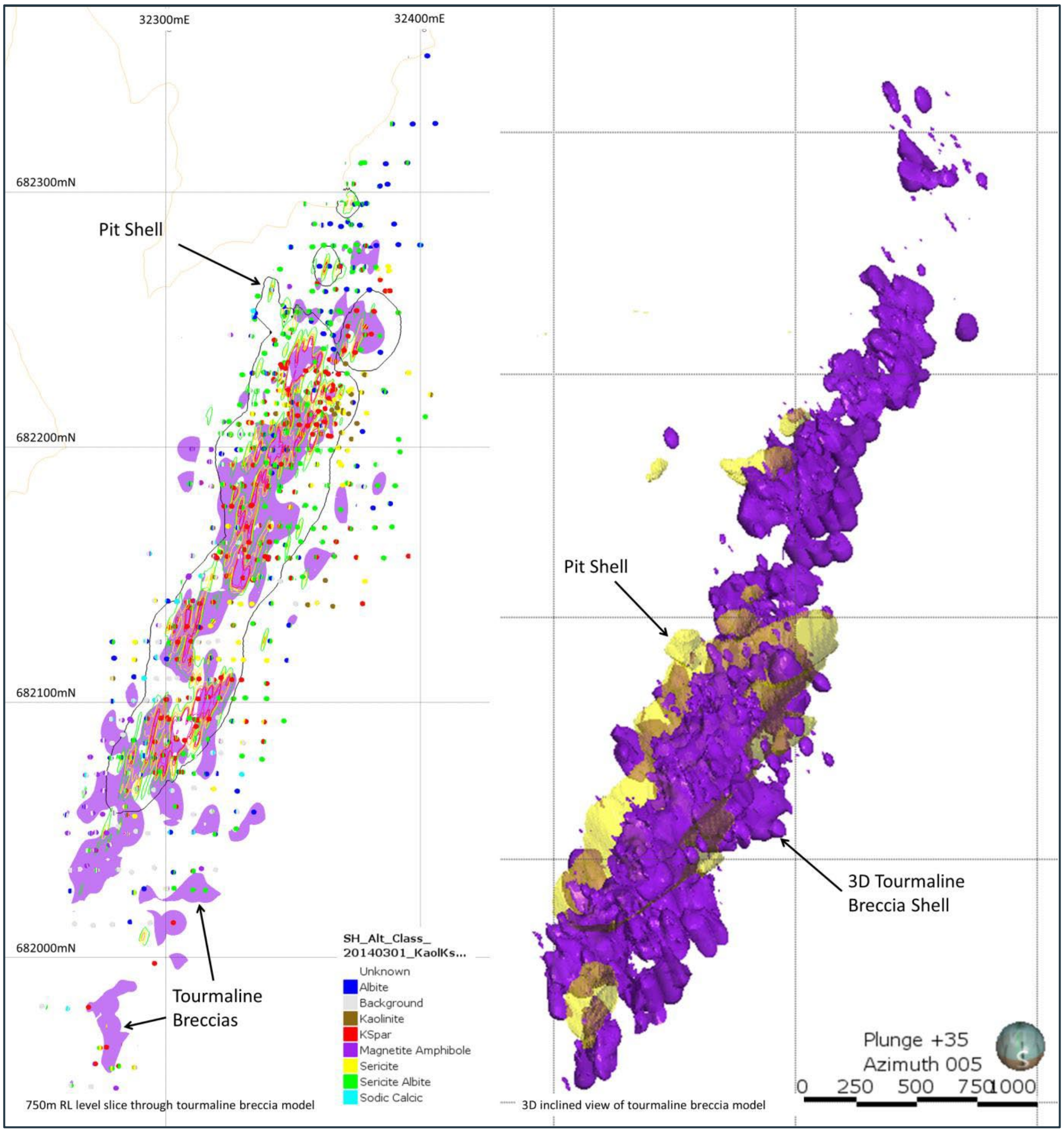
Figure 14.4 : 750 m RL Level Slice Through the 3D Alteration Model



14.1.6.3 3D Tourmaline Breccia Model

A preliminary 3D interpolation of the tourmaline breccia carapace was created in Leapfrog utilising a combination of geological and alteration logging, presence/absence of tourmaline and fracture frequency data (Figure 14.5).

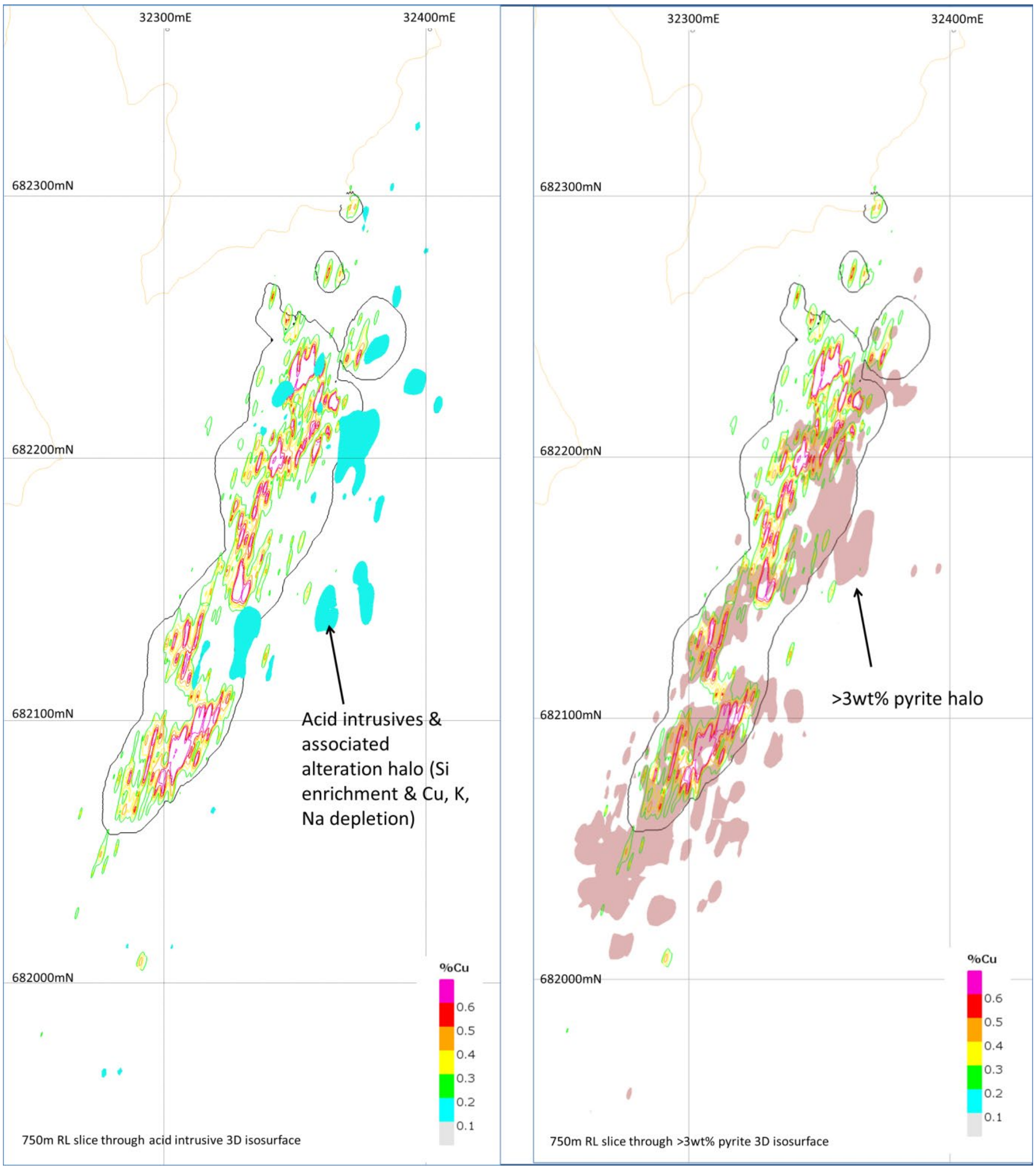
Figure 14.5 : 750 m RL Level Slice Through the 3D Tourmaline Breccia Model with Leapfrog 3D Copper Isosurface Contours (Left), 3D View of Breccia Model (Right)



14.1.6.4 Pyrite Enrichment Model

Weight percent pyrite was calculated for each sample interval from the multi-element geochemistry. The >3wt% 3D isosurface (Figure 14.6) demonstrates the relationship between pyrite enrichment, copper grades and the acid alteration domain. It is evident that in the northern part of the deposit there is not a 1:1 relationship between pyrite abundance and copper grades, however the greatest pyrite enrichment does appear to have a close (but inverse) spatial correlation with the acid assemblages.

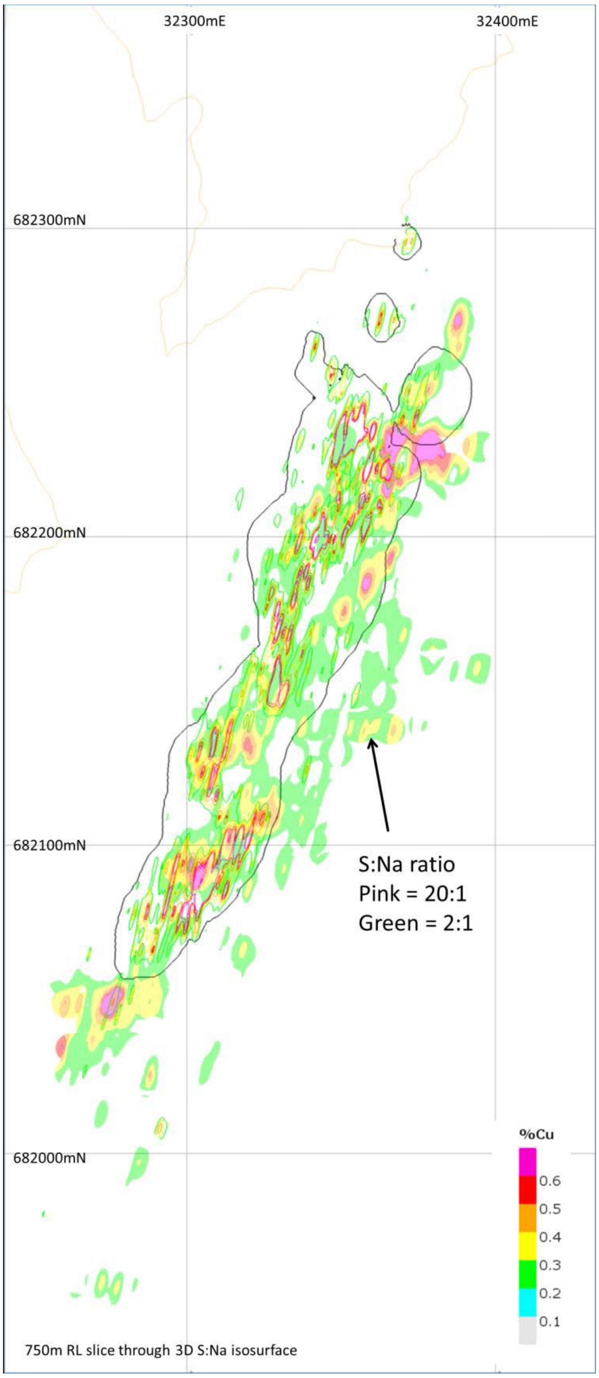
Figure 14.6 : 750 m RL Slices Through 3D Model of Acid Intrusives and Associated Acid Alteration Carapace (Left), 3D Isosurface of >3 wt% Pyrite Domain (Right)



14.1.6.5 S:Na Alteration Intensity Index

S:Na ratios (Figure 14.7) provide a workable proxy for a magmatic hydrothermal alteration intensity index, demonstrating a broad correlation between elevated S:Na ratios (>2:1) and elevated copper grades. Importantly, this indicator seems to highlight proximity to Habanero-type mineralisation on the margins of the acid intrusives.

Figure 14.7 : S:Na Ratio - 750 m RL Slice Through S:Na Ratio Model



14.1.7 2013 Predictive Modelling Files

2013 (Generation 1) files are listed below:

- Albite_20131213
- Background_20131213

- Kaolinite_20131213
- KSpar_20131213
- Magnetite Amphibole_20131213
- Sericite Albite_20131213
- Sericite_20131213
- Sodic Calcic_20131213
- Unknown_20131213.

2014 (Generation 2) files are listed below:

- pyrite_wt_pct3.0
- s_na2.0
- tourmaline-breccia_60.

14.1.8 Weathering Domain Modelling

The 2024 Productora Resource utilised an updated approach to the modelling of weathering surfaces, with 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.

The updated weathering model was produced using the following three-step process:

1) Create initial weathering 'domains' in Leapfrog using predominantly copper-sulphur ratio (Cu:S).
2) Run indicator estimates on 6 weathering related variable within each domain (using suitable variograms, search neighbourhoods, and appropriate anisotropy).

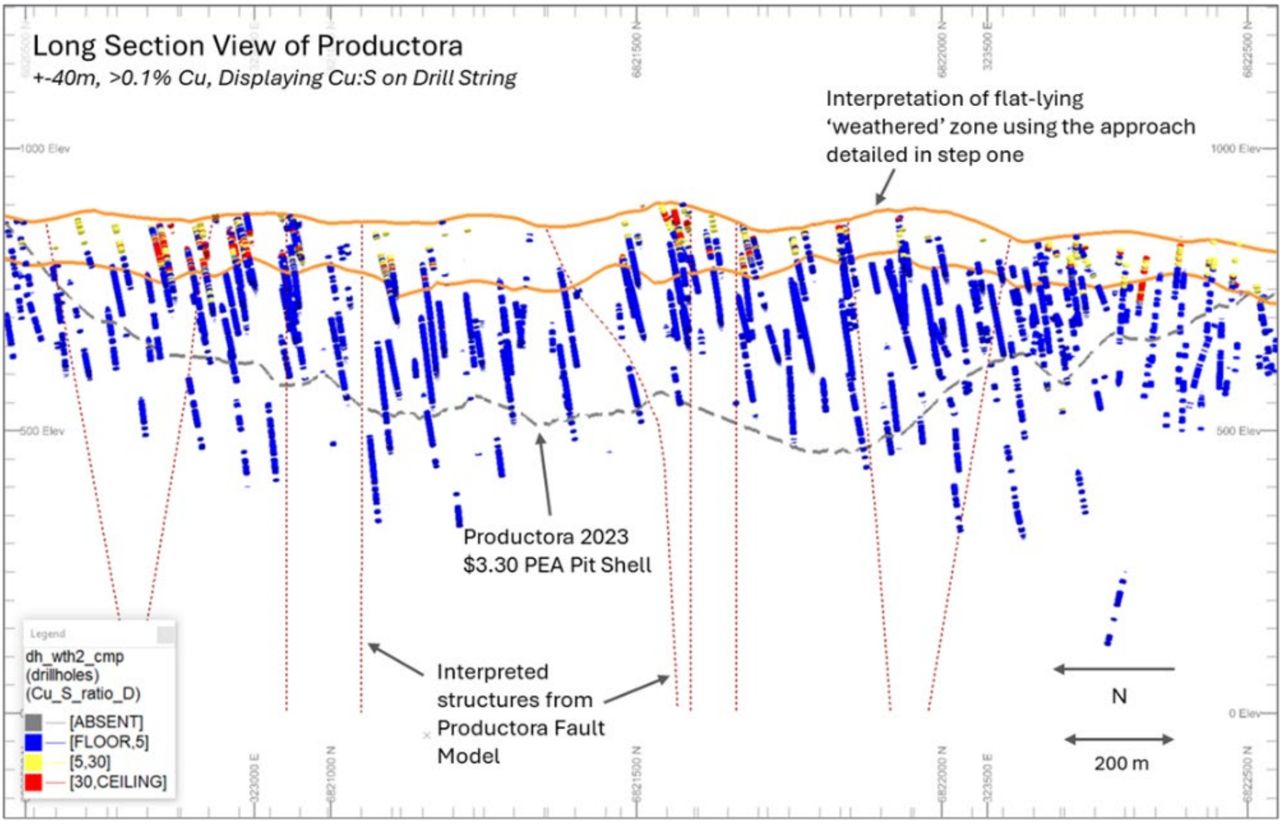
   Assign weightings to each of the indicators based on confidence in the data, and then determine the likely weathering type (oxide/transition/fresh etc) within a block (at 5 m (X) x 5 m (Y) x 5 m (Z) scale).

The initial weathering domains were created to 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, using predominantly Cu:S ratio (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). When composited to 10m, and filtered above 0.1% copper, it produces a reasonably tabular top-of-fresh-rock (TOFR) surface.

This approach mimics that used to create weathering domains previously at Productora, but which did not account for the structural complexity observed by geologists at Productora from surface mapping and drillhole logging. In Figure 14.8, 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.

Figure 14.8 : Initial Weathered and Fresh Domains at Productora



In total, six variables have been used to determine weathering classification (Table 14.3). 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 14.3 : 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.

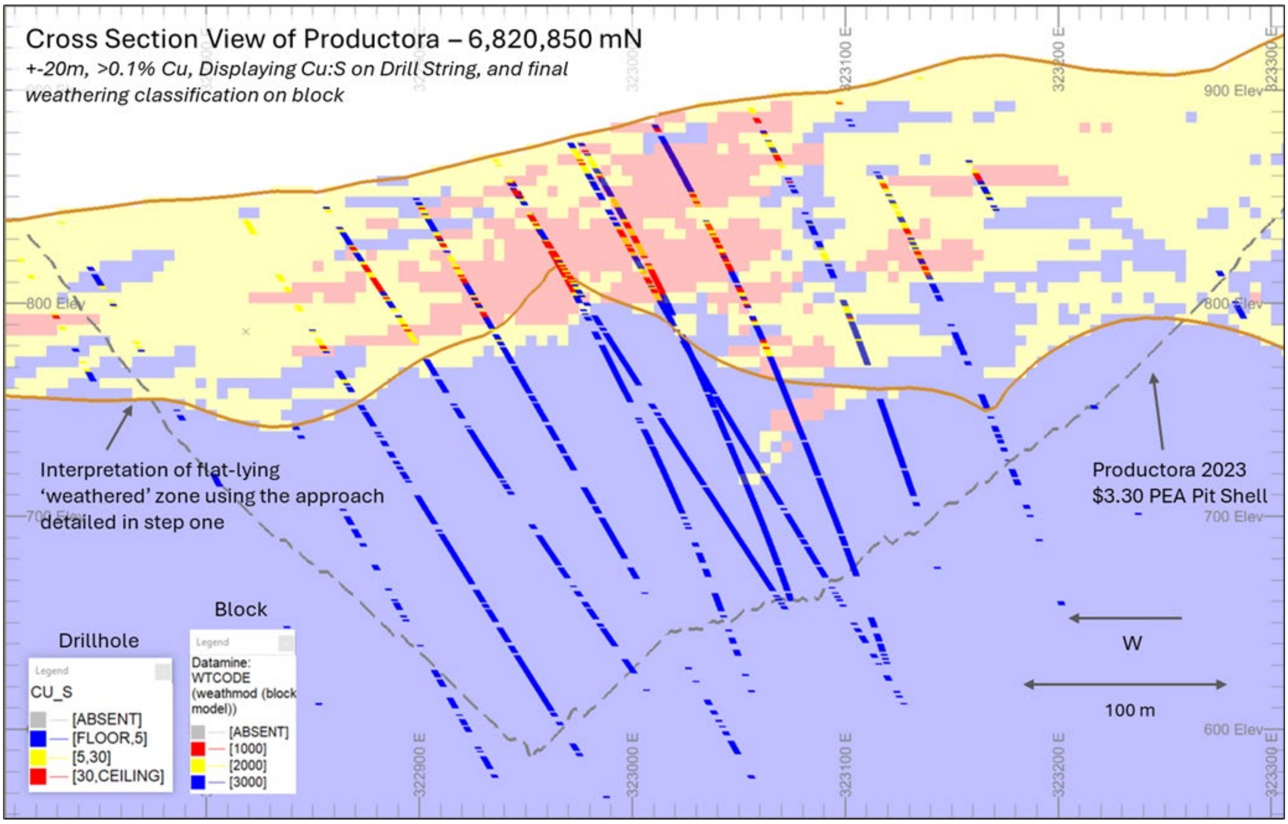
For the flat lying weathered zone, all six variables are estimated. For the vertical zone, there was no information available for ratio of soluble copper to total copper, so only five variables are estimated.

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 14.9 below.

Figure 14.9 : 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.



14.1.9 Data Flagging

Geology, mineralisation, alteration, structure, and weathering wireframes were used to flag the drill hole data and block model. The wireframes used are listed in Table 14.4 with an explanation of the fields and their coding in Table 14.5.

Table 14.4 : Wireframes Used for Flagging Drill Holes and Models

Wireframe	Zone Field	Type
2021_f27_limited	FAULTBLOCK	Surface
2021_faultblock_10	FAULTBLOCK	Solid
2021_faultblock_6	FAULTBLOCK	Solid
2021_faultblock_7	FAULTBLOCK	Solid
2021_faultblock_8	FAULTBLOCK	Solid
2021_faultblock_9	FAULTBLOCK	Solid
2023_mod_lim	MOD_LIM	Surface
2023_weathered	WEATH	Surface
2023_fresh	WEATH	Surface
interp_1_central_split	CENDOM	Solid
v2015_a_tou_brec_clean	TOU	Solid
v2015_fault_rancho	FAULTBLOCK	Surface
v2015_topo_clean	TOPO	Surface

Table 14.5 : Drill Holes and Models Coding

Field	Code	Description
FAULTBLOCK	40 000	South of Serrano fault
FAULTBLOCK	50 000	Between Serrano and Rancho faults
FAULTBLOCK	60 000	North of Rancho fault sub-domain
FAULTBLOCK	70 000	North of Rancho fault sub-domain
FAULTBLOCK	80 000	North of Rancho fault sub-domain
FAULTBLOCK	90 000	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	1 000	Weathered
WEATH	3 000	Fresh

14.1.10 Block Modelling

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 14.6.

Table 14.6 : Block Model Dimensions					
Dimension	Minimum	Maximum	Extent (m)	Block Size (m)	
				Parent	Minimum
Easting	321 850	325 050	2 800	5.0	5.0
Northing	6 819 000	6 827 400	8 400	20.0	5.0
Elevation	200	1 200	1 000	5.0	5.0

14.1.11 Dynamic Anisotropy Modelling

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 14.7.

Table 14.7 : Dynamic Anisotropy Trend Wireframes		
Wireframe	Zone Field	Code
DA_TREND_V5_02_STHOF27	FAULTBLOCK	40 000
DA_TREND_V5_02_NTHOF27	FAULTBLOCK	50 000
DA_TREND_V2_NTH_FB678_B	FAULTBLOCK	60 000
DA_TREND_V2_NTH_FB678_B	FAULTBLOCK	70 000
DA_TREND_V2_NTH_FB678_B	FAULTBLOCK	80 000
DA_TREND_V2_NTH_FB910_B	FAULTBLOCK	90 000
DA_TREND_V2_NTH_FB910_B	FAULTBLOCK	100 000

14.1.12 Categorical Domaining

14.1.12.1 Strategy

The 2022 Productora model update incorporated the use of geochemical associations to help define domains for estimation. The 2024 Productora model has further refined these associations.

The following observations were made regarding the wireframe interpretations from the 2014 and 2015 interpretation work:

- 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 large 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.

14.1.12.2 Categorical Domaining Data

Drill hole data was coded with binary indicator fields ('1' being above the grade/value specified, '0' being below) as described in Table 14.8. Various ratios were also calculated and applied as shown in Table 14.9.

A total of 18 indicators and 17 element ratios were tested along with the calculated silica.

Table 14.8 : 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 14.9 : 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

Table 14.9 : Ratio Calculation for Drill Holes	
Ratio Field	**Calculation of Ratio**
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
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*

14.1.12.3 Categorical Domaining Variography

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 are presented in Table 14.10 for the mid area, Table 14.11 for the FAULTBLOCK = 40000 (south area) and Table 14.12 for FAULTBLOCK = 60000 to FAULTBLOCK = 100000 (north area). The directions modelled represent the various orientations along the Productora corridor.

Table 14.10 : 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
							20		70		100
							10		20		60
4	COIND1	100	110	0	0.1	0.46	50	0.14	120	0.30	600
							35		135		280
							20		105		230
5	COIND2	100	110	0	0.1	0.36	20	0.20	30	0.34	200
							30		90		95
							20		60		90

Table 14.10 : 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
6	SIND	115	160	0	0.06	0.28	50	0.23	180	0.43	1 100
							50		80		380
							20		180		180
7	CALC_SI	110	100	0	0.15	0.30	60	0.30	70	0.25	800
							25		80		250
							15		55		250
8	R_P_CA	120	50	0	0.11	0.33	40	0.22	200	0.34	1 050
							40		350		560
							40		200		300
9	R_P_CA	120	90	0	0.11	0.31	40	0.24	200	0.34	1 050
							70		300		300
							30		140		380
10	R_K_AL	105	110	0	0.09	0.47	40	0.17	105	0.27	460
							40		150		260
							20		65		200
11	R_CU_S Ox/Tr	110	170	0	0.03	0.41	50	0.27	170	0.29	800
							15		200		200
							15		100		130
12	R_CU_S	110	110	0	0.11	0.21	50	0.34	60	0.34	800
							15		80		320
							15		40		280
13	AGIND03	120	120	0	0.22	0.16	30	0.62	150	-	-
							30		150		-
							15		40		-

Table 14.11 : Categorical Variogram Models – South Area (FAULTBLOCK = 40000)

VREF	Variable	Rotation (ZXZ)			Nugget	Structure 1		Structure 2		Structure 3	
					C0	C1	R1	C1	R1	C1	R1
21	CUIND05	110	115	0	0.12	0.46	70	0.17	160	0.25	600
							70		200		260
							60		100		100
22	CUIND15	110	115	0	0.12	0.40	20	0.29	35	0.19	120
							50		60		80
							15		20		20
23	MOIND	110	110	0	0.15	0.26	20	0.30	25	0.29	100
							20		45		110
							10		20		30
24	COIND1	100	110	0	0.1	0.46	50	0.14	120	0.30	600
							35		15		280

Table 14.11 : Categorical Variogram Models – South Area (FAULTBLOCK = 40000)

VREF	Variable	Rotation (ZXZ)			Nugget	Structure 1		Structure 2		Structure 3	
					C0	C1	R1	C1	R1	C1	R1
							20		105		230
25	COIND2	100	110	0	0.1	0.36	30	0.34	50	0.20	200
							30		60		110
							20		30		40
26	SIND	115	160	0	0.06	0.30	80	0.37	1 200	0.27	1 200
							80		900		900
							40		150		150
27	CALC_SI	110	100	0	0.12	0.30	60	0.29	75	0.29	800
							25		40		150
							15		60		325
28	R_P_CA	120	50	0	0.11	0.26	40	0.36	100	0.27	670
							15		135		230
							15		65		180
29	R_P_CA	120	90	0	0.11	0.47	40	0.15	100	0.27	670
							15		230		230
							25		100		420
30	R_K_AL	105	110	0	0.09	0.47	40	0.14	140	0.30	600
							70		250		250
							20		190		200
31	R_CU_S Ox/Tr	110	170	0	0.03	0.20	50	0.27	80	0.50	200
							40		50		150
							10		20		30
32	R_CU_S	110	110	0	0.03	0.35	90	0.22	180	0.40	900
							30		140		240
							30		140		240
33	AGIND03	130	130	0	0.13	0.23	160	0.64	200	-	-
							160		200		-
							40		120		-

Table 14.12 : Categorical Variogram Models – South Area (FAULTBLOCK = 40000)

VREF	Variable	Rotation (ZXZ)			Nugget	Structure 1		Structure 2		Structure 3	
					C0	C1	R1	C1	R1	C1	R1
21	CUIND05	110	115	0	0.12	0.46	70	0.17	160	0.25	600
							70		200		260
							60		100		100
22	CUIND15	110	115	0	0.12	0.40	20	0.29	35	0.19	120
							50		60		80
							15		20		20
23	MOIND	110	110	0	0.15	0.26	20	0.30	25	0.29	100

Table 14.12 : Categorical Variogram Models – South Area (FAULTBLOCK = 40000)											
VREF	Variable	Rotation (ZXZ)			Nugget	Structure 1		Structure 2		Structure 3	
					C0	C1	R1	C1	R1	C1	R1
							20		45		110
							10		20		30
24	COIND1	100	110	0	0.1	0.46	50	0.14	120	0.30	600
							35		15		280
							20		105		230
25	COIND2	100	110	0	0.1	0.36	30	0.34	50	0.20	200
							30		60		110
							20		30		40
26	SIND	115	160	0	0.06	0.30	80	0.37	1 200	0.27	1 200
							80		900		900
							40		150		150
27	CALC_SI	110	100	0	0.12	0.30	60	0.29	75	0.29	800
							25		40		150
							15		60		325
28	R_P_CA	120	50	0	0.11	0.26	40	0.36	100	0.27	670
							15		135		230
							15		65		180
29	R_P_CA	120	90	0	0.11	0.47	40	0.15	100	0.27	670
							15		230		230
							25		100		420
30	R_K_AL	105	110	0	0.09	0.47	40	0.14	140	0.30	600
							70		250		250
							20		190		200
31	R_CU_S Ox/Tr	110	170	0	0.03	0.20	50	0.27	80	0.50	200
							40		50		150
							10		20		30
32	R_CU_S	110	110	0	0.03	0.35	90	0.22	180	0.40	900
							30		140		240
							30		140		240
33	AGIND03	130	130	0	0.13	0.23	160	0.64	200	-	-
							160		200		-
							40		120		-

Table 14.13 : Categorical Variogram Models – North Area (FAULTBLOCK 60000 to 100000)											
VREF	Variable	Rotation (ZXZ)			Nugget	Structure 1		Structure 2		Structure 3	
					C0	C1	R1	C1	R1	C1	R1
41	CUIND05	115	15	0	0.22	0.21	40	0.21	130	0.41	150
							40		60		90
							10		40		60

Table 14.13 : Categorical Variogram Models – North Area (FAULTBLOCK 60000 to 100000)											
VREF	**Variable**	**Rotation (ZXZ)**			**Nugget**	**Structure 1**		**Structure 2**		**Structure 3**	
					C0	**C1**	**R1**	**C1**	**R1**	**C1**	**R1**
42		115	15	0	0.17	0.19	50	0.38	200	0.32	270
	CUIND1						50		50		110
							20		20		30
43		115	15	0	0.11	0.27	40	0.21	130	0.41	240
	CUIND3						40		60		150
							10		40		40
		115	40	0	0.11	0.29	40		80		170
44	MOIND						40	0.36	60	0.28	150
							10		30		40
45		115	15	0	0.07	0.30	20	0.23	80	0.33	180
	COIND1						20		60		90
							10		30		40
46		115	15	0	0.14	0.30	20	0.23	60	0.33	180
	COIND2						20		60		90
							10		20		40
47		115	15	0	0.14	0.30	20	0.23	60	0.33	180
	SIND						20		60		90
							10		20		40
48		115	15	0	0.14	0.30	20	0.23	60	0.33	180
	CALC_SI						20		60		90
							10		20		40
49		115	90	0	0.14	0.36	40	0.31	90	0.22	380
	R_P_CA						15		60		260
							10		40		100
50		115	40	0	0.11	0.29	40	0.36	80	0.28	270
	KALIND						40		60		150
							10		30		40
51		115	15	0	0.07	0.27	40	0.21	130	0.41	240
	CUSIND						40		60		150
							10		40		40
52		115	15	0	0.11	0.21	40	0.21	160	0.41	150
	CUSIND2						40		60		90
							10		40		60
53		125	100	0	0.17	0.27	40	0.36	100	0.26	550
	R_CA_NA						25		55		300
							10		40		60
54		110	115	0	0.11	0.21	40	0.21	130	0.41	150
	CUIND05a						40		60		90
							10		40		60

Table 14.13 : Categorical Variogram Models – North Area (FAULTBLOCK 60000 to 100000)											
VREF	Variable	Rotation (ZXZ)			Nugget	Structure 1		Structure 2		Structure 3	
					C0	C1	R1	C1	R1	C1	R1
55	INDCOMB	110	115	0	0.17	0.21	40	0.21	130	0.41	150
							40		60		90
							10		40		60

14.1.12.4 Categorical Domaining Estimation

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 into the three fault block areas, as defined above.

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 each area and variable estimated is contained in Table 14.14, Table 14.15 and Table 14.16 with the combination of search and variogram parameters listed in Table 14.17, Table 14.18 and Table 14.19.

Table 14.14 : 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 14.15 : Search Strategy for Categorical Estimation for South Area (FAULTBLOCK = 40000)												
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	
21	Dynamic			50	50	20	4	6	12	6	12	3
26	130	170	0	50	50	20	4	6	12	6	12	3
28	110	50	0	50	50	20	4	6	12	6	12	3
29	110	90	0	50	50	20	4	6	12	6	12	3
31	110	170	0	50	50	20	4	6	12	6	12	3

Table 14.16 : Search Strategy for Categorical Estimation for North Area (FAULTBLOCK = 60000 to 100000)												
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	
41	Dynamic			150	150	60	-	6	12	-	-	3
42	110	150	150	150	150	60	-	6	12	-	-	3
43	90	150	150	150	150	60	-	6	12	-	-	3

Table 14.17 : 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

Table 14.18: Combination of Variogram and Search for Categorical Estimation South Area (FAULTBLOCK = 40000)				
Variable	Model Variable	SREF	VREF	Description
CUIND05	CUIND05	21	21	Using dynamic search
CUIND3	CUIND3	21	22	Using dynamic search
MOIND	MOIND	21	23	Using dynamic search
COIND1	COIND1	21	24	Using dynamic search
COIND2	COIND2	21	25	Using dynamic search
SIND	SIND	26	26	Using static search
CALC_SI	CALC_SI	21	27	Using dynamic search
CAPIND	CAPIND	28	28	Using static search
CAPIND	CAPIND2	29	29	Using static search
KALIND	KALIND	21	30	Using dynamic search
CUSIND	CUSIND	31	31	Using static search
CUSIND2	CUSIND2	21	32	Using dynamic search
AGIND03	AGIND03	22	33	Using dynamic search

Table 14.19 : Combination of Variogram and Search for Categorical Estimation North Area (FAULTBLOCK 60000 to 100000)

Variable	Model Variable	SREF	VREF	Description
CUIND05	CUIND05	41	41	Using dynamic search
CUIND1	CUIND1	41	41	Using dynamic search
CUIND3	CUIND3	41	41	Using dynamic search
MOIND	MOIND	41	41	Using dynamic search
COIND1	COIND1	41	47	Using dynamic search
COIND2	COIND2	41	47	Using dynamic search
SIND	SIND	41	47	Using dynamic search
CALC_SI	CALC_SI	41	47	Using dynamic search
R_P_CA	R_P_CA	41	46	Using dynamic search
KALIND	KALIND	41	44	Using dynamic search
CUSIND	CUSIND	42	43	Using static search
CUSIND2	CUSIND2	41	41	Using dynamic search
R_CA_NA	R_CA_NA	41	45	Using dynamic search
CUIND05	CUIND05A	43	41	Using static search
IND_COMB	IND_COMB	41	41	Using dynamic search
AGIND03	AGIND03	41	41	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.

14.1.12.5 Categorical Domaining Interpretation

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 high- and low-grade copper sub-domains.

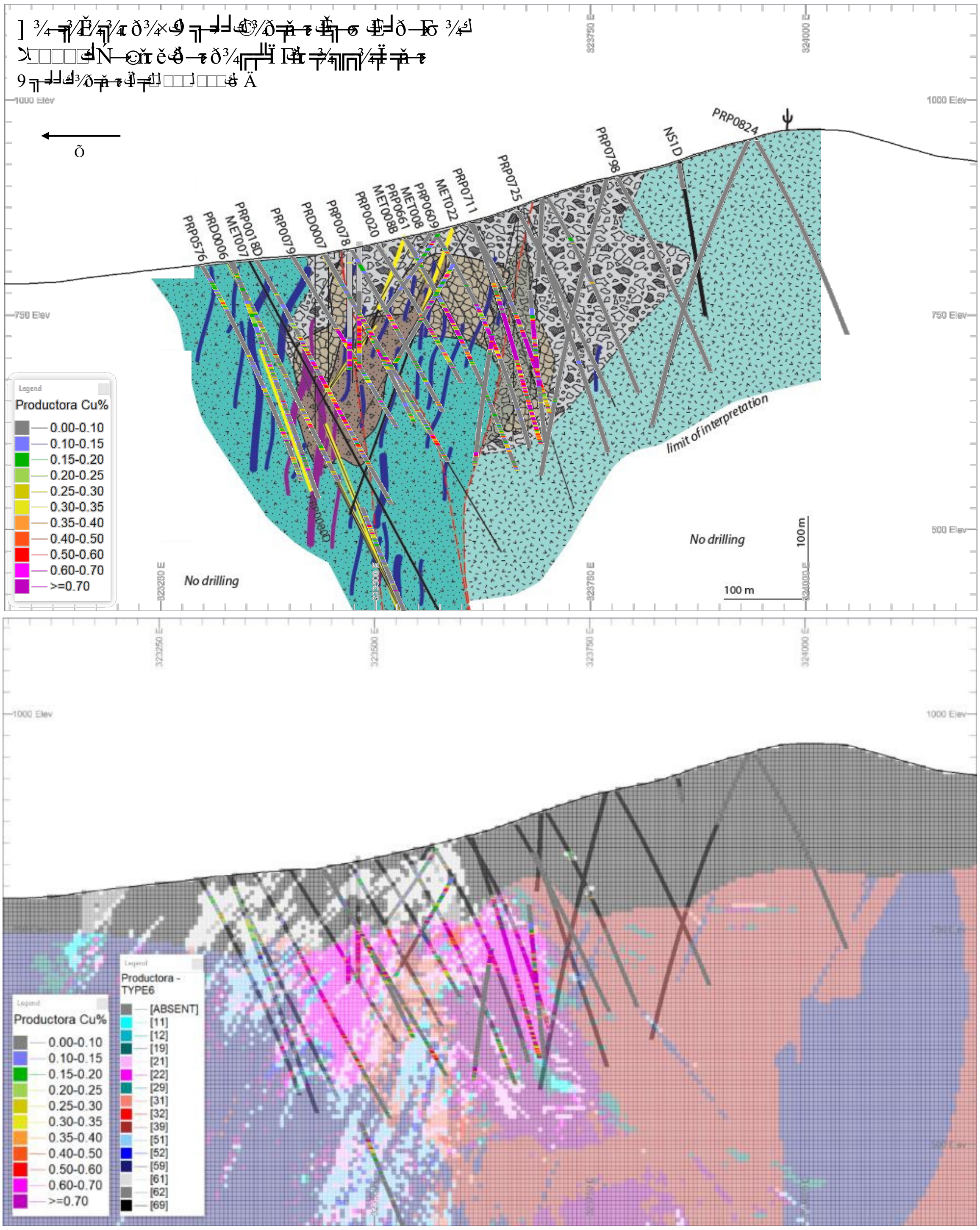
14.1.12.6 Copper Domains

Model 'Types' were coded based on a combination of indicator and ratio as shown in Table 14.20 and compared to the Escolme interpretation. An example east-west section at 6 822 215 mN is shown in Figure 14.19.

Table 14.20 : 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 14.10: Comparison of Conceptual Interpretation (Top), to domain coded Block Model (Bottom) on east-west Section 6822215 mN



In Figure 14.10, 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 – light-pink, and Stage 2 – pink) match well across both interpretations.

14.1.12.7 Soluble Copper Domains

The model was coded with a combination of indicators for soluble copper domains as:

- CUSOL = 100 Soluble copper near surface where material is coded as oxide or transitional and CUSIND >= 0.5
- CUSOL = 200 Soluble copper in faults where material is coded as fresh and CUSIND2 >= 0.5.

14.1.12.8 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 on a combination of indicator and ratio as shown in Table 14.21.

Table 14.21 : 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

14.1.12.9 Cobalt Domains

It is understood the early pyrite alteration is associated with elevated cobalt. The model was coded with a combination of the pyrite wireframe and indicators for cobalt domains as:

- CODOM = 1 Within pyrite wireframe and COIND1 >= 0.5
- CODOM = 2 Within pyrite wireframe and COIND1 < 0.5 or absent
- CODOM = 3 Outside pyrite wireframe and COIND1 >= 0.5
- CODOM = 4 Outside pyrite wireframe and COIND1 < 0.5 or absent.

14.1.12.10 Calcium Domains

Calcium and phosphorus assays broadly outline a wedge of high calcium material between Productora and Alice which is generally very poorly mineralised. The model was coded with a combination of the two Ca/P ratio indicators as:

- CADOM = 5 where CAPIND >= 0.2 AND CAPIND2 >= 0.2

- CADOM = 9 remainder of material.

14.1.12.11 Molybdenum Domains

The model was coded with a combination of indicators and ratios for molybdenum domains as shown in Table 14.22.

Table 14.22 : 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
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

14.1.12.12 Iron and Sulphur Domains

A combination of coded lithology, pyrite and sulphur interpolants, and indicator estimate results were used to generate iron and sulphur domains as shown in Table 14.23.

Table 14.23 : Model Coding for Iron and Sulphur Domaining

Description	Condition	FESDOM
Facies 3B(1,2), Stage 2	Outside pyrite and outside sulphur wireframes and SIND >= 0.30 and MOIND >= 0.10	1 500 000
Facies 3B(1,2), Stage 2	Within pyrite and outside sulphur wireframes and SIND >= 0.30 and MOIND >= 0.10	1 500 001
Facies 3B(1,2), Stage 2	Outside pyrite and within sulphur wireframes and SIND >= 0.30 and MOIND >= 0.10	1 500 010
Facies 3B(1,2), Stage 2	Within pyrite and within sulphur wireframes and SIND >= 0.30 and MOIND >= 0.10	1 500 011
Rhyolite/Rhyodacite lapilli tuff	Outside pyrite and outside sulphur wireframes and CAPIND >= 0.20 and CAPIND2 >= 0.20	5 000 000
Rhyolite/Rhyodacite lapilli tuff LG	Within pyrite and outside sulphur wireframes and CAPIND >= 0.20 and CAPIND2 >= 0.20	5 000 001
Rhyolite/Rhyodacite lapilli tuff LG	Outside pyrite and within sulphur wireframes and CAPIND >= 0.20 and CAPIND2 >= 0.20	5 000 010
Rhyolite/Rhyodacite lapilli tuff LG	Within pyrite and within sulphur wireframes and CAPIND >= 0.20 and CAPIND2 >= 0.20	5 000 011
Background	Outside pyrite and outside sulphur wireframes and (CAPIND < 0.20 or CAPIND2 < 0.20)	9 900 000
Background	Outside pyrite and outside sulphur wireframes and (SIND < 0.30 or MOIND < 0.10)	9 900 000
Background	Within pyrite and outside sulphur wireframes and (CAPIND < 0.20 or CAPIND2 < 0.20)	9 900 001
Background	Within pyrite and outside sulphur wireframes and (SIND < 0.30 or MOIND < 0.10)	9 900 001
Background	Outside pyrite and within sulphur wireframes and (CAPIND < 0.20 or CAPIND2 < 0.20)	9 900 010
Background	Outside pyrite and within sulphur wireframes and (SIND < 0.30 or MOIND < 0.10)	9 900 010
Background	Within pyrite and within sulphur wireframes and (CAPIND < 0.20 or CAPIND2 < 0.20)	9 90 0011
Background	Within pyrite and within sulphur wireframes and (SIND < 0.30 or MOIND < 0.10)	9 900 011

14.1.12.13 Potassium and Aluminium Domains

A combination of coded lithology and indicator estimate results were used to generate potassium and aluminium domains as shown in Table 14.24.

Table 14.24 : Model Coding for Potassium and Aluminium Domaining

Description	Condition	KALDOM
Facies 3B(1,2) LG	KALIND >= 0.1 and KALIND < 0.5 and SIND >= 0.30 and MOIND >= 0.10 and CUIND05 >= 0.60	111
Facies 3B(1,2) HG	KALIND >= 0.1 and KALIND < 0.5 and SIND >= 0.30 and MOIND >= 0.10 and CUIND3 >= 0.35	112
Facies 3B(1,2) VLG	KALIND >= 0.1 and KALIND < 0.5 and SIND >= 0.30 and MOIND >= 0.10 and CUIND05 < 0.60	119
Stage 2 LG	KALIND >= 0.1 and KALIND < 0.5 and SIND >= 0.30 and MOIND >= 0.10 and CUIND05 >= 0.60 and COIND2 >= 0.1	121
Stage 2 HG	KALIND >= 0.1 and KALIND < 0.5 and SIND >= 0.30 and MOIND >= 0.10 and CUIND3 >= 0.35 and COIND2 >= 0.1	122
Stage 2 VLG	KALIND >= 0.1 and KALIND < 0.5 and SIND >= 0.30 and MOIND >= 0.10 and CUIND05 < 0.60 and COIND2 >= 0.1	129
Rhyolite/Rhyodacite lapilli tuff LG	KALIND >= 0.1 and KALIND < 0.5 and CAPIND >= 0.20 and CAPIND2 >= 0.20 and CUIND05 >= 0.60	131
Rhyolite/Rhyodacite lapilli tuff HG	KALIND >= 0.1 and KALIND < 0.5 and CAPIND >= 0.20 and CAPIND2 >= 0.20 and CUIND3 >= 0.35	132
Rhyolite/Rhyodacite lapilli tuff VLG	KALIND >= 0.1 and KALIND < 0.5 and CAPIND >= 0.20 and CAPIND2 >= 0.20 and CUIND05 < 0.60	139
Background LG	KALIND >= 0.1 and KALIND < 0.5 and Not previously coded and CUIND05 >= 0.60	151
Background VLG	KALIND >= 0.1 and KALIND < 0.5 and Not previously coded and CUIND05 < 0.60	159
Stage 2 LG	KALIND >= 0.5 and SIND >= 0.30 and MOIND >= 0.10 and CUIND05 >= 0.60 and COIND2 >= 0.1	221
Stage 2 HG	KALIND >= 0.5 and SIND >= 0.30 and MOIND >= 0.10 and CUIND3 >= 0.35 and COIND2 >= 0.1	222
Stage 2 VLG	KALIND >= 0.5 and SIND >= 0.30 and MOIND >= 0.10 and CUIND05 < 0.60 and COIND2 >= 0.1	229
Rhyolite/Rhyodacite lapilli tuff LG	KALIND >= 0.5 and CAPIND >= 0.20 and CAPIND2 >= 0.20 and CUIND05 >= 0.60	231
Rhyolite/Rhyodacite lapilli tuff HG	KALIND >= 0.5 and CAPIND >= 0.20 and CAPIND2 >= 0.20 and CUIND3 >= 0.35	232
Rhyolite/Rhyodacite lapilli tuff VLG	KALIND >= 0.5 and CAPIND >= 0.20 and CAPIND2 >= 0.20 and CUIND05 < 0.60	239
Background LG	KALIND >= 0.5 and Not previously coded and CUIND05 >= 0.60	251
Background VLG	KALIND >= 0.5 and Not previously coded and CUIND05 < 0.60	259
Facies 3B(1,2) LG	KALIND < 0.1 and SIND >= 0.30 and MOIND >= 0.10 and CUIND05 >= 0.60	911
Facies 3B(1,2) HG	KALIND < 0.1 and SIND >= 0.30 and MOIND >= 0.10 and CUIND3 >= 0.35	912
Facies 3B(1,2) VLG	KALIND < 0.1 and SIND >= 0.30 and MOIND >= 0.10 and CUIND05 < 0.60	919
Stage 2 LG	KALIND < 0.1 and SIND >= 0.30 and MOIND >= 0.10 and CUIND05 >= 0.60 and COIND2 >= 0.1	921

Table 14.24 : Model Coding for Potassium and Aluminium Domaining		
Description	**Condition**	**KALDOM**
Stage 2 HG	KALIND < 0.1 and SIND >= 0.30 and MOIND >= 0.10 and CUIND3 >= 0.35 and COIND2 >= 0.1	922
Stage 2 VLG	KALIND < 0.1 and SIND >= 0.30 and MOIND >= 0.10 and CUIND05 < 0.60 and COIND2 >= 0.1	929
Rhyolite/Rhyodacite lapilli tuff LG	KALIND < 0.1 and CAPIND >= 0.20 and CAPIND2 >= 0.20 and CUIND05 >= 0.60	931
Rhyolite/Rhyodacite lapilli tuff HG	KALIND < 0.1 and CAPIND >= 0.20 and CAPIND2 >= 0.20 and CUIND3 >= 0.35	932
Rhyolite/Rhyodacite lapilli tuff VLG	KALIND < 0.1 and CAPIND >= 0.20 and CAPIND2 >= 0.20 and CUIND05 < 0.60	939
Background LG	KALIND < 0.1 and Not previously coded and CUIND05 >= 0.60	951
Background HG	KALIND < 0.1 and Not previously coded and CUIND3 >= 0.35	952
Background VLG	KALIND < 0.1 and Not previously coded and CUIND05 < 0.60	959

14.1.12.14 Drill Hole Coding

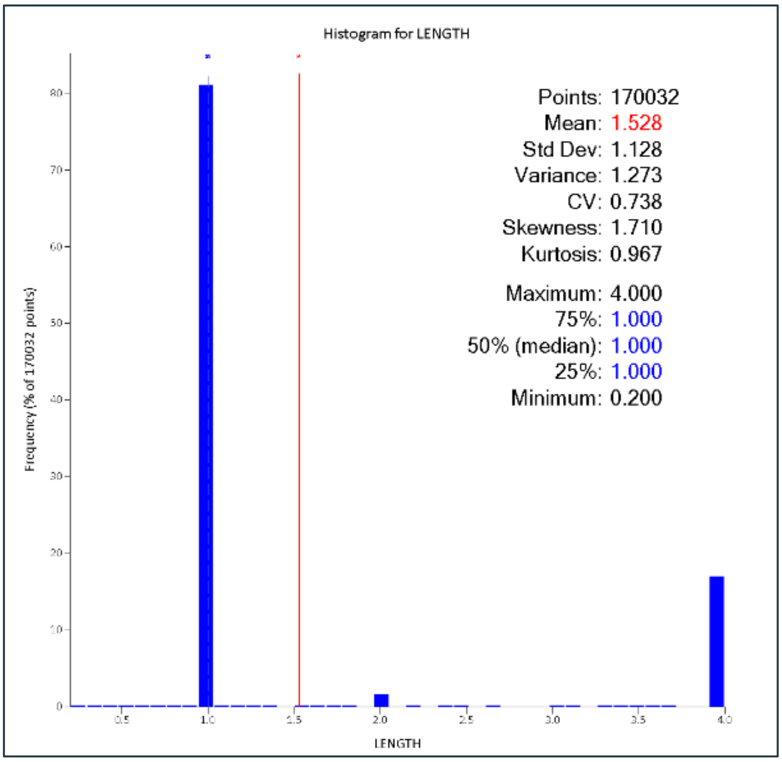
Block model domains were back-flagged onto the drill holes to ensure consistency between the model and the data used for grade estimation.

14.1.13 Compositing

82% of the drill sampling at Productora was at 1 m intervals (Figure 14.11). This proportion increased to 97% in potentially economic mineralised domains where copper was greater than 0.1%. 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.

The compositing process used the relevant domain and weathering as a boundary to ensure no composites were created across domains.

Figure 14.11 : Productora Sample Lengths within Model Limit Area



14.1.14 Statistical Analysis

14.1.14.1 Introduction

Statistical analysis of copper, gold, molybdenum, silver, cobalt, calcium, potassium and aluminium 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.

14.1.14.2 Top Cutting

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 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.

For Productora, due to the comprehensive domaining applied, only 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 key elements in Table 14.25 to Table 14.27 below.

Table 14.25 : Copper and Gold Statistics

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%	-10.5%

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 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%	-16.4%
62	OxFlat	HG Au	1 935	0.001	3.07	0.09	1.64	1.50	5	0.09	1.32	-2.3%	-19.6%
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%	-15.5%
61	OxFlat	LG Au	4 648	0.001	14.15	0.05	4.42	1.50	8	0.05	1.99	-6.7%	-55.1%
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%	-10.0%
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	-24.6%	-79.4%
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%

Table 14.26 : Molybdenum Statistics

Domain				Raw					Top-cut	%Diff Raw to Top-cut			
Code	Lith	Min	Number	Min	Max	Mean	CV	Value	Num cut	Mean	CV	Mean (%)	CV (%)
111	F3B(1,2) LG Cu	HG Mo	1 531	0.5	4 330	127	1.84	2 700	1	126	1.70	-0.8%	-6.3%
112	F3B(1,2) HG Cu	HG Mo	1 011	0.5	6 580	245	1.93	7 000	-	245	1.93	0.0%	0.0%
119	F3B(1,2) VLG Cu	HG Mo	5 997	0.5	6 970	118	1.97	2 000	13	115	1.57	-2.6%	-20.1%
121	Stg2 LG Cu	HG Mo	8 676	0.5	8 450	134	1.81	4 000	4	133	1.68	-0.6%	-7.3%
122	Stg2 HG Cu	HG Mo	8 720	0.5	10 000	207	1.49	5 000	5	206	1.37	-0.5%	-8.1%
129	Stg2 VLG Cu	HG Mo	14 491	0.5	10 000	139	1.70	2 000	23	137	1.40	-1.7%	-17.6%
131	Bckg LG Cu	HG Mo	2 769	0.5	3 690	104	1.34	2 700	1	104	1.26	-0.3%	-5.5%
132	Bckg HG Cu	HG Mo	387	2	2 120	119	1.29	1 000	1	116	1.07	-2.4%	-17.6%
139	Bckg VLG Cu	HG Mo	5 625	0.5	2 190	103	1.20	1 000	18	101	1.04	-1.4%	-12.8%
151	RRLT LG Cu	HG Mo	1 727	0.5	4 360	94	1.71	1 000	5	91	1.31	-2.7%	-23.3%
152	RRLT HG Cu	HG Mo	252	0.5	905	103	1.26	600	5	101	1.15	-2.8%	-8.7%
159	RRLT VLG Cu	HG Mo	2 079	0.5	1 160	73	1.28	700	3	72	1.23	-0.7%	-4.0%
911	F3B(1,2) LG Cu	LG Mo	337	0.5	878	42	1.99	350	4	38	1.46	-8.5%	-26.2%
912	F3B(1,2) HG Cu	LG Mo	104	0.5	874	78	1.71	400	4	70	1.38	-11.0%	-19.0%
919	F3B(1,2) VLG Cu	LG Mo	1 814	0.5	1 760	31	2.44	400	8	29	1.61	-6.9%	-34.1%
921	Stg2 LG Cu	LG Mo	1 514	0.5	1 500	38	1.63	350	6	36	1.15	-3.9%	-29.4%
922	Stg2 HG Cu	LG Mo	424	1	2 960	65	2.54	400	9	57	1.33	-12.3%	-47.8%
929	Stg2 VLG Cu	LG Mo	2 778	0.5	1 280	35	1.70	450	8	35	1.46	-2.4%	-14.2%
931	Bckg LG Cu	LG Mo	7 662	0.5	800	21	1.45	600	4	20	1.39	-0.3%	-4.4%
932	Bckg HG Cu	LG Mo	747	0.5	846	38	1.61	400	3	37	1.42	-2.4%	-11.6%
939	Bckg VLG Cu	LG Mo	87 999	0.5	3 125	10	2.82	350	57	10	2.03	-2.1%	-28.0%
951	RRLT LG Cu	LG Mo	9 997	0.5	1 170	19	2.28	1 000	2	19	2.23	-0.2%	-1.8%
952	RRLT HG Cu	LG Mo	1 140	0.5	2 300	34	2.92	600	6	32	2.06	-7.0%	-29.6%
959	RRLT VLG Cu	LG Mo	74 898	0.5	2 080	7	2.97	600	6	7	2.49	-0.8%	-16.1%

Table 14.27 : Silver Statistics

Domain				Raw				Top-cut		%Diff Raw to Top-cut			
Code	Lith	Min	Number	Min	Max	Mean	CV	Value	Num cut	Mean	CV	Mean (%)	CV (%)
1015	F3B(1,2)	HG Ag	16 955	0.01	101	0.72	3.30	3.5	141	0.64	0.84	-10.2%	-74.5%
1050	RRLT	HG Ag	14 259	0.01	101	0.62	3.48	3.5	57	0.56	0.73	-8.8%	-79.0%
1060	OxFlat	HG Ag	20 287	0.01	100	0.66	1.53	3.5	101	0.64	0.72	-2.9%	-53.0%
1099	Bckg	HG Ag	3 422	0.01	25.9	0.59	1.39	3.5	22	0.56	0.83	-3.9%	-40.4%
9015	F3B(1,2)	LG Ag	25 216	0.01	77.8	0.27	2.78	1.5	77	0.25	0.65	-4.4%	-76.5%
9050	RRLT	LG Ag	46 324	0.01	100	0.25	2.29	1.5	88	0.24	0.65	-2.4%	-71.6%
9060	OxFlat	LG Ag	50 350	0.01	44.4	0.24	1.14	1.5	69	0.24	0.58	-1.1%	-48.9%
9099	Bckg	LG Ag	26 912	0.01	18.7	0.23	0.91	1.5	30	0.23	0.64	-1.0%	-29.6%

14.1.14.3 Correlation

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 14.28 for Facies 3B (1,2) low grade copper, Table 14.29 for Facies 3B (1,2) high grade copper, Table 14.30 for Stage 2 low grade copper and Table 14.31 for 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 14.12.

Table 14.28 : Correlation Matrix – 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

Table 14.29 : Correlation Matrix – 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

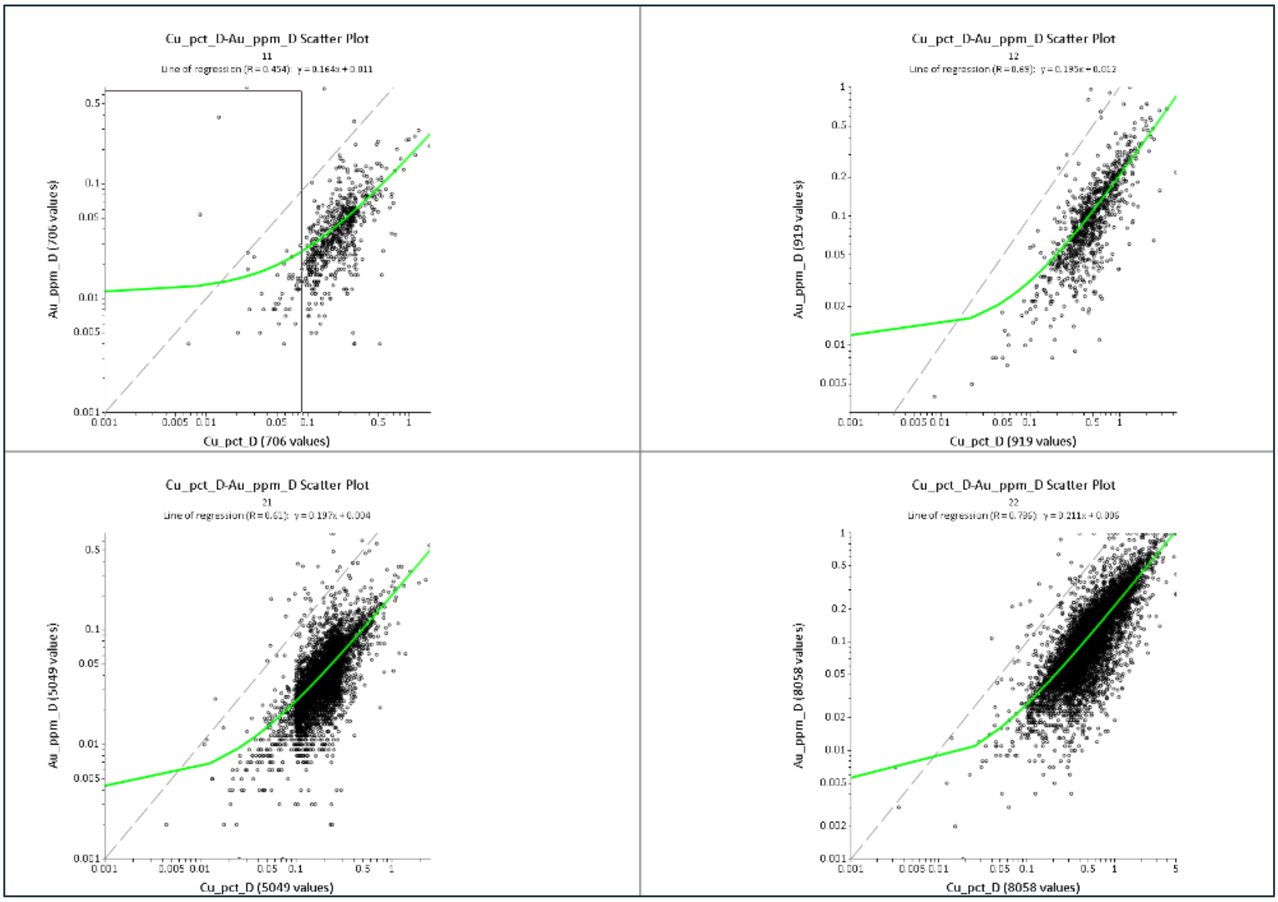
Table 14.30: Correlation Matrix – Mid Area – Stage 2 LG Cu – 50021

	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

Table 14.31 : Correlation Matrix – 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
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 14.12 : Scatterplots Between Copper and Gold for Highest Grade Domains



14.1.14.4 Variography

Snowden Supervisor Version 8.14.3.2 software was used to generate and model the variograms for each of the elements and domains. The correlation coefficients 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.

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.

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 are shown for copper and gold in Table 14.32 to Table 14.34, for molybdenum in Table 14.35 to Table 14.37, and for silver in Table 14.38 to Table 14.40.

The normal score variogram model for mineralised copper is illustrated in Figure 14.13.

Table 14.32 : Grade Variogram Models – Cu/Au Estimate - Mid Area (FAULTBLOCK = 50000)

VREF	Variable	Rotation (Datamine ZXZ)			Nugget	Structure 1		Structure 2		Structure 3	
					C_0	C_1	R_1	C_1	R_1	C_1	R_1
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
19	Au (Flat Oxide)	120	170	0	0.09	0.63	20	0.19	60	0.09	260
							20		60		260
							10		40		40

Table 14.33 : Grade Variogram Models – Cu/Au Estimate – South Area (FAULTBLOCK = 40000)

VREF	Variable	Rotation (Datamine ZXZ)			Nugget	Structure 1		Structure 2		Structure 3	
					C_0	C_1	R_1	C_1	R_1	C_1	R_1
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

Table 14.34 : Grade Variogram Models – Cu/Au Estimate – North Area (FAULTBLOCK = 60000 to 100000)

VREF	Variable	Rotation (Datamine ZXZ)			Nugget C_0	Structure 1		Structure 2		Structure 3	
						C_1	R_1	C_1	R_1	C_1	R_1
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

Table 14.35 : Grade Variogram Models – Mo Estimate - Mid Area (FAULTBLOCK = 50000)

VREF	Variable	Rotation (Datamine ZXZ)			Nugget C_0	Structure 1		Structure 2		Structure 3	
						C_1	R_1	C_1	R_1	C_1	R_1
5	Mo F3B(1,2) HG Indicator	110	90	-90	0.27	0.29	15	0.19	25	0.25	160
							15		40		125
							15		25		70
6	Mo RLLT+ Bckg, HG Indicator	110	90	-90	0.21	0.37	15	0.24	25	0.18	120
							15		20		90
							15		25		70
7	Mo F3B(1,2) LG Indicator	110	90	-90	0.33	0.26	15	0.19	25	0.22	190
							15		40		190
							15		25		70
8	Mo RLLT, LG Indicator	110	90	-90	0.27	0.47	25	0.17	50	0.09	220
							15		50		655
							15		80		80
9	Mo Bckg, LG Indicator	110	90	-90	0.17	0.37	30	0.28	180	0.18	400
							60		10		1400
							30		150		230

Table 14.36 : Grade Variogram Models – Mo Estimate - South Area (FAULTBLOCK = 40000)

VREF	Variable	Rotation (Datamine ZXZ)			Nugget C_0	Structure 1		Structure 2		Structure 3	
						C_1	R_1	C_1	R_1	C_1	R_1
32	Mo F3B(1,2) HG Indicator	125	135	0	0.36	0.46	50	0.08	115	0.10	150
							20		40		90
							10		15		60
33	Mo	125	135	0	0.27	0.45	50	0.11	115	0.17	150
							20		40		90

Table 14.36 : Grade Variogram Models – Mo Estimate - South Area (FAULTBLOCK = 40000)

VREF	Variable	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
	RLLT+ Bckg, HG Indicator						10		15		60
34	Mo F3B(1,2) LG Indicator	125	135	0	0.33	0.45	30	0.13	100	0.09	650
							20		80		350
							15		30		30
35	Mo RLLT+ Bckg, LG Indicator	125	135	0	0.15	0.63	40	0.12	250	0.10	970
							20		120		350
							15		45		50

Table 14.37 : Grade Variogram Models – Mo Estimate – North Area (FAULTBLOCK = 60000 to 100000)

VREF	Variable	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
1	Mo	120	90	0	0.25	0.25	50	0.25	100	0.25	150
							20		40		100
							10		20		50

Table 14.38 : Grade Variogram Models – Ag Estimate - Mid Area (FAULTBLOCK = 50000)

VREF	Variable	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
1	Ag F3B(1,2)	120	40	0	0.30	0.42	30	0.28	150	-	-
							30		120		-
							30		100		-
2	Ag RLLT	130	60	0	0.17	0.36	40	0.23	100	0.25	185
							40		80		150
							40		80		150
3	Ag Flat Oxide	-70	100	180	0.13	0.27	50	0.32	150	0.27	250
							50		150		250
							50		150		250
4	Ag Bckg	110	90	-90	0.01	0.41	100	0.23	200	0.35	400
							100		200		400
							100		200		400

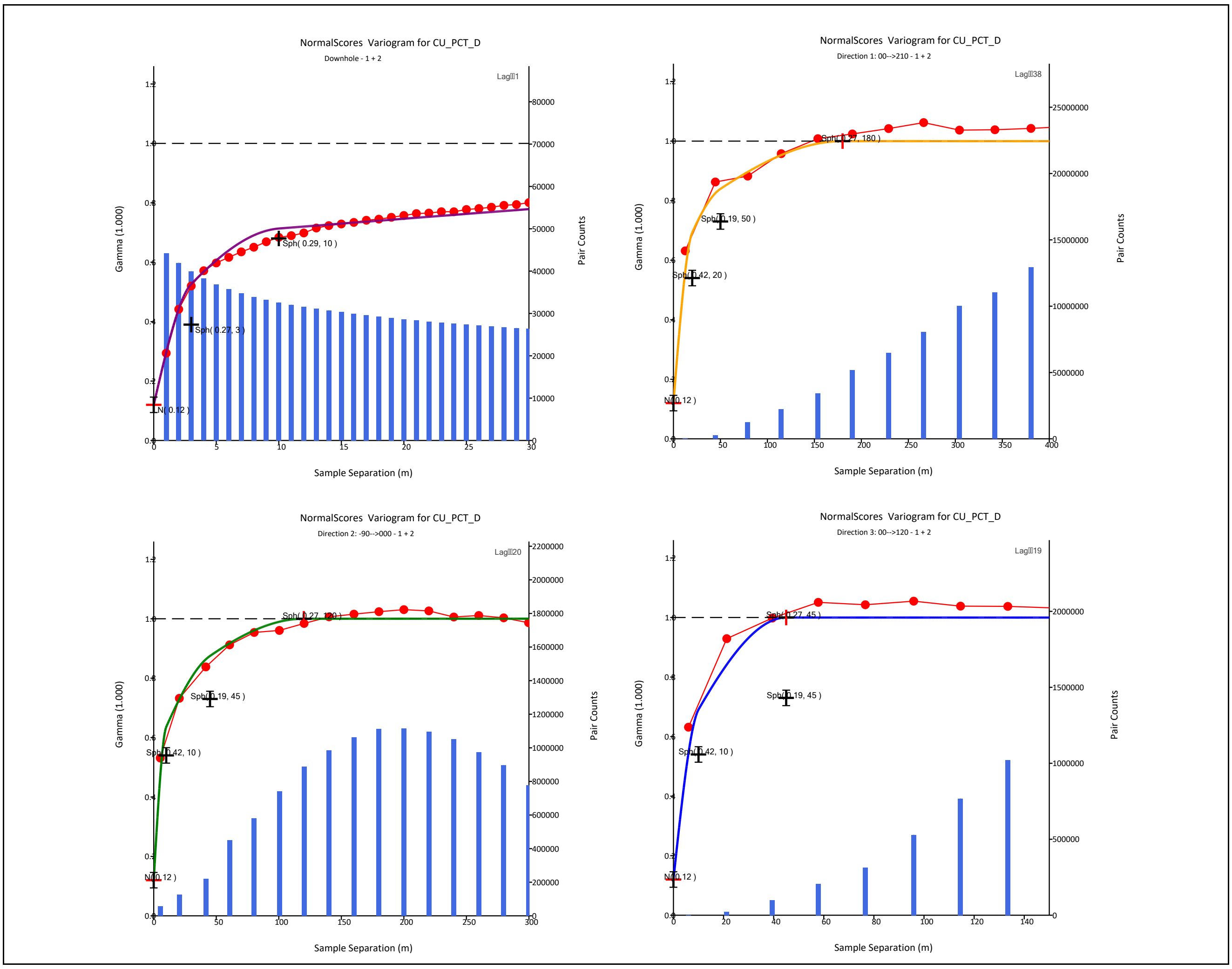
Table 14.39 : Grade Variogram Models – Ag Estimate - South Area (FAULTBLOCK = 40000)

VREF	Variable	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
5	Ag F3B(1,2)	120	40	0	0.30	0.42	30	0.28	150	-	-
							30		120		-
							30		100		-
6	Ag RLLT	130	60	0	0.17	0.36	40	0.23	100	0.25	185
							40		80		150
							40		80		150
7	Ag Flat Oxide	-70	100	180	0.13	0.27	50	0.32	150	0.27	250
							50		150		250
							50		150		250
8	Ag Bckg	110	90	-90	0.01	0.41	100	0.23	200	0.35	400
							100		200		400
							100		200		400

Table 14.40 : Grade Variogram Models – Ag Estimate – North Area (FAULTBLOCK = 60000 to 100000)

VREF	Variable	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
9	Ag	120	40	0	0.30	0.42	30	0.28	150	-	-
							30		120		-
							30		100		-

Figure 14.13 : Normal Score Variogram Model for Mid Area Copper High Grade and Low Grade Combined



14.1.15 Estimation

A suite of elements (copper, gold, molybdenum, cobalt, calcium, iron, sulphur, potassium and aluminium) was estimated using 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.

Mineralisation was estimated using hard boundaries according to the domain conditions for each element. The boundaries between oxidation states were soft.

There was a hard boundary between domains cut by the Serrano fault and the Rancho fault but soft boundaries between other fault blocks in the north area.

The composite data was top-cut prior to estimation as discussed in Section 14.1.14.

For the estimation, 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.

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, faultblock-specific dynamic anisotropy trend wireframes. An example of dynamic anisotropy is included in Figure 14.13.

The search strategy for each element and domain is shown in Table 14.41 to Table 14.49 below.

No octant searches were used.

Table 14.41 : Search Strategy for Grade Estimation – Cu/Au – Mid Area (FAULTBLOCK = 50000)

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	
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

Table 14.42 : Search Strategy for Grade Estimation – Cu/Au – South Area (FAULTBLOCK = 40000)

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	
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

Table 14.43 : Search Strategy for Grade Estimation – Cu/Au – North Area (FAULTBLOCK = 60000 to 100000)

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	
Cu/Au	23	Dynamic Anisotropy			100	100	50	2	6	12	6	12	5

Table 14.44 : Search Strategy for Grade Estimation – Mo – Mid Area (FAULTBLOCK = 50000)												
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	
Mo F3B(1,2) HG Indicator	5	Dynamic Anisotropy			200	150	100	3	6	12	6	12	7
Mo RLLT+ Bckg, HG Indicator	6	Dynamic Anisotropy			200	150	120	3	6	12	6	12	7
Mo F3B(1,2) LG Indicator	7	Dynamic Anisotropy			200	200	80	3	6	12	6	12	7
Mo RLLT, LG Indicator	8	Dynamic Anisotropy			200	200	80	3	6	12	6	12	7
Mo Bckg, LG Indicator	9	Dynamic Anisotropy			200	200	115	3	6	12	6	12	7

Table 14.45 : Search Strategy for Grade Estimation – Mo – South Area (FAULTBLOCK = 40000)													
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	
Mo	29	Dynamic Anisotropy			100	100	50	3	6	12	6	12	5

Table 14.46 : Search Strategy for Grade Estimation – Mo – North Area (FAULTBLOCK = 60000 to 100000)													
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	
Mo	1	Dynamic Anisotropy			100	100	50	3	6	12	6	12	7

Table 14.47 : Search Strategy for Grade Estimation – Ag – Mid Area (FAULTBLOCK = 50000)													
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	
Ag	1	Dynamic Anisotropy			200	150	100	3	6	12	6	12	5

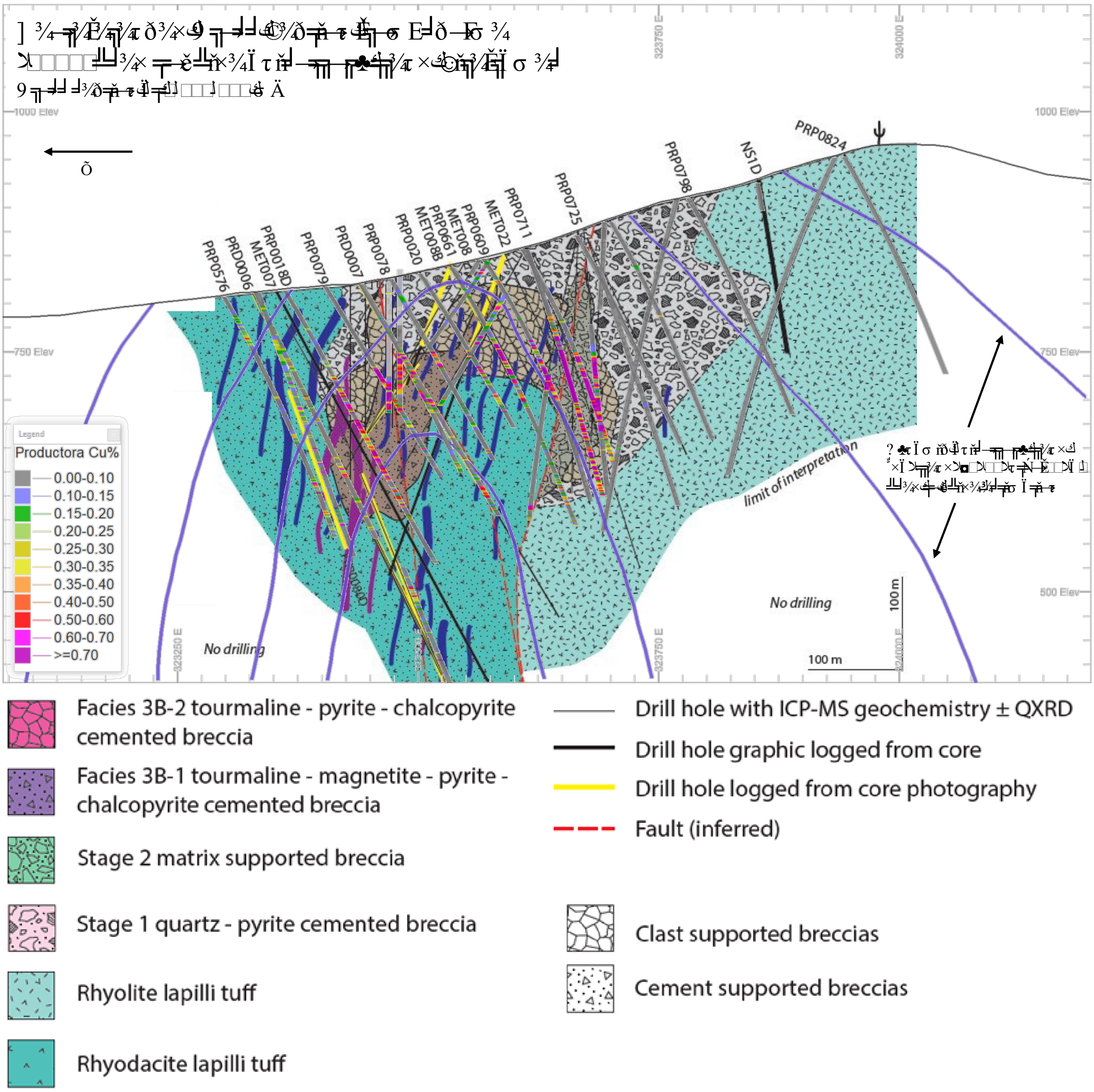
Table 14.48 : Search Strategy for Grade Estimation – Ag – South Area (FAULTBLOCK = 40000)

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	
Ag	2	Dynamic Anisotropy			200	150	100	3	6	12	6	12	5

Table 14.49 : Search Strategy for Grade Estimation – Ag – North Area (FAULTBLOCK = 60000 to 100000)

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	
Ag	2	Dynamic Anisotropy			200	150	100	3	6	12	6	12	5

Figure 14.14 : Cross Section at 6822215 mN Displaying a Georeferenced Interpreted Distribution of Breccia Facies from Escolme (2016)



14.1.16 Model Validation

14.1.16.1 Introduction

The 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.

14.1.16.2 Composites vs. Estimates

The global comparisons were completed using the hard boundary domain coding. The global comparison for the copper domains is presented in Table 14.50. 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.

The validation by northing, in Section 14.1.16.3 illustrates these effects.

Table 14.50 : Global Comparison of Copper Composites and Estimates

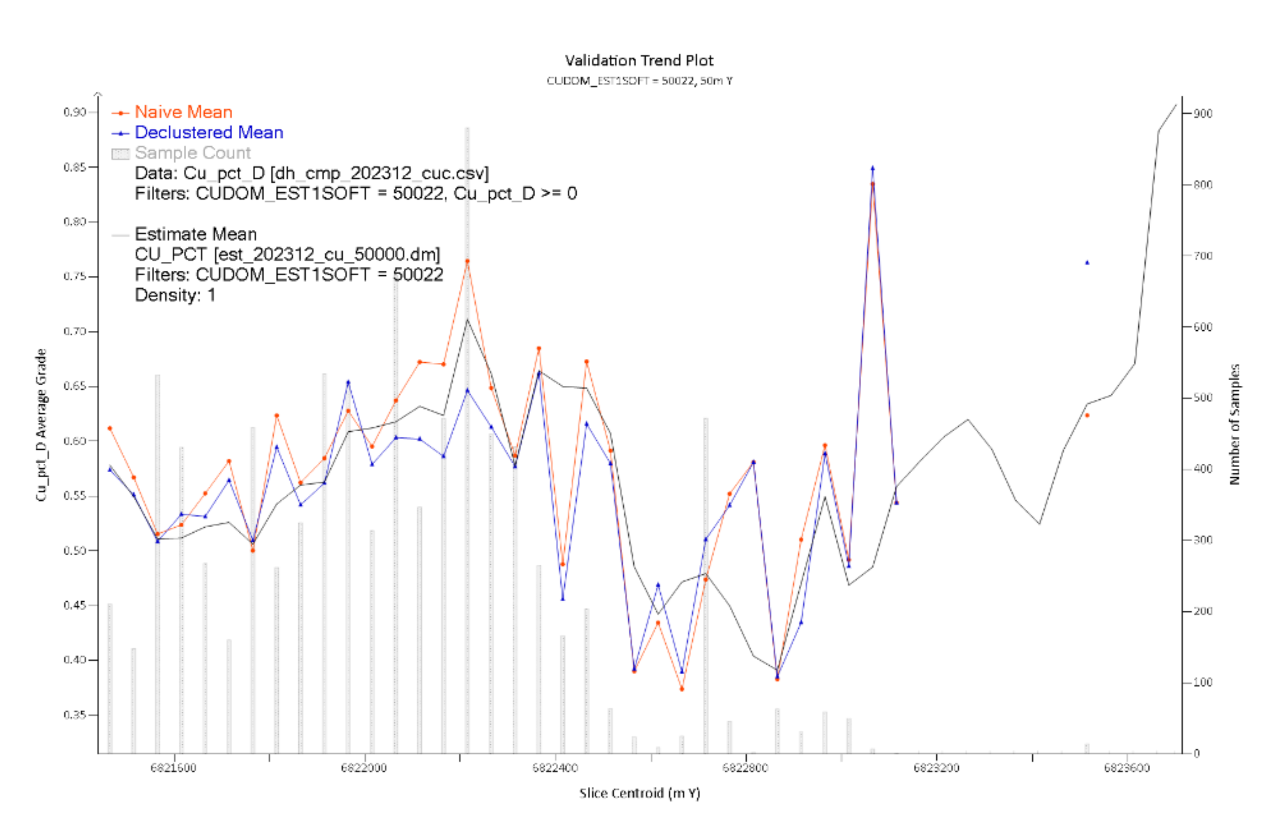
CUDOM	Composites		Model	% Difference		% Est	Model	% Difference	
	Top-cut	Top-cut Declustered	1st pass	Top-cut	Top-cut Declustered	1st pass	All	Top-cut	Top-cut Declustered
50022	0.60	0.56	0.57	-5.3%	+1.7%	23%	0.57	-5.9%	+1.0%
50012	0.52	0.52	0.54	+3.2%	+3.0%	80%	0.53	+1.7%	+1.5%
50032	0.56	0.51	0.61	+5.6%	+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%

14.1.16.3 Trend Plots

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.

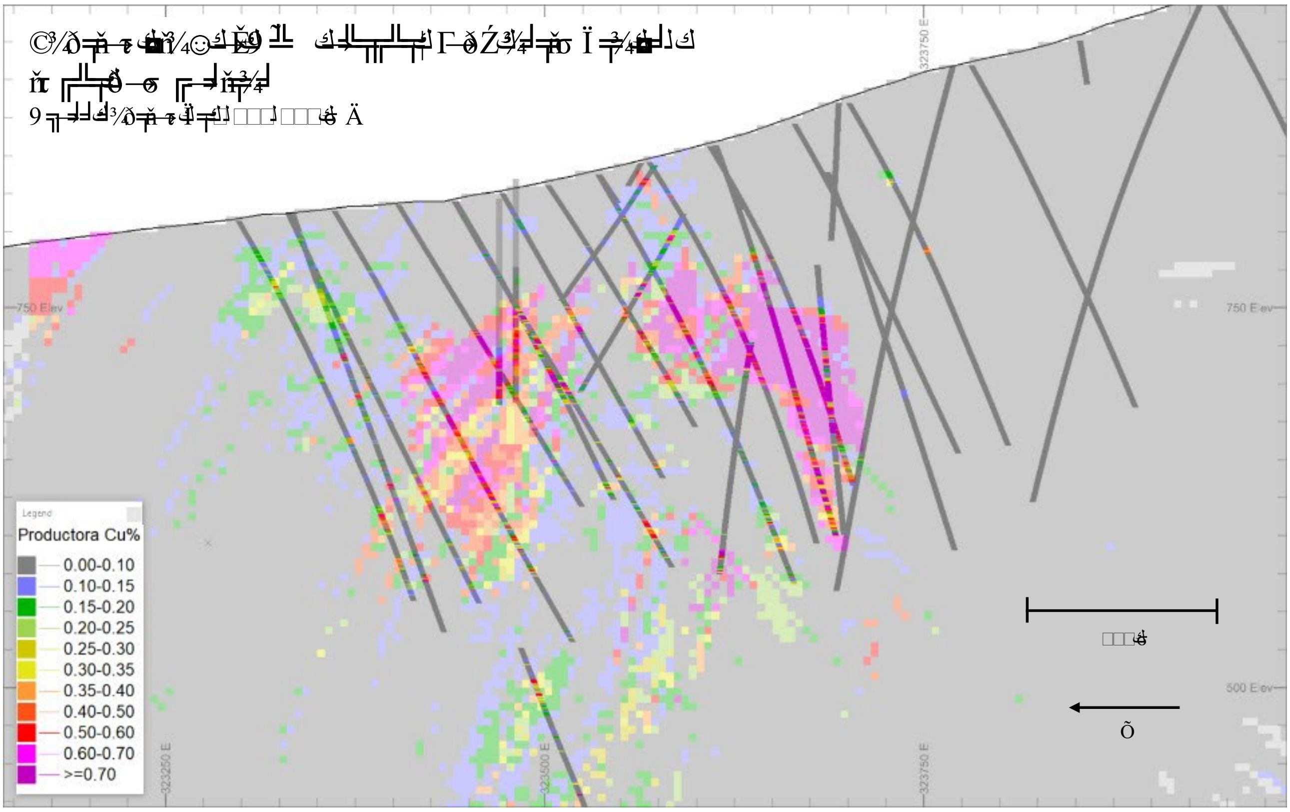
Figure 14.15: Northing Trend Validation for Copper in Domain 50022



14.1.16.4 Visual Validation

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 : Visual Validation of Copper for East-West Section 6822215 mN



14.1.17 Bulk Density

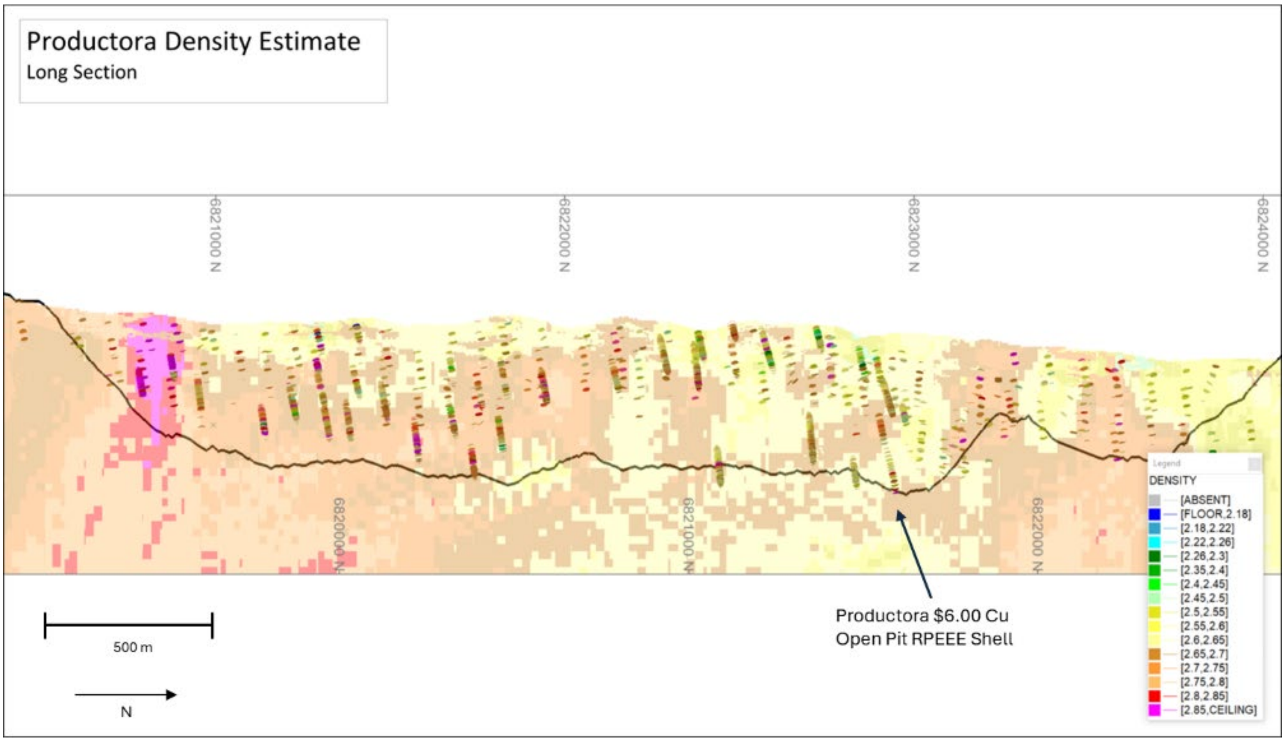
A Total of 12 481 measurements were available for estimation of bulk density at Productora.

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 long section view of the final bulk density estimate is included in Figure 14.17 below.

Figure 14.17 : Final Coded Bulk Density Values for Productora



14.1.18 Mineral Resource Classification

The Productora Mineral Resource estimate has been classified considering the guidelines provided in the 2014 CIM Definition Standards and were estimated in accordance with the CIM 2019 Best Practices Guidelines, as required by NI43-101.

A range of criteria was considered in determining the Mineral Resource classification, including:

- Drill data density
- Sample/assay confidence
- 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:

- 2023_prod_indicated
- 2023_prod_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.

14.1.19 Reasonable Prospects for Eventual Economic Extraction (RPEEE)

Reporting for Productora considers Reasonable Prospects for Eventual Economic Extraction (RPEEE) for an open-pit mining scenario, with the cost basis for cut-off grade analysis utilising costs from the 2023 PEA.

An open pit shell was generated using the industry-standard Lerchs-Grossman algorithm and the parameters listed in Table Table 14.51. 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 14.51: Key open pit optimisation parameters applied to generate a pit shell defining Reasonable Prospects of Eventual Economic Extraction

Productora 2024 RPEEE Parameter Table		
Value	**Units**	**Description**
6	USD/t	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
6.32	USD/t	Oxide Processing Cost
49°	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
90 / 70	%	Sulphide Copper Recovery (Fresh/Transitional)
52 / 50	%	Sulphide Gold Recovery (Fresh/Transitional)
53 / 46	%	Sulphide Molybdenum Recovery (Fresh/Transitional)
20 / 20	%	Sulphide Silver Recovery (Fresh/Transitional)

The controlling surface for Productora RPEEE is shown in Figure 14.18 and Figure 14.19. These figures also include sensitivity testwork completed at different copper prices, showing that the RPEEE surface is not particularly sensitive to copper price at Productora.

Figure 14.18 : Long-Section View Looking West Showing Pit Shell Surface Shapes Used to Define RPEEE

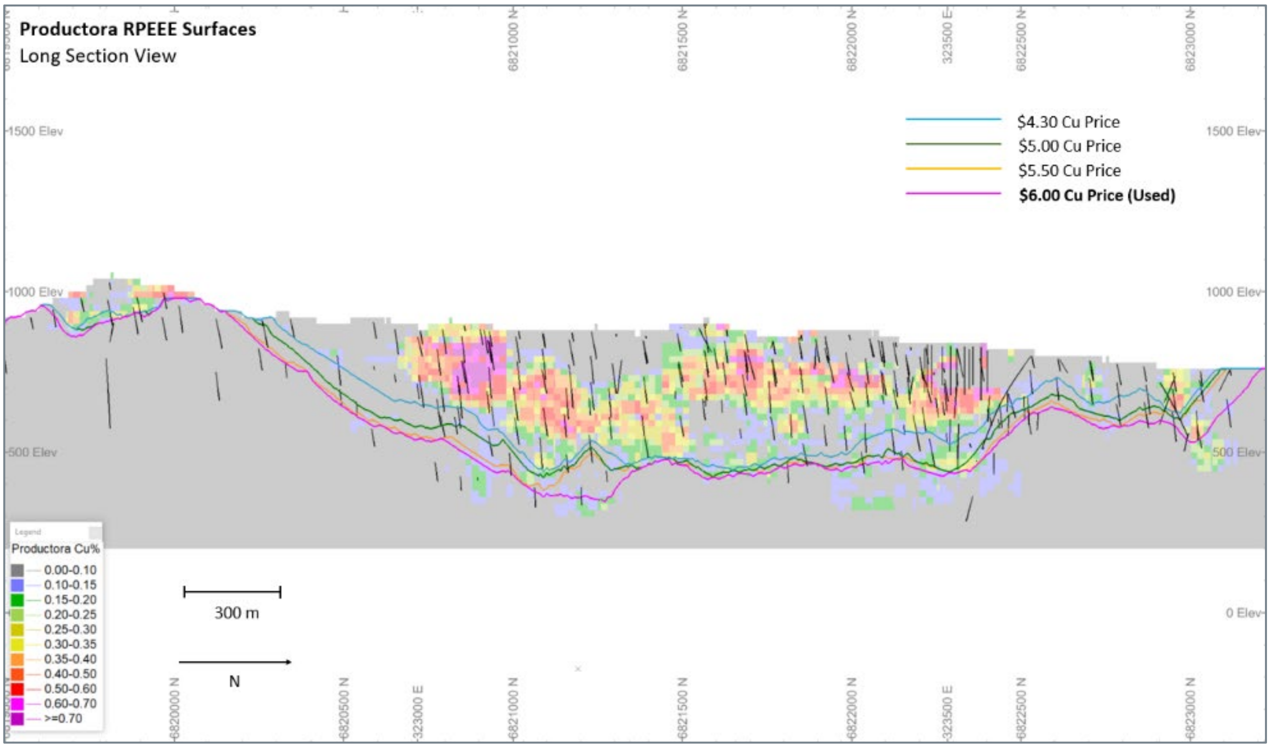
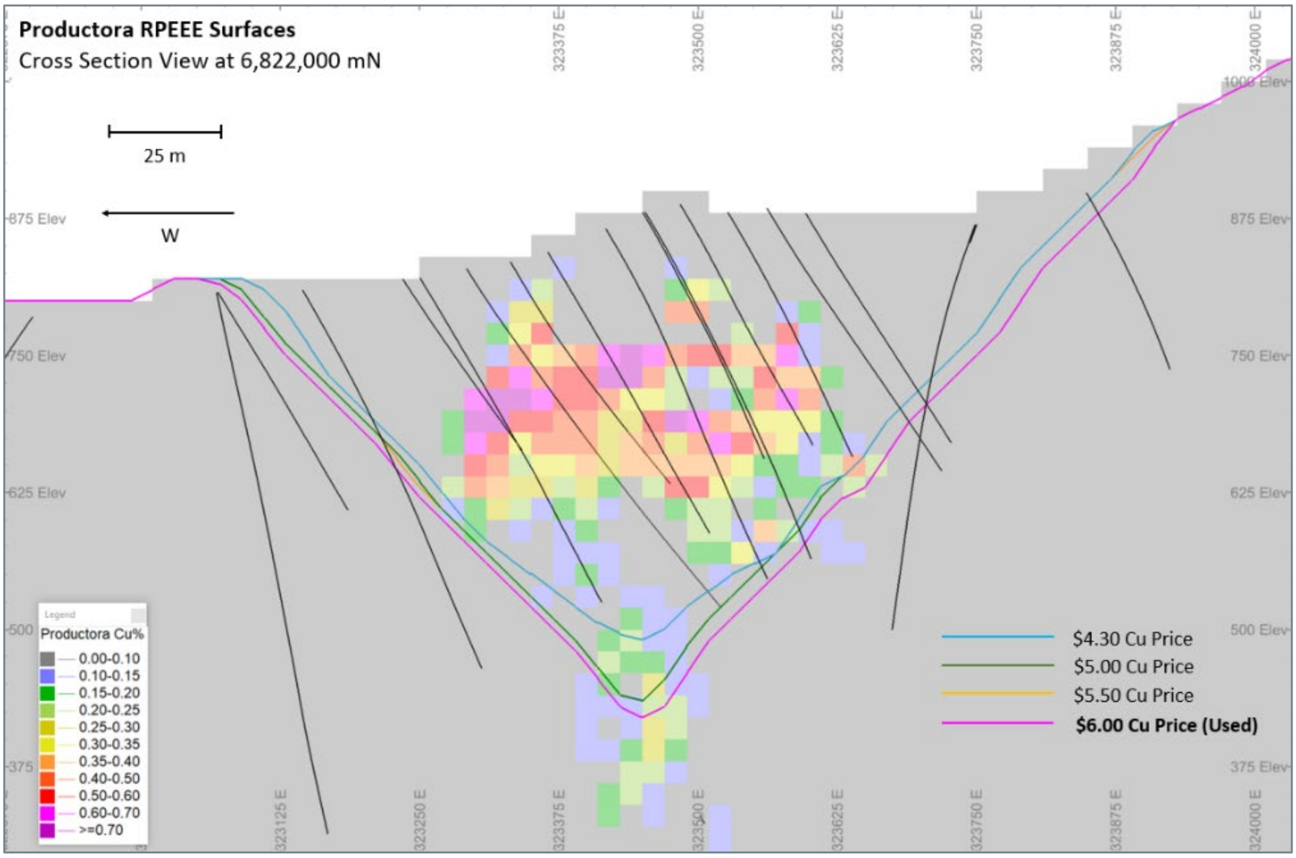


Figure 14.19 : Cross Section View Looking North Showing Pit Shell Used to Define RPEEE



14.2 Alice

14.2.1 Introduction

The current Alice Mineral Resource was updated in February 2024 for inclusion in the combined Costa Fuego Mineral Resource Release (26 February 2024).

14.2.2 Database Validation

The Alice database was validated through the following checks:

- Drill collars – comparison of planned vs. actual survey pick up coordinates, checked in 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
- Lithology – reviewed based on surrounding drill holes, with drill core or RC chips reviewed and logging updated where necessary
- Mineralisation intensity - logs completed for each drill hole, and compared with assay results
- 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.

14.2.2.1 Summary of Data Used in Estimate

All drilling data is stored in the HCH acQuire drill hole database.

Data table records consist of:

- Collar – Alice_Collar.csv
- Survey – Alice_Survey.csv
- Assays – Alice_Assay.csv
- Assays – Alice_Cusol.csv
- Geology – Alice_Lith.csv
- Geology – Alice_Weathering.csv
- Geology – Alice_Vein.csv
- Geology – Alice_Min.csv

No holes were excluded from the Alice Resource Estimate.

A summary of the drilling data used to compile the estimate is listed in Table 14.52. This table only includes drillholes that pass through the limits of the outermost estimation domain at Alice (i.e. excludes some Alice regional exploration drilling).

Table 14.52 : Drill Holes Included in Estimate Database

Drilling Method	Holes	Metres
RC	53	16 028
DD	2	1 020
RC with DD tail	5	1 745

14.2.3 Data Manipulation

Negative-grade values are used in the assay database table to indicate specific events or conditions. To ensure these assays are suitable for use in the mineral resource, edits were made to remove any non-numeric, zero and negative values.

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 (0.0025 g/t Au, 0.005% Cu, 0.5 ppm Mo, 0.005 ppm Ag)
- If the assay value was equal to zero and there was no recorded weight of sample the assay was set to ‘absent’.

14.2.4 Domaining

Field observations and geological logging of diamond core and reverse circulation (RC) chips suggested that several alteration and mineralogical domains were present at Alice. Exploratory analysis was undertaken in software packages including SURPAC, IoGAS, and Snowden Supervisor.

Initial data flagging confirmed the western boundary of Alice mineralisation as the Alice Fault. Rock in the hanging wall of the Alice Fault is primarily a felsic volcanic succession which has undergone advanced argillic alteration. This alteration was flagged in IoGas as intense or moderate and defined a coherent non-mineralised hard boundary domain.

Mineralised domain interpretation was completed in Leapfrog 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, interpolants were created at 0.4%, 0.2%, and 0.025% (CuEq) based on natural breaks in the grade population. This differs from the approach used in the previous estimate, which used manually interpreted chalcopyrite and pyrite interpolants as proxies for high-grade and low-grade domains, respectively.

Given the strong correlation between copper and gold, it was considered appropriate to use the copper mineralised domains for the gold estimate. No significant differences in gold grade population were present between the 0.4%, 0.2%, and 0.025% (CuEq) domains which resulted in these domains being combined for a single estimate.

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.

14.2.5 Weathering Model

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 a majority of 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.

14.2.6 Data Flagging

Geology, alteration, and weathering wireframes were used to flag the drill hole data and block model. The wireframes used are listed in Table 14.53 with an explanation of the fields and their coding in Table 14.54.

Table 14.53 : Wireframes Used for Flagging Drill Holes and Models

Wireframe	Zone Field	Type
Geol_1	LTCODE	Solid
Geol_3	LTCODE	Solid
Geol_20	LTCODE	Solid
alice_oxide_20221108	WTCODE	Solid
alice_trans_20221108	WTCODE	Solid
alice_fresh_20221108	WTCODE	Solid
alice0.025cueq20221108	CUEQAREA	Solid
alice_0.4_cu_20221108	DOM_CU1	Solid
alice_0.2_cu_20221108	DOM_CU1	Solid
alice_0.2_ag_20230918	DOM_AG1	Solid
alice_50_mo_20221108	DOM_MO1	Solid
alice_20_mo_20221108	DOM_MO1	Solid
alice_60_co_20221108	DOM_CO1	Solid
alice_1p0_k_20230904	DOM_K1	Solid
alice_1p5_ca_20230904	DOM_CA1	Solid
alice_1p0_py_20230905	DOM_PY1	Solid

Table 14.54 : Drill Holes and Models Coding

Field	Code	Description
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

14.2.7 Block Modelling

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 14.55.

Table 14.55 : Block Model Dimensions					
Dimension	Minimum	Maximum	Extent (m)	Block Size (m)	
				Parent	Minimum
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

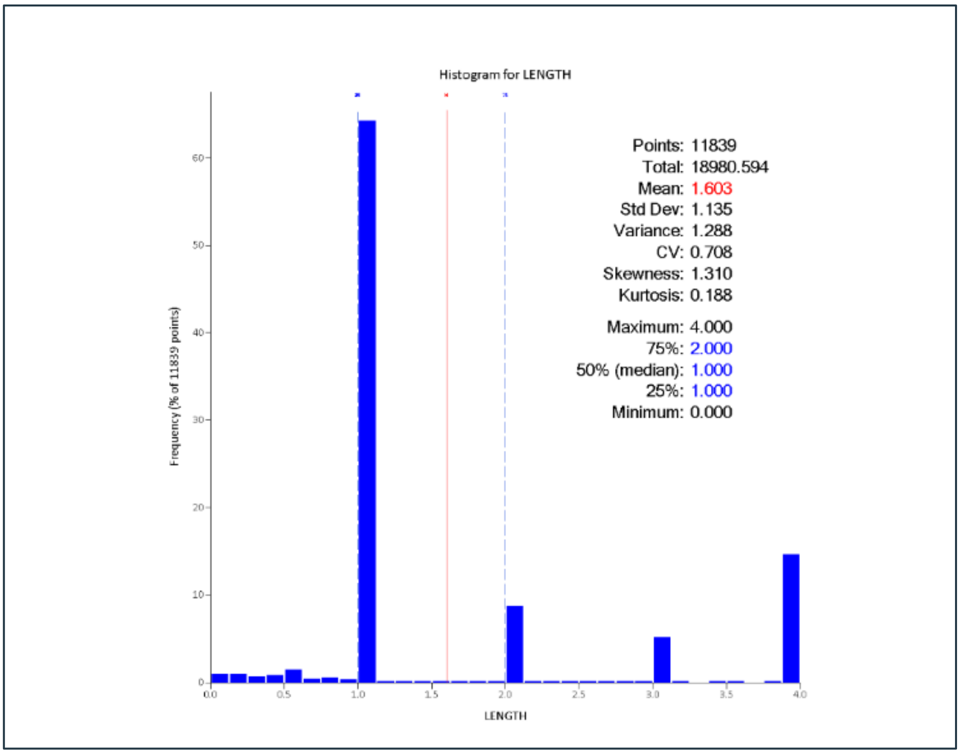
Density was assigned to blocks using averages from each lithology and weathering classification. Where no oxide or transitional value was available for a lithology a nominal 20% or 10%, respectively, was subtracted from the fresh value.

14.2.8 Compositing

70% of the drill sampling at Alice was at 1 m intervals (Figure 14.20). This proportion increased to 91% in potentially economic mineralised domains where copper was greater than 0.1%. 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.

The compositing process used the relevant domain and weathering as a boundary to ensure no composites were created across domains.

Figure 14.20 : Alice Sample Lengths within Model Limit Area



14.2.9 Statistical Analysis

14.2.9.1 Introduction

Statistical analysis of copper, gold, molybdenum, silver, cobalt, calcium, potassium and aluminium 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.

14.2.9.2 Top Cutting

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 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.

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 14.56. This is supported by the absence of any genuine outliers within the domains, as well as the low coefficient of variation (CV).

Table 14.56 : Raw vs Top Cut Statistics

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%

14.2.10 Variography

Snowden Supervisor Version 8.14.3.2 software was used to generate and model the variograms for each of the elements and domains.

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.

For domains where a coherent variogram could not be formed, a constructed variogram with a nominal nugget of 0.2 and isotropic search of 200 m was used to estimate grade. These areas are predominately lower grade, and distal to the Alice mineralised porphyry.

The back-transformed variogram model parameters used for resource estimation are shown Table 14.57 to Table 14.60.

The normal score variogram model for mineralised copper is illustrated in Figure 14.21.

Table 14.57 : 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
	130	100	0	0.27	0.06	50	0.38	180	0.29	210

Table 14.57 : 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 = 20						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 14.58 : Grade Variogram Models – Au 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, 20, 50	140	90	120	0.16	0.45	25	0.31	90	0.08	180
						25		90		180
						25		90		180
DOM_CU1 = 90	0	0	-90	0.2	0.8	200	-	-	-	-
						200		-		-
						200		-		-

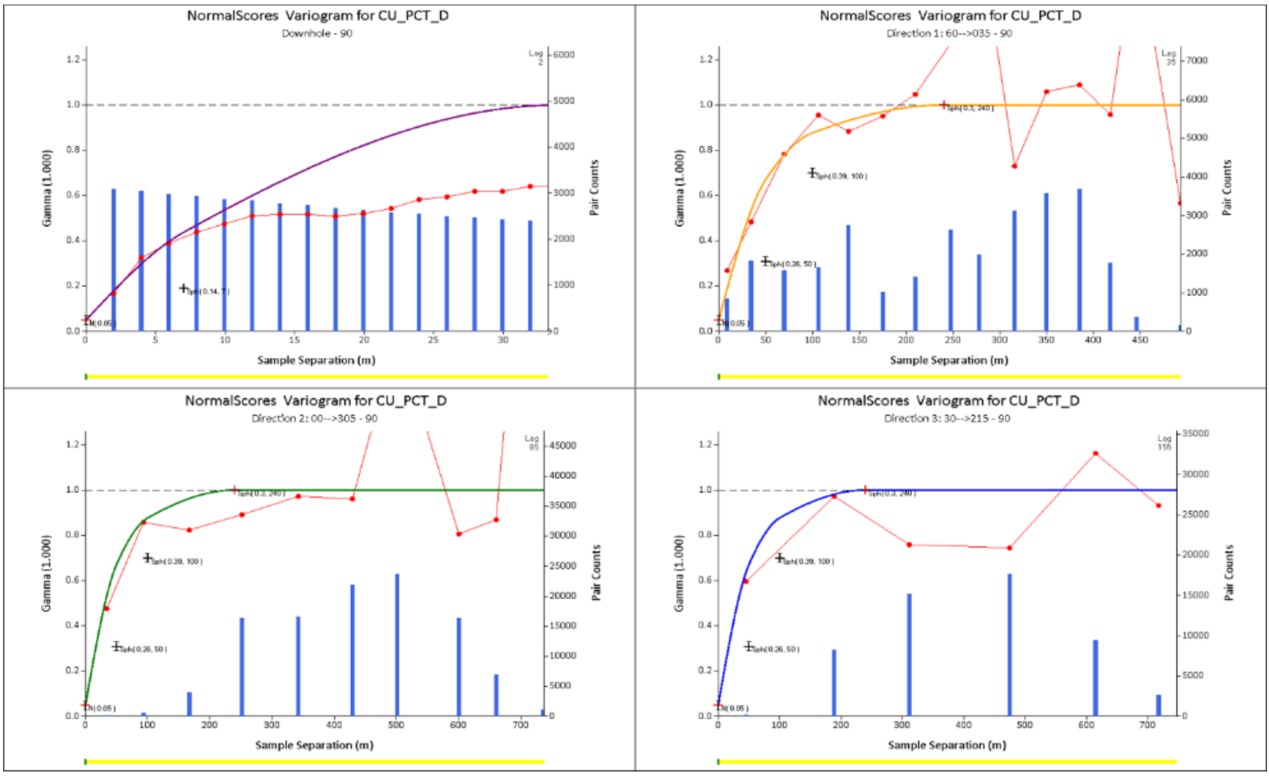
Table 14.59 : Grade Variogram Models – Ag Estimate

Domain	Rotation (Datamine ZXZ)			Nugget C_0	Structure 1 C_1	Structure 1 R_1	Structure 2 C_1	Structure 2 R_1	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		-		-

Table 14.60 : Grade Variogram Models – Mo 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_MO1 = 10	120	100	90	0.17	0.26	25	020	80	0.38	160
						25		80		130
						25		60		80
DOM_ MO 1 = 20	130	80	90	0.30	0.19	30	0.17	90	0.34	250
						30		60		110
						30		60		90
DOM_ MO 1 = 50	160	120	90	0.12	0.47	75	0.21	115	0.21	140
						30		90		140
						30		90		140
DOM_ MO 1 = 90	0	0	-90	0.20	0.80	200	-	-	-	-
						200		-		-
						200		-		-

Figure 14.21 : Normal Score Variogram Model for Copper High Grade Estimate



14.2.11 Estimation

A suite of elements (copper, gold, molybdenum, silver, cobalt, calcium, iron, sulphur, potassium and aluminium) was estimated using 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.

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 weathering domains were soft.

The composite data was top-cut prior to estimation as discussed in Section 14.2.9.2.

The search strategy for each element and domain is shown in Table 14.61 to Table 14.64.

No octant search was used.

	Table 14.61 : Search Strategy for Grade Estimation – Cu											
Domain	Orientation			Search			2nd Search Factor	Number of Composites				
	Rot 1	Rot 2	Rot 3	D1	D2	D3		First Search		Second Search		Max samples across Domain Boundary
								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

	Table 14.62 : Search Strategy for Grade Estimation – Au											
Domain	Orientation			Search			2nd Search Factor	Number of Composites				
	Rot 1	Rot 2	Rot 3	D1	D2	D3		First Search		Second Search		Max samples across Domain Boundary
								Min	Max	Min	Max	
DOM_CU1 = 10, 20, 50	140	90	120	120	120	120	1.5	12	24	12	24	6
DOM_CU1 = 90	0	0	-90	200	200	200	1.5	6	12	6	12	4

Domain	Table 14.63 : Search Strategy for Grade Estimation – Ag											
	Orientation			Search			2nd Search Factor	Number of Composites				
	Rot 1	Rot 2	Rot 3	D1	D2	D3		First Search Min	First Search Max	Second Search Min	Second Search Max	Max samples across Domain Boundary
DOM_AG1 = 10	-60	-60	0	160	100	80	1.5	10	20	10	20	6
DOM_AG1 = 50	140	90	170	140	100	70	1.5	6	16	6	16	5
DOM_AG1 = 90	0	0	-90	200	200	200	1.5	6	12	6	12	5

Domain	Table 14.64 : Search Strategy for Grade Estimation – Mo											
	Orientation			Search			2nd Search Factor	Number of Composites				
	Rot 1	Rot 2	Rot 3	D1	D2	D3		First Search Min	First Search Max	Second Search Min	Second Search Max	Max samples across Domain Boundary
DOM_MO1 = 10	120	100	90	100	80	50	1.5	10	20	10	2	6
DOM_MO1 = 20	130	80	90	160	70	60	1.5	12	24	12	24	6
DOM_MO1 = 50	160	120	90	90	90	90	1.5	10	20	10	20	6
DOM_MO1 = 90	0	0	0	200	200	200	1.5	6	12	6	12	5

14.2.12 Model Validation

The 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.

The Alice estimation was considered acceptable in honouring the input data.

Examples of a northing trend plot and cross section of the copper estimate are shown in Figure 14.22 and Figure 14.23.

Figure 14.22 : Northing Trend Validation for Alice Copper Estimate

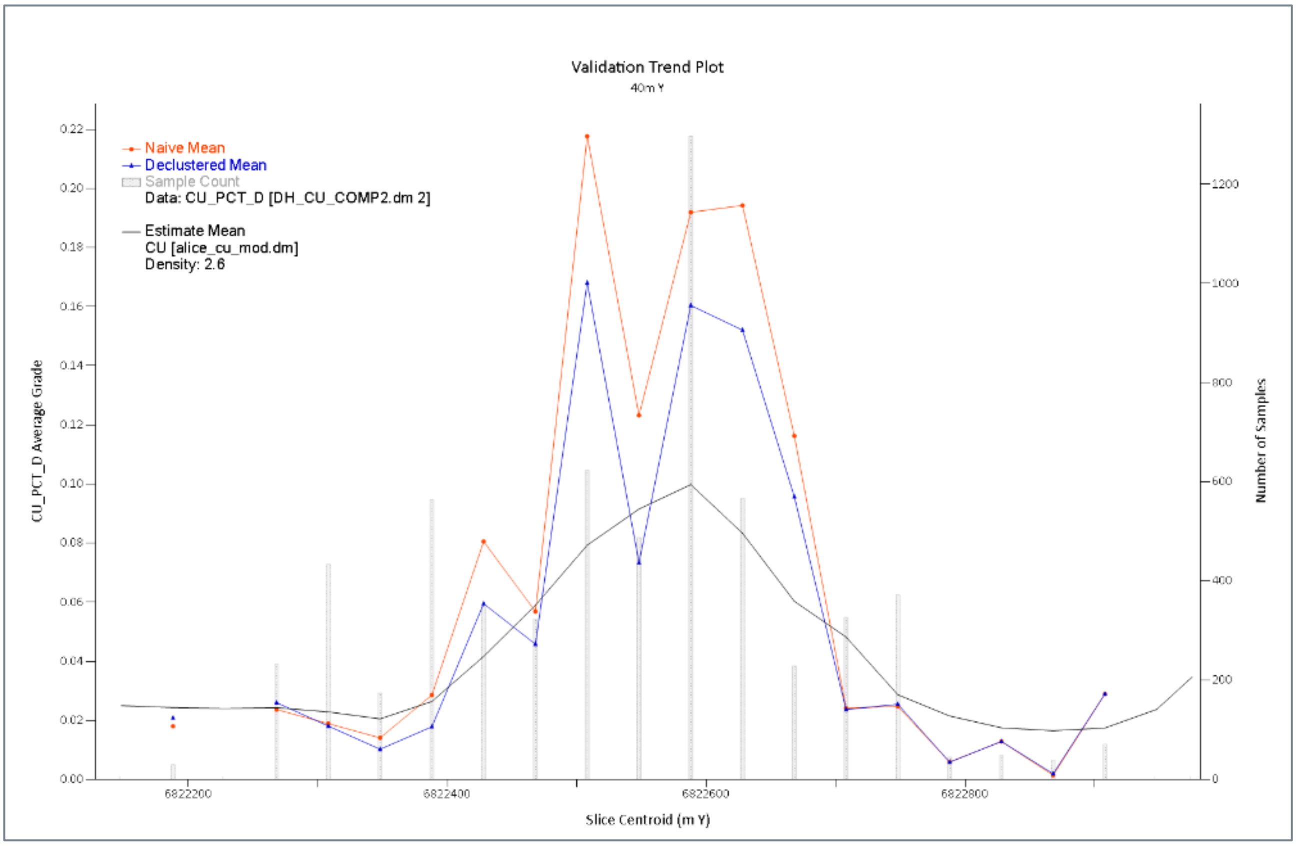
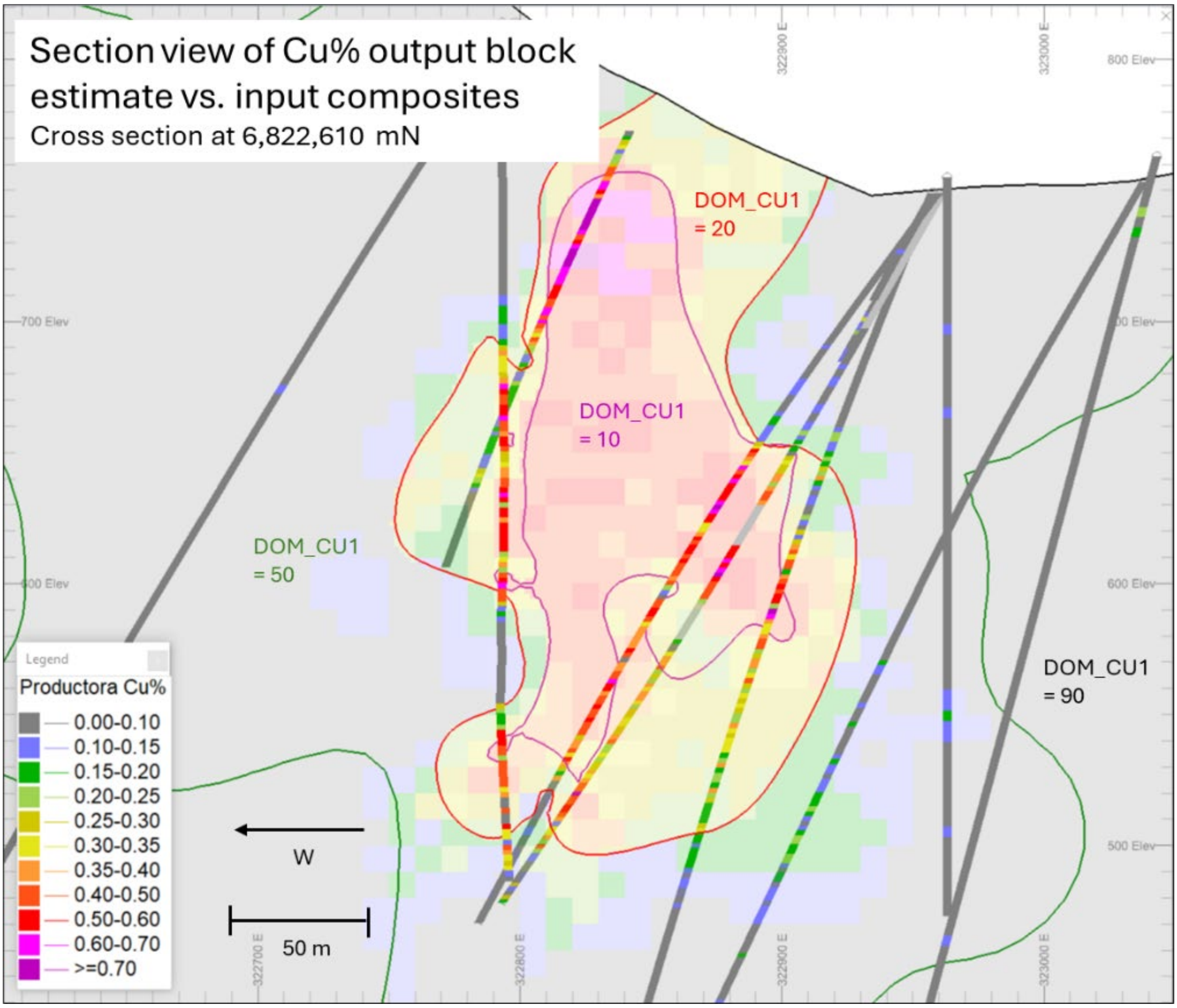


Figure 14.23 : Visual Validation of Alice Copper Estimation for East-West Section 6822215 mN



14.2.13 Resource Classification

The Alice Mineral Resource estimate has been classified considering the 2014 CIM Definition Standards and were estimated in accordance with the CIM 2019 Best Practices Guidelines, as required by NI43-101.

A range of criteria was considered in determining the Resource classification, including:

- Drill data density
- Sample/assay confidence
- 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.

The following wireframes were constructed to flag the blocks as Indicated or Inferred:

- Alice_RC2_20231004_1
- Alice_RC3_20231004_1.

14.2.14 Reasonable Prospects for Eventual Economic Extraction (RPEEE)

Reporting for Alice considers Reasonable Prospects for Eventual Economic Extraction (RPEEE) for an open-pit mining scenario, with the cost basis for cut-off grade analysis utilising the costs from the 2023 PEA.

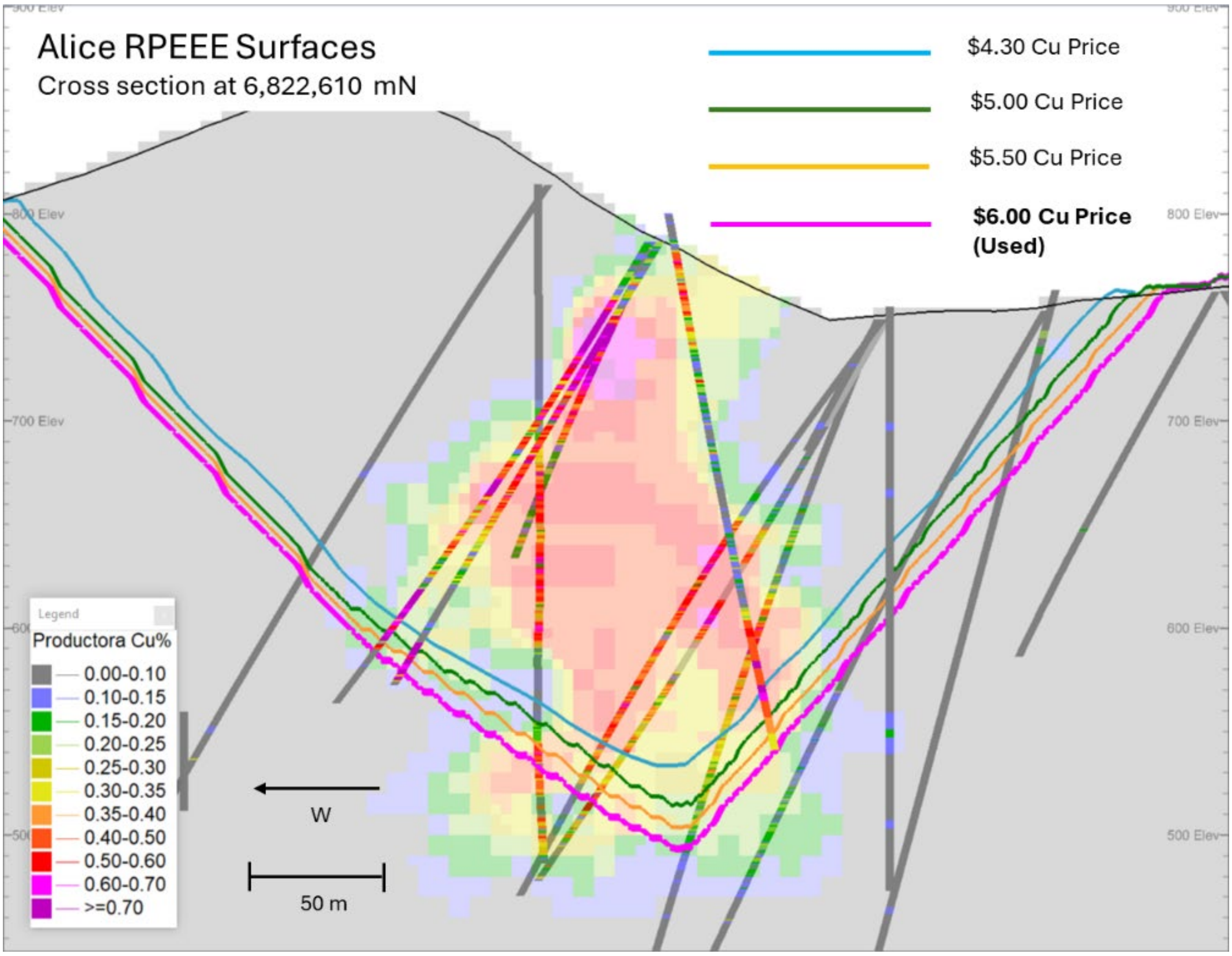
An open pit shell was generated using the industry-standard Lerchs-Grossman algorithm and the parameters listed in Table 14.65. Selling costs incorporate transport and handling costs, as well as any treatment and refining costs. Payable terms assume that the copper concentrate is sold into the Asian smelter market, for which these terms are standard.

Table 14.65 : Key open pit optimisation parameters applied to generate a pit shell defining Reasonable Prospects of Eventual Economic Extraction

Alice 2024 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
6.32	USD/t	Oxide Processing Cost
45°	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
91 / 81	%	Sulphide Copper Recovery (Fresh/Transitional)
51 / 51	%	Sulphide Gold Recovery (Fresh/Transitional)
57 / 57	%	Sulphide Molybdenum Recovery (Fresh/Transitional)
40 / 40	%	Sulphide Silver Recovery (Fresh/Transitional)

The controlling surface for Alice RPEEE is shown in Figure 14.24. This also includes sensitivity testwork completed at different copper prices, showing that the RPEEE surface is not particularly sensitive to copper price at Alice.

Figure 14.24 : Cross Section View Looking North Showing Pit Shell Surface Shapes Used to Define Reasonable Prospects of Eventual Economic Extraction by Open Pit Mining



14.3 Cortadera

14.3.1 Introduction

The current Cortadera Mineral Resource was updated in February 2024 for the combined Costa Fuego Mineral Resource Release (26 February 2024).

14.3.2 Summary of Data Used in Estimate

All drilling data is stored in the HCH exploration acQuire drill hole database. The system is backed up daily to a server based in Perth. Five data tables were exported from the acQuire drill hole database to create the estimation database.

Data table records consist of:

- Collar – Collars.csv
- Survey – Survey.csv
- Assays – Cort_AssayPref.csv
- Copper soluble assays – CuSol.csv
- Lithology – Cort_Lith.csv
- Weathering – Weathering.csv
- Alteration – Alteration.csv
- Mineralisation – Cort_Minerisation.csv.

The drill hole database used for the Mineral Resource Estimate (MRE) has been validated by several methods including checking of QA/QC data, extreme outlier values, zero values, negative values, possible miscoded data based on geological domaining and assay values, sample overlaps, and inconsistencies in length of drill hole surveyed, length of drill hole logged and sampled, and sample size at laboratory.

No holes were excluded from the Cortadera Resource Estimate.

A summary of the drilling data used to compile the estimate is listed in Table 14.67. Note that this table includes all drillholes containing the Cortadera Project Code (i.e., includes some regional Cortadera exploration drilling).

Table 14.66 : Drill Hole Included in Estimate Database

Drilling Method	Holes	Metres
RC	148	28 878
DD	45	30 073
RC with DD tail	58	48 849

14.3.3 Data Manipulation

Negative-grade values are used in the assay database table to indicate specific events or conditions. To ensure these assays are suitable for use in the mineral resource, edits were made to remove any non-numeric, zero and negative values.

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 (0.0025 g/t Au, 0.005% Cu, 0.5 ppm Mo, 0.005 ppm Ag)
- If the assay value was equal to zero and there was no recorded weight of sample the assay was set to 'absent'.

14.3.4 Geological and Mineralisation Interpretation

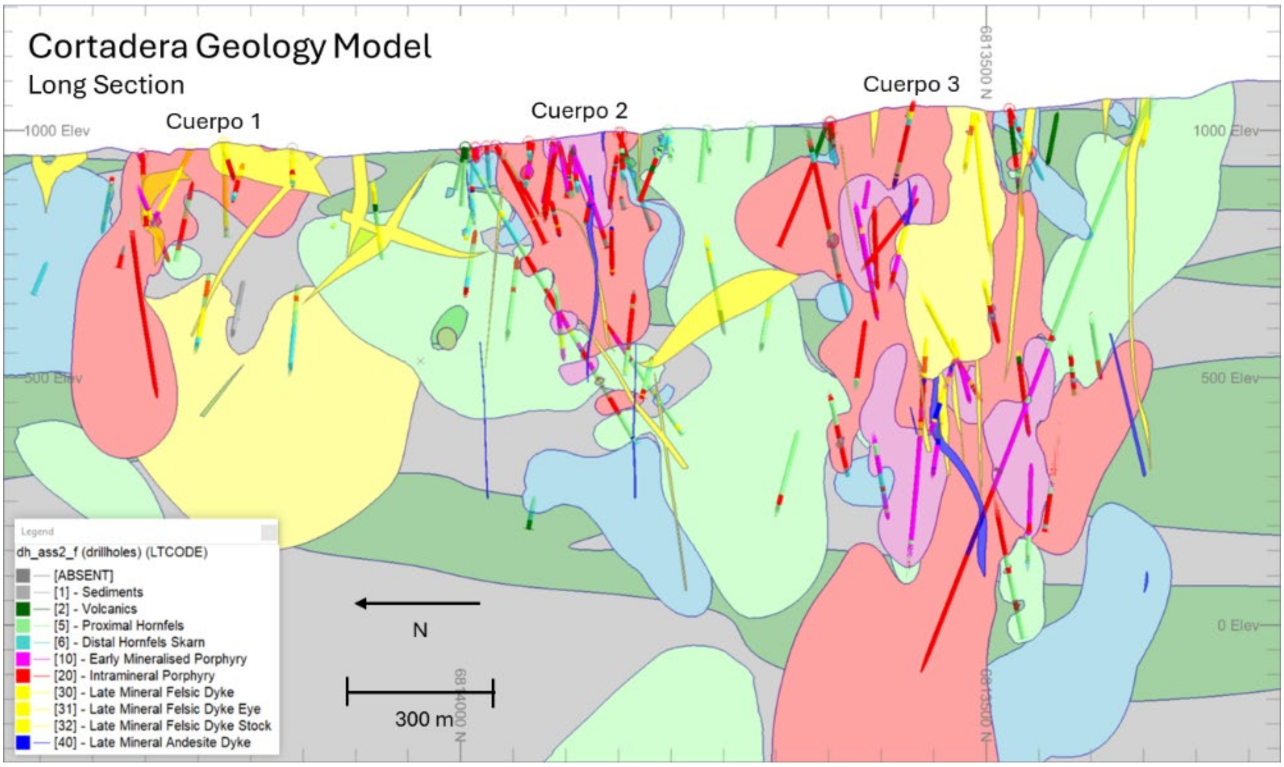
14.3.4.1 Geological Model

The geological model is based on surface mapping, diamond and RC drill hole logging, assay data (including rock geochemistry) and field data.

The Cortadera deposit is characterised by early- (10-series) and intra-mineralisation (20-series), porphyritic tonalitic to quartz dioritic intrusions and adjacent volcano-sedimentary wall-rocks that have been recrystallised to hornfels and skarn. The hydrothermal alteration consists of moderate- to strong-phyllic (+ chloritic) alteration, characterised by quartz/silica, sericite, and lesser amounts of chlorite.

Figure 14.25 shows a long-section view of the lithology model used to define the mineral resource.

Figure 14.25 : Long Section Looking Northeast (+-20m) Showing Lithological Domains Comprising Cuerpo 1, 2 and 3 (HCH, 2024)



Lithological domains were built using the intrusion tool in Leapfrog Geo. In areas of sparser data density, edits were made using polylines and points which forced the interpretation to conform with the broader geological model.

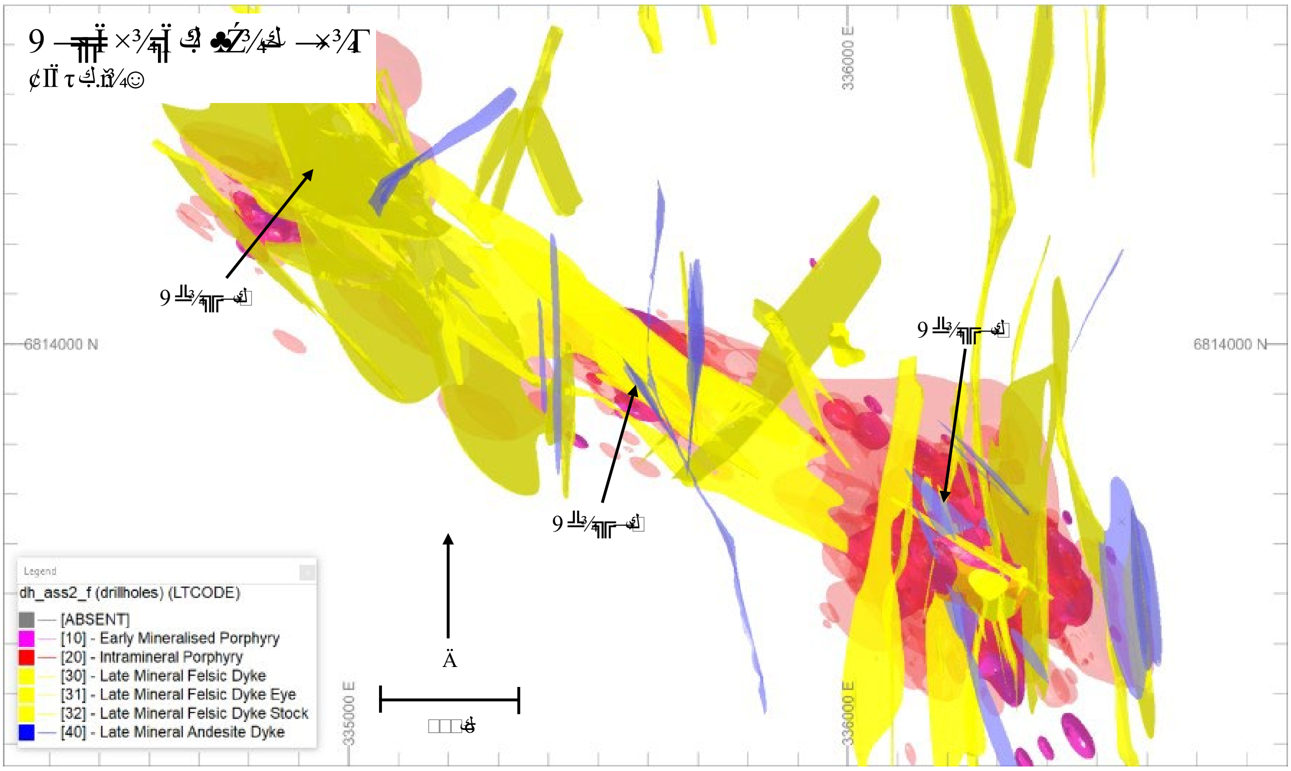
The felsic and andesitic dykes (30- and 40-series) are more discrete geological units, with clear hanging wall and footwall contacts, and have been modelled with the vein system tool using a combination of drill hole and surface mapping data.

To build the vein models, an interval selection table is created so intervals can be assigned to a specific dyke (using nomenclature i.e. 30_01 and 40_01). Polylines based upon surface mapping were digitised on topography. Additional polylines have been used to guide the interpretation, especially in areas where drill holes are oriented sub-parallel to the dyke.

The orientations of the overall dyke package were best supported by surface mapping, with only a few structural measurements from diamond core at Cortadera. The orientation is currently interpreted as predominantly north-south striking with near-vertical dip (although there is some northwest-southeast oriented 30- and 40-series dykes).

Figure 14.26 shows a plan view of the dyke model used to inform the mineral resource estimate, relative to 10- and 20-series porphyry models.

Figure 14.26 : Plan View Showing the Cortadera Dyke Model Used for February 2024 Mineral Resource Estimate (HCH, 2024)



The dyke systems are significant to the mineral resource estimate as they comprise barren volumes which need to be depleted from the block model. Due care has been taken to model dykes as accurately as possible. It is also noted that it is not possible to model all the dykes present at Cortadera, as continuity is not always exhibited between multiple drill holes.

The volume of barren 30- and 40-series dykes which has not been modelled is not considered material to the mineral resource.

Dyke validation within Leapfrog Geo involved merging data (copper assay data, lithology, and total quartz-vein abundance), and applying a query filter filtering and cross-checking against the drill hole logs. This aimed to reduce any intervals within the dyke volume which were not logged as dyke (i.e. were logged as 10- or 20-series porphyry) and that graded greater than 0.4 Cu% (first pass) and greater than 0.1 Cu% (second pass).

In some instances, validation checks resulted in intervals logged as 30- or 40-series dyke being re-logged. These intervals were corrected in the acQuire database and the datasets re-exported.

14.3.5 Mineralisation Model

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 Geo independently for each estimated element using cut-off grades guided by breaks in each element’s grade distribution, shown in Table 14.68 below.

Mineralisation domains for gold, silver and molybdenum were created using grade interpolants on validated drill holes, composited to 10 m.

Copper mineralisation domains were created using a set of geological conditions (as described below) on validated drill holes composited to 10 m intervals.

- 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>
<tr><th colspan="5">Table 14.67 : Cut-Off Grades Used for Cortadera Mineralisation Domains</th></tr>
<tr><th>Cuerpo</th><th>Element</th><th>HG Domain Cut-off (DOM = 20)</th><th>LG Domain Cut-off (DOM = 30)</th><th>SLG Domain Cut-off (DOM = 50)</th></tr>
<tr><td rowspan="4">1</td><td>Cu</td><td>1.0% cpy</td><td>0.15% Cu</td><td rowspan="4">0.05% CuEq</td></tr>
<tr><td>Au</td><td>0.07 g/t Au</td><td>-</td></tr>
<tr><td>Ag</td><td>1.0 g/t Ag</td><td>0.6 g/t Ag</td></tr>
<tr><td>Mo</td><td>70 ppm Mo</td><td>50 ppm Mo</td></tr>
</table>

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% copper equivalent (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 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.

14.3.6 Weathering Domains

The modelling of weathering domains is based on:

- Copper species flagging using Cu:S ratio

- Total contained sulphide (S^{2-}) percent
- Regolith logging from diamond core and RC chips (validated by viewing photographs, where possible).

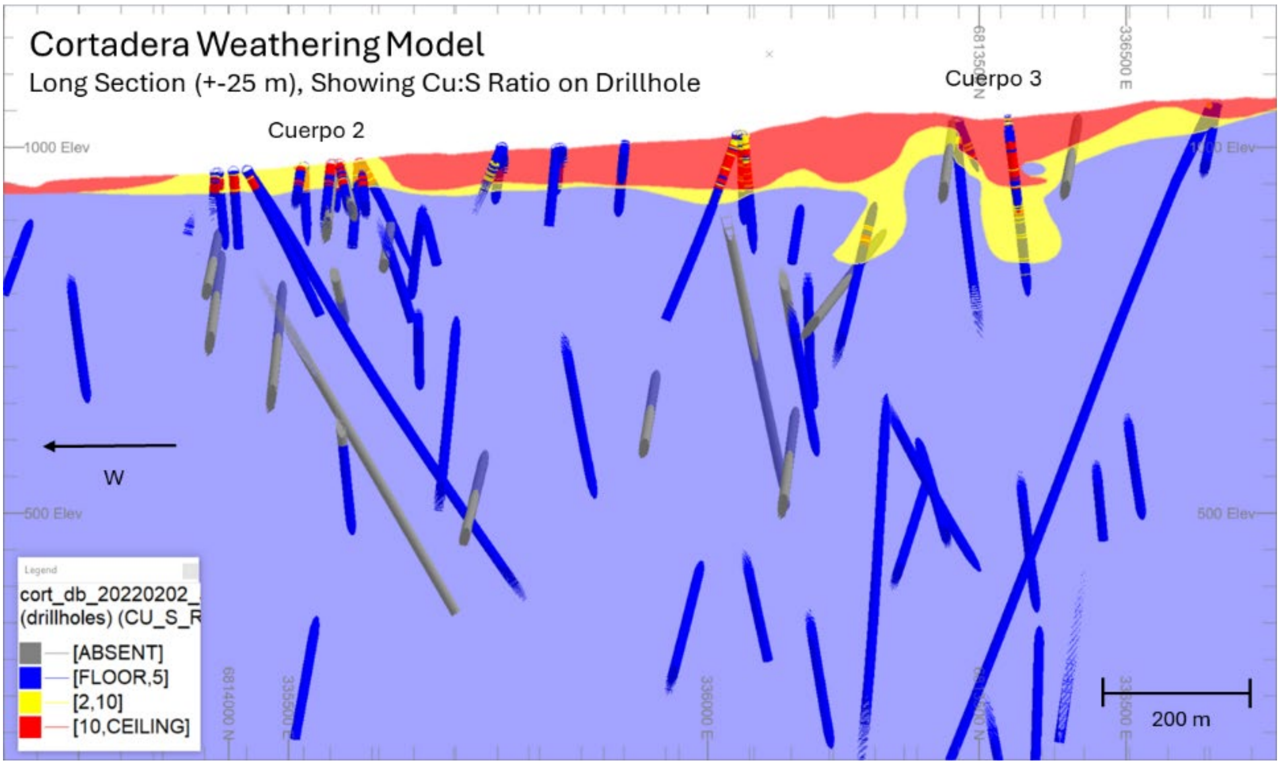
Where a quantitative method was used for interpretation of the weathering domains (i.e. Cu:S or total contained sulphide), validation was completed using the regolith logging from diamond core and RC chips.

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 14.27 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 14.27 : Cross Section (Looking North-West) Showing Final Weathering Model (Fresh is Blue, Transitional is Yellow, Oxide is Red) Compared to Drill Holes Displaying Cu:S



14.3.7 Data Flagging and Compositing

Geology, mineralisation, and weathering wireframes were used to flag the drill hole data and block model. Additional wireframes were used to flag each of the porphyry intrusions (Cuerpos). Wireframes used to create the volume model are listed in Table 14.69.

Table 14.68 : List of Final Wireframes Used for Estimation Coding

Wireframe	Field	Code	Description	Type
cuerpo1_extent	CUERPO	1	Cuerpo 1 model area	Solid
cuerpo2_extent	CUERPO	2	Cuerpo 2 model area	Solid
cuerpo3_extent	CUERPO	3	Cuerpo 3 model area	Solid
06_20230905	LTCODE	6	Distal Skarn lithology coding	Solid
05_20230905	LTCODE	5	Proximal Skarn lithology coding	Solid
02_20230905	LTCODE	2	Volcanic host coding	Solid
20_20230905	LTCODE	20	Intramineral porphyry coding	Solid
10_20230905	LTCODE	10	Early mineralised porphyry coding	Solid
30_dyke_stock_20230905	LTCODE	32	Late mineral felsic dyke stock coding	Solid
30_dyke_eye_20230905	LTCODE	31	Late mineral felsic dyke eye coding	Solid
30_dyke_system_20230905	LTCODE	30	Late mineral felsic dyke system coding	Solid
40_dyke_system_20230905	LTCODE	40	Late mineral andesitic dyke system coding	Solid
topo_2023	TOPO	10000	Topographic surface coding	Surface
20230408_weath_oxide	WTCODE	1	Oxide weathering coding	Solid
20230408_weath_trans	WTCODE	2	Transitional weathering coding	Solid
20230408_weath_fresh	WTCODE	3	Fresh rock weathering coding	Solid

14.3.8 Mining and Tenement Flagging in the Model

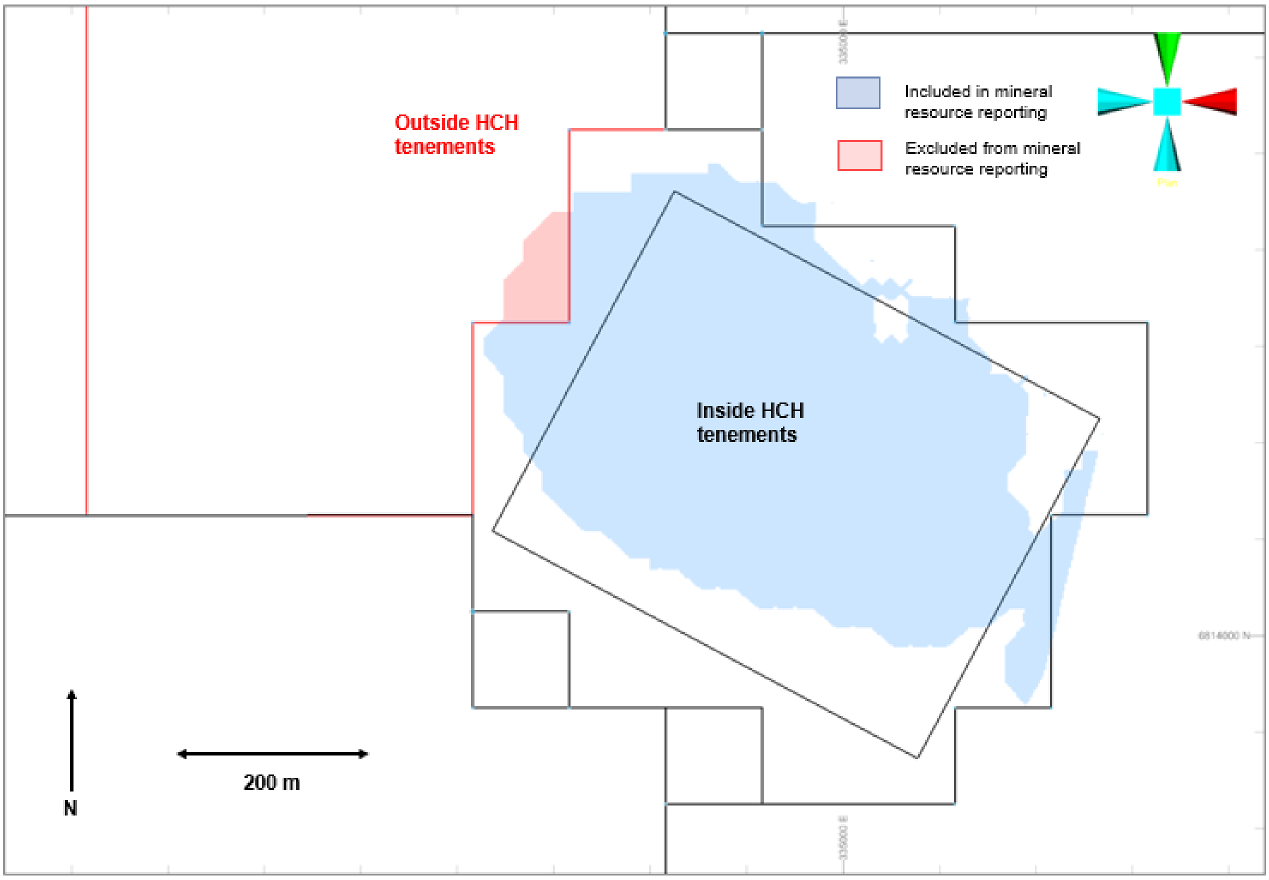
Cortadera tenements are flagged into the 'TENEMENT' field in the final estimate using binary code (0 = inside of HCH tenement boundary and 1 = outside of tenement boundary).

As shown below (Figure 14.28) a small section of Cuerpo 1 lies outside the current HCH tenement boundary. This material has been excluded from the Mineral Resource reported in February 2024.

In November 2022, HCH executed an Option Agreement with Antofagasta Minerals S.A. (AMSA) to acquire a 100% interest in five mining rights, including those hosting the remainder of the Cortadera Mineral Resource (see 'Hot Chili Executes Option to Secure Major Extension to Cortadera' dated 28 November 2022).

Only limited surface workings have been completed in the Cortadera model area. Depletion of this material is completed by use of a recent topography model.

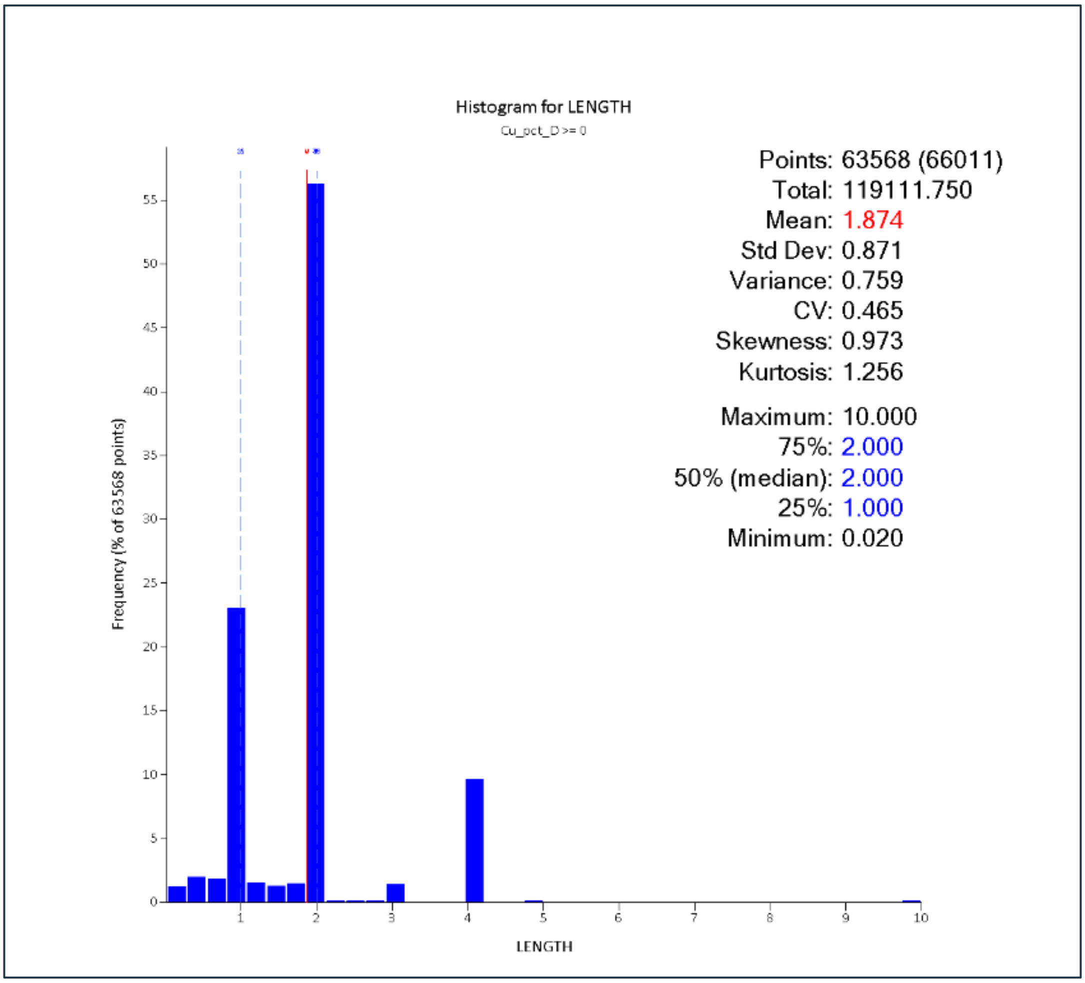
Figure 14.28 : Plan View of the Area of Cuerpo 1 Which Lies Outside the Current HCH Tenement Boundary



14.3.9 Compositing

80% of the drill sampling at Cortadera was at either 1 m or 2 m intervals (Figure 14.29). This proportion increased to 95% within mineralised domains where copper was greater than 0.1%. 2 m was chosen as a suitable composite length, with the compositing process using mineralisation, lithology, and weathering domains as boundaries to ensure no composites were created across domains.

Figure 14.29 : Histogram of Cortadera Sample Lengths



14.3.10 Statistical Analysis

The statistical, spatial, and geological characteristics of the data were examined within each estimation domain.

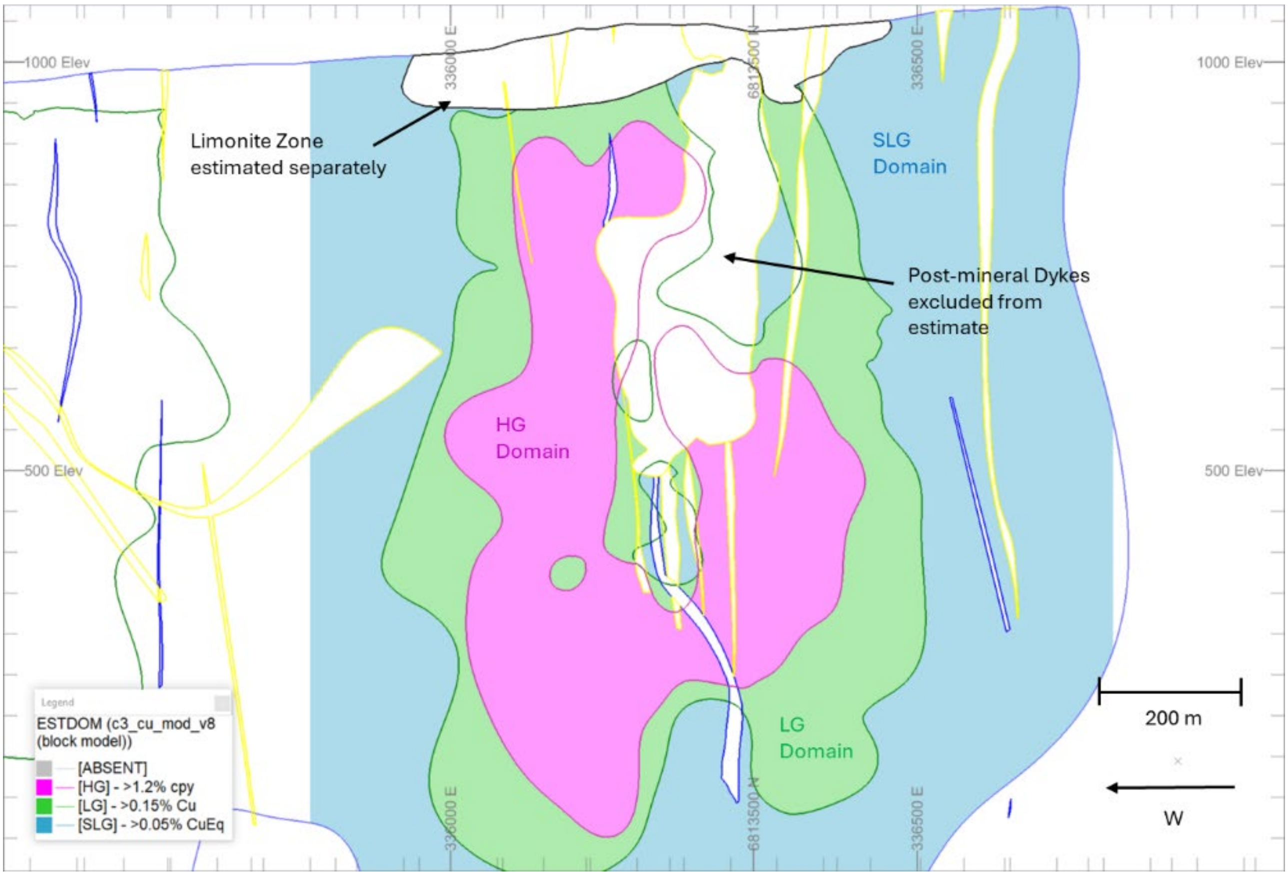
The SLG which comprises the low-grade boundary was analysed individually for each Cuerpo, but the output statistics suggested no hard boundary existed. For this reason, the SLG domain was analysed as a single domain straddling all three Cuerpos.

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 limonite, copper and gold estimates, further domaining was applied through use of the categorical indicator kriging estimation methodology (CIK).

Figure 14.30 below shows the domaining applied for the estimation of Copper in Cuerpo 3.

Figure 14.30 : Oblique Cross Section of Cuerpo 3 Showing Cu% Estimation Domains Used for Mineral Resource



14.3.11 Categorical Domaining

14.3.11.1 Strategy

The 2024 Cortadera model update uses categorical indicator kriging for copper (Cuerpo 1, 2, and 3) and gold (Cuerpo 2 and 3) to help define final domains for estimation.

14.3.11.2 Categorical Domaining Data

Drill hole data was coded with binary indicator fields (‘1’ being above the grade/value specified, ‘0’ being below) as described in Table 14.70.

Table 14.69 : 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

14.3.11.3 Categorical Domaining Variography

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 14.71.

Table 14.70 : 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
		-140	90	20	0.33	0.13	30	0.24	80	0.30	140

Table 14.70 : 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						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

14.3.11.4 Categorical Domaining Estimation

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 14.72.

Table 14.71 : 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	Rot 3	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 14.71 : Search Strategy for Categorical Estimation

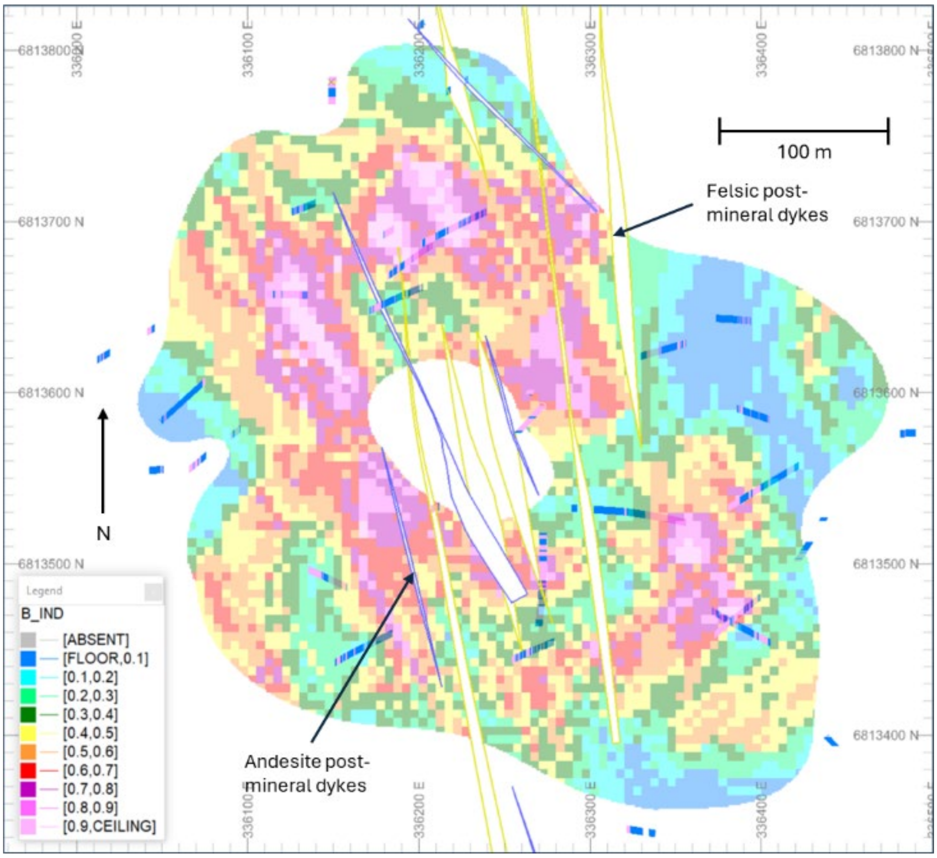
AREA	Domain	Orientation			Search			2nd Search Factor	No. of Composites				
									First Search		Second Search		Max Per Drill Hole
		Rot1	Rot2	Rot 3	D1	D2	D3		Min	Max	Min	Max	
Cuerpo 1	DOM_CU1 = 50	-140	80	20	200	100	100	1.5	10	20	10	20	6
Cuerpo 2	DOM_CU1 = 20	-130	60	60	90	70	50	1.5	10	20	10	20	6
Cuerpo 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.

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 14.31 below.

Figure 14.31 : Indicator Estimate on Binary Fields Showing the Probability of a Block Being Above 0.5% for Cuerpo 3 HG Domain. Plan View Section at 450 mRL

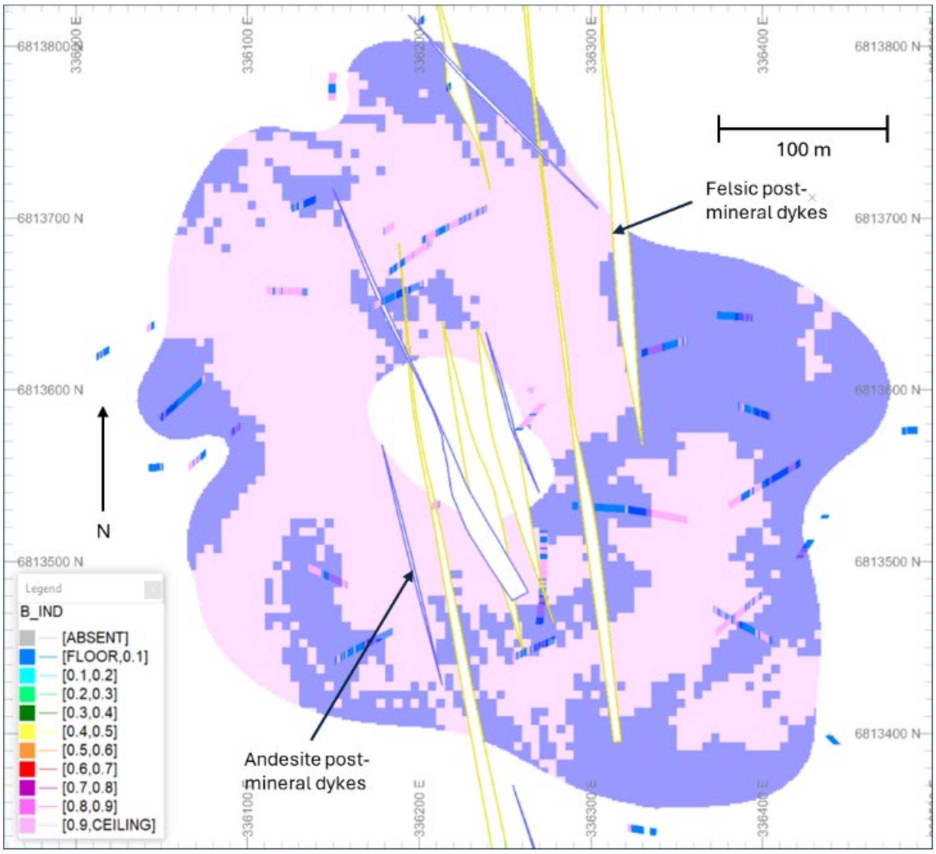


14.3.11.5 Estimation Subdomains

Estimation subdomains were coded based on indicator thresholds as shown in Table 14.73. An example of indicator estimate subdomains for the Cuerpo 3 HG domain (DOM_CU1=20) is shown Figure 14.32.

Table 14.72 : 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
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 14.32 : Indicator Estimate Subdomains for Cuerpo 3 HG Domain . Plan View Section at 450 mRL



14.3.12 Top-Cuts

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), an subsequent estimation of grades. Using outliers in an estimate can result in material overestimation of grade and metal. Top-cuts were selected for each element based on statistical analysis of the data (log histograms, log probability plots, and grade disintegration) and are summarised in Table 14.74.

Table 14.73 : Cortadera Mineralised Domains Top-Cut Analysis

Element	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

Table 14.73 : Cortadera Mineralised Domains Top-Cut Analysis

Element	AREA	Domain	CIK Sub-domain	Raw				Top cut			
				Number	Max	Mean	CV	Value	# Cut	Mean	CV
		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
Ag	Cuerpo 1	DOM_AG1 = 20	-	444	8.70	1.23	0.82	-	-	1.23	0.82
		DOM_AG1 = 30	-	775	10.3	0.69	1.18	5.0	5	0.68	1.11
		DOM_AG1 = 50	-	4 907	3.00	0.16	1.93	1.0	131	0.15	1.80
	Cuerpo 2	DOM_AG1 = 20	-	440	9.00	1.23	0.95	-	-	1.23	0.95
		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

Table 14.73 : Cortadera Mineralised Domains Top-Cut Analysis

Element	AREA	Domain	CIK Sub-domain	Raw				Top cut			
				Number	Max	Mean	CV	Value	# Cut	Mean	CV
	Cuerpo 3	DOM_AG1 = 20	-	2 932	100	1.28	1.56	3.5	30	1.22	0.51
		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	Cuerpo 1	DOM_MO1 = 20	-	429	523	91.9	0.68	250	14	89.8	0.61
		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
	Cuerpo 2	DOM_MO1 = 20	-	195	581	122	0.78	350	6	118	0.68
		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
	Cuerpo 3	DOM_MO1 = 20	-	817	2 980	339	0.91	1 250	14	328	0.77
		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 the one-way semi-soft boundaries, 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 semi-soft one-way 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 14.75.

Table 14.74 : 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

14.3.13 Variography

Snowden Supervisor Version 8.14.3.2 software was used to generate and model the variograms for each of the elements, domains, and indicator subdomains.

A normal scores transformation was applied to the element data prior to the construction of the experimental variogram. Final variograms were then back-transformed to traditional space adjusting for variance.

Knowledge of the underlying geological conditions was always considered when constructing variograms. Variogram orientations were overlain onto each lode during the validation step to ensure trends are coherent and agree with the broader geological model.

Care was taken to ensure reasonable correlation between sub-domains (orientations, nugget effect and variogram ranges). For instance, it would not be considered reasonable for this style of deposit to have variogram orientations for adjacent sub-domains at right angles to one another.

The variogram model parameters used for grade estimation of key elements are shown below (Table 14.76 to Table 14.79).

Table 14.75 : Grade Variogram Models – Cu Estimate

AREA	Domain	CIK Sub 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
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
								40		80		200

Table 14.75 : Grade Variogram Models – Cu Estimate

AREA	Domain	CIK Sub 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
								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
	DOM_CU1 = 30	HG_CIK	-120	70	80	0.34	0.38	25	0.13	50	0.15	220
								25		50		150

Table 14.75 : 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
		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	0.15	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

Table 14.76 : 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
Cuerpo 1	DOM_CU1 = 20, 30	-	-150	80	90	0.33	0.29	15	0.20	50	0.18	150
								15		50		150
								15		30		75
	DOM_CU1 = 50	-	-150	80	0	0.18	0.33	50	0.25	150	0.24	350
								20		70		200
								20		70		200
Cuerpo 2	DOM_CU1 = 20	HG_CIK	-140	60	160	0.17	0.30	30	0.23	50	0.30	225
								10		25		150
								10		25		100
		LG_CIK	-140	60	160	0.17	0.30	30	0.23	50	0.30	225
								10		25		150
								10		25		100
	DOM_CU1 = 30	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
		HG_CIK	-150	70	0	0.19	0.27	50	0.23	100	0.32	200

Table 14.76 : Grade Variogram Models – Au Estimate

AREA	Domain	CIK Sub 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
								20		50		200
								20		50		150
	DOM_CU1 = 50							50		100		200
		LG_CIK	-150	70	0	0.19	0.27	20	0.23	50	0.32	200
								20		50		150
								30		50		110
		HG_CIK	-150	70	70	0.39	0.10	30	0.12	50	0.39	110
								30		50		110
	DOM_CU1 = 20							20		50		150
		LG_CIK	-110	60	90	0.23	0.21	20	0.27	50	0.17	125
								20		50		100
								25		50		220
		HG_CIK	-120	70	80	0.34	0.38	25	0.13	50	0.13	150
								25		50		150
Cuerpo 3	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	0.13	150
								25		50		150
	DOM_CU1 = 50							60		100		150
		LG_CIK	-130	100	50	0.34	0.18	40	0.31	80	0.17	125
								40		60		100

Table 14.77 : Grade Variogram Models – Mo Estimate

AREA	Domain	CIK Sub 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
								80		240		450
	DOM_MO1 = 20	-	60	170	80	0.16	0.17	80	0.36	240	0.30	450
Cuerpo 1								80		240		450
								60		95		200
	DOM_MO1 = 30	-	50	120	180	0.31	0.28	20	0.23	40	0.18	120
								20		40		120

Table 14.77 : Grade Variogram Models – Mo Estimate

AREA	Domain	CIK Sub Domain	Rotation (Datamine ZXZ)			Nugget	Structure 1		Structure 2		Structure 3	
						C_0	C_1	R_1	C_1	R_1	C_1	R_1
	DOM_MO1 = 50	-	40	110	150	0.19	0.26	55	0.29	150	0.26	250
								20		70		200
								20		70		175
Cuerpo 2	DOM_MO1 = 20	-	0	0	-90	0.20	0.80	200	-	-	-	-
								200		-		-
								200		-		-
	DOM_MO1 = 30	-	0	0	-90	0.20	0.80	200	-	-	-	-
								200		-		-
								200		-		-
	DOM_MO1 = 50	-	-140	50	90	0.24	0.30	50	0.21	100	0.26	230
								50		100		230
								50		100		230
Cuerpo 3	DOM_MO1 = 20	-	-110	100	70	0.48	0.14	20	0.30	100	0.08	200
								20		75		125
								20		75		125
	DOM_MO1 = 30	-	-10	160	90	0.47	0.34	40	0.10	120	0.10	250
								20		80		150
								10		20		50
	DOM_MO1 = 50	-	40	150	-90	0.05	0.32	30	0.32	120	0.36	320
								30		120		320
								10		120		300

Table 14.78 : Grade Variogram Models – Ag Estimate

AREA	Domain	CIK Sub Domain	Rotation (Datamine ZXZ)			Nugget	Structure 1		Structure 2		Structure 3	
						C_0	C_1	R_1	C_1	R_1	C_1	R_1
Cuerpo 1	DOM_AG1 = 20	-	-170	110	150	0.36	0.27	120	0.15	160	0.24	215
								120		160		215
								80		120		175
	DOM_AG1 = 30	-	140	100	0	0.38	0.28	50	0.20	125	0.14	320
								50		125		320
								50		125		200
	DOM_AG1 = 50	-	-160	40	90	0.18	0.58	120	0.16	200	0.08	400
								90		180		350

Table 14.78 : Grade Variogram Models – Ag Estimate													
AREA	Domain	CIK Sub 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	
								90		180		350	
Cuerpo 2	DOM_AG1 = 20	-	0	0	-90	0.20	0.80	200	-	-	-	-	
								200		-		-	
								200		-		-	
	DOM_AG1 = 30	-	-165	100	40	0.22	0.18	80	0.20	250	0.40	400	
								80		250		300	
								50		100		200	
	DOM_AG1 = 50	-	-110	130	90	0.28	0.39	50	0.21	150	0.12	350	
								50		100		200	
								50		80		150	
Cuerpo 3	DOM_AG1 = 20	-	-150	100	80	0.35	0.12	20	0.22	40	0.30	100	
								20		40		100	
								15		30		70	
	DOM_AG1 = 30	-	60	110	-90	0.06	0.47	30	0.24	80	0.23	300	
								30		80		250	
								30		50		150	
	DOM_AG1 = 50	-	0	0	-90	0.20	0.80	200	-	-	-	-	
								200		-		-	
								200		-		-	

14.3.14 Block Modelling

Two different parent block sizes were used for the grade estimates to account for different data densities present in separate areas of the deposit.

- 20 mE by 20 mN by 20 mRL
- 10 mE by 10 mN by 10 mRL.

Sub-blocking to a minimum of 2 m was completed to honour the volume and geometry of the wireframes.

Kriging neighbourhood analysis ensured the suitability of parent block sizes selected.

Block model parameters are shown in Table 14.80.

Table 14.79 : Block Model Dimensions						
Dimension	**Minimum**	**Maximum**	**Extent**	**Size**	**Number**	**Sub-cell**
Easting (mE)	334 100	337 100	3 000	10	300	2
Northing (mN)	6 812 700	6 814 780	2 080	10	208	2
Elevation (mRL)	-400	1 280	1 680	10	168	2
Easting (mE)	334 100	337 100	3 000	20	150	2
Northing (mN)	6 812 700	6 814 780	2 080	20	104	2
Elevation (mRL)	-400	1 280	1 680	20	84	2

14.3.15 Grade Estimation

A suite of elements (copper, gold, molybdenum, silver, cobalt, calcium, iron, sulphur, potassium, and aluminium) was estimated using 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.

Kriging neighbourhood analysis was completed for each domain (or CIK subdomain) and used as a guide for the selection of sample counts, search distances, discretisation points and block size.

The search strategy for each element and domain is shown in Table 14.81 to Table 14.84 below.

The composited data was top-cut prior to estimation as discussed in Section 14.3.12.

No octant search was used.

Table 14.80 : 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
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

Table 14.80 : 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	
	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 14.81 : 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

Table 14.82 : Search Strategy for Grade Estimation – Ag

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_AG1 = 20	-	-170	110	150	140	140	120	1.5	8	18	8	18	6
	DOM_AG1 = 30	-	140	100	0	210	210	130	1.5	8	18	8	18	4

Table 14.82 : Search Strategy for Grade Estimation – Ag

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	
	DOM_AG1 = 50	-	-160	40	90	260	230	230	1.5	10	20	10	20	4
Cuerpo 2	DOM_AG1 = 20	-	0	0	-90	100	100	100	1.5	10	20	10	20	6
	DOM_AG1 = 30	-	-165	100	40	260	200	130	1.5	10	20	10	20	4
	DOM_AG1 = 50	-	-110	130	90	230	130	100	1.5	12	22	12	22	4
Cuerpo 3	DOM_AG1 = 20	-	-150	100	80	100	100	50	1.5	10	20	10	20	6
	DOM_AG1 = 30	-	60	110	-90	200	170	100	1.5	10	20	10	20	4
	DOM_AG1 = 50	-	0	0	-90	100	100	100	1.5	10	20	10	20	4

Table 14.83 : Search Strategy for Grade Estimation – Mo

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_AG1 = 20	-	60	170	80	300	300	300	1.5	8	18	8	18	6
	DOM_AG1 = 30	-	50	120	180	130	80	80	1.5	10	20	10	20	6
	DOM_AG1 = 50	-	40	110	150	170	130	120	1.5	10	20	10	20	6
Cuerpo 2	DOM_AG1 = 20	-	0	0	-90	300	300	300	1.5	8	18	8	18	5
	DOM_AG1 = 30	-	0	0	-90	300	300	300	1.5	10	20	10	20	4
	DOM_AG1 = 50	-	140	50	90	150	150	150	1.5	10	20	10	20	6
Cuerpo 3	DOM_AG1 = 20	-	-110	100	70	130	80	80	1.5	10	20	10	20	6
	DOM_AG1 = 30	-	-10	160	90	160	100	40	1.5	12	22	12	22	6
	DOM_AG1 = 50	-	40	150	-90	200	200	180	1.5	10	20	10	20	5

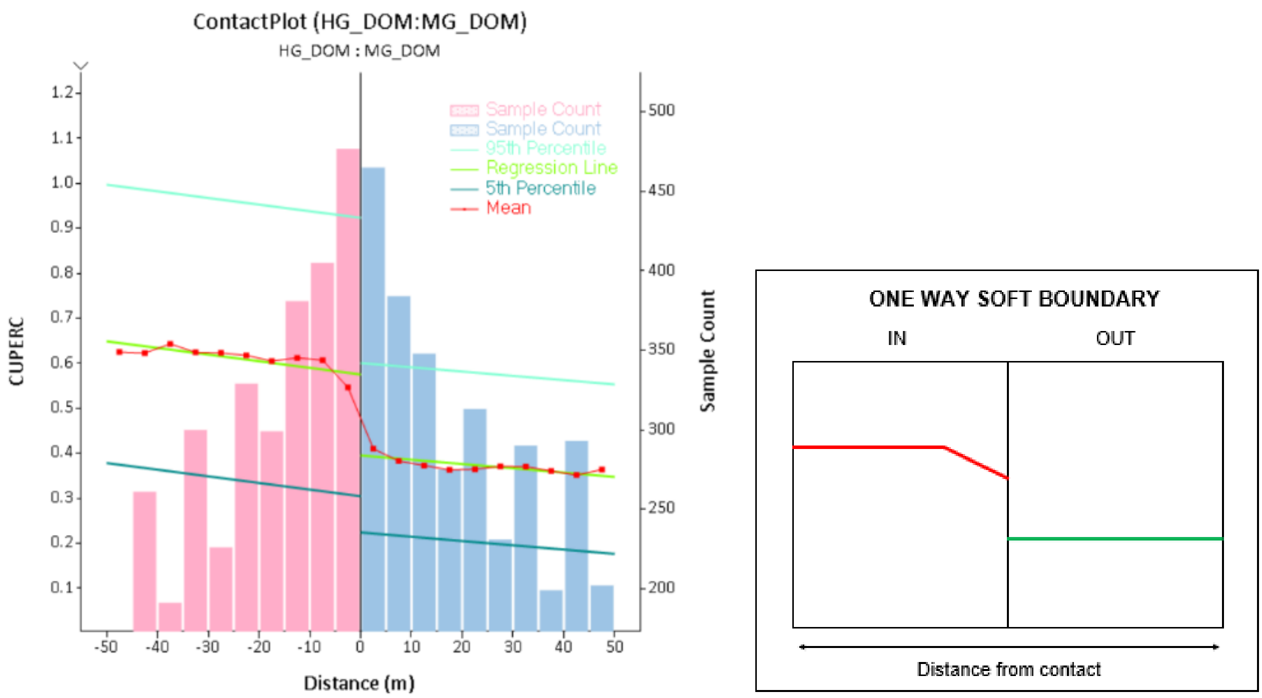
Mineralisation was estimated using one-way semi-soft boundaries according to the domain conditions for each element. This approach is based on the observation that the mineralised system comprises a high-grade “core” with gradational copper grade decreasing outwards to the edge of the porphyry intrusion 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 14.33).

For copper and gold estimates, high-grades across the semi-soft boundary are controlled using the ‘cap distance’ in Datamine, discussed in Section 14.3.12.

The boundaries between oxidation states were soft.

Figure 14.33 : 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 14.85 details the soft boundaries used for each estimate, including minimum and maximum sample counts for the estimate.

Table 14.84 : 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.

14.3.15.1 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.

14.3.16 Bulk Density

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 14.86.

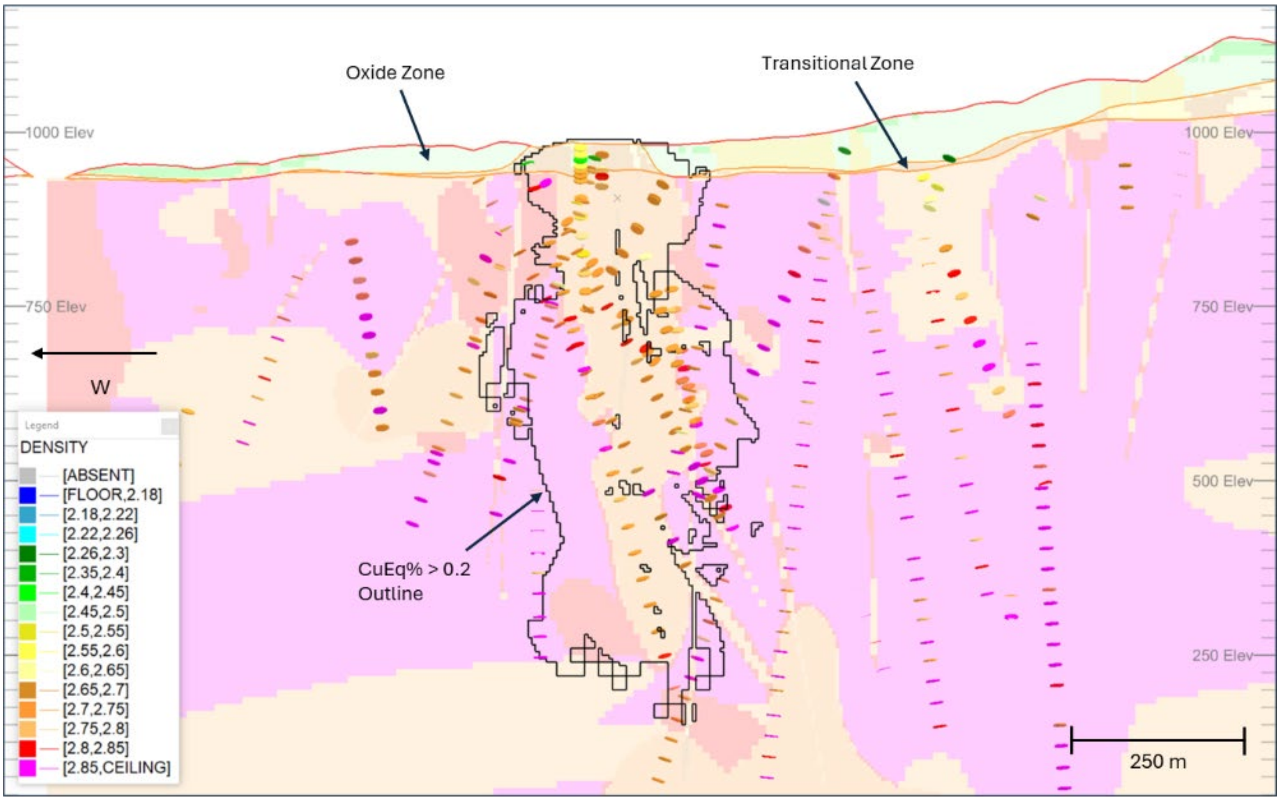
Table 14.85 : Bulk Density Value Assignment

LTCODE	WTCODE	DENSITY ASSIGNED
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

Table 14.85 : Bulk Density Value Assignment		
LTCODE	WTCODE	DENSITY ASSIGNED
32 = Late Mineral Dyke stockwork		2.77
40 = Post Mineral Andesitic Dyke		2.64

Figure 14.34 shows the densities as coded in the final block model.

Figure 14.34 : Cross Section (at 6 813 900 mN) looking North Through Cuerpo 2 Showing the Bulk Density Sampling and Assignment to the Block Model



14.3.17 Estimate Validation

Estimates for all elements were validated using a three-stage comparison between top-cut composites and the estimated variables.

The first stage involved calculating the global statistics of the composites compared to the tonnage weighted averages of estimated variables. The second stage involved comparing statistics in elevation slices along the mineralisation and the third involved a detailed visual comparison by section to ensure the estimated variables honour the input composite data.

CIK sub-domains were validated independently (by sub-domain) and as a combined estimate and dataset (sub-domains combined).

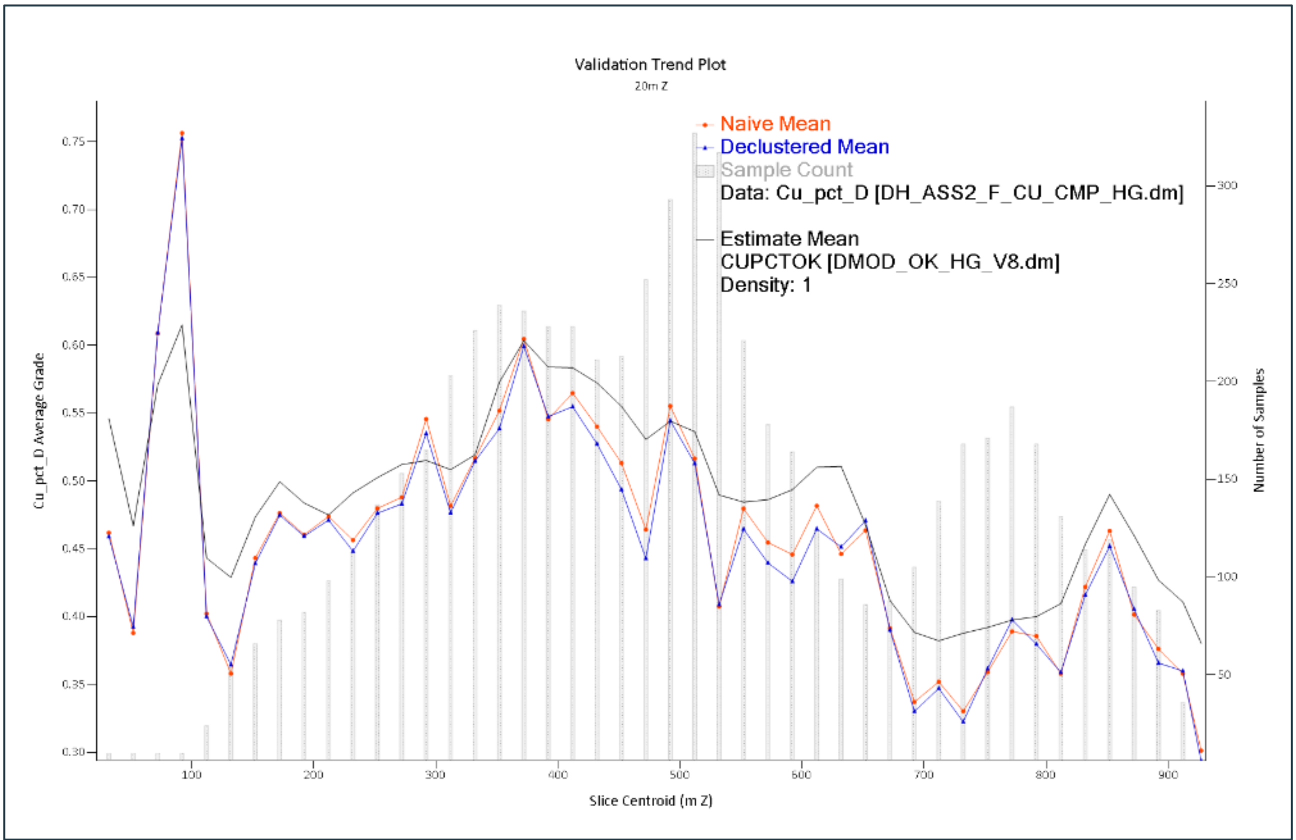
The comparison of average grade of the composites to the block grade estimate (excluding blocks estimated on third pass) for a selection of elements is in Table 14.87.

Table 14.86 : Estimation Validation – Comparison of Top-Cut, Declustered 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%

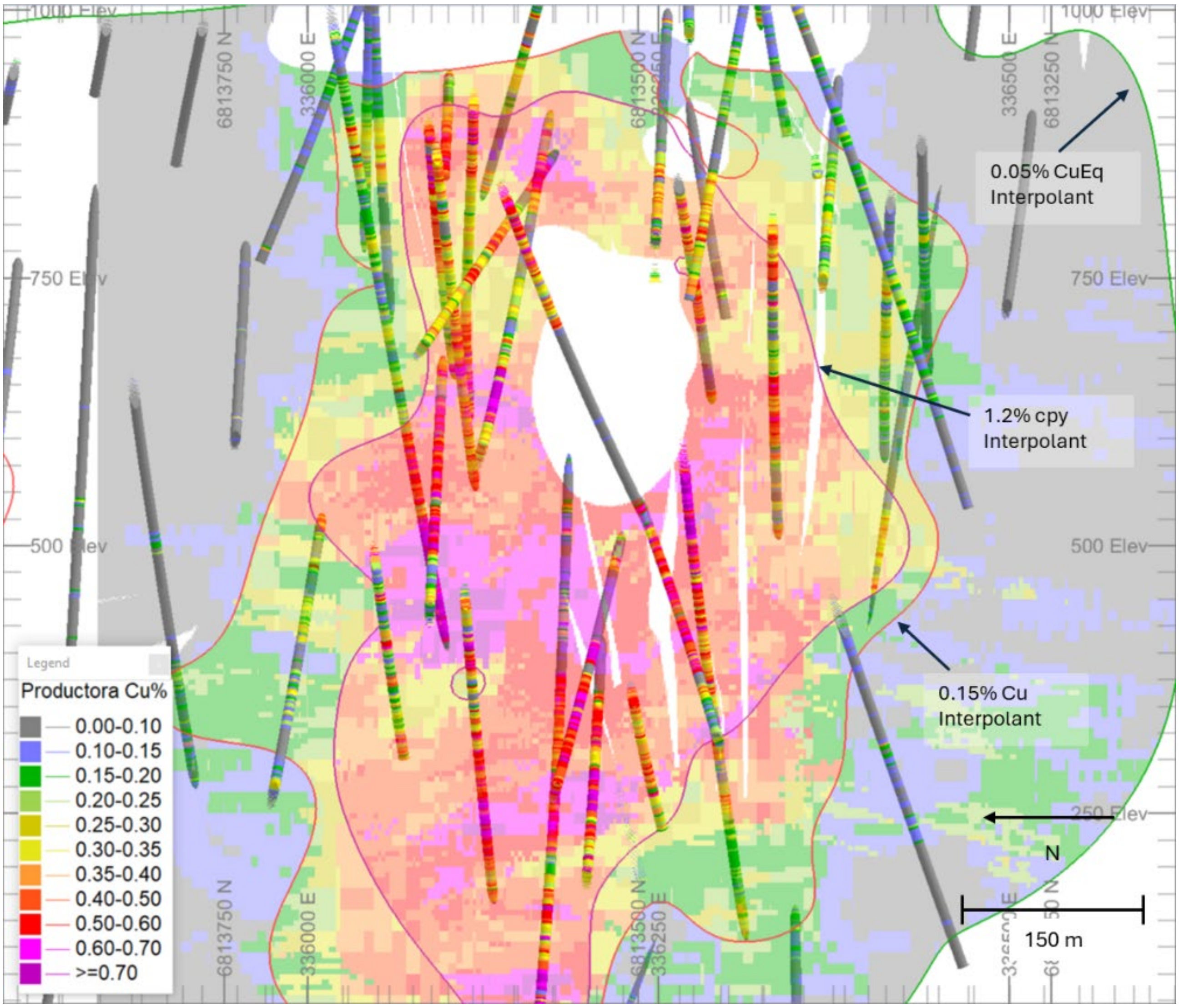
Trend analysis was completed within each domain in all three dimensions (x-axis, y-axis, and z-axis). An example is shown in Figure 14.35 for the High-Grade Cuerpo 3 copper estimate.

Figure 14.35 : 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 14.36). 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 14.36 : Visual Validation of Drill Holes Versus Modelled Cu Grade. Plan View (Top) Showing Location of Sections A-A′ and B-B′; Cross Sections Shown on Bottom



14.3.18 Resource Classification

The Cortadera Mineral Resource estimate has been classified considering the 2014 CIM Definition Standards and were estimated in accordance with the CIM 2019 Best Practices Guidelines, as required by NI43-101.

A range of criteria was considered in determining the Resource classification, including:

- Drill data density
- Sample/assay confidence
- 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 in Datamine to control the extents of classification, with the following file names:

- Cuerpo 1 Inferred – 20230623_rescat_3_inf_c1
- Cuerpo 2 Inferred – 20230623_rescat_3_inf_c2
- Cuerpo 3 Inferred – 20230623_rescat_3_inf_c3
- Cuerpo 1 Indicated – 20230623_rescat_2_ind_c1
- Cuerpo 2 Indicated – 20230623_rescat_2_ind_c2
- Cuerpo 3 Indicated – 20230623_rescat_2_ind_c3

No Measured resource has been defined in this Mineral Resource estimate.

14.3.19 Reasonable Prospects for Eventual Economic Extraction

Reporting for Cortadera considers Reasonable Prospects for Eventual Economic Extraction (RPEEE) for both open-pit and underground mining scenarios, with the cost basis for cut-off grade analysis utilising the costs from the 2016 study at Productora (nominally inflated and adjusted to current values) and a benchmarking study into block-cave mining (Wood, 2020).

The key revenue and cost inputs, plus other relevant controls, are provided in Table 14.88 and Table 14.89. Payable metals (other than copper) use the same prices as the Mineral Resource Estimate.

Table 14.87 : 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 14.88 : 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

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 it might be included in a draw point if included in the typical footprint of the cave shape). This geometry was considered a reasonable size to initiate a cave albeit assuming pre-stress and a hydraulic radius of +20 m.

The resulting cave shapes produced by the MSO software serve to identify material that has reasonable prospects of eventual economic extraction (RPEEE) by exploring a typical block cave mining strategy. The geotechnical work for a potential block cave propagation at Cortadera is subject to further study and these are based on preliminary concept geotechnical parameters.

Since mineralisation extends from the surface, an open pit shell was also generated using the industry-standard Lerchs-Grossman algorithm and the parameters listed in Table 14.90. 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 14.89: Key open pit optimisation parameters applied to generate a pit shell defining Reasonable Prospects of Eventual Economic Extraction

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
6.32	USD/t	Oxide Processing Cost
45°	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
83 / 70	%	Sulphide Copper Recovery (Fresh/Transitional)
56 / 50	%	Sulphide Gold Recovery (Fresh/Transitional)
83 / 46	%	Sulphide Molybdenum Recovery (Fresh/Transitional)
37 / 30	%	Sulphide Silver Recovery (Fresh/Transitional)
40	%	Dump Leach Copper Recovery (Fresh/Transitional)
50	%	Oxide Copper Recovery

The controlling surfaces for RPEEE are shown in Figure 14.37 and display 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.

Figures also include sensitivity testwork completed at different copper prices, showing that the RPEEE surface is not particularly sensitive to copper price at Cortadera.

Figure 14.38 shows the final coded RPEEE classification (Open Pit or Underground) for Cortadera for the final $6.00 copper price scenario.

Figure 14.37 : Long-Section View Looking Showing Pit Shell Surfaces (Above) and Block Cave Mining Shapes (Below) Used to Define Reasonable Prospects of Eventual Economic Extraction (HCH, 2024)

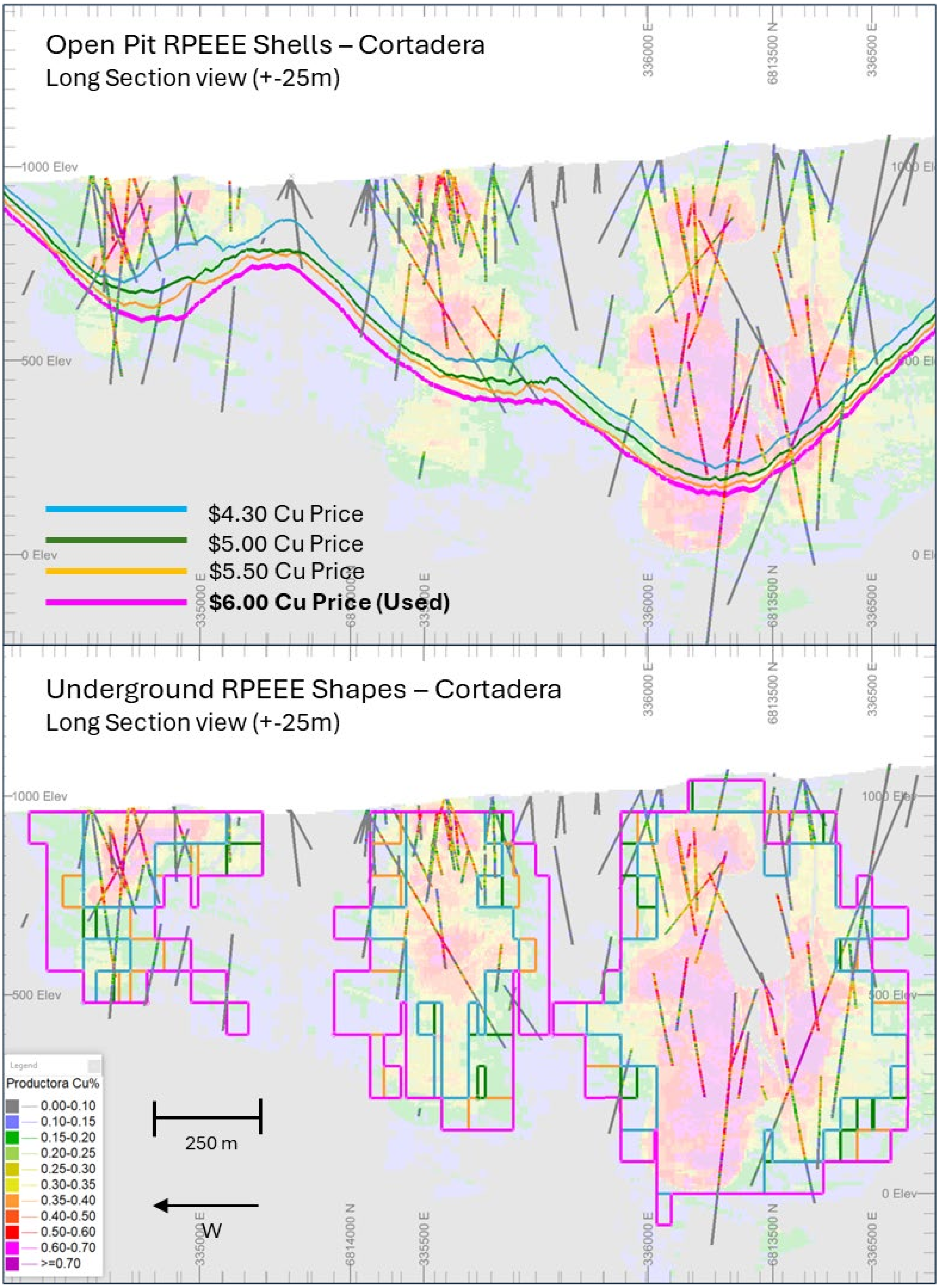
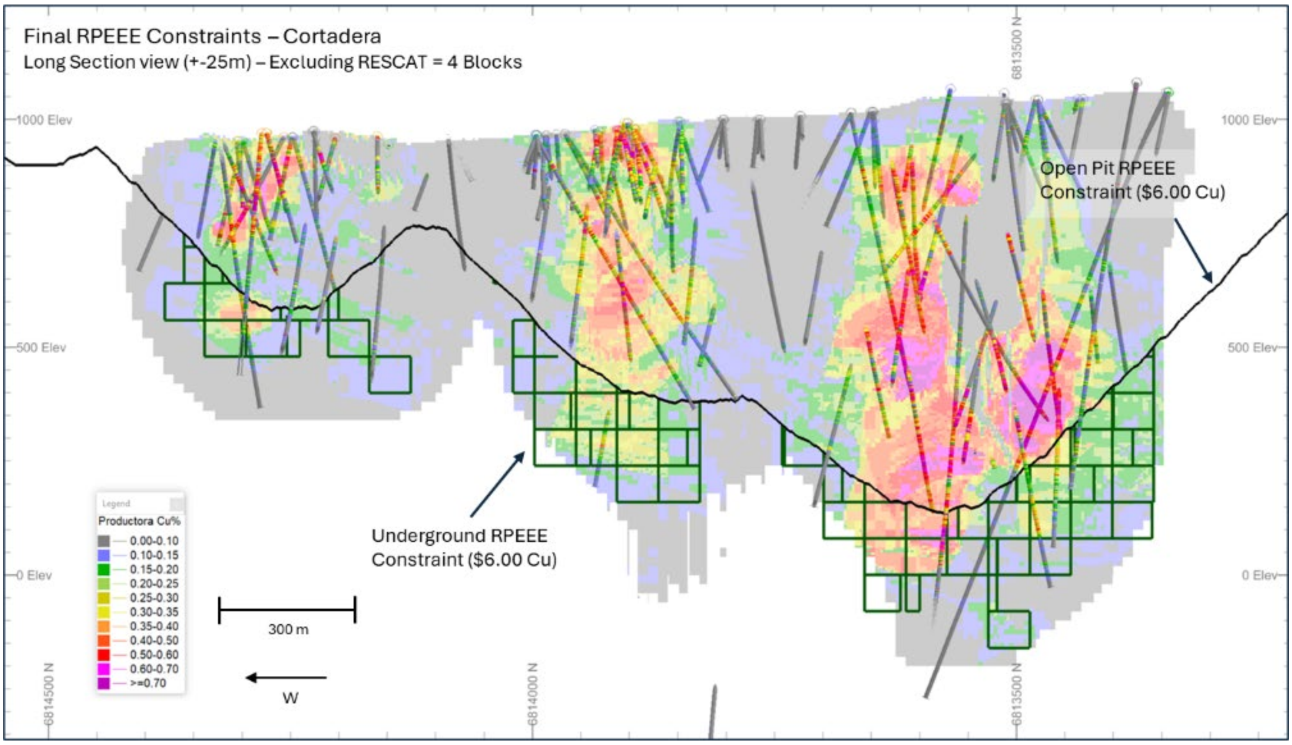


Figure 14.38 : Long Section View Showing Coding of Final Block Model With Open Pit and Block Cave Mining Reasonable Prospects of Eventual Economic Extraction – Excluding Unclassified Blocks (HCH, 2024)



14.4 San Antonio

14.4.1 Introduction

The current San Antonio Mineral Resource was updated in February 2024 for the combined Costa Fuego Mineral Resource Release (26 February 2024).

14.4.2 Database Validation

The San Antonio database was validated through the following checks:

- Drill collars – comparison of planned vs. actual survey pick up coordinates, checked in 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
- Lithology – reviewed based on surrounding drill holes, with drill core or RC chips reviewed and logging updated where necessary
- Mineralisation intensity - logs completed for each drill hole, and compared with assay results
- 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.

14.4.2.1 Summary of Data Used in Estimate

All drilling data is stored in the HCH acQuire drill hole database.

Data table records consist of:

- Collar – collar.csv
- Survey – survey.csv
- Assays – San_Antonio_assay.csv
- Geology – San_Antonio_lith.csv
- Geology – San_Antonio_weathering.csv
- Geology – San_Antonio_vein.csv
- Geology – San_Antonio_mineralisation.csv
- Geology - San_Antonio_alteration
- Geology - San_Antonio_structure

No holes were excluded from the San Antonio Resource Estimate.

A summary of the data used to compile the estimate is listed in Table 14.92. Note that this table only includes drillholes that pass through the limits of the outermost estimation domain at San Antonio (i.e., excludes some regional exploration drilling).

Table 14.90 : Data Included in Estimate Database		
Method	**Holes**	**Metres**
RC	54	6 931
DD	3	495
Underground Drillhole	69	4 994

14.4.3 Data Manipulation

Negative-grade values are used in the assay database table to indicate specific events or conditions. To ensure these assays are suitable for use in the mineral resource, edits were made to remove any non-numeric, zero and negative values.

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 (0.0025 g/t Au, 0.005% Cu, 0.5 ppm Mo, 0.005 ppm Ag)
- If the assay value was equal to zero and there was no recorded weight of sample the assay was set to 'absent'.

14.4.4 Interpretation and Modelling

14.4.4.1 Geology Domains

An updated San Antonio lithology and structural model was prepared for the 2024 release, informed by surface mapping (with campaigns completed in 2018 and 2022), underground mapping (2018 and 2019), drillhole logging, assay data (2018 and 2022), and an RC chip relogging campaign (2022).

The structural model for San Antonio has been built based off surface mapping and observations of shear and/or damaged intervals intersected during diamond drilling.

The geology model combines mapped and interpreted structures, with a total of nine diorite dykes modelled (although it is noted that many more exist, only those considered significant to the mineralisation have been modelled).

Dykes tend to intrude along the NNE-SSW trending structural corridor, with along-strike continuity variable from metre to hundred-metre scale.

14.4.4.2 Mineralisation Domains

Mineralised copper domain interpretation was completed in Leapfrog based primarily on surface mapping and the resultant mafic dyke model, which is thought to be a primary control on mineralisation. This updates the interpretation used for the 2022 estimate, which used a primarily grade-based interpretation (based on the best information available at the time).

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.

Due to the update in interpretation methodology for the 2024 Resource, there has been a mineralised lode volume decrease which can be attributed to two main factors:

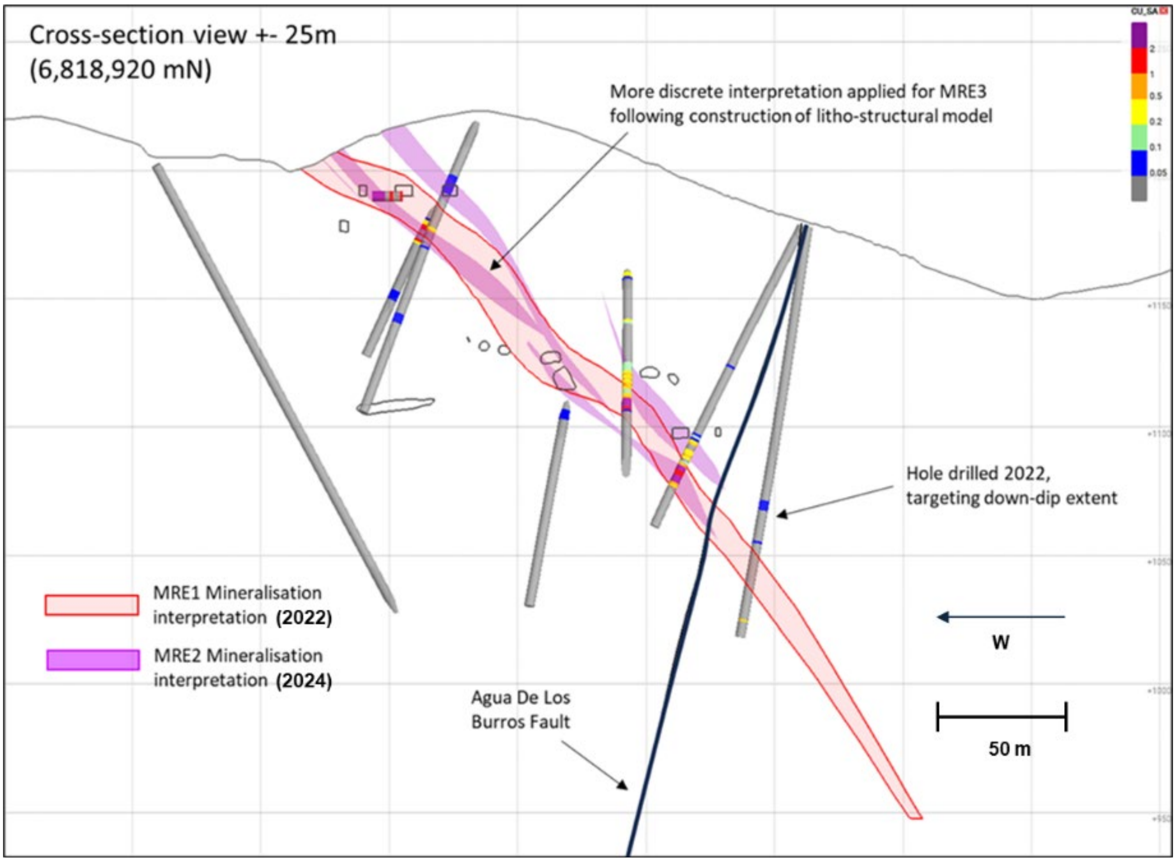
- Implementing a more discrete lode interpretation through the densely drilled and developed San Antonio central zone following construction of the lithology and structural models.

- Truncation of the lodes down-dip against the Agua de los Burros fault (proven by drilling completed during the 2022 campaign).

A representative cross section view comparing the 2022 and 2024 lode interpretations is included in Figure 14.39.

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.

Figure 14.39 : Cross Section Looking North Showing Changes in San Antonio Mineralisation Interpretation (HCH, 2024)



14.4.4.3 Weathering Domains and Bulk Density

Drilling completed after release of the 2022 Resource confirmed that a weathering profile was present which has since been modelled in Leapfrog. The 2022 Resource had assumed rock was fresh from surface.

Fresh domain density was calculated as 2.93 t/m^3 for the 2024 Resource based on an average of all density 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 (2.64 t/m^3) and oxide (2.34 t/m^3) weathering domains, respectively.

14.4.5 Data Flagging and Compositing

Mineralisation wireframes were flagged to the drill hole data and block model. The wireframes used are listed in Table 14.93.

Table 14.91 : Drill Holes and Models Coding		
Wireframe	**Zone Field**	**Value**
Sa_lg_main	DOM_CU1	100
Sa_hg_main		0
Sa_hg_splay_1		1
Sa_hg_splay_2		2
Sa_hg_splay_3		3
Sa_hg_splay_4		4
Sa_hg_splay_5		5
Sa_hg_splay_6		6
Sa_hg_splay_7		7
Sa_hg_splay_8		8
Sa_hg_splay_9		9
Sa_stkpile		10
SA_topo	TOPO	1000
SAweathering_f	WTCODE	3000
SAweathering_t		2000
SAweathering_o		1000

14.4.6 Block Modelling

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 14.94.

Table 14.92 : Block Model Dimensions					
Dimension	**Minimum**	**Maximum**	**Extent (m)**	**Block Size (m)**	
				Parent	**Minimum**
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

14.4.7 Mining and Tenement Flagging in the Model

San Antonio tenements were not flagged in the Resource model. The San Antonio model is situated inside a single tenement.

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).

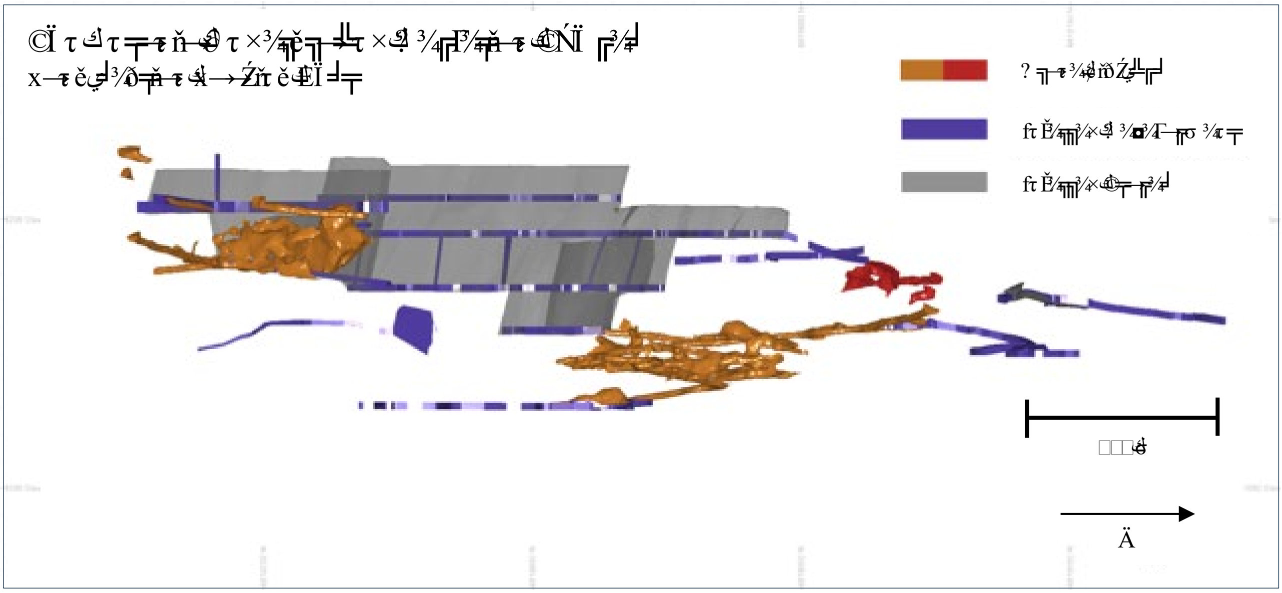
While the drone survey was successful in validating the spatial interpretation of mineralised wireframes, it was unable to access all areas of the mine due to poor ground conditions (falls of ground). As a result, assumptions have been made with regards to the depletion of the San Antonio resource.

For the 2024 Resource, 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 14.40).

The approach differs from the 2022 Resource, which used a 'cookie cutter' to deplete across the entire width of the mineralised lodes. The change in approach for the 2024 Resource provides a more realistic outcome and has been validated against available as-builts.

Total depletion for San Antonio is now 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 14.40 : Long Section Looking East Showing Inputs into Final Depletion Shape (HCH, 2024)



14.4.8 Compositing

The samples were composited to 1 m lengths. 1 m composites were chosen as the dominant drill sample length is 1 m and is most suitable to the narrow style of mineralisation. Larger composite intervals would not allow the variability of mineralisation to be investigated and may result in the grade estimation being oversmoothed.

14.4.9 Statistical Analysis

Statistical analysis of copper, gold, silver, and molybdenum were undertaken using Snowden Supervisor Version 8.14.3.2.

The analysis was completed to understand the global representative distribution of each element and account for any bias introduced by clustering of data or by extreme outliers.

14.4.10 Top-Cuts

Each element within each mineralisation domain was examined using log histograms, log probability plots, grade disintegration and the general statistics of each lode. The top-cuts were chosen to reduce the potential smearing of extremely high grades. Top-cuts for copper and silver are summarised in Table 14.95.

Table 14.93 : San Antonio Mineralised Domains Top-Cut Analysis

Element	Domain	Raw				Top-Cut			
		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

14.4.11 Grade Estimation

A conventional estimation strategy has been used for San Antonio, with the mineralised zone interpretation producing copper grade populations suitable for linear estimation (ordinary kriging on top-cut composites). 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.

Variography was attempted on copper grade for all domains. Due to low sample counts, 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.

For the molybdenum and gold estimates, a constructed variogram has been used with a nominal nugget of 0.2 and a spherical search of 150 m.

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 and Search parameters used for the San Antonio estimate are included in Table 14.96 and Table 14.97 below.

Table 14.94 : Grade Variogram Models – San Antonio

Element	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
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		-		-

Table 14.95 : Search Strategy for Grade Estimation – San Antonio

Element	Domain	Orientation Rot 1	Orientation Rot 2	Orientation Rot 3	Search D1	Search D2	Search D3	2nd Search Factor	First Search Min	First Search Max	Second Search Min	Second Search Max	Max per Drill Hole
Cu	All	120	55	20	150	80	50	1.5	10	20	10	20	6
Ag	All	120	55	20	150	80	50	1.5	10	20	10	20	6
Au	All	0	0	-90	150	150	150	1.5	10	20	10	20	6
Mo	All	0	0	-90	150	150	150	1.5	10	20	10	20	6

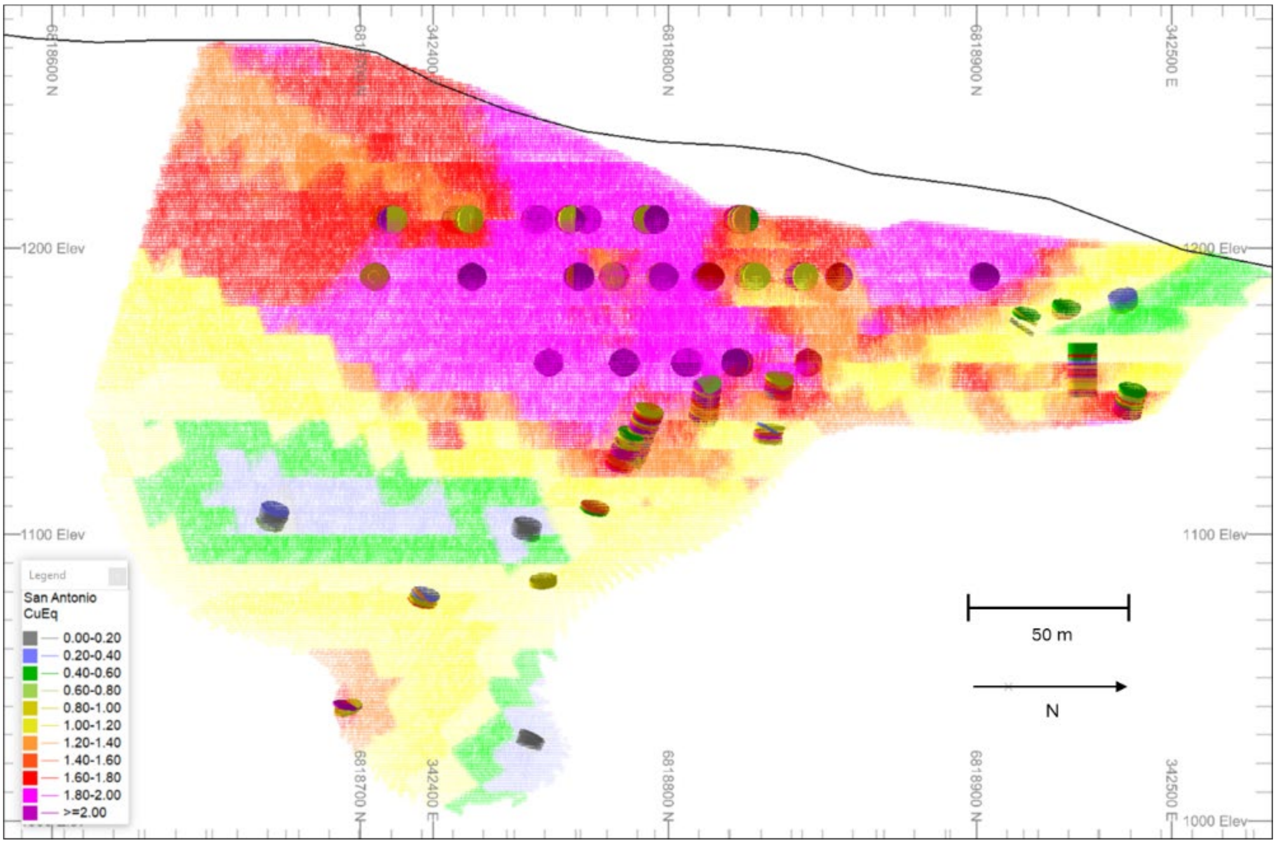
14.4.12 Estimate Validation

Estimates were validated using a three-stage comparison between top-cut composites and the estimated variables. The first stage involved calculating the global statistics of the composites compared to the tonnage weighted averages of estimated variables (Table 14.98). The second stage involved comparing statistics in elevation slices along the mineralisation and the third involved a detailed visual comparison by section to ensure the estimated variables honour the input composite data (Figure 14.41).

Table 14.96 : 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 14.41 : Visual Validation of Input Data Versus Model Copper Grade – DOM_CU1 = 0, Long Section View



14.4.13 Mineral Resource Classification

The San Antonio Mineral Resource estimate has been classified considering the 2014 CIM Definition Standards and were estimated in accordance with the CIM 2019 Best Practices Guidelines, as required by NI43-101.

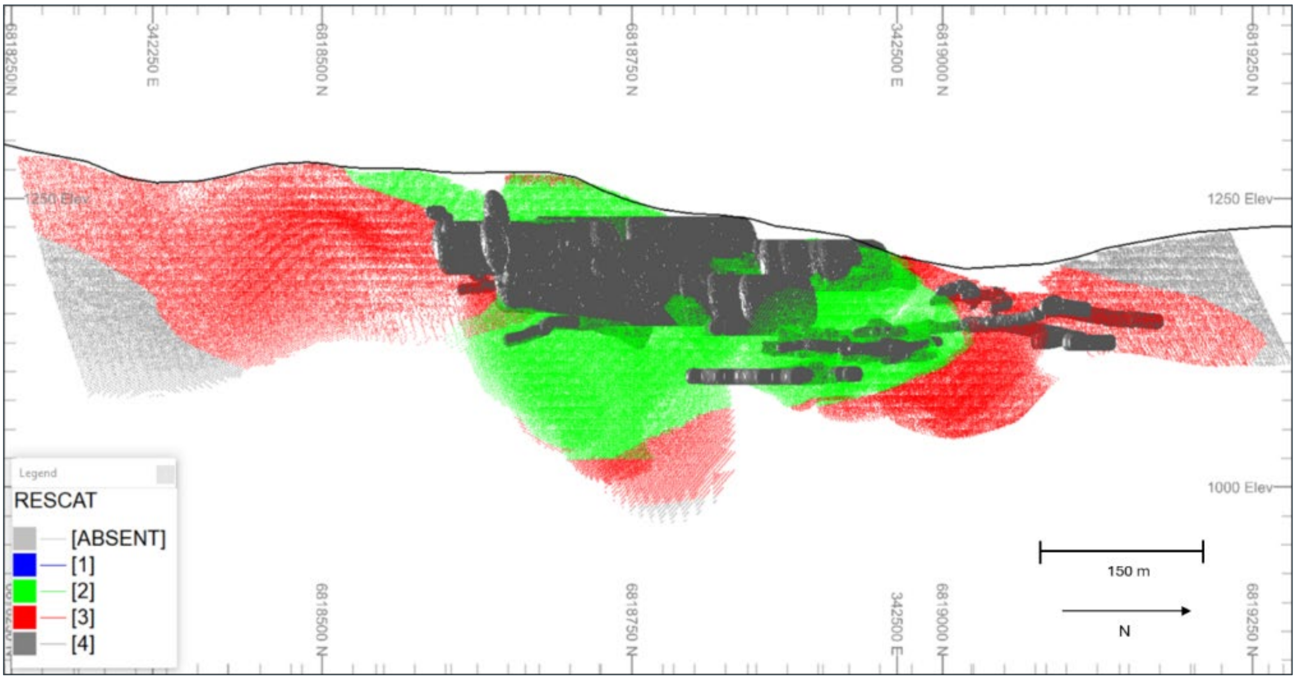
A significant outcome for the 2024 San Antonio Resource is the inclusion of an Indicated resource, with the 2022 iteration being classified as Inferred. Upgrade in classification has been possible due to:

- Creation of lithology and structural models following 2022 surface mapping and RC chip relogging campaigns
- Interpretation of updated mineralisation domains based on lithology and structural models
- Use of underground mapping to validate interpretation of discrete mineralised domains
- Update to the weathering model, including information from 2022 drilling campaigns
- Detailed depletion update, including sectional interpretation through upper levels and validation against higher-confidence as-built shapes
- Surface mapping and rock-chip sampling completed in 2024, proving the continuity of the mineralised San Antonio structure to surface.

The appropriate classification has been determined by considering a range of items including the slope of regression from the kriged estimate, drillhole spacing, proximity to inferred development/stoping, and confidence in the interpretation of the ore domain.

Classification shapes were created using sectional interpretation at 10 m intervals in plan view. Final classification is shown in Figure 14.42.

Figure 14.42 : Long Section Looking West, Showing Block Classification Across San Antonio (Green = Indicated, Red = Inferred, Grey = Unclassified)



14.4.14 Reasonable Prospect of Eventual Economic Extraction (RPEEE)

Reporting for San Antonio considers Reasonable Prospects for Eventual Economic Extraction (RPEEE) for an open-pit mining scenario, with the cost basis for cut-off grade analysis utilising the costs from the 2023 Preliminary Economic Assessment (PEA).

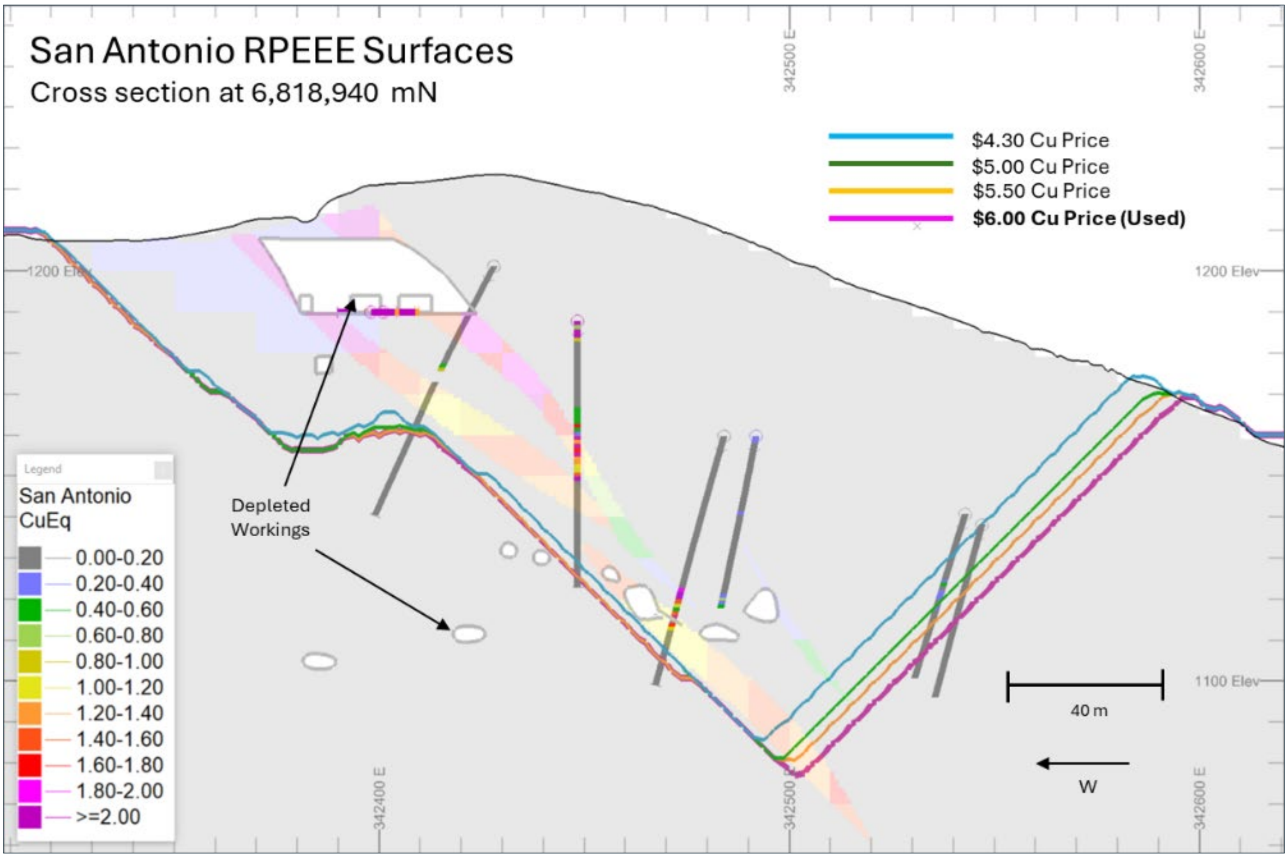
An open pit shell was generated using the industry-standard Lerchs-Grossman algorithm and the parameters listed in Table 14.99. Selling costs incorporate transport and handling costs, as well as any treatment and refining costs. Payable terms assume that the copper concentrate is sold into the Asian smelter market, for which these terms are standard.

Table 14.97 : Key open pit optimisation parameters applied to generate a pit shell defining Reasonable Prospects of Eventual Economic Extraction

San Antonio 2024 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
6.32	USD/t	Oxide Processing Cost
45°	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
88 / 88	%	Sulphide Copper Recovery (Fresh/Transitional)
72 / 72	%	Sulphide Gold Recovery (Fresh/Transitional)
88 / 88	%	Sulphide Molybdenum Recovery (Fresh/Transitional)
69 / 39	%	Sulphide Silver Recovery (Fresh/Transitional)

The controlling surface for San Antonio RPEEE is shown in Figure 14.43. This also includes sensitivity testwork completed at different copper prices, showing that the RPEEE surface is not particularly sensitive to copper price at San Antonio.

Figure 14.43 : Cross Section View Looking North Showing Pit Shell Surface Shapes Used to Define Reasonable Prospects of Eventual Economic Extraction by Open Pit Mining



14.5 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$/lb, Au=1700 US$/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%

14.6 Costa Fuego Resource Reporting

14.6.1 Mineral Resource Tables

The Cortadera, San Antonio, Productora, and Alice Mineral Resource Estimates have been combined to create the Costa Fuego Mineral Resource Estimate. The final reported tonnes and grade by classification are in Table 14.102 to Table 14.106 (Costa Fuego combined and broken down by Resource area). Grade tonnage curves for both the Open Pit and Underground Resource are in Figure 14.44.

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. The change in copper price, in combination with the latest costs, as informed by the June 2023 PEA, has reduced the breakeven grade for Costa Fuego.

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 14.101.

Table 14.98: 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 14.99 : 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 (November 29, 2019) 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 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 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.
[6] All Mineral Resource Estimates 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 Productora, Alice and San Antonio.
[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 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.
[10] The effective date of the estimate of Mineral Resources is February 26th, 2024. Refer to JORC Code Table 1 information in this announcement related to the Costa Fuego Mineral Resource Estimate (MRE) 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 Costa Fuego). Hot Chili 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.
[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 as disclosed in this Report. A detailed list of Project risks is included in 25.12.

Table 14.100 : 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**				
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 (November 29, 2019) and Mineral Reserve Estimation (September 8, 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 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 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.
[4] All Mineral Resource Estimates were assessed for Reasonable Prospects of Eventual Economic Extraction (RPEEE) using both Open Pit and Block Cave Extraction mining methods at Cortadera.
[5] 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).
[6] 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 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.

[7] 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.
[8] The effective date of the estimate of Mineral Resources is February 26th, 2024. Refer to JORC Code Table 1 information in this announcement related to the Costa Fuego Mineral Resource Estimate (MRE) 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 Costa Fuego). Hot Chili 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.
[9] 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 Project risks is included in 25.12.

Table 14.101 : 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 (November 29, 2019) and Mineral Reserve Estimation (September 8, 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 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 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.
[4] All Mineral Resource Estimates 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 Productora, Alice and San Antonio.
[5] Metallurgical recovery averages for each deposit consider Indicated + Inferred material and are weighted to combine sulphide flotation and oxide leaching performance. Process recoveries:
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).
[6] 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 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.
[7] 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.
[8] The effective date of the estimate of Mineral Resources is February 26th, 2024. Refer to JORC Code Table 1 information in this announcement related to the Costa Fuego Mineral Resource Estimate (MRE) 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 Costa Fuego). Hot Chili 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.
[9] 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 Project risks is included in 25.12.

Table 14.102 : 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 (November 29, 2019) and CIM Environmental, Social and Governance Guidelines for Mineral Resources and Mineral Reserve Estimation (September 8, 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 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.
[4] All Mineral Resource Estimates 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 Productora, Alice and San Antonio.
[5] Metallurgical recovery averages for each deposit consider Indicated + Inferred material and are weighted to combine sulphide flotation and oxide leaching performance. Process recoveries:
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).
[6] 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 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.
[7] 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.
[8] The effective date of the estimate of Mineral Resources is February 26th, 2024. Refer to JORC Code Table 1 information in this announcement related to the Costa Fuego Resource Estimate (MRE) 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 Costa Fuego). Hot Chili 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.

[9] 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 Project risks is included in 25.12.

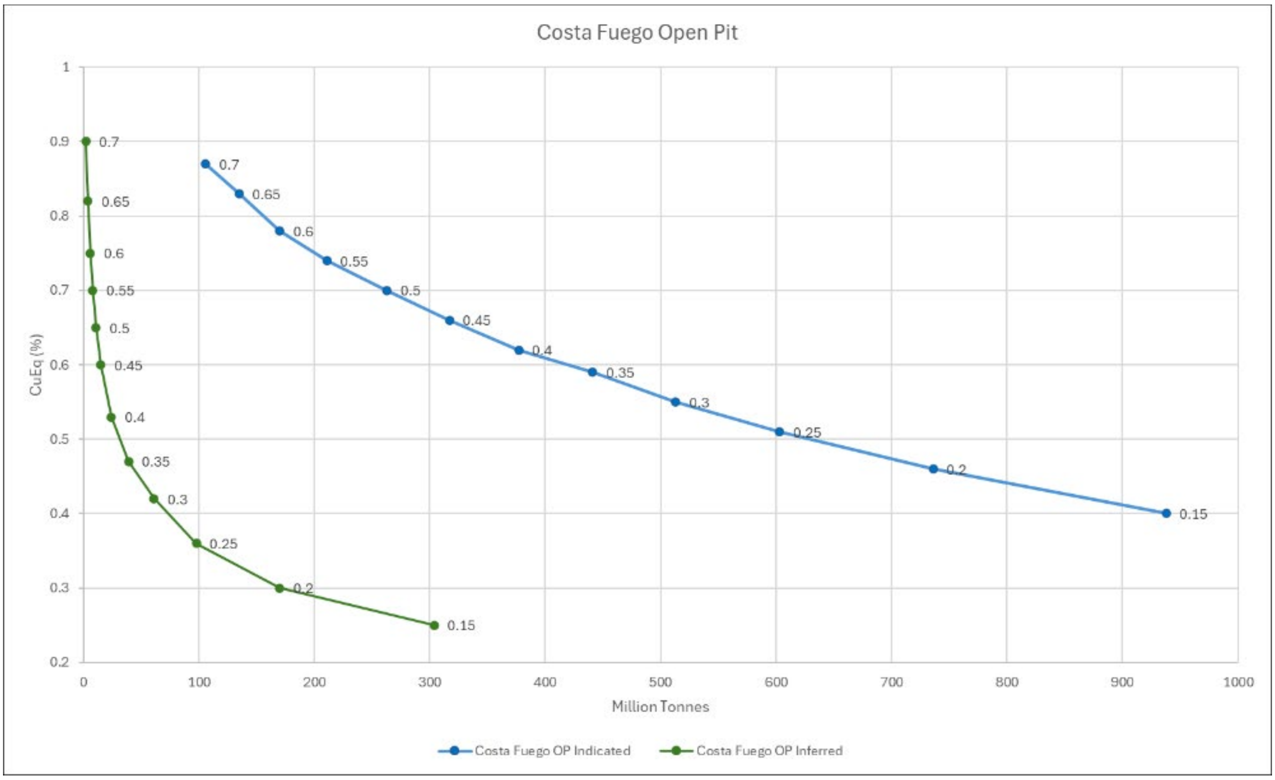
Table 14.103 : Alice Mineral Resource Summary – Reported by Classification (26 February 2024)											
Alice Total Resource		**Grade**					**Contained Metal**				
Classification	Tonnes	CuEq	Cu	Au	Ag	Mo	Copper Eq	Copper	Gold	Silver	Molybdenum
(+0.20% CuEq*)	(Mt)	(%)	(%)	(g/t)	(g/t)	(ppm)	(tonnes)	(tonnes)	(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	-	-	-	-	-	-	-	-	-	-	-

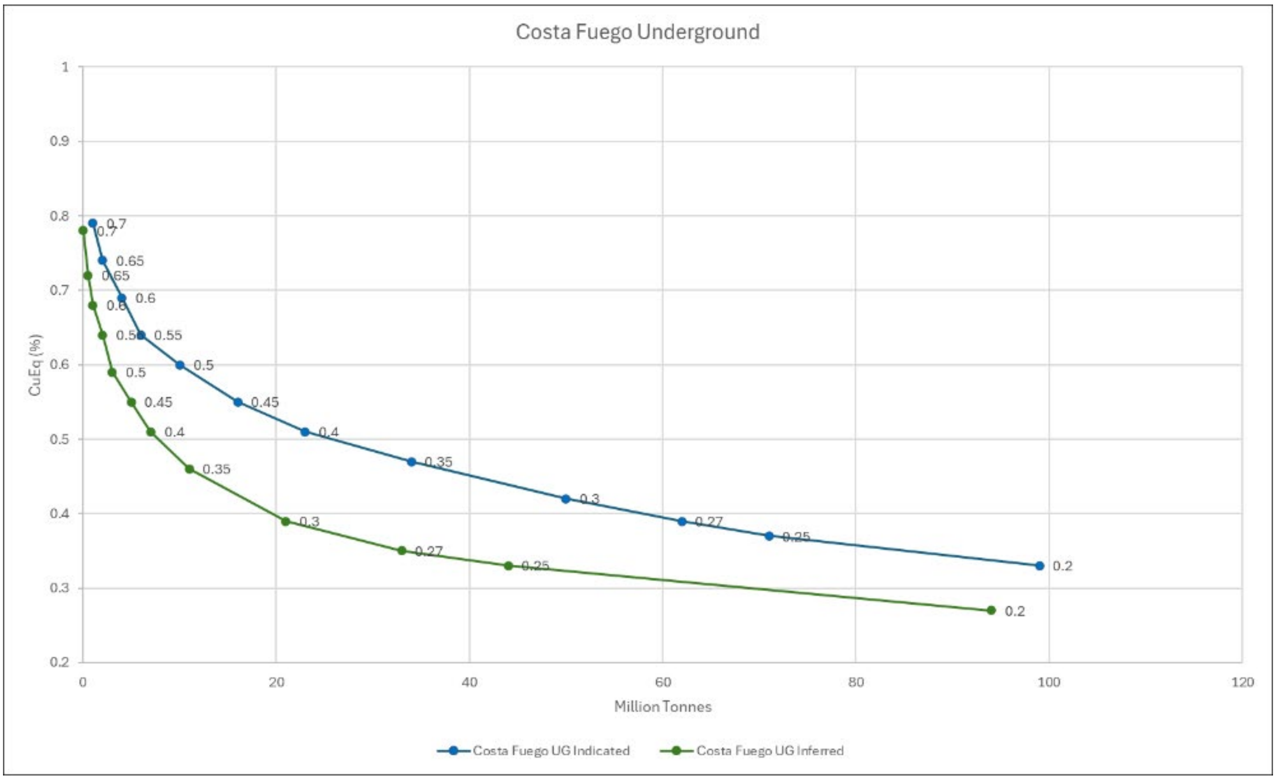
[1] Mineral Resources are reported on a 100% Basis. All figures are rounded, reported to appropriate significant figures and reported in accordance with the Joint Ore Reserves Committee Code (2012) and NI 43-101. Mineral resource estimation practices are in accordance with CIM Estimation of Mineral Resource and Mineral Reserve Best Practice Guidelines (November 29, 2019) 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 Productora deposit (including Alice) 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 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.
[4] All Mineral Resource Estimates 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 Productora, Alice and San Antonio.
[5] Metallurgical recovery averages for each deposit consider Indicated + Inferred material and are weighted to combine sulphide flotation and oxide leaching performance. Process recoveries:
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).
[6] 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 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.
[7] 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.
[8] The effective date of the estimate of Mineral Resources is February 26th, 2024. Refer to JORC Code Table 1 information in this announcement related to the Costa Fuego Resource Estimate (MRE) 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 Costa Fuego). Hot Chili 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.
[9] 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 Project risks is included in 25.12.

Table 14.104 : 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**
0.15	938	0.40	0.32	0.09	0.44	77	0.15	304	0.25	0.20	0.05	0.33	52
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**
0.25	71	0.38	0.30	0.07	0.53	81	0.25	44	0.33	0.27	0.07	0.39	44
0.20	99	0.33	0.26	0.07	0.49	71	0.20	94	0.27	0.22	0.05	0.37	40

Figure 14.44 : Costa Fuego Grade-Tonnage Curves – (Open Pit (Top) and Underground (Bottom))





14.7 Relevant Factors Affecting Resource Estimates

There is currently no known mining, metallurgical, infrastructure, environmental, permitting, legal, title, taxation, socio-economic, marketing, political or other relevant factors which could affect the Mineral Resource estimates. A detailed list of Project risks is included in 25.12.

14.8 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 2019 Best Practices Guidelines, as required by NI43-101. The mineral resources reported in Table 14.102 to Table 14.106, inclusively, are constrained and reported using economic and technical criteria such that the Mineral Resource has reasonable prospects for eventual economic extraction.

The Mineral Resource estimation 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 three deposits within the Costa Fuego project.

15 Mineral Reserve Estimates

Mineral Reserves are not estimated in this Preliminary Economic Assessment.

16 Mining Methods

This section has been taken directly from the technical report titled, "Costa Fuego Copper Project: NI 43-101 Technical Report & Preliminary Economic Assessment (PEA)" dated August 14, 2023 with an effective date of June 28, 2023. The Mineral Resource estimate update, as described in this Report, does not materially affect the mineral resource inventory that formed the basis of the 2023 PEA and no additional technical or scientific information has been generated on the project that would change the basis of the assumptions and parameters used to prepare the PEA and, therefore, the results and conclusions of the 2023 PEA are considered current and have been restated in this Report.

The reader is cautioned that the 2023 PEA is preliminary in nature and utilises all mineral resource categories in reported mining physicals, including Inferred Mineral Resources, which are too geologically speculative in nature to form a true economic valuation to state Mineral Reserves. As such there is no certainty that the values stated in this report will be realised.

Mineral Resources that cannot be converted to Mineral Reserves do not have demonstrated economic viability.

16.1 Introduction

Open pit mining methods have been selected as the key exploitation technique for the Costa Fuego deposits, these methods are supported by near surface mineralisation which allows for low waste/mineral strip ratios and associated cashflow.

There is additional underground block cave exploitation potential at the Cortadera Cuerpo 3 deposit.

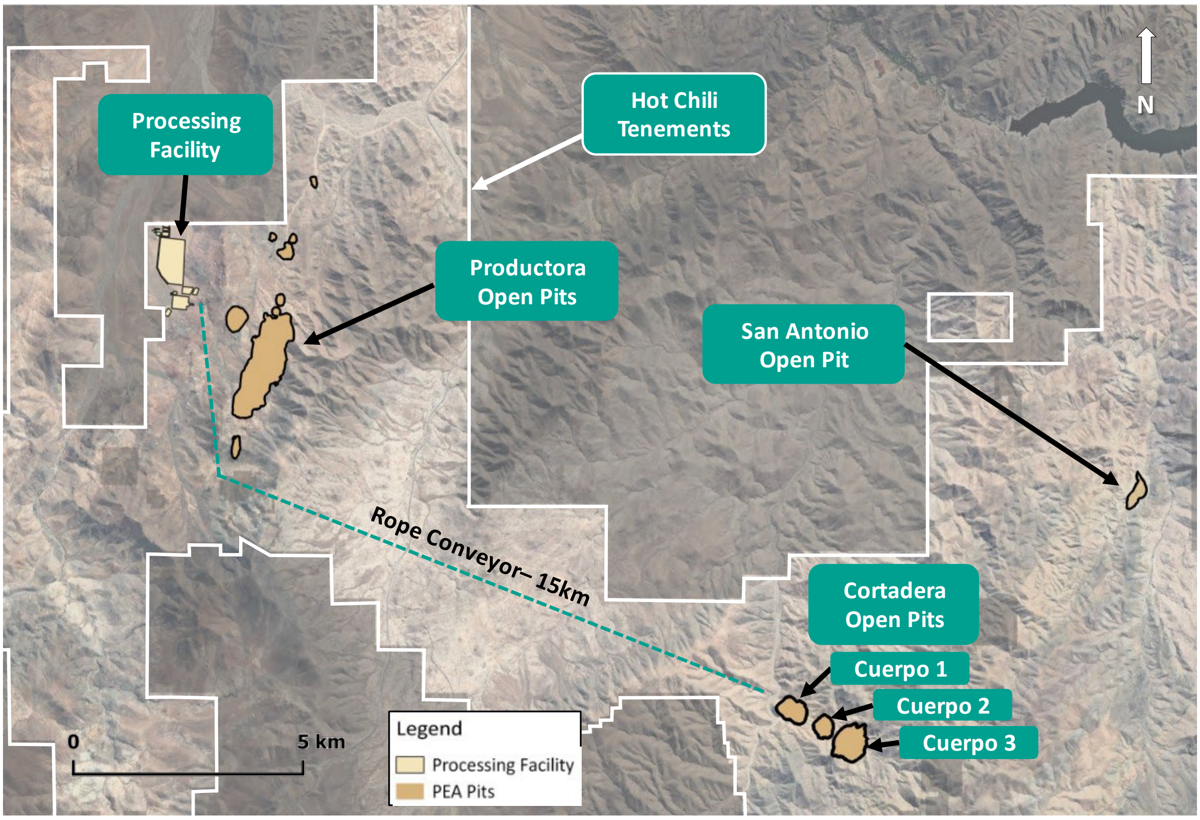
16.2 Economic Limits to Mining

16.2.1 Methodology

The Project consists of the following mining areas (Figure 16.1):

- Productora (open pit)
- Cortadera (three deposits)
  - Cuerpo 1 (open pit only potential)
  - Cuerpo 2 (open pit only potential)
  - Cuerpo 3 (open pit and underground block cave potential)
- San Antonio (relatively small, flatter dipping deposit with open pit potential)).

Figure 16.1 : Costa Fuego Mining Areas



16.2.1.1 Open Pit

An 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 an economic value to each block contained within the respective (regularised) 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 16.2, 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 NPV and helping to create an optimised mining sequence and schedule.

Figure 16.2 : NPVS Process Flow

In addition to the NPVS work, Material Allocation Optimiser (MAO) and MultiMine (MMS) software was used to refine the initial mining schedules to account for material blending, thereby allowing an optimal process feed blend to be presented to the concentrator and heap leach. This software allows for modified mining rates to extract and process higher grade process feed earlier and stockpiling lower grade material for future processing. More information and detail on this topic can be found in Section 16.7.

16.2.1.2 Underground

The Cortadera Cuerpo 3 underground block cave potential was studied using Geovia PCBC™ Footprint Finder software (PCBC). The software was used to assess the vertical elevation, lateral extent, and potential economic value of a likely block cave footprint. It determined the quantity and geometric distribution of potential economic mineralisation and identified block caving as the most likely underground mining method.

To perform this initial PEA, all material classes (Indicated and Inferred) in the Resource model were considered. 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. 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.

The Resource model (20b20b20_2022ugfin2.csv) was regularised to approximate eventual drawpoint dimensions to be modelled in more detail in PCBC when more information is available, typically 20 m cell sizes. PCBC considers a potential block cave layout (footprint) within a user-defined range of vertical elevations (columns). Typically, a potential block cave footprint will be evaluated on each level of the Resource model, i.e. every 20 m elevation for the regularised Cortadera block model. For each footprint, pre-vertical mixing is applied, based on Laubscher's mixing algorithm, to each column in the Resource model. This process assumes unconstrained vertical caving and provides acceptable results for early evaluation. More detailed analysis, such as cave back propagation, lateral movement of material and differential material movement rates will be assessed in PCBC once appropriate data is available, typically during a Prefeasibility and Feasibility Study.

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 such as access from surface to the footprint elevation, surface infrastructure and other surface capital, which is assumed to remain relatively constant between different footprints 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). Footprint Finder (FF) will determine the height at which the maximum column value is obtained (Best Height of Draw, or BHOD) in order 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 sequence and vertical height in the column, in order 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 open pit potential material of Cuerpo 3 was depleted and the resulting block model was then analysed in FF software to identify the following key block cave information and data:

- Best/most economical Footprint
- Relative Level (RL) with the highest value to consider as the undercut level
- The most economical column heights within the most economic footprint
- Develop a schedule of draw within the most economical Footprint and cave heights.

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 the next study phase detailed mine design and scheduling and first principles cost modelling and financial evaluation should be used to further refine these results and to determine a likely cave value.

The FF results for the selected footprint were written into a 3D (.csv) block model file with all the resource block model fields coupled with diluted mining grades and a draw schedule field for the selected Cuerpo 3 block cave footprint. This was incorporated into the overall NPVS schedule and combined with the open pit model.

Cuerpo 2 may have secondary underground potential (from visual investigation of the block model). However, the concern is obtaining a sufficient caving front, footprint and viable column heights. Cuerpo 2 was tested in FF but did not yield a reasonable underground block cave option. There may be potential to apply bulk open stoping methods at Cuerpo 2, however, this was not investigated in the PEA due to marginal economic viability and feed quantities.

16.2.2 Dilution

It is becoming common to develop Resource models which consider potential dilution within the blocks, or adopt selective mining units (SMUs) as part of the Resource modelling process.

The 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 Resource model added approximately 5% to 8% dilution (reduction in average grade). At a variable cut-off of 0.1% to 0.15% CuEq the impact was approximately 5% to 8% on process feed loss (reduction in process feed blocks above the marginal cut-off grade in a sub-celled model but are now sub-economic). The combination of more tonnage at a lower grade resulted in a 0.5% reduction for contained copper, therefore, the process feed dilution and mining recoveries do not have a significant impact on the Project.

All open pits utilise 15 m high benches for planning purposes. At bulk loading rates and with bulk loading equipment, 15 m bench heights (barring geotechnical acceptance) are on the upper productivity levels when using larger diesel hydraulic loading units.

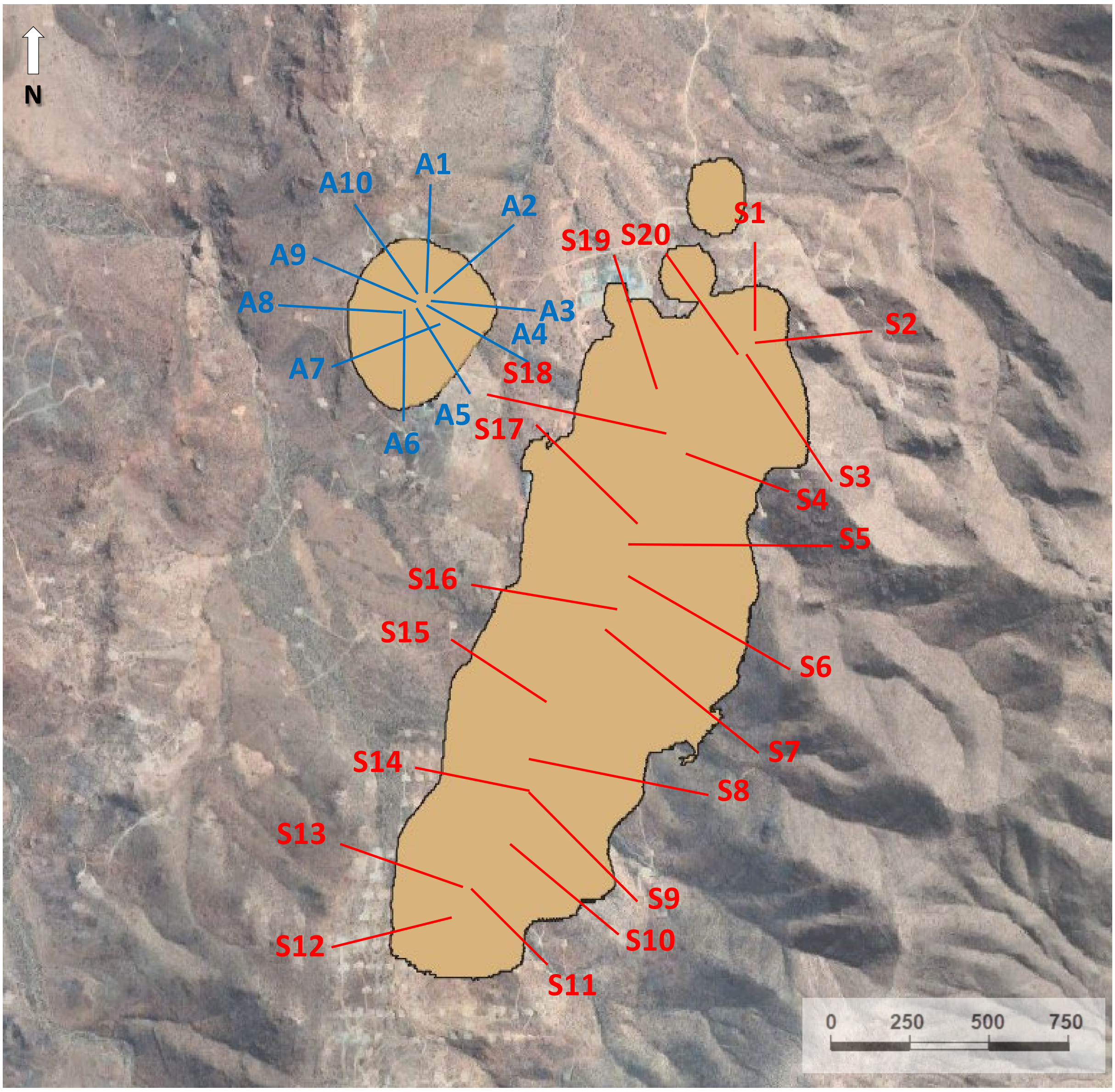
The underground block cave model has been regularised to 20 m x 20 m x 20 m to approximate drawpoint column dimension. PCBC FF uses Laubscher's pre-vertical mixing during footprint optimisation, with a height of interaction of 160 m and First Point of Dilution Entry of 60% selected for the footprint optimisations. Caving is a non-selective, bottom-up mining method. FF will include 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 was used for all columns.

16.2.3 Geotechnical

16.2.3.1 Productora

The following figures and table summarise the Productora (including the Alice satellite pit) geotechnical slope criteria applied in the study. Figure 16.3 displays the location and orientation of the geotechnical sections while Table 16.1 and Table 16.2 summarise the stability results. Further detail on geotechnical analysis can be found in section 16.5.

Figure 16.3 : Location and Orientation of the Geotechnical Sections of Productora and Alice Pits



(Blue Lines for the Alice Sections and Red Lines for Productora Sections)

Table 16.1 : Productora Pit Final Stability Results Summary								
Section	**Design**			**Interramp**		**Overall Slope Angle**		
	Bench Height	**Overall Design Angle**	**Bench Face Angle**	**Interramp Angle**	**Interramp Height**	**Ritchie Criterion (24 m bench Height)**	**Structural Analysis**	**Rotational Analysis**
	(m)	**(°)**	**(°)**	**(°)**	**(m)**	**(°)**	**(°)**	**(°)**
S1	160	50	75	57	120	57	84	>50
S2	175	53	75	57	120	57	82	>53
S3	320	53	75	57	120	56	77	>53
S4	300	54	75	57	120	56	84	54
S5	410	49	75	57	120	55	50	>49
S6	440	50	75	57	120	55	66	>50
S7	400	46	75	57	120	55	49	>46
S8	340	51	75	57	120	56	70	>51
S9	360	47	75	57	120	55	60	>47
S10	320	51	75	57	120	56	62	>51
S11	310	50	75	57	120	56	62	>50
S12	200	50	75	57	120	57	55	>50
S13	240	51	75	57	120	57	59	>51
S14	280	48	75	57	120	57	56	>48
S15	260	54	75	57	120	57	71	>54
S16	300	52	75	57	120	56	72	>52
S17	310	44	75	57	120	56	69	>44
S18	200	50	75	57	120	57	75	>50
S19	180	53	75	57	120	57	83	>53
S20	160	55	75	57	120	57	86	>55

Table 16.2 : Alice Pit Final Stability Results Summary								
Section	Design			Interramp		Overall Slope Angle		
	Bench Height	Overall Design Angle	Bench Face Angle	Interramp Angle	Interramp Height	Ritchie Criterion (24 m bench Height)	Structural Analysis	Rotational Analysis
	(m)	(°)	(°)	(°)	(m)	(°)	(°)	(°)
A1	180	47	75	57	120	57	65	>47
A2	150	45	75	57	120	57	70	>45
A3	160	42	75	57	120	57	53	>42
A4	170	43	75	57	120	57	51	>43
A5	200	45	75	57	120	57	51	>45
A6	180	42	75	57	120	57	59	>42
A7	180	39	75	57	120	57	75	>39
A8	220	52	75	57	120	57	69	>52
A9	230	50	75	57	120	57	66	>50
A10	240	50	75	57	120	57	66	>50

16.2.3.2 Cortadera

Cortadera and San Antonio open pit slope criteria are conceptual, although assumptions are considered reasonable at a PEA level of study.

The Cortadera potential open pit shells assumed the following basic overall slope criteria, based on discussions and recommendations from the geotechnical consultants:

- Oxide Rock slopes – 35 degrees
- Transition Rock Slopes (semi-oxidised) – 40 degrees
- Fresh Rock slopes – 45 degrees.

A concept level block cave geotechnical analysis was carried out on Cortadera and confirmed the potential for underground mass caving methods at Cuerpo 3. Further detail on the analysis can be found in section 16.5.3.

16.2.3.3 San Antonio

San Antonio is a much smaller potential open pit with a well-developed footwall contact and reasonably flat dipping strata-bound mineralisation.

There were no oxidised rock conditions noted and therefore the following slope criteria were assumed:

- Transition (semi weathered rock) – 43 degree overall slope
- Fresh rock – 45 degree overall slope.

Potential pit design at San Antonio will likely position the ramp on the footwall side of the orebody. Therefore, the hangingwall overall slopes might consider more aggressive overall slopes in the next phase of the Project. However, a definitive geotechnical study at San Antonio is required prior to capturing any potential benefits of more aggressive slope criteria. Due to the higher modelled grades at San Antonio, this pit is less sensitive to strip ratio compared to the other deposits.

16.2.4 Economic Parameters

The following section summarises the economic parameters used for the open pit and underground block cave optimisation studies.

The final open pit shells and the block cave assumed the 100% revenue factor (rf 100%) economic areas as developed within the respective software platforms (NPVS and FF).

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, 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. A typical benchmarking mining operating cost of US$6.55/t was used for the block cave, further detail can be found in section 21.4.3.3 of this report.

During the initial assessment, the following metal prices were used to optimise the mining inventory.

Table 16.3 : Optimisation Commodity Price Deck

Item	Unit	Value
Copper	US$/lb	3.3
Gold	US$/oz	1 700
Molybdenum	US$/lb	14
Silver	US$/oz	20

Subsequent benchmarking and changes to commodity prices prompted the price deck to be revised for input into the final economic model (see Section 22).

Mining costs, processing costs, refining costs, and processing recoveries were provided by HCH, and were maintained throughout various design and schedule iterations as shown in Table 16.4 to Table 16.12.

Table 16.4 : 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
Au Recovery (Oxide)	Not applicable	Not applicable	Not applicable	Not applicable	Not applicable
Mo Recovery (Fresh)	Rec = 0.9 x [17.342 x LN(MoFppm) - 34.65]; Max=90%	RecMo = 0.9 x [0.882 x MoFppm + 18.52]; Max=90%	50% Fixed	"Rec = 0.9 x [11.656 x LN(MoFppm) + 19.953]; Max=90%, Min=18%"	50% Fixed
Mo Recovery (Transitional)	Rec = 0.9 x [17.342 x LN(MoFppm) - 34.65]; Max=90%	Not applicable	50% Fixed	Not applicable	50% Fixed
Mo Recovery (Oxide)	Not applicable	Not applicable	Not applicable	Not applicable	Not applicable
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
Ag Recovery (Oxide)	Not applicable	Not applicable	Not applicable	Not applicable	Not applicable

Table 16.5 : Sulphide Selling Costs

Item	Unit	Productora	Alice	Cortadera OP	Cortadera UG	San Antonio
Concentrate Freight to Port Site and Load on Ship	US$/lb	0.06	0.06	0.06	0.06	0.06
Concentrate Freight from Port to Customer Port	US$/lb	0.11	0.11	0.11	0.11	0.11
Cu Concentrator Cost (Treatment Charge - TC)	US$/lb	0.16	0.16	0.16	0.16	0.16
Cu Refining Charge (RC)	US$/lb	0.09	0.09	0.09	0.09	0.09
Total Copper Selling Cost	**US$/lb**	**0.42**	**0.42**	**0.42**	**0.42**	**0.42**
Molybdenum Processing Cost	US$/lb	0.21	0.95	0.74	0.31	79.0
Molybdenum Sustaining Capital	US$/lb	0.034	0.12	0.167	0.04	4.4
Molybdenum Concentrate Freight to Smelter	US$/lb	0.12	0.12	0.12	0.12	0.12
Molybdenum TC-RC	US$/lb	0.65	0.65	0.65	0.65	0.65
Total Molybdenum Selling Cost	**US$/lb**	**1.02**	**1.85**	**1.68**	**1.12**	**85.0**

The selling cost for gold and silver was US$5/oz Au and US$0.5/oz Ag for all deposits.

Table 16.6 : Mining Operating Cost[1]						
Item	**Unit**	**Productora**	**Alice**	**Cortadera OP**	**Cortadera UG**	**San Antonio**
Reference Mining Cost	US$/t	2.03	2.23	2.03	6.55	2.23
Additional Vertical Mining Cost (per 5 m below pushback exit RL)	US$/t	0.0125	0.0125	0.0125	Not applicable	0.0125
Additional Vertical Mining Cost (per 5 m above pushback exit RL)	US$/t	0.00425	0.00425	0.00425	Not applicable	0.00425
Reference Mining Cost	US$/t	2.03	2.23	2.03	6.55	2.23

Table 16.7 : Process Plant Throughput						
Item	**Unit**	**Productora**	**Alice**	**Cortadera OP**	**Cortadera UG**	**San Antonio**
Plant throughput	Mtpa	22.3	23.2	24.2	19.4	19.4
Moisture Content in Cu Concentrate	%	8%	8%	8%	8%	8%
Moisture Content in Mo Concentrate	%	5%	5%	5%	5%	5%
Cu grade in Dry Concentrate	%	25%	25%	25%	25%	25%
Mo grade in Dry Concentrate	%	50%	50%	50%	50%	50%

[1] **MINING COST includes: Load, Haul, Drill, Blast, Ancillary fleet, Rehabilitation, Dewatering (no pioneering fleet considered at this stage)*

Table 16.8 : Sulphide Processing Cost						
Item	**Unit**	**Productora**	**Alice**	**Cortadera OP**	**Cortadera UG**	**San Antonio**
Processing Cost	US$/ore t	5.12	4.99	5.17	5.96	9.84
Concentrator Operating Cost	US$/ore t	4.82	4.69	4.55	5.34	5.22
RopeCon and Road Transport	US$/ore t	0.00	0.00	0.32	0.32	4.32
Change in Grind Size (125 – 106 µm)	US$/ore t	0.30	0.30	0.30	0.30	0.30
Additional Production Feed Mining Costs (compared to waste mining)	US$/ore t	0.12	0.12	0.12	0.12	0.12
Average Rehandle Cost	US$/ore t	0.47	0.47	0.47	0.47	0.08
G&A	US$/ore t	0.59	0.57	0.54	0.68	0.68
Sustaining Capex	US$/ore t	0.59	0.63	0.63	0.63	0.63

Table 16.9 : Open Pit Oxide Recoveries (@ 3.3 Mt/a Throughput Rate)					
Item	**Productora**	**Alice**	**Cortadera OP**	**San Antonio**	**San Antonio**
Cu Recovery	56% Fixed	46% Fixed	50% Fixed	70% Fixed	9.84

Table 16.10 : Open Pit Oxide Selling Costs ($/lb in Cathode)					
Item	**Unit**	**Productora**	**Alice**	**Cortadera OP**	**San Antonio**
Cathode Freight to Port Site and Load on Ship	$/lb	0.013	0.013	0.013	0.013
Cathode Freight from Port to Customer Port	$/lb	0.058	0.058	0.058	0.058
Cu SX-EW Cost	$/lb	0.278	0.278	0.278	0.278
Cu G&A	$/lb	0.00	0.00	0.00	0.00
Selling	$/lb	0.00	0.00	0.00	0.00
Total	**$/lb**	**0.35**	**0.35**	**0.35**	**0.35**

Table 16.11 : Open Pit Oxide Mining Cost [2]					
Item	**Unit**	**Productora**	**Alice**	**Cortadera OP**	**San Antonio**
Reference Mining Cost	$/t	2.03	2.23	2.03	2.23
Additional Vertical Mining Cost (per 5 m below pushback exit RL)	$/t	0.0125	0.0125	0.0125	0.0125
Additional Vertical Mining Cost (per 5 m above pushback exit RL)	$/t	0.00425	0.00425	0.00425	0.00425

Table 16.12 : Oxide Processing Cost					
Item	**Unit**	**Productora**	**Alice**	**Cortadera OP**	**San Antonio**
Front End Processing Cost	$/ore t	4.46	4.46	4.46	4.46
Additional Production Feed Mining Costs (compared to waste mining)	$/ore t	0.11	0.11	0.11	0.11
Average Rehandle Cost	$/ore t	0.56	0.56	0.56	4.32
G&A	$/ore t	0.00	0.00	0.00	0.00
Sustaining Capex	$/ore t	1.19	1.19	1.19	1.19
Total Leaching Cost	**$/ore t**	**6.32**	**6.32**	**6.32**	**10.08**

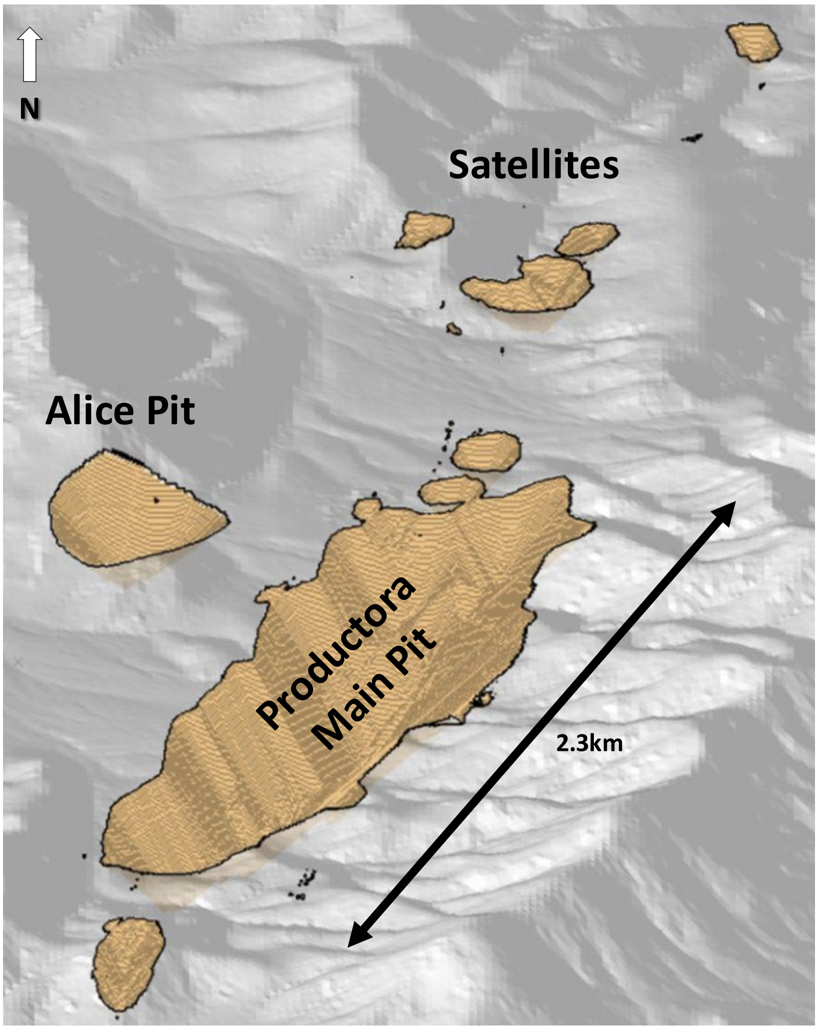
[2] **MINING COST (MCAF) includes: Load, Haul, Drill, Blast, Ancillary fleet, Rehabilitation, Dewatering (no pioneering fleet considered at this stage)*

Various royalty costs were also included. The most notable Royalty is the CCHEN royalty at Productora, as well as the Purisima Royalty applied to Cortadera (See section 22.8 for further detail).

16.2.5 Results

Optimisations for the Productora area produced several pits, consisting of Productora main pit (and satellite pits) and Alice pit, as shown in Figure 16.4. The Productora main pit is over 2 km in length and roughly 750 m wide, while the Alice pit is closer to 500 m by 500 m.

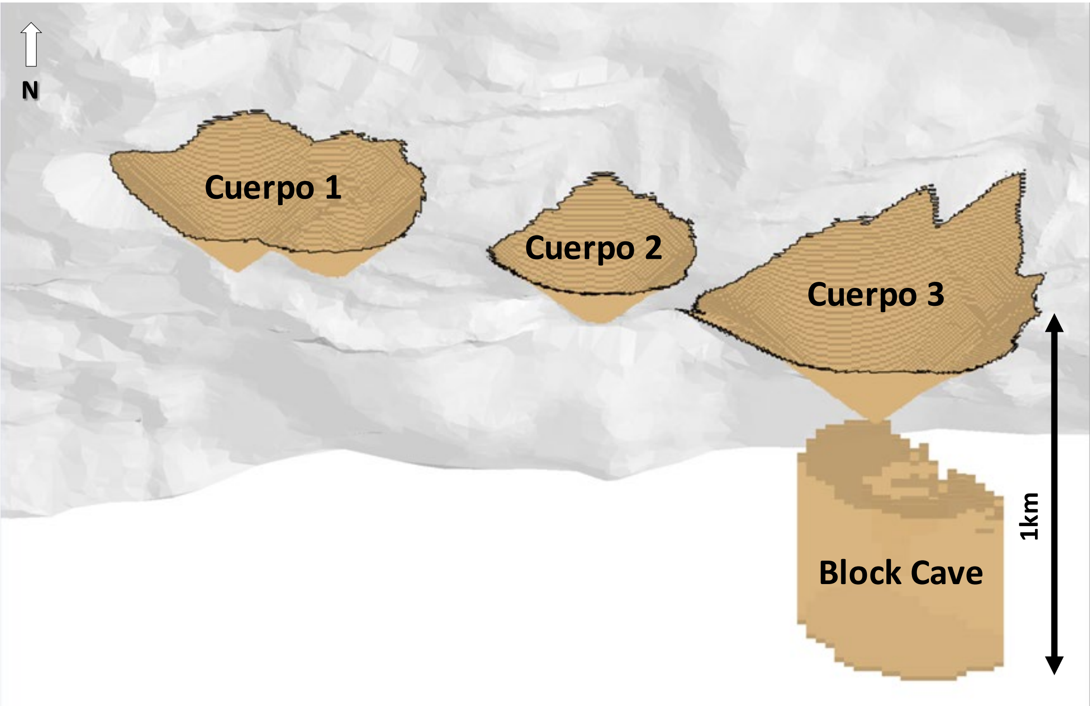
Figure 16.4 : Productora and Alice Open Pits - View Looking North



Cortadera (located roughly 15 km east-southeast from Productora) results consist of three pits, namely Cuerpo 1, 2 and 3. 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 and is depicted Figure 16.5.

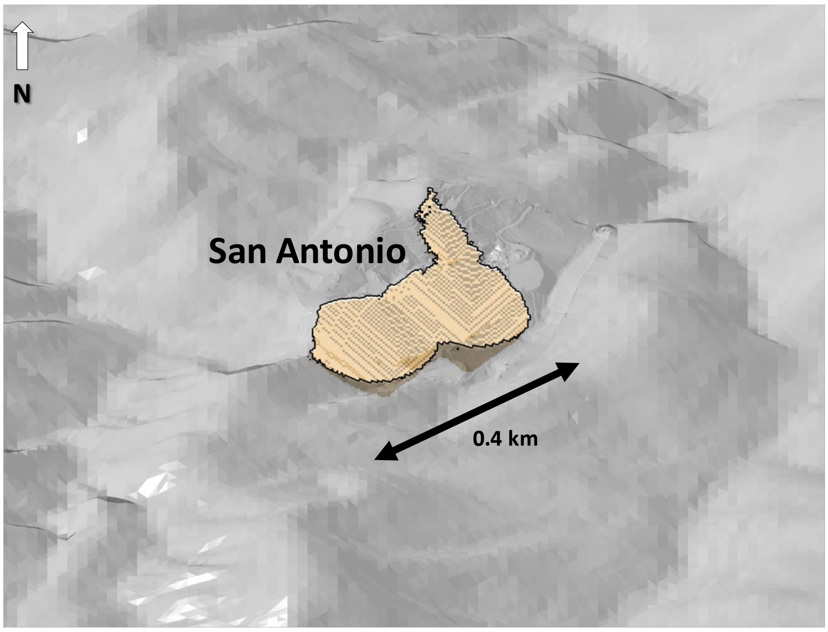
Key underground results from the FF process indicate that footprint elevation remains within ±5% of the best value over a vertical range of almost 100 m. This provides great flexibility for establishing an early footprint at a higher elevation, potentially followed by a second lift deeper down if ongoing drilling confirms further mineralisation at depth. FF indicates a best elevation of 220 mRL, which correlates to the base of the undercut for a detailed PCBC design, and an approximate extraction level elevation of 200 mRL (20 m below base of undercut). Elevations much lower than the best elevation are constrained by the maximum height of draw selected, and elevation much shallower than the best elevation is located too close to the pit and therefore does not have sufficient volumes to mine. Multi-lift lower heights of draw (example two 300 m lifts) were not evaluated and could be considered with resource model updates. FF produced a footprint of approximately 121 000 m^2, ovoid shaped of approximately 520 m long and 320 m wide.

Figure 16.5 : Cortadera Open Pits and Block Cave Shapes – View Looking North



The San Antonio pit is located roughly 8 km northeast of the Cortadera mining area and is shown in Figure 16.6.

Figure 16.6 : San Antonio Open Pit – View Looking North



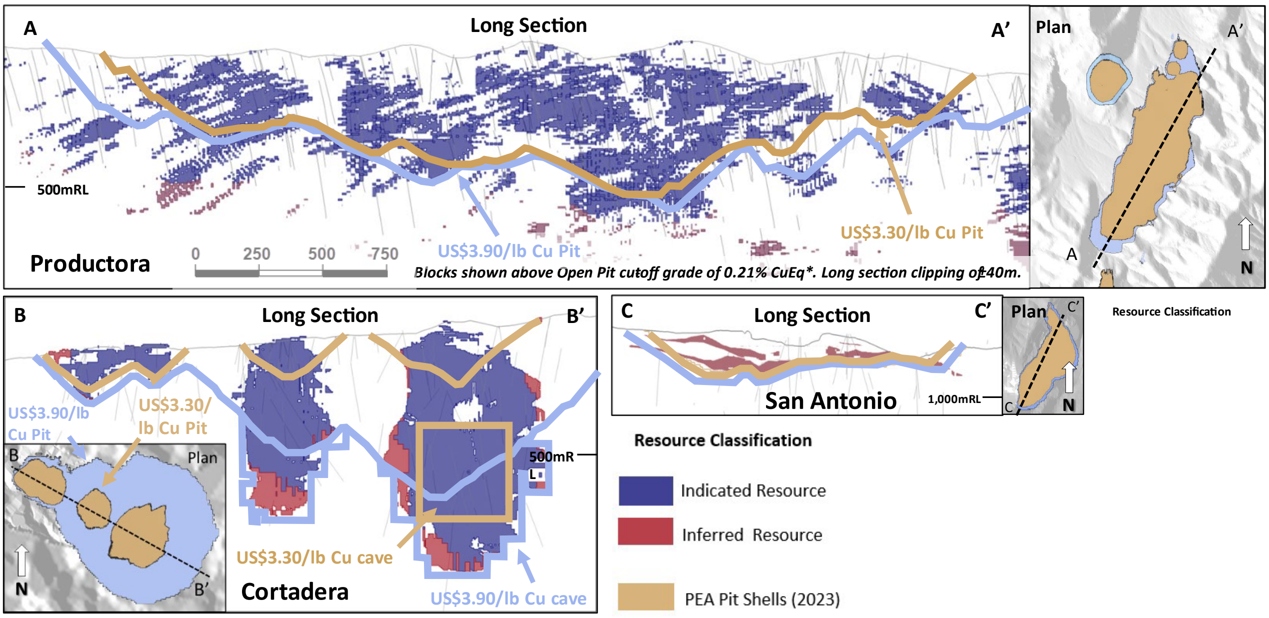
Production feed breakdown is the material which provides a positive economic contribution and is contained within the optimised shell boundaries. The production feed breakdown is the material estimated from the regularised block models and includes the material coming from blocks classified as Measured, Indicated and Inferred Resources. The diluted material from these blocks is referred to as production feed. The breakdown includes Oxide leach, Low Grade (LG) sulphide leach and Sulphide concentrator material.

The PEA only considered the open pit shells and block cave footprint for the production feed breakdown shown in Table 1.6. Open pit shells made provision for potential ramps (via pit shell slope angle) and no pit designs were completed for this study. Production feed is comprised of 97% Indicated Mineral Resource material (illustrated in Figure 16.7) providing a model fit for the PFS, where Inferred material cannot be ascribed value.

Table 16.13 : Production Feed Breakdown

Material	Units	Total
Oxide Leach	Mt	37
Cu Grade	%	0.42
LG Sulphide Leach	Mt	100
Cu Grade	%	0.14
Sulphide Concentrator	Mt	334
CuEq*	%	0.52
Cu Grade	%	0.44
Au Grade	g/t	0.12
Mo Grade	Ppm	117
Ag Grade	g/t	0.45
Waste Rock Tonnes	Mt	627

Figure 16.7 : Resource Model Blocks Filtered for Potential Production Feed and Coloured by Resource Category to Demonstrate Indicated Material Extending Beyond the Planned and Potential Pit Limits



16.2.6 Relevant Factors Affecting Production Feed Breakdown

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

16.3 Waste Rock Dumps and Stockpiles

Dumps were designed in 36 m lifts. Each lift is constructed at an approximate angle of repose of 37°. A 30 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.

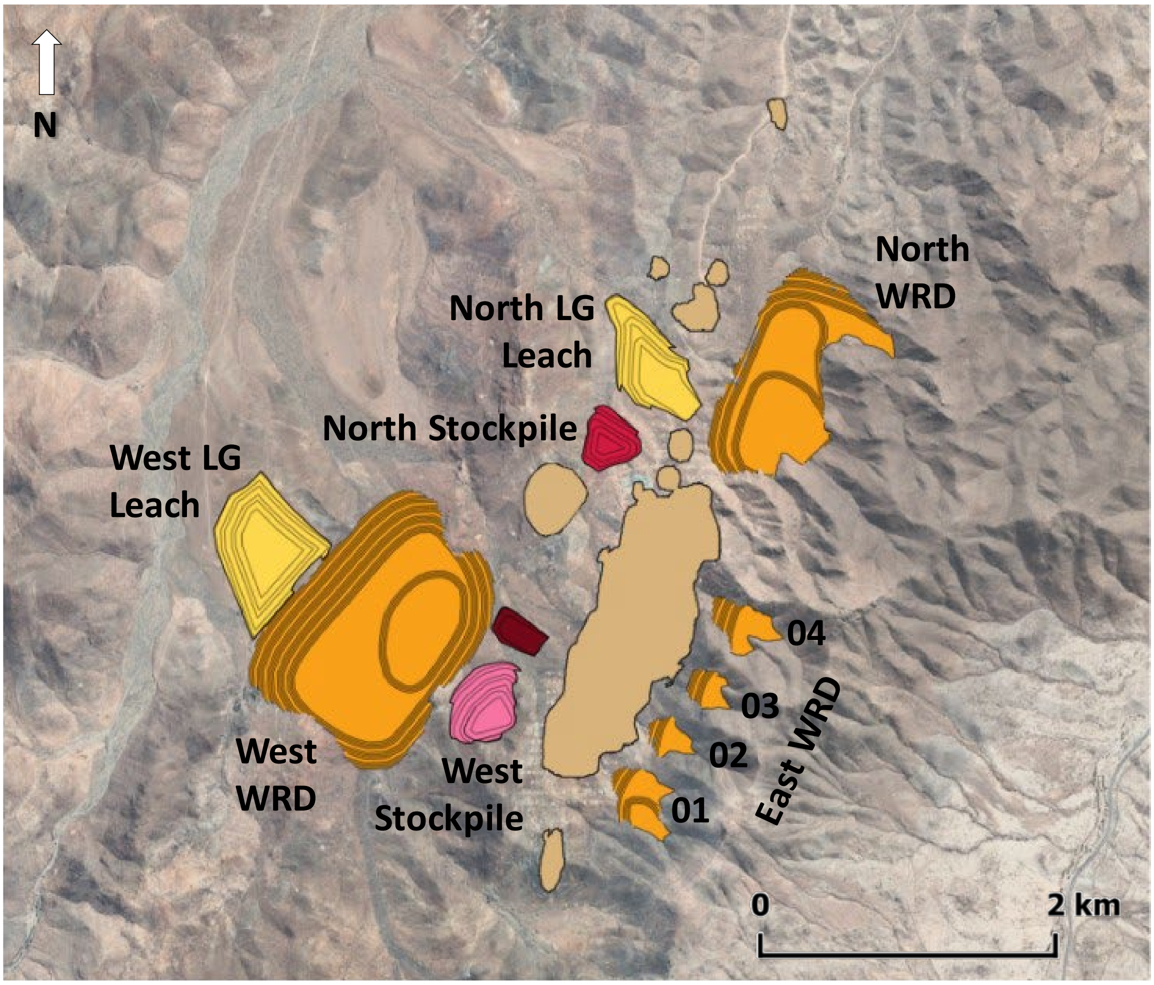
During the pre-production period, the ROM pad area will be constructed close to the primary crusher for later rehandling. The total process feed to be stockpiled during this period amounts to around 1.1 Mt.

All stockpiles are designed with the same parameters as the waste dumps: 36 m lifts, 30 m berm and 37 ° batter angle.

16.3.1 Productora

Productora utilises a combination of Waste Rock Dumps (WRD), Leach Dumps and Stockpiles, aiming to minimise the haulage costs associated with transporting and placing production inventory. Figure 16.8 shows the placement and relative size of these dumps and stockpiles.

Figure 16.8 : Plan View of the Productora WRD, Leach Dump and Stockpile Layouts



The calculated capacities of the respective dumps and stockpiles are shown in Table 16.14.

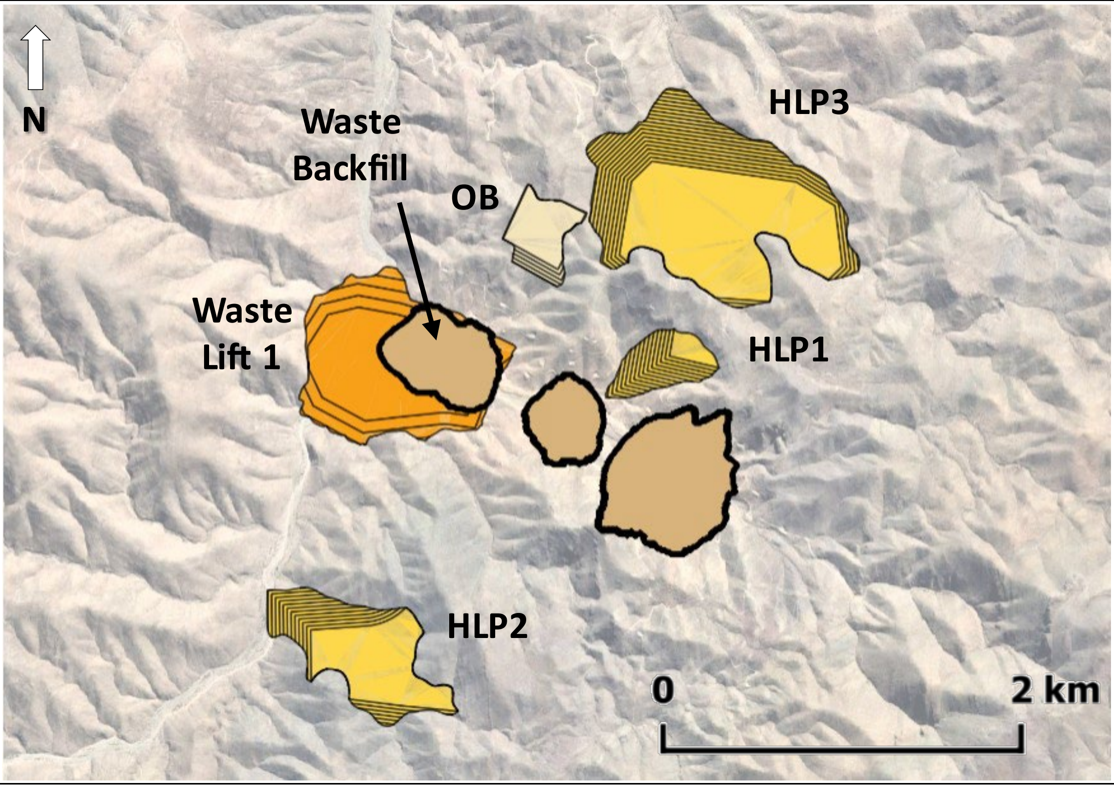
Table 16.14 : Productora Rock Dump and Stockpile Capacities	
Name	**Volume (m^3)**
North WRD	50.4
East04 WRD	2.9
East03 WRD	1.2
East02 WRD	1.2
East01 WRD	4.4
West WRD	180.5
Total Waste Capacity	**240.6**
North STKPL	5.3
West STKPL	9.9
North LG Dump Leach	11.7
West LG Dump Leach	19.5

The Productora Pit void volume is approximately 263.8 M m^3, with the total waste volume that will require placement being roughly 238.6 M m^3. Low grade (LG) sulphide dump leach capacity required is approximately 30.0 M m^3 with room for further expansion of the western dump if required in the future.

16.3.2 Cortadera

Cortadera utilises a combination of smaller and larger Waste Rock Dumps and Leach Dumps, aiming to minimise the haulage costs associated with transporting and placing production inventory. Figure 16.9 shows the placement and relative size of these dumps.

Figure 16.9 : Plan View of the Cortadera PEA Waste Dump and Heap Leach Layouts



The calculated capacities of the respective waste dumps and heap leach pads are shown in Table 16.15

Table 16.15 : Cortadera Waste Rock Dump and Heap Leach Capacity

Name	Volume (m^3)
Heap Leach Phase 1 (HLP1)	4.2
Heap Leach Phase 2 (HLP2)	13.4
Heap Leach Phase 3 (HLP3)	53.9
Total Leach	**71.6**
Cuerpo 1 Waste Backfill	20.4
Cuerpo 1 Waste Lift 1	43.2
Office Block area (OB)	1.2
Total Waste	**64.7**

The Cortadera Pit waste volume that will require placement being roughly 43.7M m^3, with a further 2.5M m^3 associated with UG waste generated.

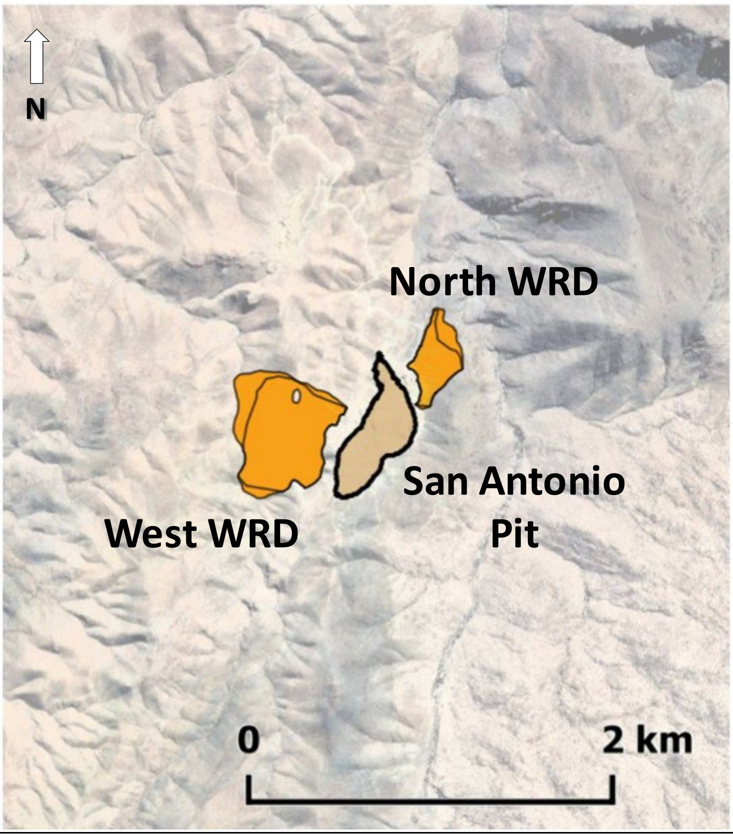
To ensure that the environmental approvals process could start as soon as possible and yet remain valid through subsequent levels of study with likely upgrades in Resource tonnage, all the dump outlines were expanded a further 100 m metres laterally in all directions. This provides another ~10% - 15% of waste capacity and ~15% - 20% sulphide leach capacity.

An alternative waste storage location that may be considered is the area between Heap Leach Phase 2 and the Cuerpo 3 Pit, which was earmarked as a waste dump in previous iterations of the dump designs. It will be important to complete sterilisation drilling for all proposed locations to ensure future economic material is not sterilised.

16.3.3 San Antonio

San Antonio utilises a combination of two larger Waste Rock Dumps and no Leach Dumps. All production feed will be transferred to the Productora process plant. Figure 16.10 shows the placement and relative size of these dumps.

Figure 16.10 : Plan View of the San Antonio Waste Dump Layouts



The calculated capacities of the respective waste dumps are shown in Table 16.16

Table 16.16 : San Antonio Waste Rock Dump Capacity

Name	Volume (m^3)
North Waste Rock Dump	9.9
West Waste Rock Dump	1.3
Total Waste	**11.2**

The San Antonio Pit waste volume that will require placement being roughly 10.2 M m^3.

16.4 Mine Operations and Equipment

16.4.1 Mining Strategy

Open pit mining of copper deposits is a well understood activity in Chile where some of the largest copper mines in the world can be found, including Chuquicamata, Escondida and Collahuasi. Mining engineering expertise for pit optimisation, pit and waste dump designs, mine scheduling, mine operation and cost estimation is generally well understood by most mining companies and consultancy firms.

Block caving is a well-known method in South America, and worldwide block caving experience has grown significantly in recent years and continues to grow through the operation of large international block caves such as Cadia, Grasberg and Oyu Tolgoi. It is expected that the required caving experience and skills will be available at the time a potential block cave at Cortadera requires these skills.

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 to:

- Define standards and guidance for mining activities
- Manage mining production and safety
- Manage resource definition and resource estimate
- Manage process feed control activities

- Manage geotechnical activities
- Run long-term and medium-term mine planning
- Manage mining costs through administration of mining contractor agreement.

16.4.2 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, resource modelling, mine and waste dump surveying, process feed sampling, geotechnical support, mine statistics and mine contract analysis.

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.

16.4.3 Mine Equipment Selection

Mine equipment was selected to perform the following duties:

- Construct roads to the initial mining areas and to the crusher, waste storage areas and stockpiles, construct additional roads as needed to support mining activity
- The pre-production development required to expose process feed for initial production
- Mine and transport process feed to the primary crushers
- Mine and transport waste from the pit or underground to the waste storage areas
- Maintain all the mine work areas, in-pit haul roads and external haul roads; and maintain the waste storage areas
- Rehandle the process feed and marginal process feed (load, transport and auxiliary equipment) from the stockpiles to feed the primary crusher.

No specific mining units were specified for the PEA, but it is expected that the contractor will have suitable main waste and process feed production equipment: typically around a 34 m^3 bucket excavator in waste and a 22 m^3 hydraulic excavator for the process feed, both loading 220 t trucks.

Underground mine equipment is assumed to be 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) (circa 17 t to 21 t capacity), trucks (circa 60 t capacity), ground support equipment (shotcrete sprayers, agitator trucks), charge-up rigs, secondary breaking rigs, integrated tool carriers, boxhole rigs (drawbell development) and other auxiliary support equipment. 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.

16.4.4 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.

16.4.5 Loading and Hauling

The Productora main pit and Cortadera Cuerpo 2 and 3 assume that the mining operation could use 34 m^3 hydraulic excavators for waste and 22 m^3 hydraulic excavators for process 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 suitable grade control. 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, Satellite oxide and Cortadera Cuerpo 1 pits assume the use of 15 m^3 front-end loaders or a combination of hydraulic excavators with similar bucket capacities and 90 t trucks.

Mining at San Antonio is expected to utilise smaller mining units, similar to the Alice pit and Satellite oxide pits, as described previously.

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.

16.4.6 Open Pit Grade Control

Blast hole sampling is necessary for process feed control purposes. Blast holes will be sampled and tested for copper, gold and molybdenum to define the sulphide process feed, transitional process feed, oxide process feed and waste. Cortadera and San Antonio also contain silver, which will be included in the grade control testing suite.

HCH personnel will conduct the sample collection and deliver grade control samples to an external company for sample preparation and laboratory analysis in Vallenar. An estimated grade control operating cost of US$0.5 M per year is 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.

Careful grade control must be carried out during mining to minimise misplaced process feed, due to the effect of head grade on recovery. These efforts should include the following standard procedures:

- Implement an intense and systematic program of sampling, mapping, laboratory analyses and reporting
- Utilise specialised in-pit, bench sampling drills for sampling well ahead of production drilling and blasting
- Use of back hoes to selectively mine process feed zones
- Maintain high quality laboratory staff, equipment, and procedures to provide accurate and timely assay reporting

- Utilise trained geologists and technicians to work with shovel operators in identifying, marking and selectively mining and dispatching process feed and waste
- Ongoing interim and final pit wall mapping to continue development of the geological, alteration, structural and mineralisation models by mine geologists.

16.4.7 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 process feed from the stockpiles to the primary crushers.

The low grade and heap leach process 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 Satellite oxides pits.

Equipment utilisation will be analysed in more detail during the next study phase.

16.5 Geotechnical Analysis

16.5.1 Productora Open Pit

16.5.1.1 Setting

Ingeroc (Ingeniería de Rocas Ltda.) was engaged to undertake a pit and waste dump slope stability study to provide recommendations for pit and dump slope angles to be used for this study.

Previous studies were carried out in March 2014, which reviewed all relevant data derived from the resource development drilling in addition to a specific geotechnical surface mapping campaign over several drill hole platform excavations (40 pads) and two existing underground mines. An overall slope angle for Productora final pit was recommended for different geotechnical rock domains with the data available at the time.

Field work started at the end of 2014 with a geotechnical drilling campaign. Structure orientations were taken from oriented core (Reflex Technique). Core orientations were not often achieved. To improve the structural orientation information, bore hole scanning techniques (optical and acoustic methodologies) were undertaken in 39 of the bore holes (from either geotechnical or exploration drilling campaigns). These cover areas of both Alice and Productora pits.

Along with the geotechnical logging, Point Load Testing (PLT) and core sample selection for laboratory testing were undertaken in order to assist the geotechnical characterisation of the rock mass.

In addition to bore holes, field work was executed mainly in the Alice and waste dump areas. In the Alice area, 33 geotechnical windows were mapped in order to obtain geotechnical properties of the rock mass. The information provided by these new geotechnical windows was merged with the one available from 2014 field work. In the waste dump areas, geotechnical mapping of several trial pits was executed on the location defined in areas of the base of the preliminary dump designs.

The consideration of available sources of information, characterisation, analysis and process using either geotechnical or geological properties of the rock mass, led to the definition and characterisation of the geotechnical units that define the rock mass where Alice and Productora projects are placed. This geotechnical characterisation was carried out considering the following geotechnical classification systems:

- Rock Mass Rating (RMR, Bieniawsi, 1989)
- Mining Rock Mass Rating (RMR-MRMR, Laubscher, 1990)
- NGI-Q System (Q, Barton, 2000)
- Geological Strength Index (GSI, Hoek&Brown, 1997).

16.5.1.2 Productora Pits Slope Stability Evaluation

Pit slope stability was analysed using three separate methodologies:

- Rotational Analysis
- Structural Analysis
- Ritchie Criterion.

Results for the rotational analysis (static and pseudo-static limit equilibrium analysis) evidence that all the Factors of Safety (FoS) of the rotational analysis are above the established limits (1.3 for the static case and 1 for the pseudo-static case). In addition, analysing the stability through the structural analysis in the inter-ramp case and the overall slope angle, shows again all the FoS are above the established limit for the deterministic analysis (1.3) as well as the probabilistic analysis (percentages below 5 and 30% for FoS below 1.0 and 1.3 respectively).

16.5.1.3 Productora Waste Dumps and Slope Stability Evaluation

Stability analysis for waste dump designs has considered the maximum capacity option of each waste dump.

In order to evaluate waste dump stability, trial pits were defined to evaluate ground conditions and select soil samples for laboratory tests.

In total, eight trial pits were constructed and developed to characterise the soil over which waste dumps will be placed. These were designed to a minimum depth of 5 m, unless this was not possible due to geological issues.

Based on geological interpretation of the different soil levels identified during the site visit, seven samples were collected and sent to Universidad de Chile laboratory (FCFM) for classification tests such as granulometry, Atterberg limits and USCS Classification definition. Soluble salts content was also tested for three of the samples (Calicata 3, 4 and 6).

None of the trial pits identified ground water levels. As a result, stability analysis was done considering dry substratum.

Considering results from the trial pits evaluation, the laboratory tests and consultant experience on similar cases, the geotechnical parameters defined for the different relevant geological units, defined as their characteristic soils, are presented in Table 16.17.

Table 16.17 : Geotechnical Parameters Defined for the Identified Relevant Geological Units that Form the Waste Dump Substratum

Geological Unit	Friction Angle (°)	Cohesion (Kpa)	Density (g/cm³)	E (Mpa)
Sandy silt with isolated gravels	26	0 00	1.60	4 - 8
Clast-supported gravels	32 - 35	0,00 - 50 00	2.00 - 2.10	100 – 500
Rock mass	42	200 00	2.60	500 - 8 000

Waste dump stability was investigated based on rotational analysis. Sections through waste dumps were evaluated considering static and pseudo-static cases and factor of safety (FoS) results calculated. Results (assuming a minimum failure width of 20 m) indicate that waste dump designs are stable when at maximum capacity.

16.5.2 Cortadera Pit Geotechnical Review

Ingeroc was engaged to provide recommendations for pit slope angles to be used for this study, based on the geotechnical surveys carried out in drill holes and geotechnical mapping.

The statistical analysis of data regarding the geological alterations indicates that propylitic and phyllic alterations are associated with the best geotechnical qualities. The inter-argillic alteration has lower geotechnical qualities.

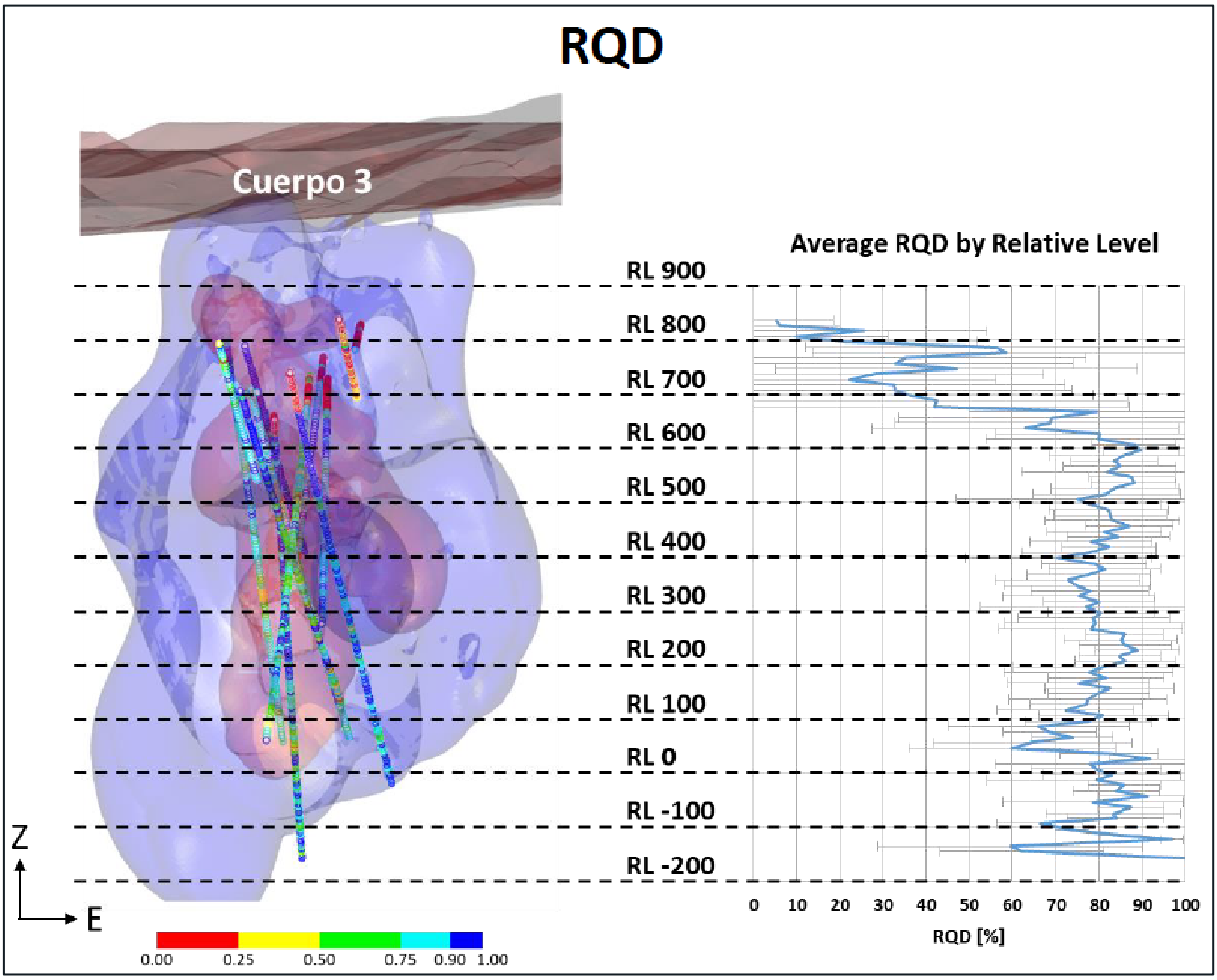
According to the Rock Mass Rating (RMRb) index, the porphyric and skarns rocks have Fair (40-60) to Good (60-80) geotechnical qualities.

16.5.3 Cortadera Block Cave Geotechnical Analysis

A concept level study was conducted by Geomechanics, Mining and Technology (GMT) to review the geological and geotechnical information available to review the rock mechanics aspects of a potential block cave for Cuerpo 3 underground. The analysis is based primarily on empirical data and will be refined in the next study phase.

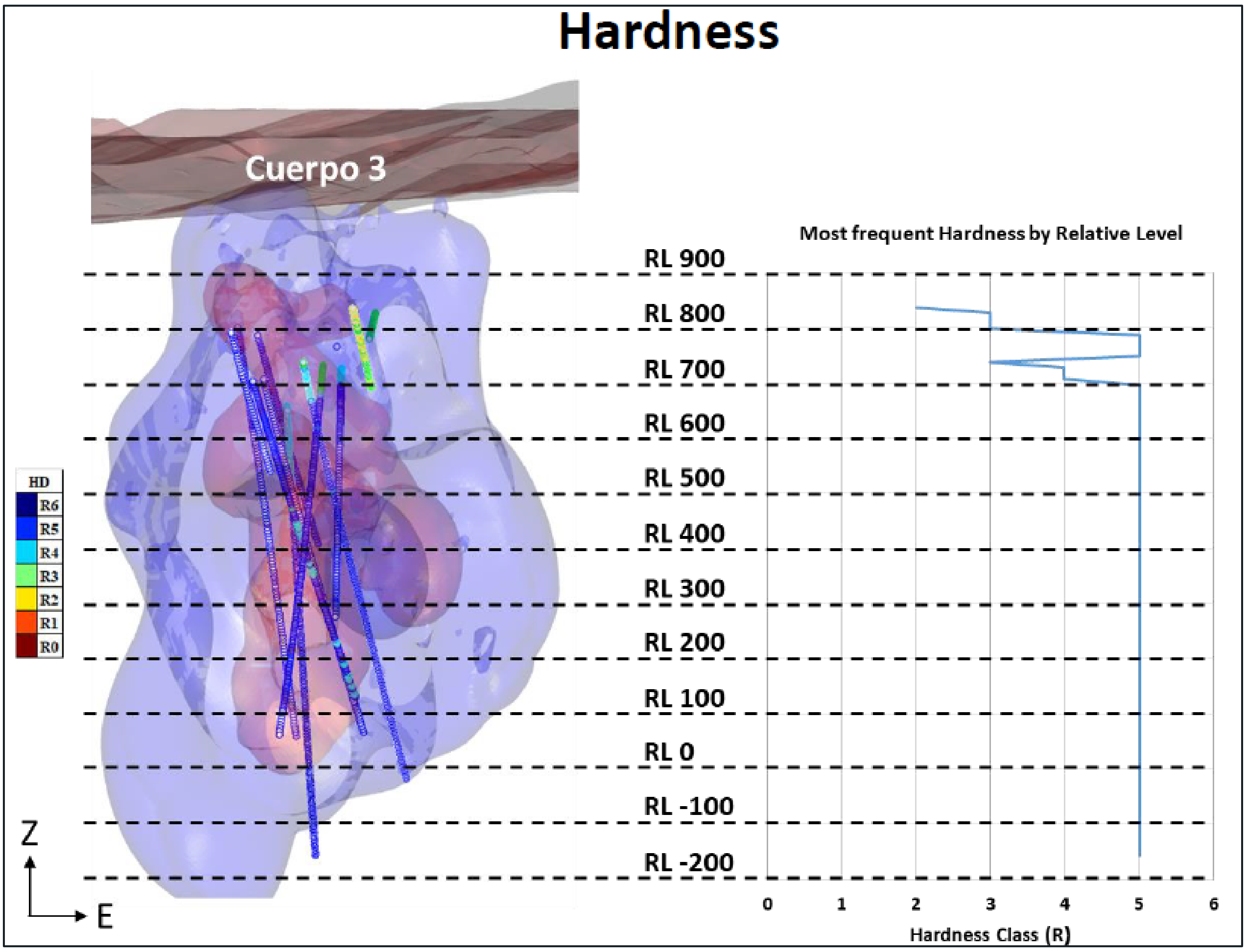
Geological information was obtained from logging DD holes and reported alteration and mineralisation, weathering, and major structures (5 drill holes, ~780 m). Fracture frequency and Rock Quality Designation (RQD) logging was conducted on 13 drill holes (~6 476 m).

Figure 16.11 : Average RQD by Relative Level



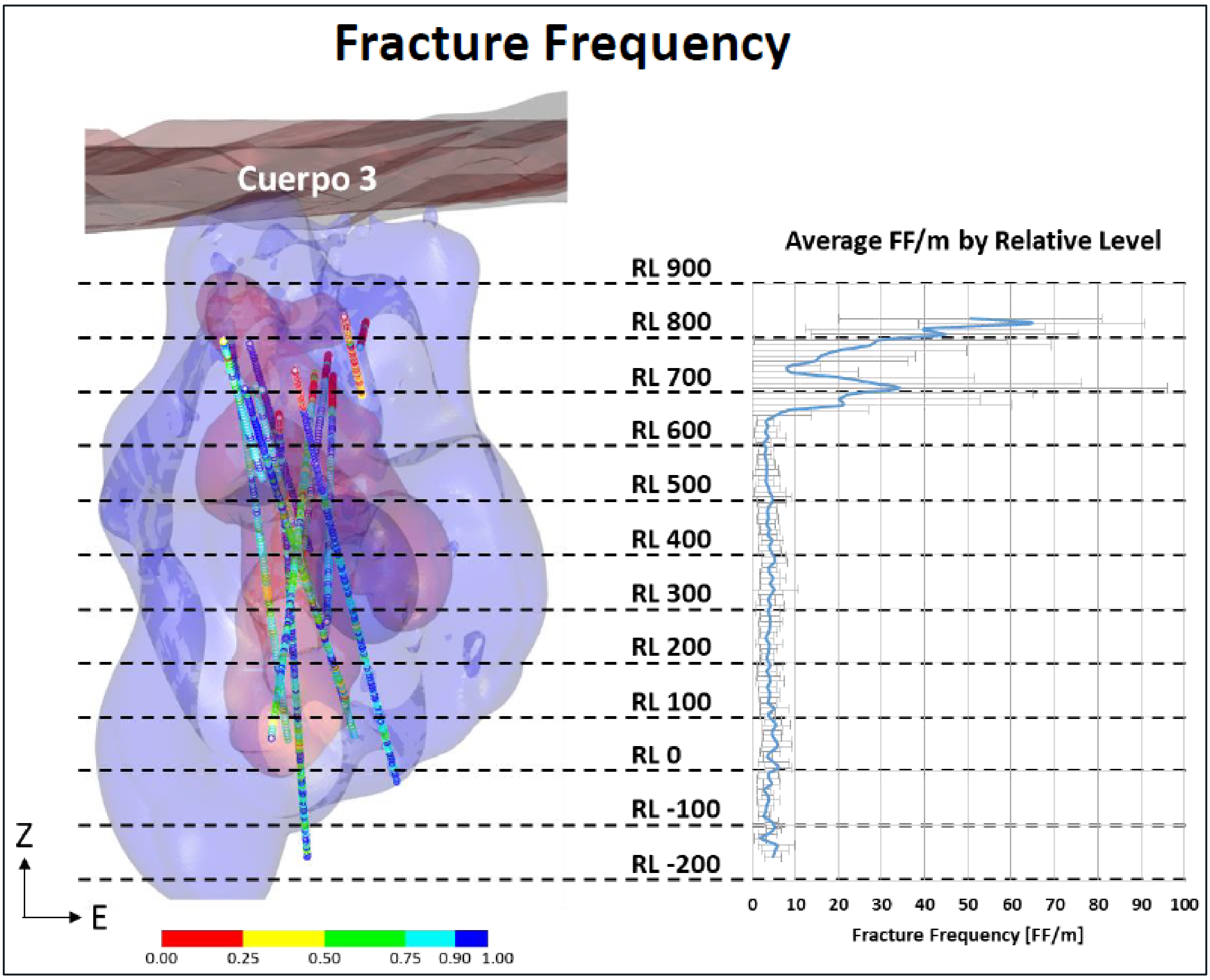
Note : GMT, 11 February 2021, Review of the geological/geotechnical information and evaluation of the rock mechanics aspects for an underground mass mining caving method

Figure 16.12 : Rock Hardness by Relative Level



Note : GMT, 11 February 2021, Review of the geological/geotechnical information and evaluation of the rock mechanics aspects for an underground mass mining caving method)

Figure 16.13 : Average Fracture Frequency/m by Relative Level



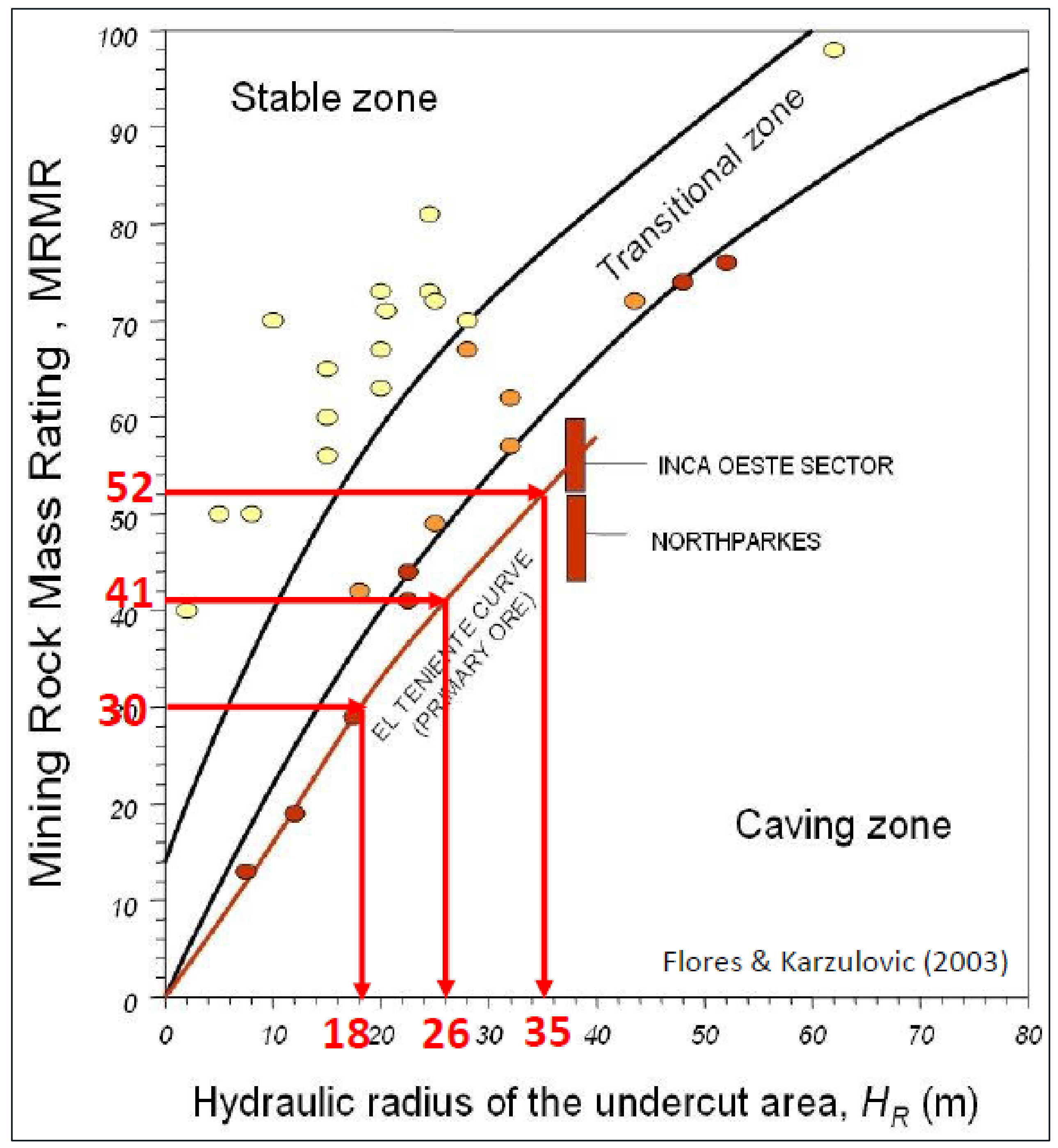
Note : GMT, 11 February 2021, Review of the geological/geotechnical information and evaluation of the rock mechanics aspects for an underground mass mining caving method

The analysis provided the following results:

- Rock Mass Rating (RMR) (Laubscher – 1977): 37 to 65 (rock quality of poor to good), based on:
  - Hardness (Uniaxial Compressive Strength, UCS): 100 to 250 Mpa, assumed intact rock strength is equal to UCS
  - Fracture Frequency (FF)/m: 3-7/m
  - Joint condition rating: 15 to 27 (assumed average of 31 with 30% variability)
- Mine Rock Mass Rating (MRMR): 30 to 52 based on adjustments:
  - A_M (Weathering adjustment) = 1 (no weathering)
  - A_O (Joint orientation adjustment) = 0.8 (three joint sets)
  - A_T (Blasting adjustment) = 1 (no blast damage)
  - A_S (Mining induced stress adjustment) = 1 (mining induced stress, conservative)

- Hydraulic radius: 18 to 35 (used 35 to start caving, maximum >90 for full footprint)
  - Based on El Teniente caving curve and analysis results above.

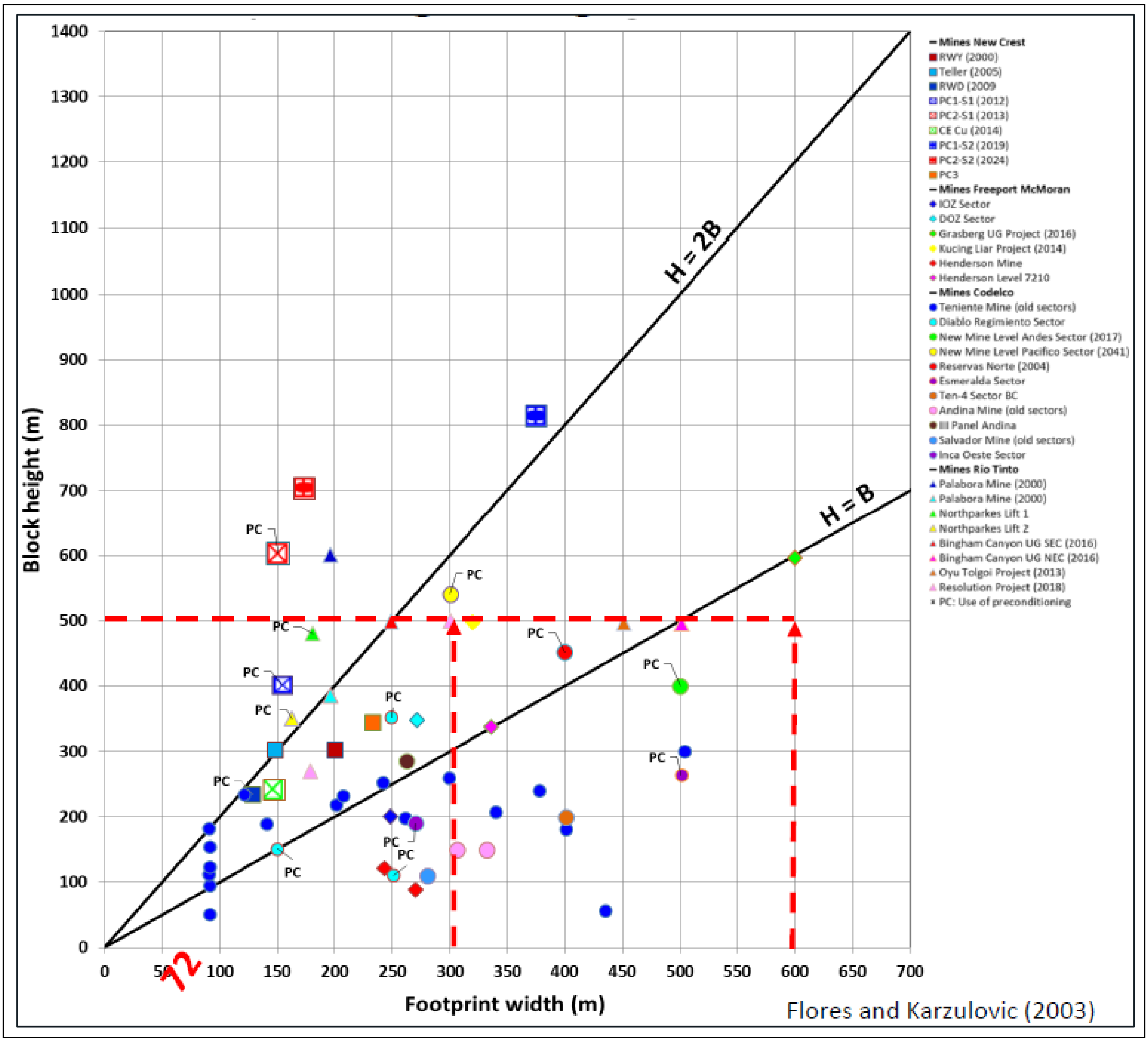
Figure 16.14 : Block Cave Hydraulic Radius



Wood, 30 June 2021, Cortadera Deposit – Identification Study for Block Caving.

A maximum cave height of 400 to 500 m is anticipated, with footprint widths of 300 to 600 m, placing the potential cave well within current benchmark aspect ratios of cave width versus height.

Figure 16.15 : Maximum Cave Height



Wood, 30 June 2021, Cortadera Deposit – Identification Study for Block Caving.

Since the potential cave is located directly below a large open pit, major surface subsidence it is not anticipated to be a concern and will be further investigated in the next study phase.

The average fracture frequency of 5 FF/m indicates fragmentation of 0.1 to 2.0 m can be expected, after Laubscher 1994. More detailed fragmentation analysis and precondition will be considered in the next study phase.

Based on the analysis above, it is considered that block caving is feasible at a PEA level. Further data gathering and analysis will be conducted in the next study phase.

16.6 Other Mining Engineering Items

16.6.1 Hydrogeological Field Campaign, Pore Pressure and Pit Dewatering Analysis

16.6.1.1 Hydrogeological Field Campaign

The hydrological and hydrogeological studies of the Project were undertaken by Artois Consulting Ltda. from 2012.

The hydrogeological field campaign 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)
- Permeability testing of soils and rock formations within the TSF area by Ausenco (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).

16.6.1.2 Productora Groundwater and Pore Pressure Analysis

Based on the geological setting, groundwater flow in the Project area is likely to occur along the weathered units near surface, and as fracture flow within the igneous rock formations, inducing a northwest-ward regional flow direction. At greater depth, flow is confined to the regional fault zones and the discontinuities associated with the mineralised zone, such as the stock work and the breccia.

In addition, a conceptual model was generated as a representation of the groundwater flow regime within the Project area. This model is described as follows:

- The 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-ward 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 done assuming a final pit depth in the order of 500 metres above mean sea level (mamsl), Both 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 inntercept 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 6821500 mN and 6822000 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.

16.6.1.3 Productora Pit Dewatering

The arid climate of Chile Region III, 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.

16.6.1.4 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 a1:50 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 waste rock dump (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 hydrochemical 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.

16.6.2 Underground Design

To gauge the timing for the Cuerpo 3 block cave tonnage within the overall Project and to determine infrastructure requirements outside the cave Footprint (as this is accounted for in the FF component), a conceptual development design was completed within Deswik.CAD. The design utilises a twin decline layout, keeping access near long term vertical ventilation rises to facilitate practical development down to the 150 m RL 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.

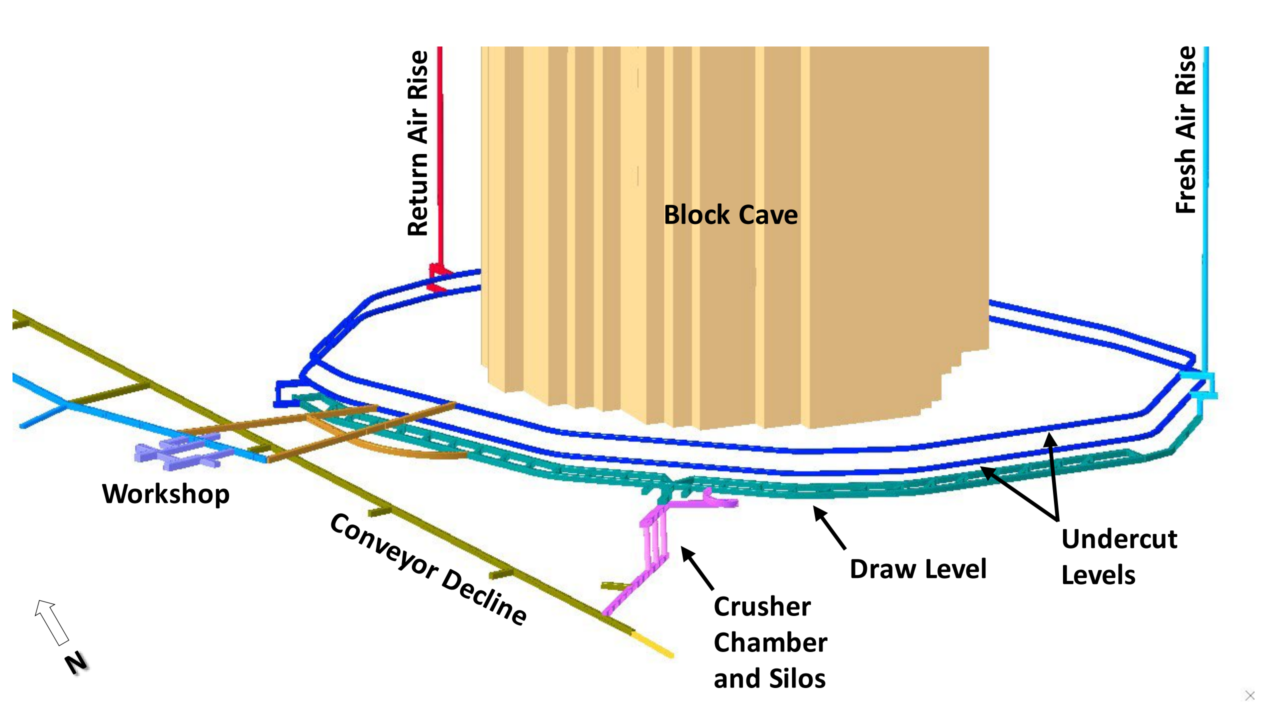
FF results indicated that the block cave draw level should be located on the 200 m RL level, and the undercut levels located on the 220 m RL and 240 m RL levels to initiate caving. The block cave footprint design makes allowance for a larger footprint to place long term infrastructure outside the cave abutment zone and to ensure the estimated development quantities will suffice if the resource increases, following additional resource drilling. It also ensures sufficient time is allowed to establish the initial capital infrastructure from a reasonably conservative approach.

The undercut levels will utilise a set of perimeter drives that connects the mine workings with a Fresh Air Rise and a 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 draw level. This fleet will consider conventional diesel propulsion as well as fully electrical solutions in the next stage of study. 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.

The workshop design will need to 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 conceptual draw level design feeds an underground crusher complex and a series of three vertical crushed process feed silos that may be used for grade control purposes and surge capacity in the event of production delays. The optimal draw level layout, number of crushers and silos will be optimised in subsequent phases of study and is significantly dependent on the ultimate production target for the Cortadera block cave. If higher production targets are considered, it may be prudent to allow for two to three crushers and multiple silos, reducing the bogging distance from draw points and resulting in a more robust system through redundancy of the crushers and total surge capacity. The design in Figure 16.16 shows the layout for a single crusher and triple silo arrangement.

Figure 16.16 : Concept Block Cave Layout



Standard excavation profiles were utilised and will be optimised in the next phase of study.

16.7 Mine Production Scheduling

16.7.1 Underground Schedule

The underground development/access schedule was developed in Deswik.CAD and Deswik.SCHED. Following the timing of when the undercut level is accessed and when undercut and the block cave production levels could commence, the block cave draw schedule was developed first in the Footprint Finder (FF) module in Geovia's PCBC software and then imported into Datamine's MultiMine Scheduler (MMS) software to combine the open pits and underground schedules (see section 16.7.2).

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 that the block cave should not reach hydraulic radius and associated caving whilst there is still 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. This is quite 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.

The block cave undercut will be developed at roughly four to six draw bells per month steady state, or 20 000 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.

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.

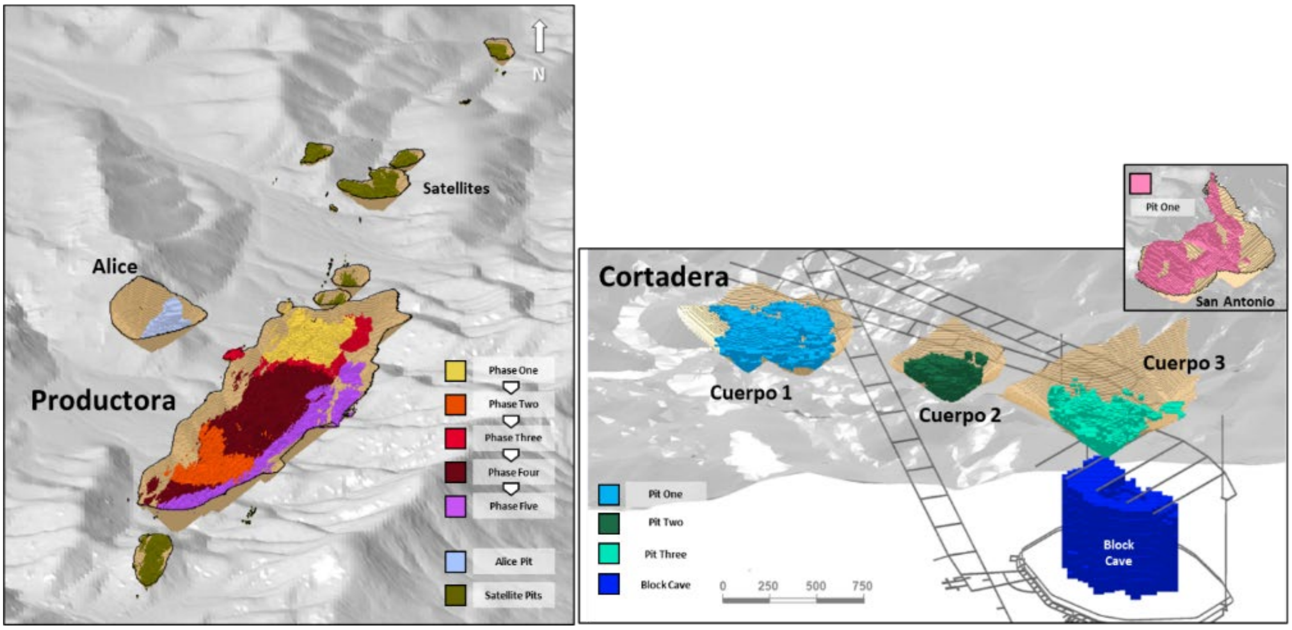
16.7.2 Combined Schedule

A 20 Mt/a sulphide concentrate processing and 10-12 kt/a cathode SX-EW production throughput rate was considered during the PEA mine scheduling.

The mine/production scheduling was completed for all the open pits and the block cave option within Datamine's Multi-Mine Scheduler (MMS) and further refined with Material Allocation Optimiser (MAO) software. The block cave blocks, and sequence was exported from Footprint Finder and imported into MMS and MAO software to enable a complete/combined mine production schedule for all the open pits and the underground block cave.

The Productora deposit comprises the largest volume of open-pit mineralisation, with six pit pushbacks that are phased throughout the LOM. The Cortadera deposit consists of three separate pits, with the mining sequence commencing with the Cuerpo 1 pit, which has the largest volume of higher-grade, near-surface mineralisation. The block cave is timed with the Cuerpo 3 pit so as not to influence the pit operations. Satellite pits at Productora as well as pits at Alice and San Antonio are mined in a single phase. Figure 16.17 illustrates the mining phases for the project.

Figure 16.17 : Costa Fuego Mining Phases



As the software preferentially selects maximum NPV material for prioritised processing, various adjustments to the schedule were implemented to consider practical and financial constraints, which resulted in the generation of a robust schedule reflecting the anticipated mine sequence and operational considerations.

Industry standard assumptions on direct feed ratios were utilised to arrive at a weighted ROM rehandle cost (ROM loader only); with the software utilised to re-code and apply deferred stockpile rehandle costs for a loader and truck combination which attracted a higher cost for material movement. Vendor quotes were utilised for other key inputs such as rope conveying from Cortadera to Productora.

Whilst there is similar information in section 16.2.4, Table 16.18 and Section 16.7.2, these figures are used for different purposes - economic limit inputs in Section 16.2.4 are utilised to generate pit shells and viable block cave options, while scheduling inputs in Table 16.18 are applied to the pit shells and block cave footprint for mine scheduling purposes and the subsequent optimisation of the schedule to achieve a realistic, pragmatic and financially attractive solution.

The process feed cut-off strategy, defined by the mining schedule, requires deferring any low-grade process feed produced to maximise the plant's throughput rate and the project NPV. The low-grade process feed will be placed on independent sulphide heap leach dumps, bringing the leached product forward in the production schedule. Oxide stockpiles will be fed into the processing plants through rehandling operations later in the schedule. The estimated process feed rehandling differs for the different processing capacity options considered for the PEA. The Project benefits from a faster mining/depletion rate making higher grade process feed available to the concentrator processing plant/mill. By leaching the low-grade sulphide process feed material earlier in the schedule, stockpiles which may have been impacted by long term exposure and therefore unintended metal leaching, has been addressed.

Table 16.18 : Scheduling Inputs and Parameters (Compiled October 2022)						
Category / Item	**Unit**	**Productora**	**Alice**	**Cortadera**	**San Antonio**	**Note**
2015 – 2022 Escalation Factor	% from 2015 US$	20.4%	20.4%	20.4%	20.4%	Section 21.4.2.1
2015 – 2022 Scaling Factor	% from 15 Mt/a	86.7%	86.7%	86.7%	86.7%	2015: 14.4 Mt/a 2022: 20 Mt/a
Selling Prices	US$/lb US$/oz	Var.	Var.	Var.	Var.	Ref: Table 16.10
Mining Recoveries						Applies during mining operations
Process Feed Loss	%	5.0%	5.0%	5.0%	5.0%	
Process Feed Dilution	%	3.0%	3.0%	3.0%	3.0%	
Metallurgical Recoveries	%	Var.	Var.	Var.	Var.	Applies during processing operations Ref: Table 16.4
Mining Operating Costs (used in Multimine)						
Open Pit Cost	$/t	$2.03	$2.03	$2.03	$2.23	Base RL
Open Pit Cost	$/t per 12 m RL decrease	$0.03	$0.03	$0.03	$0.03	Cost increase per bench below base
Open Pit Cost	$/t per 12 m RL increase	$0.01	$0.01	$0.01	$0.01	Cost increase per bench above base
Block Cave Cost	$/t	N/A	N/A	$6.55	N/A	Benchmarked
Processing Operating Costs (used in Multimine)						
Heap Leach Oxide *HL1, HL3*	$/t	$4.33 *HL1*	$4.33 *HL1*	$5.09 *HL3*	N/A	Formula driven in block model
Concentrator *PC1, PC3, PC5, PC8a*	$/t	$5.50 *PC1*	$5.50 *PC1*	$5.57 (OP) *PC3* $3.64 (BC) *PC5*	$15.10 *PC8a*	PC1 formula, PC3, 5 calculated PC8a – batched Lower cost for PC5 due to UG crush and conveyor process
Deferred Stockpile/Rehandling Costs (used in MAO)						
Heap Leach Oxide *HL2, HL4*	$/t	$0.84 *HL2*	$0.84 *HL2*	$0.84 *HL4*	N/A	MAO applies cost where deferred stockpile material is processed. Higher cost for PC6 due to additional conveyancing

Table 16.18 : Scheduling Inputs and Parameters (Compiled October 2022)						
Category / Item	**Unit**	**Productora**	**Alice**	**Cortadera**	**San Antonio**	**Note**
Concentrator *PC2, PC4, PC6, PC8*	$/t	$1.16 *PC2*	$1.16 *PC2*	$0.43 (OP) *PC4* $0.53 (BC) *PC6*	N/A	
Selling Costs	various					Ref: Table 16.5 and Table 16.10
Additional Royalties		4% of revenue (CCHEN)	N/A	N/A	N/A	Royalty on CCHEN (1:70) excluding Productora (1:16) and CMP - Cu

The following graphs depict the mine production schedule and copper production profile for the 20 Mt/a concentrator processing throughput option.

Figure 16.18 : Costa Fuego Mine Production Schedule

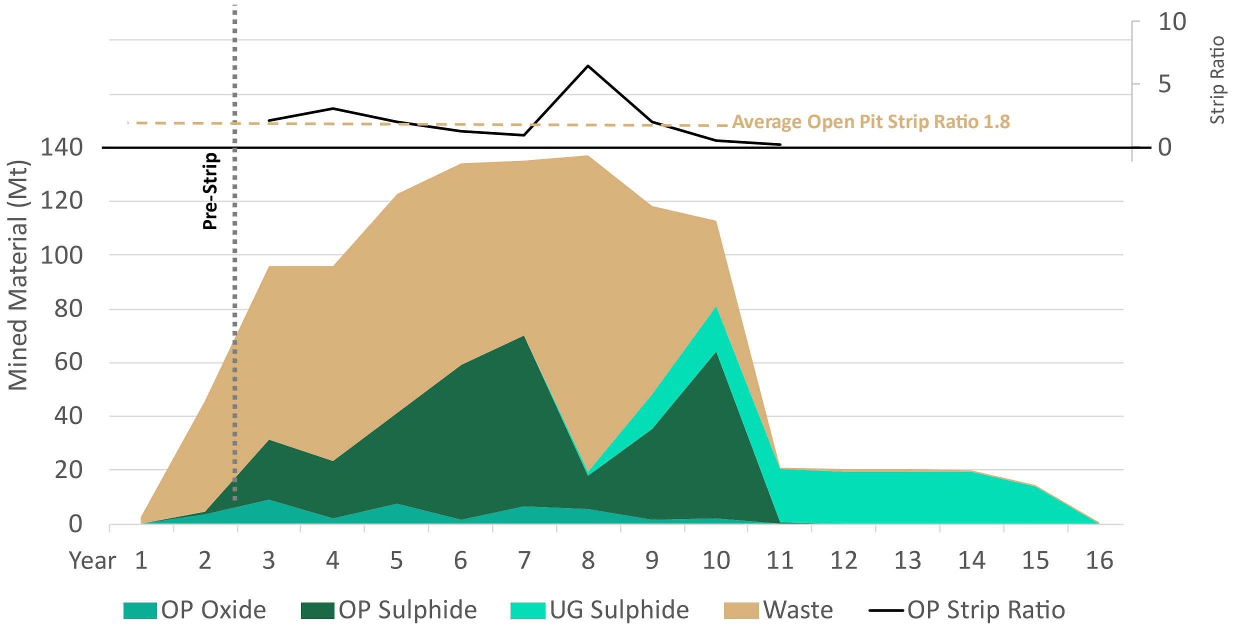


Figure 16.19: Costa Fuego Mine Production Schedule by Area

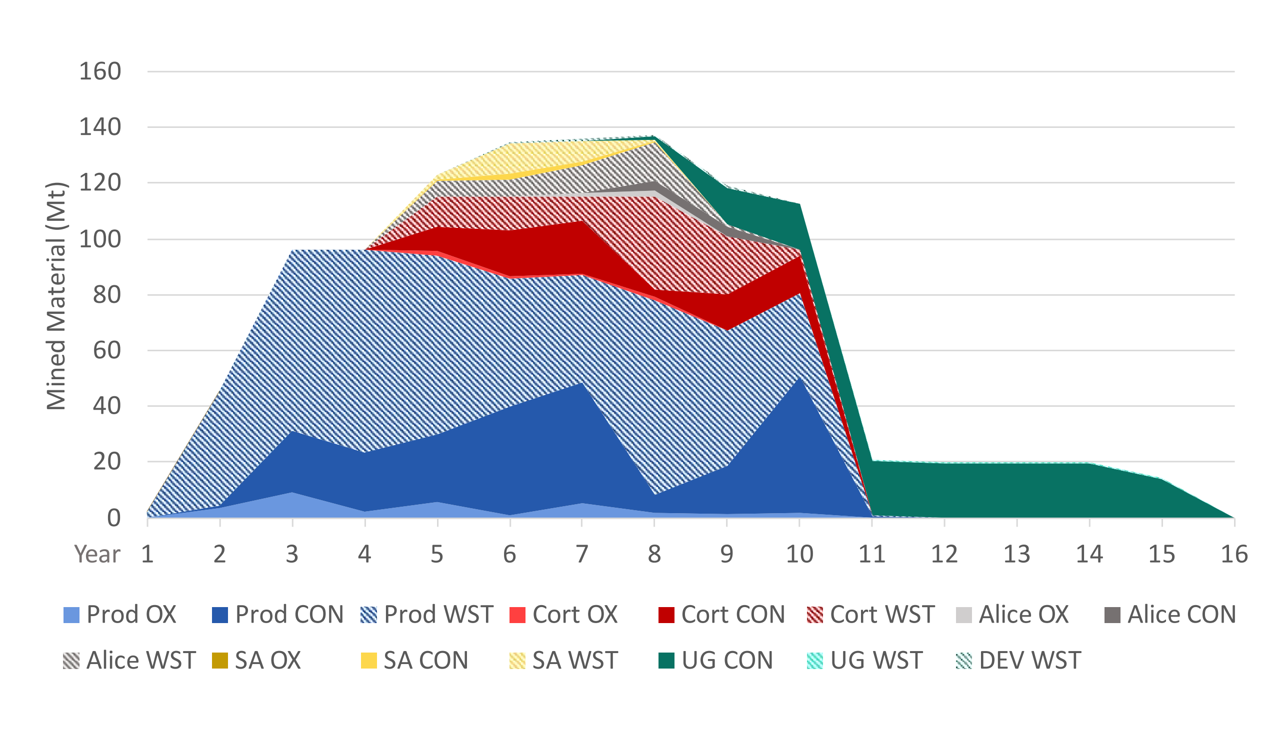


Figure 16.20: Concentrator Production Feed Schedule

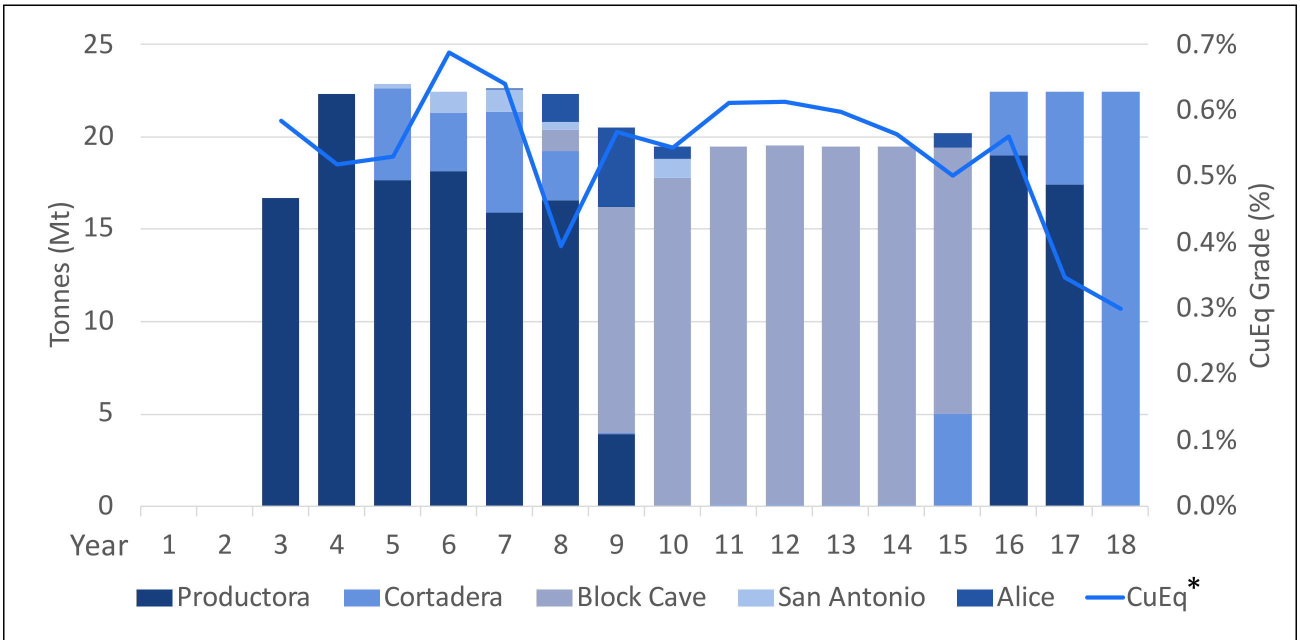
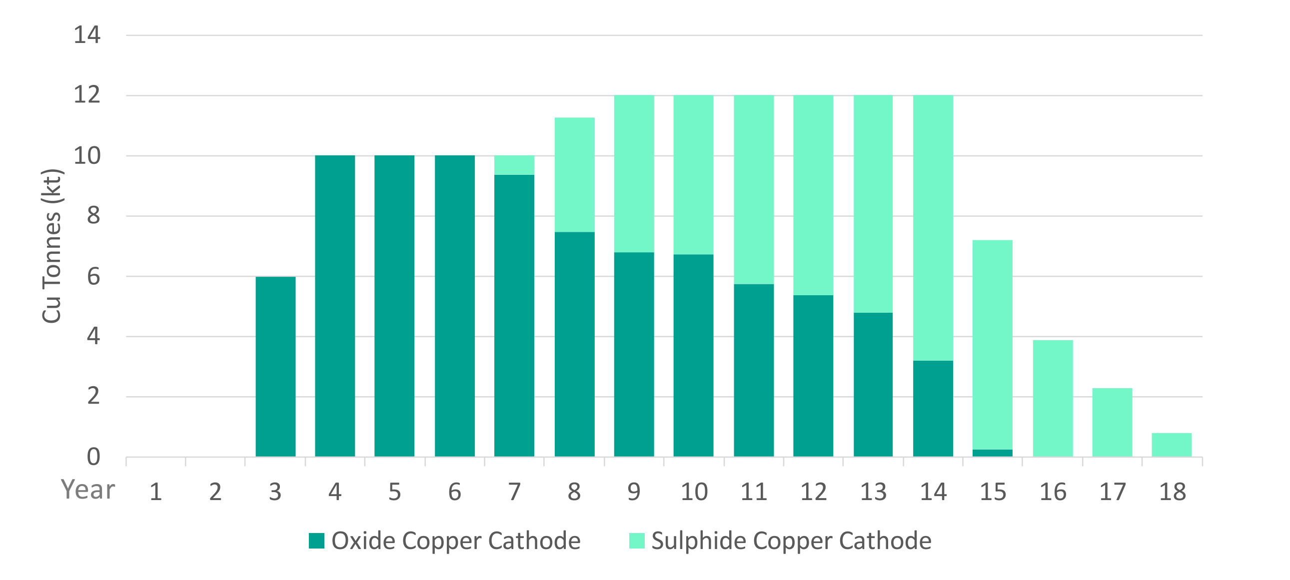
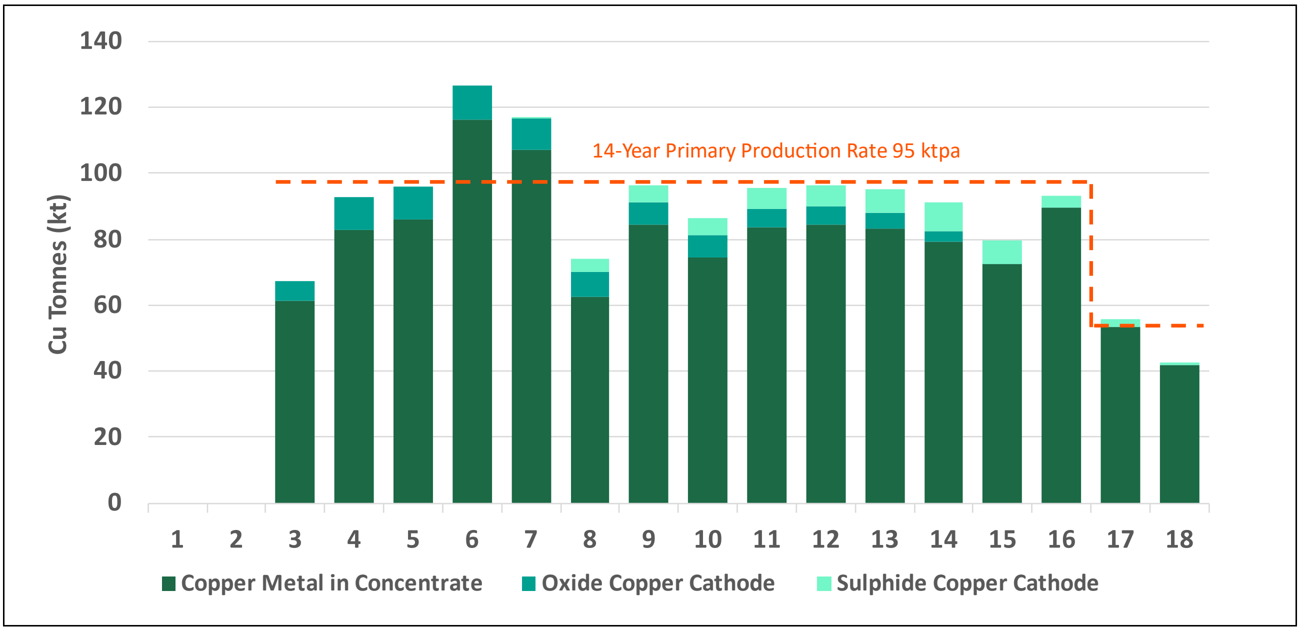


Figure 16.21 : Cathode Copper Metal Produced



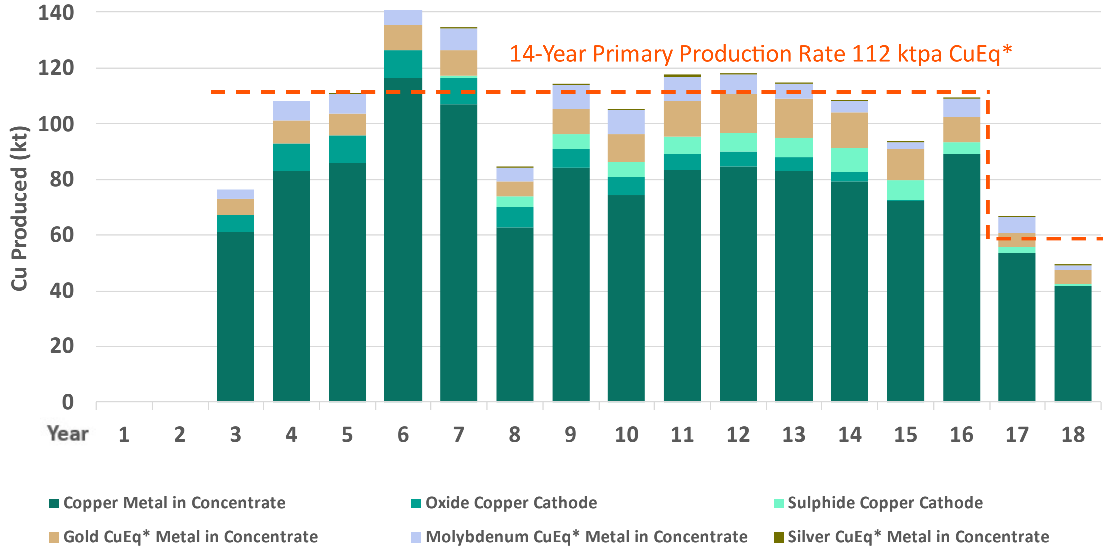
An average copper production of +95 Kt/a was maintained for 14 years from Year 3 through to year 16 (see Figure 16.22).

Figure 16.22 : Total Copper Metal Produced



An average copper equivalent production of +112 Kt/a was maintained for 14 years from Year 3 through to year 16 (see Figure 16.23).

Figure 16.23 : Total Copper Equivalent* Metal Produced



*The copper-equivalent (CuEq) annual production rate was based on the combined contribution of processing feed from all production sources and was estimated to match the combined revenues anticipated using long-term commodity prices of: Copper US$ 3.85/lb, Gold US$ 1750/oz, Molybdenum US$ 17/lb, and Silver US$21/oz; and estimated metallurgical recoveries for the production feed to the following processes: Concentrator (87% Cu, 56% Au, 37% Ag, 58% Mo), Oxide Leach (55% Cu only), and Low-grade Sulphide Leach (40% Cu only).

Figure 16.24 : Gold and Silver Metal Produced

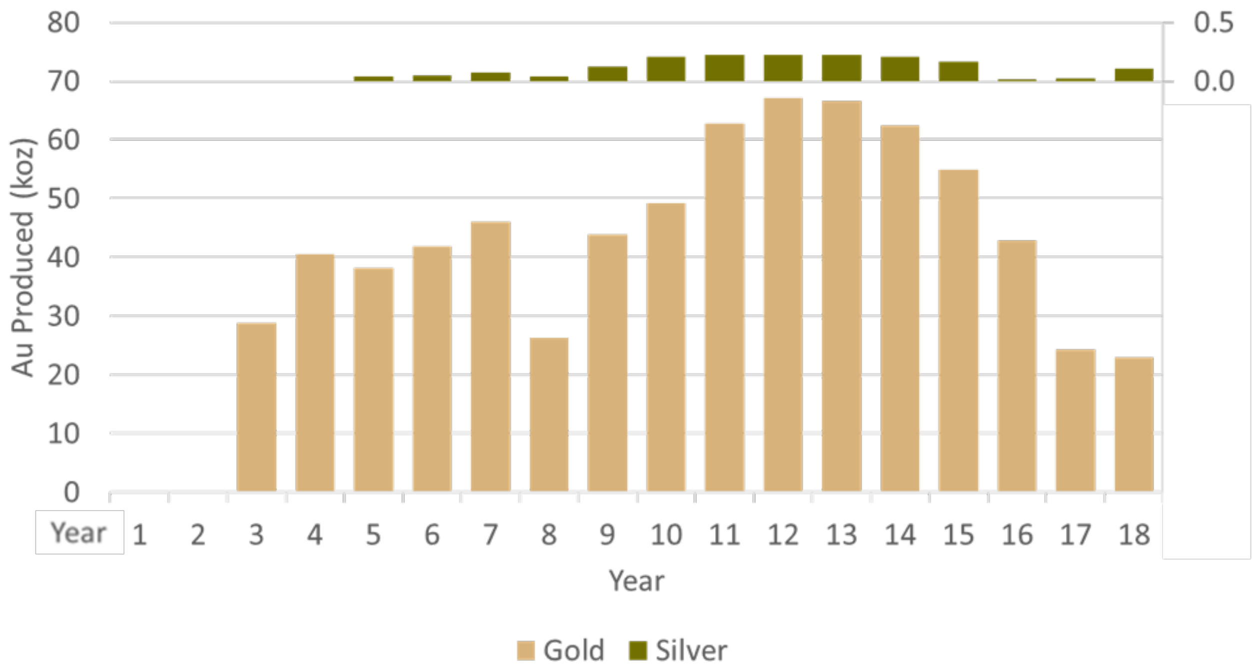
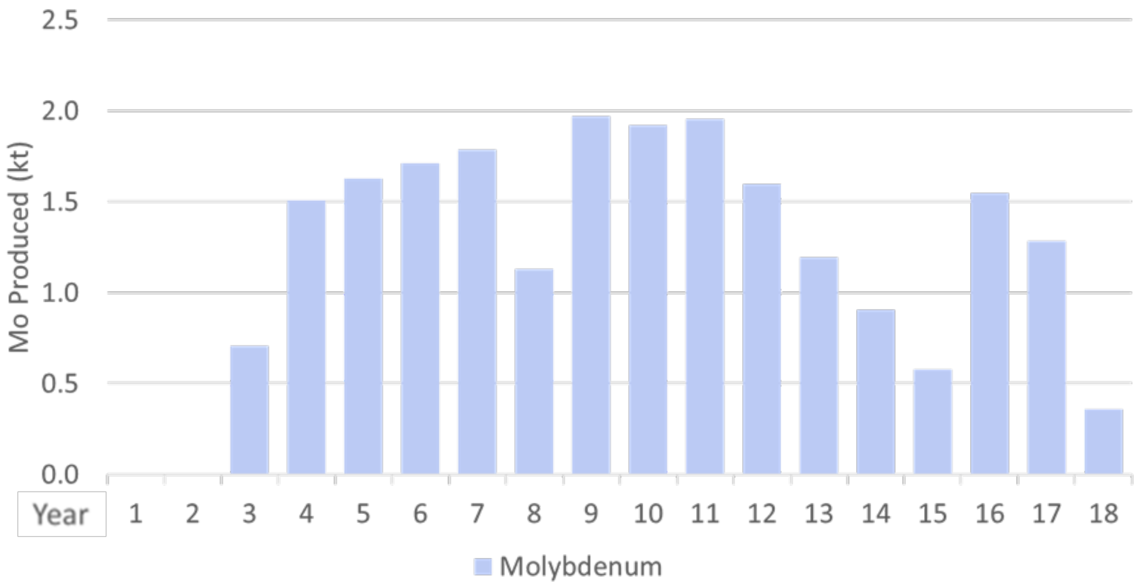


Figure 16.25 : Molybdenum Produced



The mine production schedule and process schedules can be viewed in Section 22.

16.8 Comments on Section 16

This mine plan is conceptual in nature and some elements require further study to refine concepts and increase accuracy of the estimates.

The next stage of study will include:

- Investigate a large single open pit scenario for Cortadera, trade-off investigations for large open pit vs. underground block cave for the Cortadera area
- Detailed studies and costings to support a PFS.

Future geotechnical work required to support further mining studies includes (but not limited to):

- Additional Geotech drilling to support increasing study detail
- Pit and waste dump slope stability studies based on final waste dump designs and capacities for all areas
- Dump leach location and stability investigations.

16.8.1 Comments on Production Schedule

The production scheduling for the open pits utilised Multi-Mine Scheduler and Mine Allocation Optimiser (MAO) which enabled the integration and optimisation of stockpile strategies and optimising feed value for each period from the various production feed sources. The scheduling of the open pits furthermore used practical sinking rate constraints and pushback volumes making the scheduling approach fit-for-purpose and adequate for this level of study.

The block cave assessment is at a concept level of accuracy and was conducted to support the PEA only. Based on the PEA level of assessment it was concluded that the block cave makes a positive contribution to the overall mine and processing plan and should therefore be considered in the overall decision-making process, such as processing, infrastructure, mine life, etc. To be conservative and to align with the Cortadera open pit schedule, the cave production is scheduled for processing at the tail end of the overall open pit processing profile.

A detailed design and schedule for this initial cave establishment and ramp-up was not constructed during this assessment and will be part of the next study phase. Such a schedule should consider potential delays in cave access development and establishment, as well as potential shortfall in the open pit production to ensure a smooth processing transition from open pit to underground material. This design and schedule will provide further support for a detailed first principles schedule and cost estimate for access development, infrastructure and cave establishment.

The assumptions used during the caving assessment were based on international benchmarks and can vary significantly based on local conditions as more data is gathered and further study work is completed. Further work is required to confirm the input assumptions supporting the mine design and schedule, processing and economic assessment for the block cave. The cave footprint from this PEA assessment is large enough and has a suitable geometry compared to other world class block caves to support the selected 20Mt/a production rate based on industry benchmarks. Should further data gathering and analysis during the next study phase confirm these assumptions, it will still require a world class operator to fully realise the potential of this deposit. The PEA assumptions could be considered optimistic for a first-time block cave operator, and especially the block cave production ramp-up schedule. The block cave conceptual schedule and assessment were chosen during the assessment as a reflection of the preliminary potential of the deposit rather than current capability of the operator but it is noted that the schedule ramp-up requires additional design and schedule detail to refine in future studies. Processing and surface infrastructure would have been in operation for multiple years through the various open pit operations and is not expected to change significantly through the introduction of a block cave.

17 Recovery Methods

This section has been taken directly from the technical report titled, "Costa Fuego Copper Project: NI 43-101 Technical Report & Preliminary Economic Assessment (PEA)" dated August 14, 2023 with an effective date of June 28, 2023. The Mineral Resource estimate update, as described in this Report, does not materially affect the mineral resource inventory that formed the basis of the 2023 PEA and no new scientific or technical information has been developed that would materially affect the outcome of the 2023 PEA and, therefore. the results and conclusions of the 2023 PEA are considered current and have been restated in this Report.

17.1 Introduction

The process facilities are designed to process sulphide and oxide material from all deposits in the Project.

The Project would produce saleable flotation concentrates of copper and molybdenum and would also produce copper cathode from the heap leach (both oxide and sulphide). The Project has potential to generate other products, but costs and revenues associated with these products are not considered in the PEA.

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

The Project has a processing ramp-up time of one year for both the concentrator and oxide heap leach with annual throughput for the sulphide flotation circuit averaging 21.5 Mt/a for the Project life (Figure 17.1).

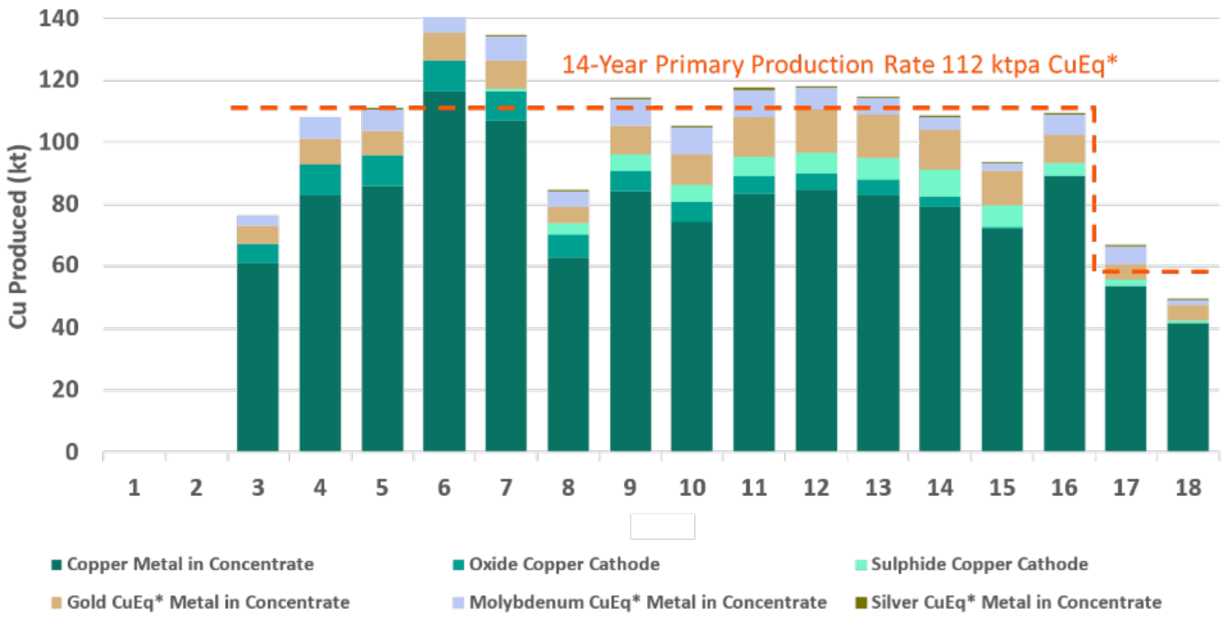
Table 17.1 : Average Processing Variable Throughput Rates by Deposit

Deposit	Concentrator (Mt/a)	# Samples	% of Production Feed
Productora	22.3	27	46%
Alice	23.2	3	2%
Cortadera Open Pit	24.2	4	12%
Cortadera Block Cave	19.4	22	37%
San Antonio	19.4	1	3%
Average	**21.5**		

Sulphide material below variable mill cut-off grade would be stockpiled to be processed via a low-grade sulphide dump leach. The low-grade sulphide leach option replaces tapering oxide production and helps maintain consistent copper metal production through the SX-EW plant to the end of mine-life.

Annual metal production across the three processing streams averages 95 kt Cu, 49 koz Au, 121 koz Ag and 3,294 klb Mo for the primary production period of the first 14 years. LOM annual metal production across the 16-year mine life, including ramp down and residual leaching, averages 88 kt Cu, 45 koz Au, 121 koz Ag and 2,999 klb Mo. Figure 1.16 shows a breakdown of the yearly copper equivalent production over the life of the Project.

Figure 17.1 : Yearly Copper-Equivalent* Production Over Life-Of-Mine



17.2 Overall Project Layout

The majority of the process infrastructure for the Project would be located near the Productora deposit, this includes:

The sulphide concentrator, oxide Run of Mine (ROM) pad, oxide crusher, heap leach area and Solvent Extraction/Electrowinning (SX/EW) plant would be sited in the area to the west of both the Alice and Productora pits. The sulphide ROM pad and primary crusher would be located adjacent to the main haul road close to the Alice and Productora pits. The Low Grade Heap Leach and Dump Leach pads would be located near the Cortadera pits.

At Cortadera, there would be a ROM pad and a primary crusher directly loading on to a sacrificial conveyor which feeds the overland rope conveyor to Productora. The Cortadera facility would be able to accept feed from any of the deposits in the vicinity.

At the Productora end of the overland rope conveyor, there would be a switching system which allows sulphide and oxide production feed to be sent to their respective locations. It is planned that oxide and sulphide production feed would be delivered in distinct campaigns to minimise the need to switch between destinations.

17.3 Copper Sulphide Concentrator

17.3.1 General

The process design philosophy is to provide a concentrator that is “fit for purpose” and is able to process the design tonnage of production feed annually, regardless of feed sources.

The climate at the Project allows for the use of an "open air" design. In the concentrator area, the reagent storage and product filtration areas would be the only enclosed buildings. The design incorporates a single processing line, including comminution, flotation, thickening of concentrate and tailings, filtration of concentrates and truck load-out facilities.

The process plant design utilises conventional and proven processing methods and equipment. The sizing and duty rating of equipment are in accordance with equipment supplier standards for plants of this size and nature. Process plant operations would be monitored and controlled by a Plant Control System (PCS) comprising of a supervisory control and data acquisition (SCADA) system over PCS architecture. Central control facilities would be located adjacent to the flotation area. The PCS is accessed by a data management system, which provides a high level of production reporting.

Sampling and process performance measurement is achieved by automatic sample collection systems and chemical assay of shift composite samples. Manual sampling at the concentrate load-out facility would complete the metallurgical accounting system. For real-time plant performance monitoring, automatic samplers feed a multi-stream chemical analyser and a particle size analyser.

17.3.2 Copper Sulphide 2016 Study Comminution Circuit Modelling

Comminution circuit modelling for the 2016 Study stage of the Project was undertaken by DMCC Pty Ltd (DMCC) who modelled three options, namely:

- Primary crushing followed by Semi-autogenous Grind (SAG) and ball milling with recycle crushing – Primary SABC (Option 1)
- Three stage conventional crushing followed by Ball Mill (BM) circuit – 3CBM (Option 2)
- Two stage crushing, High Pressure Grinding Rolls (HPGR) and ball mill circuit – 2C/HPGR/BM (Option 3).

A simple power consumption comparison of the options at two grind sizes is shown in Table 17.2. The PEA power was simply estimated using the ratio of throughput rates (20 Mt/14 Mt).

Table 17.2 : DMCC Comminution Modelling Comparison of Flowsheets

Option	**1a**	**2a**	**3a**	**1b**	**2b**	**3b**
Configuration	**SABC**	**3CBM**	**2C/HPGR/BM**	**SABC**	**3CBM**	**2C/HPGR/BM**
Grind P80 (µm)	106	106	106	125	125	125
Total Specific Energy (kWh/t)	22.0	20.3	18.3	20.4	18.6	16.8
PFS Power for 14 Mt/a (MW)	38.5	35.5	32.0	35.7	32.6	29.4
PEA Power for 20 Mt/a (MW)	55.0	50.8	45.8	51.0	46.5	42.0

Although the SABC circuit has the highest power consumption, its lower capital cost resulted in it being chosen for the 2016 Study. These values have only been presented for comparative purposes as the 2016 Study was completed before the discovery of Cortadera.

Wood conducted the Cortadera modelling and arrived at new PEA power requirements using Morrell power calculations, as did DMCC.

17.3.3 Comminution Circuit Modelling for PEA

The comminution inputs are compared in Table 17.3 below:

Table 17.3 : Comminution Properties for Costa Fuego Deposits

	Productora	Alice	Cortadera OP	Cortadera UG	DMCC Design
Samples	36	3	23	22	-
Percentile	80th	Max	80th	80th	Average
DWI	8.28	7.50	8.14	9.46	7.51
Mia	23.2	21.6	22.3	25.2	21.4
Mic	9.22	8.40	8.80	10.3	8.38
Mih	17.8	16.3	17.0	19.9	16.3
BWI	21.9	18.9	14.6	18.1	18.3
Ai	0.44	0.33	0.17	0.22	0.37
SG	2.68	2.69	2.67	2.79	2.70

Somewhat unexpectedly, the DMCC design parameters are almost identical to the maximum properties of the three tested Alice samples. Note that the DMCC design values would be exceeded because they do not include Cortadera and also because they are average values and not 80th percentile (or Max) values.

The power estimation outcomes using the Morrell method are included in Section 13 and reproduced in Table 17.4 with the equivalent information from the DMCC modelling. The maximum specific energies for each of the process steps are highlighted.

Table 17.4 : Preliminary Design (80th Percentile) Specific Energy by Equipment for Costa Fuego Deposits

Specific Energy (kWh/t)	Productora	Alice	Cortadera OP	Cortadera UG	DMCC 106 µm Design
Primary Crusher	0.10	0.09	0.12	0.10	0.10
SAG Mill	8.74	8.44	9.41	10.5	9.57
Pebble Crusher	0.24	0.22	0.23	0.27	0.21
Ball Mill	13.4	12.4	9.1	11.8	12.1
Total	**22.5**	**21.2**	**18.8**	**22.6**	**22.0**

The sum of the green highlighted specific energy values is 24.3 kWh/t. This is an increase of a little more than 10% above the DMCC design.

The specific energy values are used to select the major comminution equipment.

17.3.4 Copper Concentrator Design Basis

The comminution design criteria are summarised in Table 17.5.

Table 17.5 : Comminution Circuit Process Design Criteria			
Criteria	**Units**	**Value**	**Source**
Annual capacity	Mt/a (dry basis)	20.0	HCH
Crushing utilisation	%	70	Wood
Crushing circuit operating hours	h/a	6 132	Calculated
Crushing circuit throughput	Dry t/h	3 262	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	2 500	Calculated
Primary Grinding circuit product P_{80}	µm	125	Testwork

The specific energy requirements (power at pinion) in Table 17.4 are combined with the throughput requirements in Table 17.5 to provide the design pinion power requirements for each equipment item. The design power requirements are then escalated using factors to arrive at the likely maximum drawn power and then the installed motor power. All these values are summarised in Table 17.6.

Table 17.6: Major Equipment Power Design (Preliminary)							
	Design Specific Energy	**Pinion Power**	**Operating Headroom**	**Max Pinion Power**	**Drive Efficiency**	**Max Motor Output**	**Installed Motor**
Units	**kWh/t**	**MW**	**%**	**MW**	**%**	**MW**	**MW**
Primary Crusher	0.12	0.30	75	0.53	93	0.56	0.60
SAG Mill	10.5	26.3	10	29.7	98	30.0	30.0
Pebble Crusher	0.27	0.68	25	0.84	95	0.89	0.95
Ball Mill	13.4	33.5	5	35.2	95	37.0	38.0
Total	24.3	60.7		66.2		68.5	69.6

This method has delivered equipment with operational precedent for every item except the SAG mill. No 30 MW SAG mill has yet been constructed and the largest precedent is 28 MW. The SAG mill size and power is a direct result of the Cortadera Underground deposit power requirement of 10.5 kWh/t. The high power is partially a result of assigning a coarse SAG feed P_{80} to the UG deposit (145 mm) because the block cave mining method is known to generate less fines than open pit blasting. The soft and altered OP material has been assigned a mill F80 of 125 mm and both Productora and Alice were assigned 135 mm.

A 28 MW mill at full power supplies 9.8 kWh/t at the pinion with the assumptions used. Apart from Cortadera UG, all other deposits require less than 9.8 kWh/t at pinion. To reduce the underground production feed SAG power requirements, the feed has been made finer by the use of more primary crushing power. A finer SAG feed always leads to a coarser SAG product because there is a reduction in abrasive breakage and an increase in impact breakage. Accordingly, the UG feed transfer size used in the calculation has been increased from 1 500 µm to 2 000 µm.

The final specific energy table for plant design for the deposits is given in Table 17.7.

Table 17.7 : Final Design (80th Percentile) Specific Energy by Equipment for Costa Fuego Deposits					
Specific Energy (kWh/t)	**Productora**	**Alice**	**Cortadera OP**	**Cortadera UG**	**DMCC 106 µm Design**
Primary Crusher	0.10	0.09	0.12	0.18	0.10
SAG Mill	8.74	8.44	9.41	9.22	9.57
Pebble Crusher	0.24	0.22	0.23	0.27	0.21
Ball Mill	13.4	12.4	9.1	12.7	12.1
Total	**22.5**	**21.2**	**18.8**	**22.3**	**22.0**

The SAG size is now determined by the Cortadera OP power requirements, rather than UG. There are also now two design-determining primary crusher requirements. There would be at least four primary crushers for the Project and at least three of these would be used for sulphide production feed:

A surface primary crusher at Productora for Productora and Alice production feed

- A surface primary crusher at Productora for oxide production feed
- A surface primary crusher at Cortadera for the oxide and sulphide open pit production feed
- At least one underground primary crusher at Cortadera for underground production feed.

In reality, there should be three design-determining primary crusher requirements in Table 17.7, but the same size of crusher would be selected for use at both Productora sulphides and Cortadera surface production feed.

The final comminution power design is shown in Table 17.8.

Table 17.8: Major Equipment Final Power Design							
	Design Specific Energy	**Pinion Power**	**Operating Headroom**	**Max Pinion Power**	**Drive Efficiency**	**Max Motor Output**	**Installed Motor**
Units	**kWh/t**	**MW**	**%**	**MW**	**%**	**MW**	**MW**
Primary Crusher	0.18	0.44	75	0.77	93	0.83	1.0
SAG Mill	9.41	23.5	12	26.3	98	26.9	28.0
Pebble Crusher	0.27	0.68	25	0.85	95	0.89	0.95
Ball Mill	13.4	33.5	5	35.2	95	37.0	38.0
Total	**23.3**	**58.1**		**63.1**		**65.6**	**68.0**

Based on the power requirements, the following major equipment is recommended for processing the sulphide deposits.

Table 17.9: Major Comminution Circuit Equipment Selections

	Units	Model	Installed Power (MW)	Set (mm)	Diameter (ft)	Length (ft)	Diameter (m)	Length (m)
Productora Sulphide Primary Crusher	1	Superior 62-75 MKIII	0.6	177				
Cortadera Surface Primary Crusher	1	Superior 62-75 MKIII	0.6	177				
Cortadera Underground Primary Crusher	2	Superior 54-75 MKIII-UG	0.6	152				
SAG Mill	1	28 MW GMD	28		40	28	12.2	8.53
Pebble Crusher	1	M:O MP 1250	0.94	13				
Ball Mills	2	19 MW Twin Pinion	38		28	40	8.53	12.2

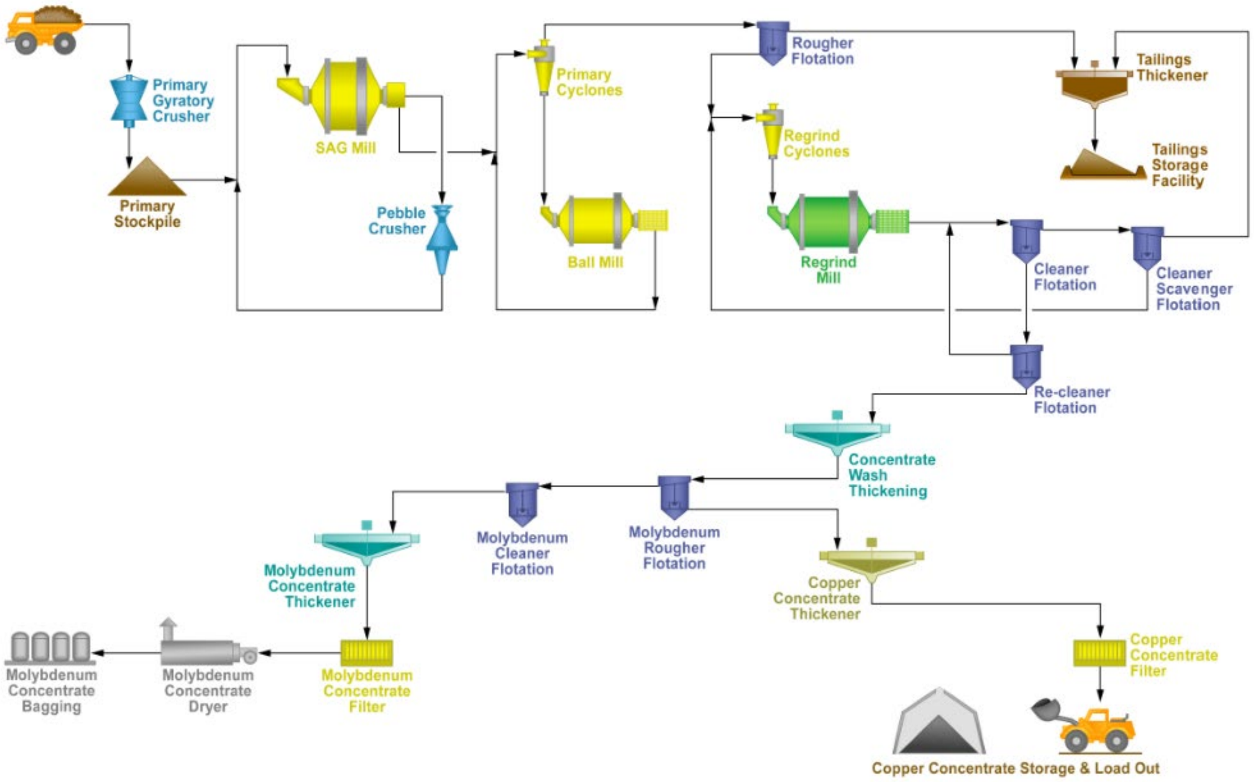
17.3.5 Copper Sulphide Circuit Design Criteria and Process Flow Diagrams

The copper sulphide process plant is designed to process open pit and underground mined sulphide production feed. The principal criteria dictating the sulphide concentrator design are listed below.

- Annual capacity (nominal): 20 Mt/a
- Concentrator throughput rate: 2500 t/h
- Design production feed Cu head grade: 0.60%
- Cu recovery: 90%
- Final concentrate Cu grade: 25%.

A schematic of the sulphide production plant is shown in Figure 17.2.

Figure 17.2 : Sulphide Process Plant Flowsheet



Note that the primary stockpile would also receive feed from Cortadera via overland conveyor.

17.3.6 Copper Sulphide Process Plant Description

17.3.6.1 Primary Crusher Productora

Run of mine (ROM) process feed from Productora pit and Alice pit would be delivered to the Productora ROM pad. The ROM pad and primary crushing station would be remotely located, approximately 1.3 km to the southeast of the sulphide process plant. The ROM bin would have a minimum capacity of 2.5 truckloads to allow trucks to direct dump into double-sided tip points to the crusher. Plant design assumes 90% direct feed from trucks to the crusher. Production feed not dumped directly would be placed on the ROM pad, which is located adjacent to the crushing plant ROM feed bin. Production feed from the ROM pad stockpile would be delivered to the ROM bin by loader.

A rock breaker mounted adjacent to the primary crusher would be used to clear blockages in the crushing chamber. The ROM bin would discharge directly into the primary gyratory crusher which would, in turn, discharge into a surge chamber under the crusher. An apron feeder would control the draw down rate on the surge chamber onto a 1.1 km long primary crushed production feed transfer conveyor delivering to the primary stockpile feed conveyor.

17.3.6.2 Primary Crusher Cortadera Open Pit

ROM production feed from the Cortadera pit would be transported by truck to the Cortadera ROM area. Oxide production feed would be stockpiled into an area nearby, but not within loader tramming distance of the ROM bin. Sulphide production feed would be direct dumped to the ROM bin. During normal operation, sulphide production feed would be the main feed to the crusher. Campaigns of oxide production feed would be run periodically using loaders or excavators filling trucks at the oxide production feed stockpile.

The Cortadera primary crusher would be identical in all aspects to the Productora primary crusher.

The surge chamber apron feeder under the crusher would control the feed onto a short sacrificial loading conveyor. The loading conveyor feeds the first section of the rope conveyor which takes the production feed to Productora. Due to the terrain the rope conveyor would be in three sections. The third rope conveyor section would deliver onto a conveyor which would either deliver to a second stockpile feed conveyor or would reposition its head drum to deliver oxide production feed to conveyors that feed oxide production feed stacking.

17.3.6.3 Underground Crusher(s) Cortadera

Underground crushers would be part of the block cave design and feed arrangements would be described elsewhere. An apron feeder under the crusher discharge surge chamber would control feed onto the first incline conveyor that brings production feed to the surface. The last incline conveyor delivers to the surface and transfers onto a surface distribution conveyor. This either feeds the sacrificial conveyor on its path to the rope conveyor or it feeds an emergency stockpile conveyor for underground production feed. The emergency stockpile capacity would be three times the maximum production feed mass from the underground crusher feed chamber to the transfer point onto the sacrificial loading conveyor.

Material from the emergency stockpile would be reclaimed with loader and trucked to the surface ROM bin.

17.3.6.4 Primary Production Feed Stockpile

The primary stockpile conveyors would deliver crushed production feed to the primary production feed stockpile, which would have a live capacity of 28 000 t (16 hours) and a total stockpile capacity of 112 000 t. The design would have an additional hour of emergency capacity built into the stockpile conveyor discharge heights in the event it is necessary to empty the overland rope conveyor onto an already full stockpile.

Three stockpile reclaim vibrating feeders, mounted underneath the stockpile, would control the flow of the material from the stockpile onto the SAG mill feed conveyor.

Two (2 off) emergency vibrating feeders, installed at the opposite outer ends of the crushed production feed stockpile, would be used to keep feeding the SAG mill when the coarse production feed stockpile level is low or when main feeders are not available for maintenance.

17.3.6.5 Primary Grinding

The SAG mill feed conveyor would discharge into the SAG feed chute where water is also added. The SAG mill feed conveyor would be fitted with a weightometer to monitor and control the feed rate to the SAG mill. The mill would be driven by a gearless motor (inherently Variable Speed Drive (VSD)) and would be highly instrumented for reliability and controllability.

A grate inside the SAG mill would allow pebbles as coarse as 75 mm to discharge. The pebble slurry mix is brought to the centre discharge trunnion by pulp lifters.

The SAG mill trunnion would discharge onto a single large vibrating screen which would have a ready spare always available on a roll out roll in system. Screen oversize pebbles would be loaded onto the pebble crusher feed conveyor and the screen undersize slurry would mix with ball mill discharge and feed the ball mill cyclones.

17.3.6.6 Pebble Crushing

Pebbles from the SAG mill discharge screen would be crushed and then return to the SAG mill via the SAG mill feed conveyor. The pebble feed conveyor would be installed with an overhead magnet to remove grinding balls, a weightometer to monitor the pebble production rate and a metal detector to further protect the pebble crusher.

17.3.6.7 Primary Classification

SAG discharge screen undersize slurry would be pumped to a distributor where half would flow to each ball mill discharge hopper. The hoppers would each feed a cyclone cluster, one for each of the ball mills.

The cyclone underflow slurry streams would be returned to the two ball mills operating in parallel for further size reduction. The cyclone overflow slurry would flow to trash screens to remove coarse particles and trash. The trash screens would operate on a basis of two-duty, one standby. Cyclones would be operated to target a grind P_{80} of 125 μm.

17.3.6.8 Secondary Grinding

Two twin pinion ball mills would operate in parallel and would operate in closed circuit with their respective cyclone clusters. The mills would be configured in overflow mode with trommel screens fitted to capture ball scats and possible rock scats. It is recommended that the mills be fitted with variable speed drives to minimise startup requirements on the supply grid. It would also provide another variable, mill speed, for controlling the grind P_{80}.

17.3.6.9 Grinding Media Charging

Automatic ball feeding would be provided to all mills to maintain optimal charge levels at all times. For the SAG mill, the balls would be fed onto the production feed stream by a ball feeder. For the ball mills, a bucket elevator would lift balls to a distribution conveyor which feeds both mills. The distribution conveyor would empty into a ball charging bin and ball would be metered out of the bin to maintain ball mill power levels.

Crane-based fast loading facilities would also be prepared for use during first fill and in the rare event a mill needs to be emptied and then refilled after major maintenance. These facilities do not need to be permanently installed and can be stored elsewhere on site.

17.3.6.10 Copper Rougher Flotation Circuit

Trash screen undersize would report to the copper rougher flotation conditioning tank. The slurry would be conditioned with collector and frother, then would feed a bank of rougher/scavenger flotation cells, operating in series. Roughers would comprise the first two cells of the bank and scavengers the remaining cells. Rougher and scavenger concentrates would be collected separately.

Rougher concentrate would either be sent to regrind feed or direct to the copper cleaner without grinding. This would be at the operator's discretion and would be guided by continuous concentrate grade monitoring.

The scavenger concentrates would always flow to the densifying cyclone feed hopper. Either rougher concentrate or rougher cleaner tails would report to the same cyclone feed hopper, depending on the operator's plant settings. Scavenger flotation tails would report to the tailings thickener feed hopper for transfer to the tailings thickener.

17.3.6.11 Regrind Mill

The regrind IsaMill is fed at about 50% solids (densifying cyclone underflow) and the discharge of the IsaMill is already internally classified to the target P_{80} (nominally 25 µm). Mill discharge slurry would be combined with the densifying cyclone overflow (also nominally 25 µm P_{80}) before pumping to the cleaner flotation circuit.

17.3.6.12 Cleaner, Cleaner Scavenger and Recleaner Flotation Circuit

The rougher cleaner would be a Jameson cell with froth washing producing a final grade concentrate in excess of 25% Cu. If the cell is unable to make 25% Cu in concentrate then the rougher concentrate will be diverted to densifying cyclone feed. The Jameson cell rougher cleaner tails reports to densifying cyclone feed, but an alternative tails destination, cleaner feed, can be selected (or its selection can be automated when rougher concentrate is being reground).

When the Jameson rougher cleaner cell is not being used for direct cleaning of rougher concentrate, it is available to reduce the load on the remainder of the cleaner circuit. Rather than sending a much higher copper loading to the copper cleaner bank, the regrind circuit product is fed to the Jameson cell and produces a copper cleaner concentrate with grade in excess of 25% Cu. This means that the concentrate from this cell always reports to final concentrate. The Jameson cleaner tailings then feeds the remainder of the cleaner circuit.

Reground copper scavenger flotation concentrate, sometimes with copper cleaner (Jameson) tailings, will feed the low grade copper cleaners. The low grade cleaners would generate a product that always requires further cleaning. The low grade cleaner tails report to the copper cleaner scavenger. The low grade cleaner concentrate reports to the Jameson recleaner cell.

Jameson recleaner concentrate is either greater than 25% Cu or it is of a grade that adds to the Jameson cleaner tailings to make a combined 25% Cu concentrate. Jameson recleaner tailings is sent back to the low grade cleaner feed.

The copper cleaner scavenger concentrate also reports to low grade cleaner feed. The copper cleaner scavenger tails joins the scavenger tails and is sent to tails thickening. An alternative manual destination for the cleaner scavenger concentrate is the densifying cyclone feed hopper as this provides potential for some additional regrinding.

17.3.6.13 Copper/Molybdenum Concentrate Thickening

The final copper circuit concentrate also contains the molybdenum. The molybdenum flotation circuit would be described separately, later in this section. However, molybdenum flotation must be carried out in fresh/RO water and the first step is to remove most of the seawater from the concentrate via the copper/molybdenum thickener. The thickener underflow, at between 65 and 70% solids, is repulped in fresh water and then fed to the molybdenum conditioner. After molybdenum rougher flotation the final copper concentrate is sent to the copper thickener.

17.3.6.14 Copper Concentrate Thickening

The final copper concentrates returning from the molybdenum flotation circuit would be thickened in a 14 m diameter high-rate thickener. The thickener overflow water is returned to a low saline process water tank for reuse in the molybdenum flotation plant.

The thickener underflow would be pumped to a concentrate filtration circuit to produce a filter cake with sufficient moisture content suitable for road transport via trucks and sea.

17.3.6.15 Copper Concentrate Filtering

The copper concentrate thickener underflow would be pumped to an agitated filter feed tank for storage and feeding to a concentrate dewatering filter. Slurry from the concentrate filter feed tank would be fed to a horizontal pressure filter to produce a filter cake with a target moisture content of 8% by weight.

Filter cake from the filter press would be discharged under gravity directly onto a concrete bunker which would then be transferred to stockpiles by front-end loader (FEL) within the concentrate storage shed before being loaded on to truck by FEL.

17.3.6.16 Tailings Disposal

Cleaner scavenger tailings and rougher tailings would be pumped to a high-rate thickener. The tailings thickener underflow, at about 70% pulp density, would be pumped to a tailings storage facility (TSF) via a two-stage pumping station. The second stage of the pumping station would be located at the toe of the TSF embankment, as it may 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 full 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 would be located near the tailing thickener. Both stations of the tailings pumping system are comprised of two duty and one standby pumps. The tailings thickener overflow water reports via a gravity pipeline to the process water tank for reuse in the concentrator.

Decant return water from the TSF would be pumped to the process water tank using a mobile decant pontoon pump and a fixed diesel (or solar) powered decant return water pump station. The two pumps would be operated on a standby – duty fashion.

17.3.7 Copper Sulphide Concentrate Handling

The filtered copper concentrate stacked on a concrete bunker would be 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 would be provided to store the filtered cake for up to 20 days on site.

17.3.8 Copper Sulphide Flotation Reagent Handling and Storage

The reagents that would be used in the copper sulphide process plant include:

- Frother
- Collector
- Diesel (supplemental collector for molybdenum flotation)
- Depressant
- Flocculant.

Reagent mixing would be done 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.

17.3.9 Copper Sulphide Utilities

17.3.9.1 Introduction

The utilities would include equipment and facilities to store and distribute the following water services:

- Raw water
- Fire water
- Filtered water
- Potable water (including safety shower/eye wash)
- Gland water
- Cooling water
- Process water
- Tailings decant water
- Underground mine dewatering water (treated)
- Air systems
- Diesel and lubricants.

17.3.9.2 Raw Water

Seawater would be used as raw water in the processing plants and at the mine. The seawater would be delivered via the infrastructure described in Section 18.0 to the seawater pond at the heap leach SX/EW plant and to the seawater pond located adjacent to the sulphide treatment plant

Two raw water pumps, operating in duty and standby mode, would supply raw water to the process plant and the fire hydrants around the plant.

A separate diesel pump would be used for the fire water supply circuit.

17.3.9.3 Process Water

Process water recovered from the thickeners would be re-used at the sulphide plant. Process water would be sourced from the following sources:

- Tailings dam decant return water pond
- Tailings thickener overflow
- Copper/molybdenum concentrate thickener overflow
- Brine reject from the seawater RO plant
- Raw water from site raw water pond
- Saline bleed from the molybdenum circuit.

Process water pumps, operating in duty and standby mode, would be used to supply process water to the sulphide plant. The make-up water for the process water tank would be from the raw water pond.

Within the process water circuit, low saline process water would be used within the molybdenum flotation circuit and stored separately in a low saline process water tank. Fresh water would be introduced to the molybdenum circuit from the seawater RO plant.

Tailings dam decant pond water would be returned to the process water tank via a decant return pipeline alongside the decant return pipeline track.

17.3.9.4 Low Saline Process Water

A water treatment plant, based on reverse osmosis (RO) technology, would treat seawater to provide low saline water for use in the flotation process (concentrate launder sprays, etc.) and in the molybdenum flotation circuit. The water treatment plant would be a vendor supplied package. The RO plant would also supply a small quantity of low saline water for further treatment to produce potable quality water for use at the plant.

17.3.9.5 Potable Water

Seawater would be treated through a RO water treatment plant located at the sulphide process plant. The majority of product from this RO plant would be directed to the sulphide flotation plant for concentrate washing. A minor stream from the RO plant would undergo chlorination and UV sterilisation to produce potable water suitable for use in applicable sections (e.g. buildings and support facilities, drinking water, emergency safety showers and eye wash stations) of the sulphide plant, oxide plant and in the mining contractor's area. A potable water storage tank would be provided at each of these locations with associated potable water distribution pumps.

17.3.9.6 Compressed Air

Compressors and blowers would provide air to the systems within the Process Plant:

- Plant air
- Instrument air.

17.4 Molybdenum Circuit

17.4.1 Introduction

Molybdenum would be floated selectively from the copper/molybdenum concentrate leaving the final copper concentrate essentially molybdenum free. A saleable molybdenum concentrate would be produced.

17.4.2 Molybdenum Circuit Design Criteria and Process Flow Diagrams

The molybdenum recovery circuit is part of the sulphide process plant which is designed to process all sulphide production feed as depicted in Figure 17.2 The principal criteria dictating the molybdenum plant design are listed below:

- Fresh production feed head grade (design) nominally 200 ppm Mo
- Molybdenum plant feed grade nominally 4600 ppm Mo and 25% Cu in the copper recleaner concentrate
- Molybdenum recovery from plant feed to copper concentrate, 70% (average value, range is from 25% to 95%)
- Molybdenum recovery from copper recleaner concentrate: 90%
- Final molybdenum concentrate grade, 50% Mo.

17.4.3 Molybdenum Recovery Circuit Plant Description

17.4.3.1 Concentrate Wash

The seawater used in the copper flotation circuit (>19 000 g/L chloride) must be displaced from the copper/molybdenum concentrate and replaced with fresh water so that molybdenum flotation can be successful. The molybdenum rougher flotation slurry water must have less than 500 ppm chloride and the detrimental effect of seawater on molybdenum flotation is noted in Section 13.8.4. The washing stage should be designed such that it would leave the copper concentrate with a chloride level of <500 ppm.

To commence the chloride removal process, the Jameson cell froth wash water and launder water would be demineralised water with <200 g/L chloride. By using the correct amount of froth wash water it is possible to replace most of the previous water that formed the flotation bubbles without losing the bubbles or the attached concentrates. This means that the final copper/molybdenum concentrate would already have a chloride level well below seawater. It is then possible to further dilute the copper/molybdenum concentrate ahead of thickening with demineralised water, so that its chloride level is low enough that a single thickening stage would achieve <500 g/L in molybdenum rougher flotation feed. The amount of dilution water required would be controlled by monitoring the conductivity of the thickener overflow water. Achieving a conductivity that corresponds to something equal to or less than 2000 g/L chloride would be the target. Once the copper/molybdenum thickener underflow is diluted to the molybdenum rougher flotation feed density the target of <500 g/L chloride would be safely achieved.

17.4.3.2 Molybdenum Rougher Flotation

In the molybdenum flotation conditioning tank, the slurry is conditioned for five minutes with copper depressant (Sodium Metabisulphite, SMBS). The conditioning tank discharge flows to the molybdenum rougher flotation circuit.

The molybdenum rougher flotation circuit would consist of conventional flotation cells operating in series. The rougher concentrate would be collected in a hopper for pumping to the head of the molybdenum cleaning circuit. The tails from the molybdenum rougher flotation circuit would flow, by gravity, to molybdenum scavenger flotation feed.

While dilution up to this point in the circuit has used a mix of freshly demineralised water with low saline process water, all water in the remainder of the molybdenum circuit would be sourced from the low saline process water tank. The low saline water would be monitored for conductivity to ensure it is always suitable for molybdenum flotation.

17.4.3.3 Molybdenum Scavenger Flotation

The molybdenum scavenger flotation circuit would consist of conventional cells operating in series. The scavenger concentrates would be pumped to the head of the molybdenum rougher flotation circuit. The scavenger tails are the final copper concentrate for the Project and would be pumped to the copper concentrate thickener ahead of filtration.

17.4.3.4 Molybdenum Cleaner Flotation

Concentrates from the molybdenum rougher circuit would be upgraded in a six-stage cleaning circuit as detailed below:

- Molybdenum rougher concentrate would be pumped to the molybdenum rougher concentrate thickener. The thickener underflow would be delivered to a concentrate surge tank that can hold up to eight hours of rougher concentrate. The surge tank volume would be utilised to stabilise the feed to the molybdenum cleaning circuit in the face of changing head grades for both molybdenum and copper.
- Pumping from the bottom of the surge tank, the molybdenum rougher concentrate would be pumped to the cleaner 1 conditioner. The high-density conditioner product would be diluted with demineralised water and then flow to the molybdenum cleaner 1 flotation cells. Stage 1a of the molybdenum cleaning flotation circuit would consist of two conventional cells operating in series. Stage 1b would consist of one flotation cell and would act as a cleaner scavenger.

- The tails from Stage 1b flotation cell would report to the head of the molybdenum scavenger flotation circuit. The Stage 1a concentrate would be pumped to Stage 2 cleaner feed. Stage 1b concentrate would be pumped to Stage 1a bank feed.
- Concentrates from the molybdenum cleaner flotation 2 circuit would be pumped to the molybdenum cleaner flotation 3 for upgrading. Tails from the molybdenum cleaner flotation 2 circuit would be pumped to the molybdenum rougher concentrate surge tank.
- Stage 3 of molybdenum cleaning flotation would consist of one flotation column. The concentrate from the molybdenum cleaner 3 flotation column would be pumped to molybdenum cleaner flotation stage 4 for further upgrading. The tails from the molybdenum cleaner flotation column 3 would be gravity fed to the head of molybdenum cleaner 2 circuit.
- Stage 4 of molybdenum cleaning flotation would consist of one flotation column. The concentrate from the molybdenum cleaner flotation column 4 would be pumped to molybdenum cleaner flotation Stage 5 for upgrading. The tails would be gravity fed to the head of molybdenum cleaner flotation circuit 3.
- Stage 5 of molybdenum cleaning flotation would consist of one flotation column. The concentrate from the molybdenum cleaner flotation column 5 would be pumped to a molybdenum cleaner flotation column 6 for upgrading. The Stage 5 column tails would be gravity fed to the head of molybdenum cleaner 4.
- Stage 6 of molybdenum cleaning flotation would consist of one flotation column. The concentrate from molybdenum cleaner flotation column 6 would be pumped to a molybdenum concentrate thickener for dewatering. Molybdenum cleaner flotation column 6 tail product would be gravity fed to the head of molybdenum cleaner column 5.

17.4.3.5 Molybdenum Concentrate Thickening

The final molybdenum concentrate would be thickened in a small (1 to 2 m diameter) high-rate thickener and produce 65% w/w solids in the underflow. The overflow water would report to the low saline process water tank.

The molybdenum concentrate thickener underflow would be pumped to a molybdenum concentrate storage tank.

17.4.3.6 Molybdenum Filtration and Drying Circuits

The molybdenum concentrate storage tank would be designed to hold 24 hours of production capacity. The thickened slurry would be pumped from the storage tank to a molybdenum vertical plate pressure filter for further dewatering. The pressure filter would produce a filter cake at a nominal 8% moisture.

The filter cakes discharged from the pressure filter would drop under gravity directly into a hopper which would then be loaded into a molybdenum concentrate dryer to further reduce its moisture to 4.5%. The dried molybdenum concentrate would be loaded into one tonne bulk bags.

17.4.3.7 Molybdenum Concentrate Handling and Transport

The bags containing dried molybdenum concentrate would be loaded onto concentrate trucks for transport by road to a smelter in Santiago.

17.4.4 Molybdenum Plant Reagent Handling and Storage

The reagents that would be used in the molybdenum sulphide process plant include:

- Collector (kerosene)
- Conditioner Sodium Metabisulphite (SMBS)
- Flocculant.

Reagent mixing would be done 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.

17.4.5 Molybdenum Recovery Circuit Plant Services

17.4.5.1 Low Saline Process Water

Low saline water would be used in all stages of the molybdenum flotation circuit. The low saline water would be stored in a low saline water tank, separate from the process water.

The low saline process water tank would be supplemented with potable quality water from the RO plant, when required to provide additional volume or when it is necessary to lower its conductivity.

17.4.5.2 Compressed Air

Refer to 17.3.9.6 for details.

17.5 Copper Oxide Plant

17.5.1 General

The oxide production feed treatment plant design is based upon equipment that is reliable and easy to maintain, arranged in a layout that has proven be convenient and efficient to operate. To counteract and minimise expected corrosion problems, due to use of low saline water, suitable material of construction was selected for all piping and key process equipment as follows:

- Rubber lined agglomerator
- High Density Polyethylene (HDPE) piping for process water pipes at the crushing plant
- Demineralised water used at the SX/EW plant.

17.5.2 Copper Oxide Process Flowsheet Selection

17.5.2.1 Front End – Crushing, Agglomeration and Heap Leaching

The design allows for the primary crusher to be fed by either a 100 t truck or a FEL. A ROM pad adjacent to the primary crusher feed hopper would allow production feed to be stored, blended and reclaimed as necessary to achieve reliable plant performance. The ROM pad would also serve as a truck marshalling area and emergency dump site in the event of unacceptably long delays for trucks dumping at the crusher.

The production feed would be scalped of fines on a grizzly and the oversize broken in a jaw crusher. The primary crushed production feed would be stockpiled at a crushed production feed stockpile. Cortadera oxide production feed, primary crushed at Cortadera and transported by the overland rope conveyor, would be delivered to this same stockpile. The primary crushed production feed reclaimed from the crushed production feed stockpile would be screened to generate three size fractions. The screen undersize would report directly to an agglomerator. The middle size fraction from the screening plant would be crushed in a tertiary crusher, while top size fraction is crushed in a secondary crusher. Both crushers operate in closed circuit with the screening plant.

An agglomeration drum would prepare the heap feed by tumbling it with acidic liquor. Agglomerator feed is nominally 100% minus 12.5 mm. Sulphuric acid, water and raffinate leach solution would be added to precondition the crushed production feed in the agglomerator. The agglomerated feed, loosely bound balls of production feed held together with the acidic liquor, would feed an overland conveyor and then to a series of grasshopper conveyors and finally the heap stacker. The overland conveyor is a fixed delivery system to the heap leach hub. The grasshopper conveyors are a series of short conveyors that can be moved to suit the current stacker position.

During pad construction, the production feed would be radially stacked to a target height of 6 m. The stacking width is the 80 m pad width and the nominal length of the pads is approximately 600 m. Heap leaching would be conducted on geotechnically engineered lined leach pads, with drainage piping installed and segmented to suit the radial stacking.

The heap pads are designed to operate based on three-stage leaching of the stacked productions feed. Seawater would be used for primary leaching of the production feed. The heap pads would drain to a “w” drain system that allows distribution of the leach solution between the intermediate liquor solution (ILS) collection pond and the pregnant liquor solution (PLS) collection pond. A pond for the raffinate would also be installed. These double HDPE lined ponds would each have a dedicated pumping system and sulphuric acid dosing system to prepare and deliver the leach solution to the leach pads as required. Dripper irrigation networks would be installed on the surface of the heaps to evenly distribute leach solution.

17.5.2.2 Back End – Solvent Extraction and Electrowinning (SX/EW)

Copper laden pregnant solution would be treated in a conventional SX/EW plant, located near the heap leach pads. The SX/EW plant would have a nominal cathode copper production capacity of 10 000 t/a.

17.5.3 Copper Oxide Design Criteria and Process Flow Diagrams

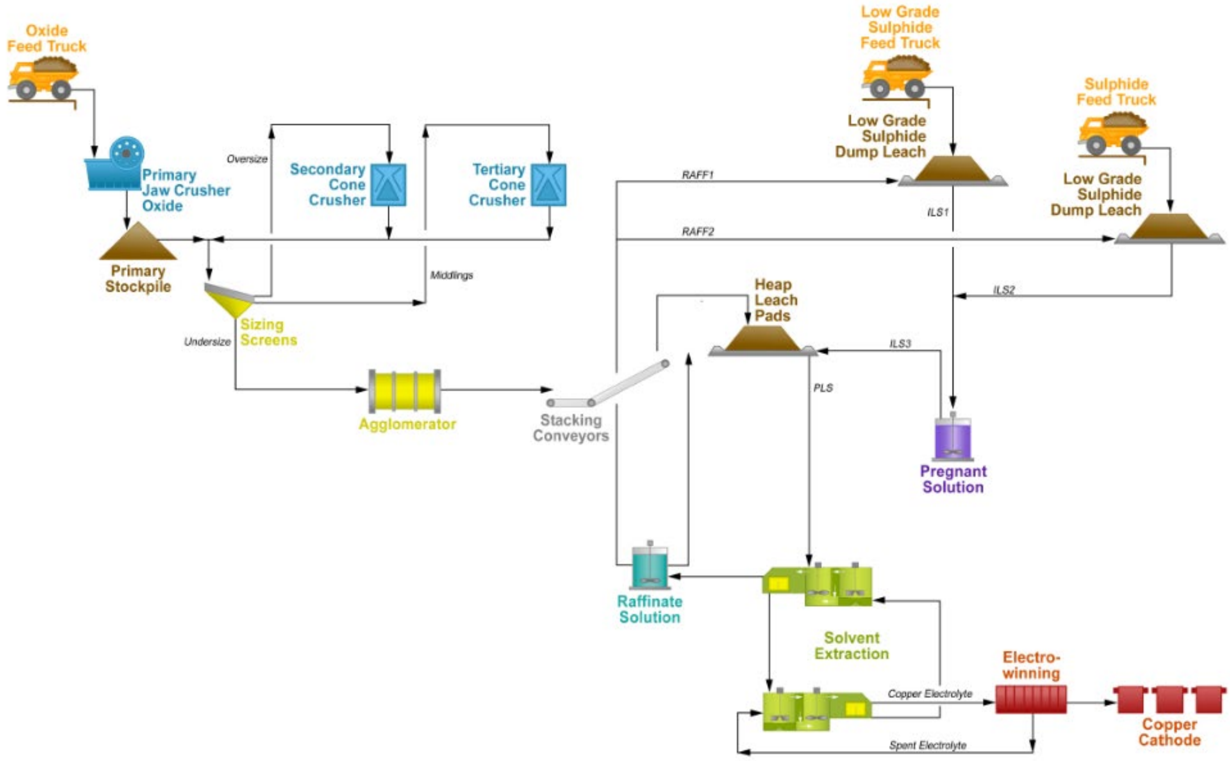
The copper oxide process plant (crushing, stacking, heap and SX/EW) is designed to process open pit mined oxide production feed. The principal criteria dictating the oxide heap leach design and SX/EW design are listed below:

- Annual oxide heap leach capacity (nominal) 3.3 Mt/a
- Crushing plant throughput rate 800 t/h, at 75% utilisation
- Production feed head grade (design) 0.60% Cu

- Metal recovery from heap leaching 5 6% of Cu
- Heap leach cycle (average) 300 days
- Acid consumption 5 to 20 kg/t depending on production feed characteristics and geometallurgical domains
- SX/EW plant would have a capacity of treating 705 m^3/h of PLS at 95% utilisation
- Three-stage leaching on the pads with expected ILS flow rate of 1410 m^3/h
- Capacity of SX/EW process plant – 10 k t/a Cu cathode (Increasing to 12 k t/a in year 8).

Topography and geotechnical ground conditions were considered during site selection and design. A schematic of the oxide process plant is shown in Figure 17.3. Note that the primary stockpile stacking conveyor for production feed from Cortadera is not shown.

Figure 17.3 : Copper Oxide Process Plant Flowsheet



17.5.4 Copper Oxide Process and Plant Description

17.5.4.1 Introduction

The SX/EW process plant would be situated to the east of the main access road to site. The crushing and agglomeration plant would be located next to the proposed heap leach pad location.

17.5.4.2 Oxide Crushing Plant

Oxide production feed from the proposed Productora open pit mine would be delivered to the ROM pad in 100 t capacity rear dump haul trucks. The target maximum particle size of production feed on the ROM pad would be 1000 mm. Production feed from the mine would be stockpiled in separate stockpiles of varying production feed types and grades to facilitate blending of the feed into the crushing plant. The ROM production feed from the stockpiles would be fed to the ROM bin, using FEL or dump trucks. Dust suppression fogging sprays would be located around the ROM bin to control dust emissions during operations.

Feed to the ROM bin would be controlled by dump/no dump traffic signals mounted at the ROM pad adjacent to the ROM bin and these would be controlled by a level sensor mounted at the front of the bin. The ROM bin would discharge to primary jaw crusher via a vibrating grizzly. Grizzly oversize would report to a primary jaw crusher. The grizzly undersize would be combined with the primary crushed production feed on a primary stockpile conveyor.

The primary crushed production feed would be transported to the primary stockpile via a sacrificial conveyor and a stockpile feed conveyor. Primary crushed production feed from Cortadera open pit mine would also be delivered to the primary stockpile. A fixed magnet would pull any large scrap metal from the primary crushed production feed stream.

17.5.4.3 Screening Plant

The crushed production feed would be reclaimed from under the stockpile by three vibrating feeders. 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 reclaim feeders would transfer production feed to a sizing screen bin feed conveyor. Two vibrating feeders would transfer production feed from the sizing screen bin to two double deck sizing screens operating in parallel. The sizing screen bin feed conveyor would be fitted with a weightometer.

- The sizing screen upper deck panels would have a nominal aperture of 50 mm. Lower deck panels would have a nominal aperture of 15 mm. The sizing screens would produce three products; +50 mm, -50+15 mm and –15 mm.
- The coarse product from the top deck (+50 mm) would be transported by a sizing screen oversize conveyor to a secondary crusher feed bin.
- The intermediate size material (-50+15.0 mm) would be transported by a sizing screen oversize conveyor to a tertiary crusher feed bin.
- The undersize material from the lower deck (-15.0 mm) is the final product and it would be transferred to an agglomeration feed bin via an agglomeration feed bin conveyor. An automatic cross cut sampler would be installed on the agglomeration feed bin conveyor for sampling of heap leach feed for metallurgical accounting purposes.

17.5.4.4 Secondary and Tertiary Crushing

The oversize material from the sizing screen top deck would be crushed in a secondary crusher. A secondary cone crusher would operate at a closed side setting of 30 mm generating nominally -60 mm crushed product. The secondary crushed material would report to the crushed production feed stockpile discharge conveyor for re-screening.

The intermediate size product (-50 mm +15 mm) would be transferred to a tertiary crusher feed bin. The tertiary cone crusher would operate at a closed side setting of 14 to 16 mm, generating a -25 mm tertiary crushed product, nominally. The tertiary crushed material would report to the stockpile discharge conveyor for re-screening.

17.5.4.5 Pre-conditioning, Agglomeration and Stacking

The agglomeration feed bin is fitted with an overflow chute feeding an emergency stockpile feed conveyor which would direct material onto an emergency stockpile, when the level in the bin reaches a high setpoint.

A vibrating pan feeder would transfer the tertiary crushed rock from the feed bin to an agglomerator feed conveyor, which feeds a drum agglomerator. The drum agglomerator would be fitted with a VSD to regulate the residence time in the agglomerator. Dilute sulphuric acid would be added in the agglomerator to pre-condition the production feed prior to stacking.

An overland conveyor would transfer the agglomerated production feed to a series of grasshopper heap stacker conveyors, which feed a radial stacker conveyor for stacking on heap pads.

17.5.4.6 Heap Leaching

A heap leach system design has been prepared by Knight Piesold (KP). The agglomerated production feed would be stacked 6 m high. When finished, a pad would be 80 m wide and 540 m long, having a volume of 260 000 m^3 and may hold up to 400 000 t of heap leach feed. The heap leach pad base would be constructed with a longitudinal slope of 1-3% and transverse slope of 0.5% and be lined with a membrane. Testing has confirmed a maximum of seven lifts is permitted with a final heap height of approximately 42 m. For illustrative purposes it has been assumed that there would be 12 cells in the ultimate design. Heap leaching would be conducted on geotechnically engineered lined leach pads, with drainage piping installed and segmented to suit the radial stacking.

Heap leaching of the agglomerated oxide production feed would be conducted based on a three-stage leaching. The primary leaching would be on freshly stacked production feed at a rate of 705 m^3/h of raffinate. The secondary and tertiary leaching stages would be conducted on the depleted cell before it is retired.

The solution from the heap pads would flow to a "w" type drain system that allows direction of the solution to either ILS or PLS solution collection ponds, depending on the grade of the solution. Diversion of the solution between ILS and PLS ponds would be manual and based on leach cycle and solution assays. A pond for storage of raffinate (solution depleted in copper because it has returned from the SX/EW circuit) would also be installed. Each solution pond would have a nominal residence time of 24 hours. The ponds would be double HDPE lined ponds with each having dedicated pumping systems for solution distribution to respective heap pads.

Dripper irrigation networks would be installed on top surface of the heap pads to evenly distribute fresh acidic leach solution at a nominal rate of 10 L/m^2/hr. The total leach cycle (covering all three stages) would be 300 days.

17.5.4.7 Solvent Extraction and Electrowinning (SX/EW)

An SX/EW plant, capable of treating seawater based PLS, would be installed to recover copper from the leach solutions. Rather than a conventional two extraction – one strip mixer-settler SX circuit, an additional wash stage using RO water has been added to allow for the removal of any entrained sea (saline) water. Therefore, the circuit would consist of four conventional mixer-settlers configured as two extractions – one wash – one strip.

17.3.4.7.1 Solvent Extraction (SX)

The SX plant is designed to allow for processing a copper concentration of 2.0 g/L. At the SX stage, industry standard SX extractants such as LIX 984N or Arcorga M5640 would be utilised. A high flashpoint, narrow cut kerosene such as Shellsol 2046 would be used as diluent. A loaded organic coalescer system would be installed to minimise the potential of entrained seawater (and associated chlorine) entering the electrowinning system.

PLS is pumped from a PLS pond to the SX area. The PLS flows through two extraction stages (E1 and E2) connected in series. Copper is transferred from a copper-rich PLS into the organic phase and the low copper raffinate is returned to a raffinate solution pond in the heap leaching area.

Organic solution flows to the loaded organic coalescer then into a loaded organic tank, from where it is pumped to washing stage (W1). In the washing stage, entrained and physically transferred impurities (e.g. chloride and iron) are washed with acidified RO water.

The washed loaded organic phase flows through a stripping stage (S1), where copper is transferred from the organic phase to a lean electrolyte and the copper loaded electrolyte (rich electrolyte) flows from the stripping stage to a rich electrolyte after settler. From the after settler, the rich electrolyte is pumped into an electrolyte circulation tank through electrolyte filters.

Crud generated at the SX area is pumped into a crud collection tank, where it is mixed with acidic water. The mixed slurry is then pumped through a filter press with filtrate returning to the SX circuit. The organic phase from the crud recovery circuit is treated by mixing with bentonite (clay) to make up a slurry. A separate filter press is used to separate the solids from the slurry, composed of bentonite and mixed organic phase. The clean organic phase is returned to the SX circuit.

17.3.4.7.2 Tank Farm

Rich electrolyte from the electrolyte filters is pumped through heat exchangers to collect in an electrolyte circulation tank and then on to the electrowinning (EW) cells. Lean electrolyte discharged from the EW cells is split into two parts, with one part returned by gravity to the lean side of the EW electrolyte circulation tank and the other part is returned to the rich side of the tank.

The lean electrolyte is pumped through heat recovery heat exchangers back to the SX area.

Guar gum and cobalt sulphate would be dosed into the electrolyte circulation tank as additives aiding cathode formation.

17.3.4.7.3 Electrowinning (EW)

The EW system would consist of permanent stainless-steel cathodes, lead alloy anodes and EW cells (pre-cast monolithic tank made from vinyl ester resin concrete). Dedicated electrolyte circulation pumps would provide fresh electrolyte to the EW cells, to ensure sufficient fresh electrolyte flow

In the EW cells the copper ions in solution are reduced to metallic copper at the cathodes. The depleted copper solution is referred to as lean electrolyte and is returned to the circulation tank.

Copper cathodes are harvested every seven days with a semi-automatic tank house crane and transferred to a semi-automatic stripping machine. Washed and stripped copper cathodes are stacked in bundles. The Grade A cathodes are transported and sold in 3 t bundles. The stripped cathode blanks are returned to the EW cells.

17.5.5 Copper Oxide Reagent Handling and Storage

The reagents that would be used in the copper sulphide process plant include:

- Sulphuric acid
- Copper extractant
- Diluent
- Cobalt sulphate
- Guar gum
- Activated clay
- Ferrous sulphate.

Reagent mixing would be done 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.

17.5.6 Copper Oxide Plant Utilities

17.5.6.1 Introduction

The utilities would include equipment and facilities to store and distribute the following water services:

- Raw water
- Demineralised water
- Potable water (including safety shower/eye wash)
- Air systems
- Diesel and lubricants.

17.5.6.2 Raw Water

Raw seawater is pumped to a raw water pond at the copper oxide plant. RO water would also be used as feed water to a demineralised water system to be used for cooling water in the EW circuit and as water addition to the heap leach and for dust suppression in the crushing plant.

17.5.6.3 Demineralised Water

Demineralised water would be produced from a deionising water treatment system which is included as part of the SX/EW plant. Demineralised water would be stored in a demineralised water tank and hot demineralised water tank for use in the crud treatment, reagent preparation, electrolyte circulation tank, cathode washing and cathode stripping machine plant areas.

17.5.6.4 Potable Water

For potable water, low saline water would undergo further treatment at a potable water plant to meet the plant's potable water requirement. Potable water would be distributed to the sulphide and oxide plants as well as to Cortadera.

17.5.6.5 Compressed Air

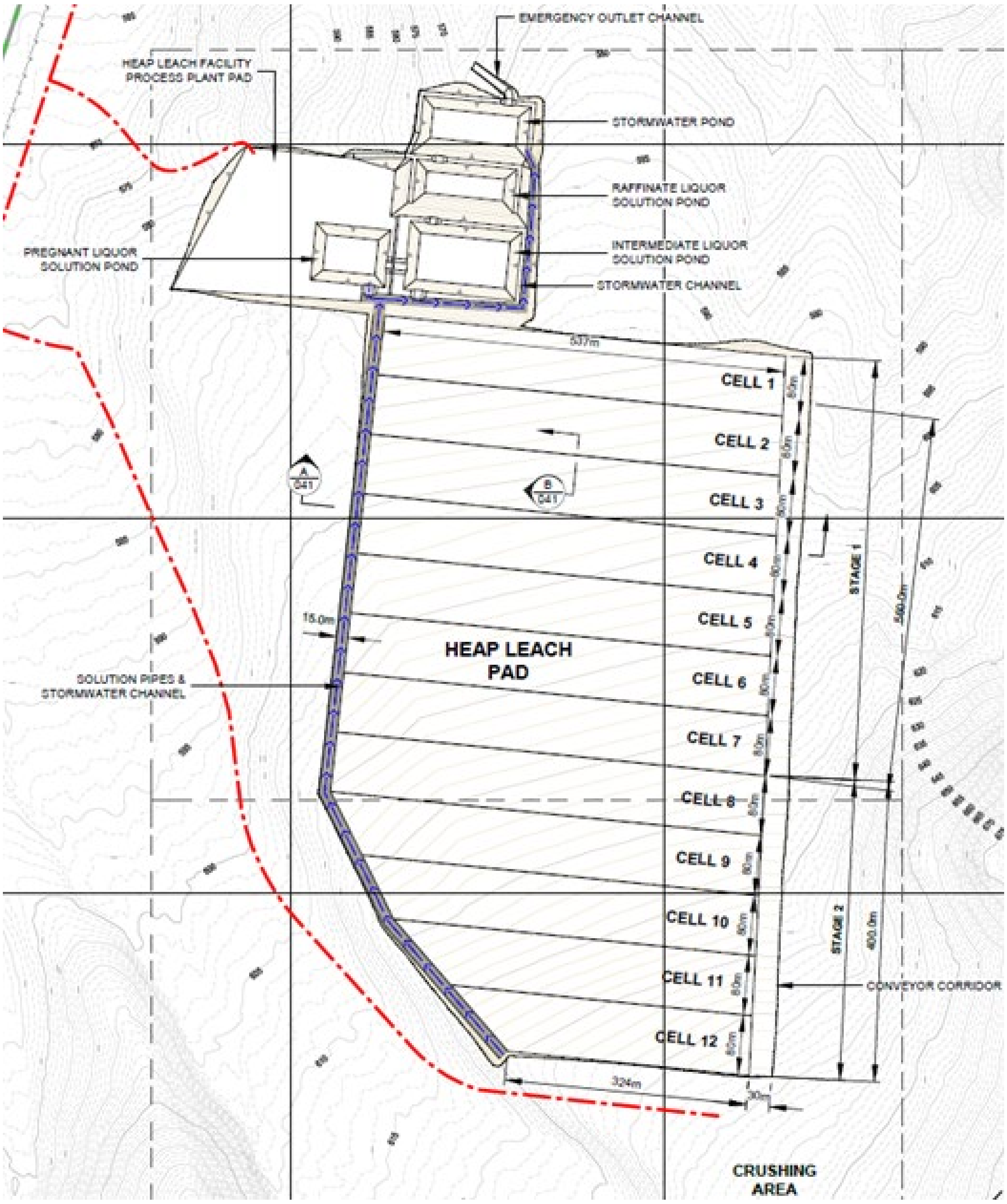
Compressors and blowers would provide air to the systems within the Process Plant:

- Plant air
- Instrument air.

17.5.7 Heap Leach Construction Sequence

The heap construction sequence is set by the stacking rate and the layout and size of the proposed 12 heap leach cells (as shown in Figure 17.4).

Figure 17.4 : Heap Leach Pad Layout



The pads would be stacked in sequence starting with cell 1. In Year 1, Cells 1, 2, 3, 4, 5 and 6 are stacked sequentially. In Year 2, Cells 7, 8, 9, 10, 11 and 12 are stacked sequentially. After heaps 1, 2 and 3, have completed their leaching cycle the drip irrigation is replaced with collection piping for the next lift, which would be stacked above.

The stacking sequence then continues in a similar fashion for the remainder of the Heap Leach LOM.

17.6 Plant Electrical Reticulation, Control Systems and Communications

Refer to Section 18.

17.7 Opportunities

Below is a list of possible alternative technologies that may prove beneficial. This list is by no means exhaustive and only some of these technologies have been the subject of initial investigation.

Process Design Opportunities:

- Crushers and HPGRs to replace SAG milling (previously considered, attractive but not incorporated, pilot testing essential)
- Vertical Roller Mill (VRM) to replace SAG milling and ball milling (recent technology advance, improved energy efficiency, pilot testing essential)
- Coarse particle copper and molybdenum flotation to replace rougher and scavenger flotation (under consideration, scoping tests arranged)
- Investigate processing of the copper concentrates by hydrometallurgical means (Albion scoping underway)
- Investigate replacement of all flotation with bulk sulphide flotation to maximise recovery of all species including pyrite. Process entire concentrate by hydrometallurgical means (successful flotation tests conducted and Albion scoping underway). Generate acid and power from the sulphide sulphur in the concentrate.
- Assess the wider use of Jameson flotation cells in the copper and molybdenum circuits.

In addition, the identified opportunities for improved recovery of copper through less selective collectors being used in scavenging, improved molybdenum recovery in copper roughers and scavengers using diesel, and recovery of free gold from the Cortadera underground deposit, would all be investigated as part of the next stage of flotation testwork.

17.8 Comments on Section 17

The QP notes that robust configurations for the sulphide and oxide process plants and for integration of Cortadera into the project have been developed for this PEA. However, opportunities for improvement, some of which have only become commercially viable since the 2016 Study, should be investigated in the next study stage as noted in Section 17.7.

18 Project Infrastructure

This section has been taken directly from the technical report titled, "Costa Fuego Copper Project: NI 43-101 Technical Report & Preliminary Economic Assessment (PEA)" dated August 14, 2023 with an effective date of June 28, 2023. The Mineral Resource estimate update, as described in this Report, does not materially affect the mineral resource inventory that formed the basis of the 2023 PEA and no new scientific or technical information has been generated on the Project that would materially affect the outcome of the 2023 PEA and, therefore, the results and conclusions of the 2023 PEA are considered current and have been restated in this Report.

18.1 Introduction

18.1.1 Existing Infrastructure and Services

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 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
- 23 kV power supply in Huasco.

Existing infrastructure associated with the supply of seawater from the coast near Huasco to the mine site and concentrate ship loading includes the following:

- Seawater would be pumped to the site partially following an existing route of two steel water pipes between pump pits at Freirina and the Huasco port facility
- Existing Las Losas port facility in Huasco bay near the city of Huasco.

18.1.2 Site Development

The proposed plant site 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 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)
- 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 and Figure 18.2 depict the existing and planned infrastructure.

Figure 18.1 : Costa Fuego Planned and Existing Regional Infrastructure

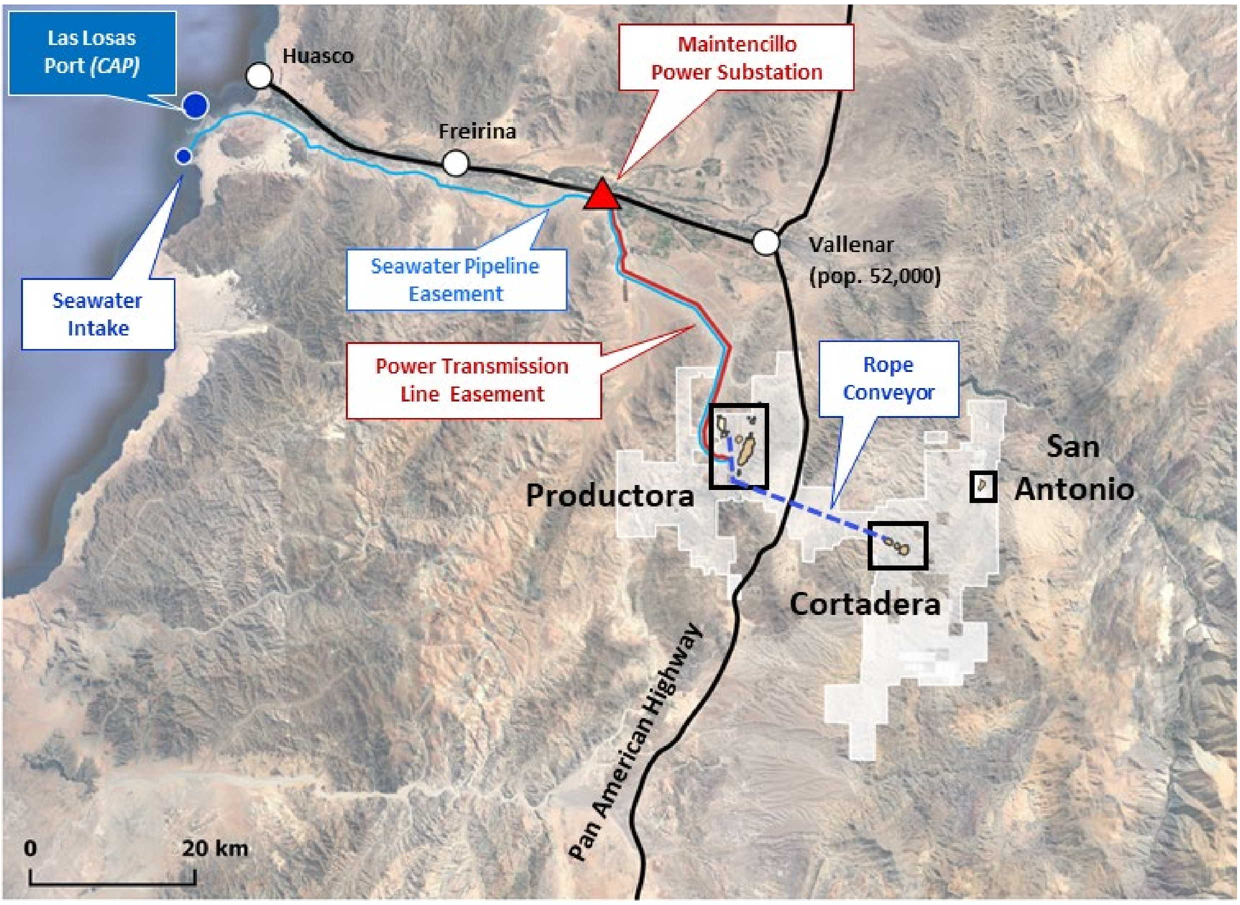
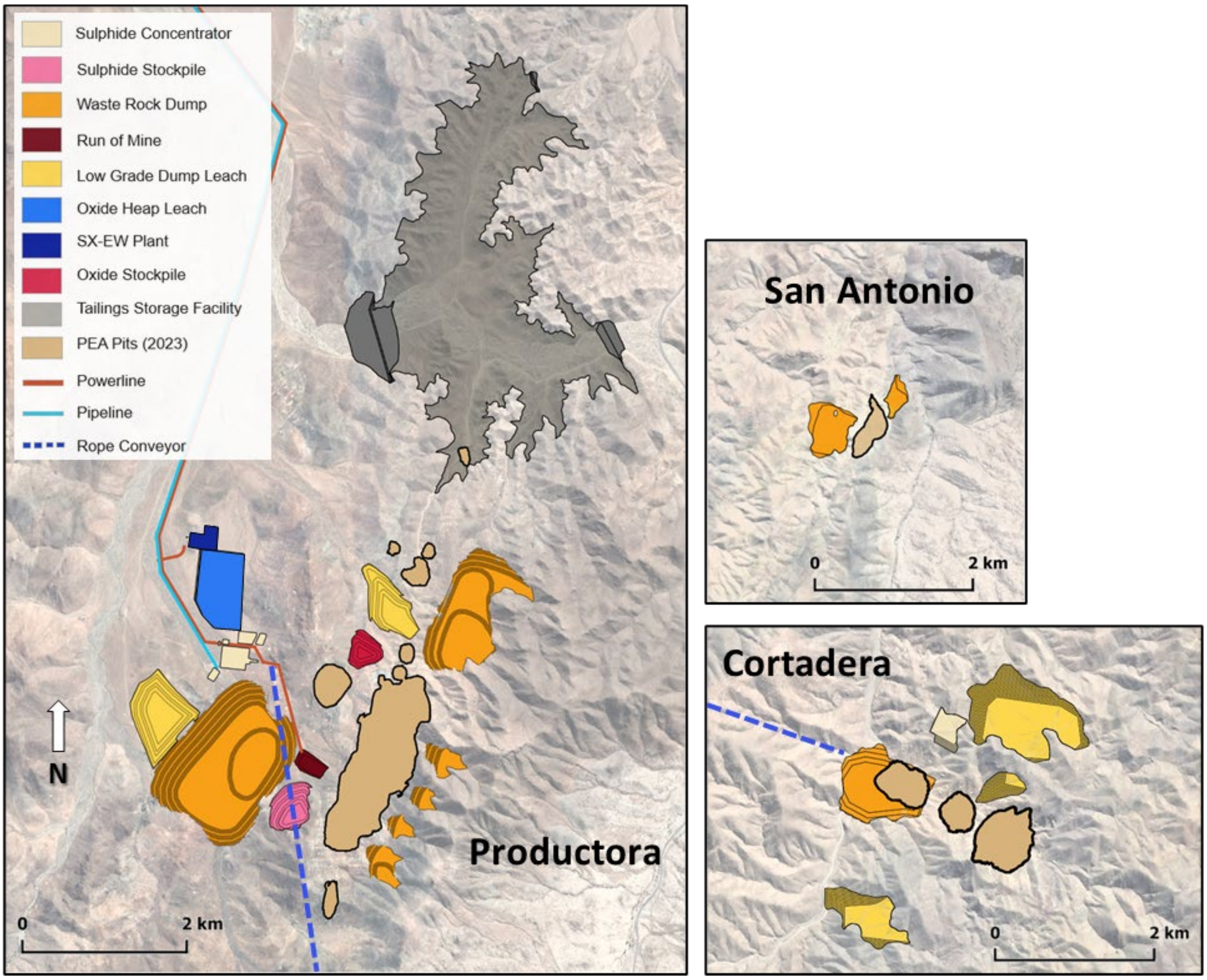


Figure 18.2 : Costa Fuego Planned and Existing Area Infrastructure



18.2 General Infrastructure

18.2.1 Area Roads

The Project is located 39 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 to the Project for work each day.

The Project can be accessed by following the main sealed Pan-American Highway connecting Vallenar to Coquimbo in the south for 25 km. From the Orito cross point, the Project is accessed by taking a gravel road (C-509) towards the southeast for 14 km before reaching the Project.

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 to the Huasco port facility. Concentrate would be transported on a route that would require partial construction, however this selected option has the cheapest net present cost and has the community benefit of avoiding Vallenar, despite having the largest length of new road to construct.

The main Project access road provides separate access to the mining contractor's area, the oxide plant area, the sulphide plant area and the TSF.

18.2.2 Plant Roads

The main Project access road terminates at the entrance to the sulphide process plant. Internal plant roads within the sulphide process plant area would 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. Internal plant roads would be provided for access to the high voltage switch yard and seawater storage pond.

There would be an access road to the oxide plant. This access road would start from a junction on the main access road (south of the access road to the mining contractor's area). The road would terminate at the oxide plant back end (SX/EW plant). Internal roads within the oxide plant back-end area would provide access to all buildings and facilities including deliveries to the reagent storage and for copper cathode transport trucks.

Internal roads within the oxide plant front end would provide access to all buildings and facilities including deliveries to the plant warehouse and for access to the crushing, agglomerator and heap leach areas.

18.2.3 TSF Roads

There would be a main TSF access road to the tailings dam. This access road would be unsealed.

There would be a track to access the decant return pump station. This access track would be unsealed and would also be utilised for the decant return pipeline to the plant.

18.2.4 Seawater Transfer System

A seawater transfer system is required to transfer water from the coast (south of the port of Huasco) to the seawater storage pond at the sulphide plant. The pipeline from coast to mine site is approximately 62 km. A transfer pipeline branching from the main seawater transfer pipeline would supply seawater to the oxide seawater pond.

The seawater transfer system would consist of one intake pump station, two seawater transfer pumping stations and buried transfer pipeline.

The seawater transfer system is capable of pumping seawater 62 km to the Project site at a required flow rate. The power for the seawater intake pump station and transfer pump station would be supplied from an existing nearby 23 kV high voltage power line.

The seawater transfer system would 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.

All other water types (e.g. raw, process, potable), usage and distribution is covered in Section 17.

18.2.5 Power and Electrical

18.2.5.1 General

The electrical supply for the Project would originate at the Maitencillo substation for the main processing facilities and from the local (Huasco) Electrical Distribution Company for the seawater pumping station located near the seawater intake facility.

The main electrical substation (Productora substation) for the processing plants would be located close to the sulphide plant grinding and classification facilities, where the main loads would be located.

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 approach is based on not providing redundancy or standby equipment. The main substation at the sulphide processing plant has space for a redundant transformer.

18.2.5.2 Point of Supply for the Productora Substation

The maximum demand is estimated at 96 MVA and the nearest location where there is sufficient capacity from a power utility is the Maitencillo substation. This substation has numerous 220 kV power lines which connect to the country's transmission and generation network. The only point of supply with sufficient capacity for the Project is from the 220 kV network. The substation has space available for a non-redundant 220 kV point of supply.

18.2.5.3 Power Distribution

The Project would utilise the following voltages:

- 220 kV feed to the main substations at the Maitencillo substation
- 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.

The power lines are a single circuit with overhead earth and supported by concrete poles. The conductors would be all aluminium. The overhead earth would be galvanised steel and include optic fibres for communication. The total length is approximately 25 km.

The assumption is made 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 is available for this Project. The power tariff metering would be at Maitencillo substation.

The Productora substation would include the main transformer, outdoor 220 kV SF6 switchgear and 23 kV air insulated switchgear.

The main transformer would be 220 kV to 23 kV and Oil Natural Air Natural/Oil Natural Air Forced (ONAN/ONAF) rated at 100 MVA/120 MVA complete with on load tap charger. The transformer size and technical specification would be based on transformers that are used within Chile.

Voltage control would be achieved using the transformer on load tap changer and power factor correction equipment.

The main substation includes space for a second identical transformer and provision for extending the 220 kV aluminium bus.

18.2.5.4 Emergency Power

The emergency generators would be diesel. The generators would deliver the rated power at 23 kV and the generators would be arranged to 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 that would progressively shut down noncritical areas and connect power to critical drives and systems.

18.2.5.5 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.

18.2.6 Plant Control Systems 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 tier would comprise the operator interface hardware and network interfacing for remote monitoring and administrative reporting.

Field instrumentation would interface to the PCS through remote 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 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.

18.2.7 Communications and Data Systems

There is currently 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 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.

18.2.8 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 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.

18.2.9 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.

18.3 Mining Infrastructure

18.3.1 Mine Surface Infrastructure

The mine surface infrastructure area, located near the mine as part of the mine surface facilities for the area would mainly 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.

18.3.2 Cortadera Materials Handling

Cortadera is located roughly 15 km east of the Productora deposit area, with the Pan American Highway (Route 5) bisecting the two deposit areas.

The Cortadera infrastructure layouts and designs are based on a Study completed in September 2021 (ABGM, 2021). A system capacity of 7.5 Mt/a was considered as a baseline and traded four main overland materials handling systems (MHS) off from a capital and operating cost perspective.

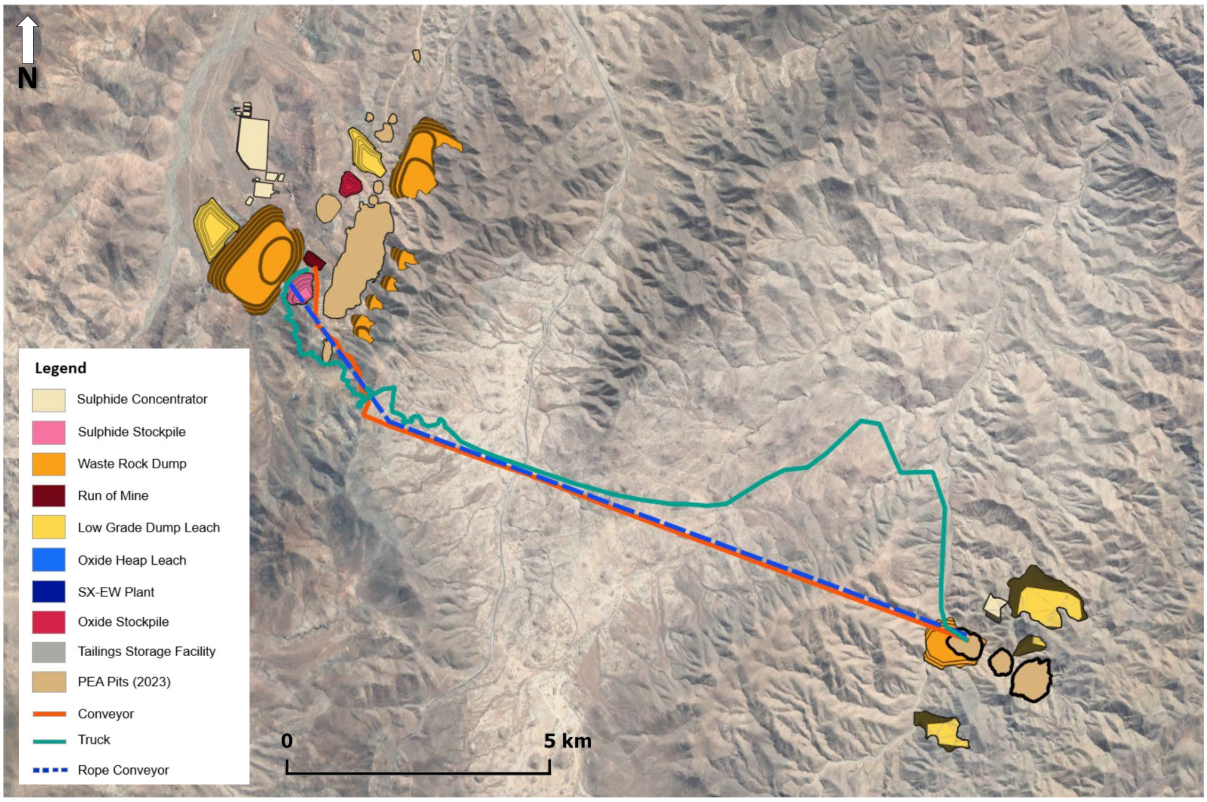
The options evaluated included:

- Trucking
- Overland conveyors (OLC)
- Aerial ropeway
- Rope conveyor (RopeCon).

Initial route designs were completed in Google Earth and exported into Deswik to ensure relative accuracy of route gradients, to determine total route distance (Figure 18.3). Once the routes were established, OEM suppliers were contacted for relevant capital and operating budgetary pricing.

Once the preferred MHS was chosen, updated pricing was obtained for the winning solution.

Figure 18.3 : Trucking, Conveyor and Rope Conveyor Routes



18.3.2.1 Cortadera Materials Handling Summary - General

The options discussed for the 7.5M t/a case were compared for total cost over a 15-year period, transporting concentrate material only, with the results shown in Table 18.1. The throughput rate takes into consideration that waste material would remain at the Cortadera site. Investigation indicated that the aerial ropeway would not be able to support the required material movement and it was subsequently removed from consideration.

Table 18.1 : Materials Handling Cost

Material Movement Option 7.5 Mt/a	Unit	Trucking	Overland Conveyor	RopeCon
Chosen Route Length	km	21.7	15.5	15.0
Unit Cost	$/tkm	0.25	0.04	0.012
Annual OPEX	US$M	27	3	1
OPEX during first 15 years	US$M	407	47	13
CAPEX	US$M	0	105.2	100

The highest capital estimate was used for both the RopeCon and OLC. During the next phase of study, either may present better value than shown here.

The RopeCon is cheaper due to its extremely low operating cost over this period and was selected for inclusion in the PEA.

RopeCon Description

Central processing at Productora allows the use of RopeCon and significantly reduces development timeframes and additional capital related to locating central processing at Cortadera.

A RopeCon is a bulk material transport system that successfully combines conveyor technology with cableway features. Material is transported on a flat belt with corrugated side walls (see Figure 18.4). As with conventional belt conveyors, the belt performs the haulage function and is driven and deflected by a drum in the head or tail station.

The axles are fixed to the belt at regular intervals to support the belt. Running wheels fitted to either end of the axles run on track ropes with fixed anchoring and guide the belt. The track ropes are arranged with fixed anchoring to their tower structures. They do not move like a cableway, although they drape analogously across the towers, to which they are fixed and elevated off the ground. Due to this limited ground contact, this suspended conveyor is well-suited for rugged terrain or to cross obstacles such as roads, rivers, or buildings.

Figure 18.4 : Close-up view - RopeCon transports material on a flat belt with corrugated side walls that is fixed to axles that run on track ropes suspended from tower structures.

The RopeCon system can provide handling capacities in excess of planned requirements, across difficult terrain whilst occupying a minimum structural footprint. Simple maintenance of the conveying line and low space requirements are the key features of this product.

The Costa Fuego RopeCon transport route spans 15 km and threads along a relatively straight route from the Cortadera Project, directly across a range of hills and then across an extended coastal plain, toward the Pan American Highway. A transfer station on the plain separates the route into two sections, with the second section crossing above the highway (see example in Figure 18.5) and then below the high-voltage powerlines of the National Grid before climbing a second range of hills to enter a valley south of the proposed Productora open pits (see Figure 18.3). Within the valley, a transfer station will allow the RopeCon route to reorient to the North, skirting a proposed waste storage facility as it climbs a bluff overlooking the planned sulphide concentrator site. From this elevated position, the suspended RopeCon will traverse the plant site to anchor on the far-side, discharging production feed directly onto the concentrator stockpile. A similar arrangement was constructed in El Limon and is shown in Figure 18.6.

Figure 18.5 : View of RopeCon Road Crossing Installation Pictured at Booysendal North, including Close-up of Shielding (below conveyor) for Road Protection.

Figure 18.6 : RopeCon installation to transport material from the crusher installation on the bluff to the plant stockpile at El Limon, Mexico.

Advantages of this system include low operating cost and energy consumption, low noise emission, high availability, reduced footprint and the easy crossing of obstacles. Maintenance of moving components is performed at a single location for each belt, reducing the need for mobile work on the line to planned inspections and repair of stationary components. Access to the fixed-line infrastructure uses an inspection carrier platform that operates independently from the top ropeway tracks and does not require system shutdown (see example in Figure 18.7).

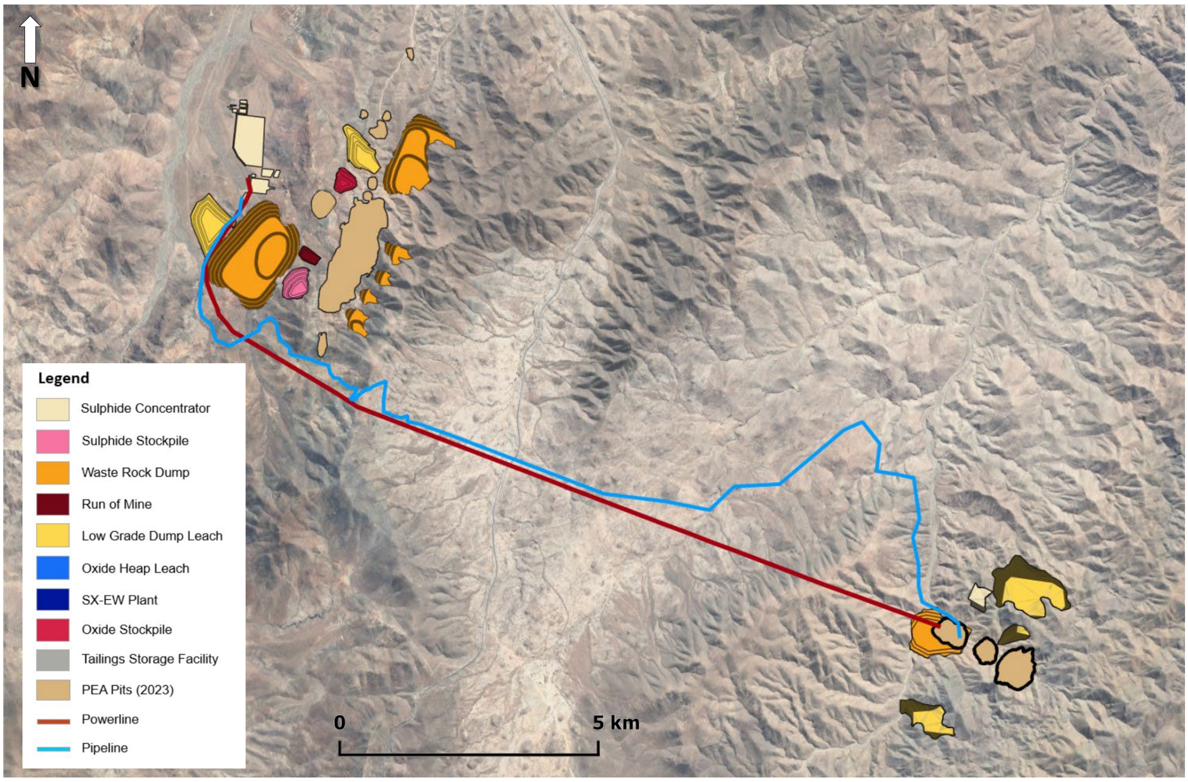
Figure 18.7 : RopeCon Inspection Carrier in Operation at Booysendal North, South Africa.

18.3.3 Cortadera Water Pipeline

With the process plant located at Productora, a nominal requirement for heap leaching, wash down and mining activities would be required at Cortadera. A water balance would be completed in the next phase of study, but the initial consideration is the route for the water pipeline to Cortadera.

The southern route was chosen as the best solution to piped water to Cortadera and follows a similar path to the southern trucking route. It starts at the sulphide seawater storage pond and skirts the southern edge of the southern WRD before passing over the saddle area toward Cortadera. The design aimed to keep the pipeline inclination at less than 8.5° over the entire route. The line is roughly 24 km long with an elevation difference of ~450 m between the pond and the saddle crest. Figure 18.8 shows the layout for the planned southern water line route.

Figure 18.8 : Cortadera Water Pipeline and Power Line Routes



18.3.4 Cortadera Power Line Route

The proposed power line route follows a similar path to the proposed water line route, but starts at the Productora Substation, and runs along the western and southern edge of the southern WRD before passing over the saddle area toward Cortadera. The design aimed to be the shortest route over the hills. The line is ~18.3 km long. The route runs parallel to the conveyor and RopeCon routes for most of its length. Figure 18.8 shows the layout for the planned power line route.

18.3.5 Block Cave UG Crusher Selection

18.3.5.1 Study Analysis (7.5 Mtpa)

During the Scoping Study investigation of alternative overland materials handling systems, targeting 7.5 Mt/a, crushing experts needed to be engaged to ensure the feed material properties required for conveyance could be achieved via primary crushing.

The Metso Outotec Group, ThyssenKrupp and MMD were contacted to provide their initial recommendation for possible crushing solutions derived from an underground block cave footprint. The largest crusher lump feed size (F100) was constrained to a top size that was reasonable for multiple common crushers of the required throughput range. As such, it was necessary to limit the largest lump size to 960 mm for each of the respective crushers, though some crushers listed here are capable of handling larger rocks. To achieve a reasonable conveyor belt width and thus obtain a reasonable match with throughput rate, the belt criteria dictated a P_{98} – P_{99} of no greater than 300 mm. Any feed material larger than F960 mm would need to be handled via a fixed grizzly and rock breaker arrangement or blasting at the respective draw points on the draw level. Alternatively, larger rocks may be placed in nearby stockpiles during the shift and at end of shift these rocks may be blasted and crushed on the following shift.

Table 18.2 provides the recommended crusher units that were proposed by the respective OEMs.

Table 18.2 : Crusher Design Characteristics Utilised						
Company	**Unit**	**Metso: Outotec Group**			**Thyssenkrupp**	**MMD**
Crusher Type		**C160 Jaw**	**C200 Jaw**	**Gyratory MKIII 50-65**	**ERC 22-20**	**1150 Twin-Shaft Sizer**
Main Characteristics						
F100 Feed Size	mm	960	1 200	1 016	960	960
Power	kW	250	400	450	300-450	400
Max CSS	mm	300	300		260	N/A
Min CSS	mm	150	175		120	N/A
Throughput at Max CSS	t/h	**1 260**	**1 575**		**3 450**	**6 000**
Throughput at Min CSS	t/h	**430**	**630**		**1 050**	
Max OSS	mm			178		N/A
Min OSS	mm			150		N/A
Throughput at Max OSS	t/h			**2 935**		N/A
Throughput at Min OSS	t/h			**2 395**		N/A
Equipment Form Factor						
Length	m	4.28	6.60	4.77	5.9	7.4
Width	m	3.70	4.08	4.46	3.95	3.35
Height	m	3.75	5.10	5.5	3.8	1.6
Weight (Basic)	t	76.3	124	153	138	60

The Metso C160 jaw crusher is the smallest (dimensions) and one of the lightest units considered during this study. At the stated throughput rate, it would be utilised at 91%, while assuming particles -300 mm are scalped from the feed via a VG860-4V vibrating grizzly feeder (VGF), removing ~700 t/h and the remaining 500 t/h being crushed by the jaw crusher.

If the C160 is not sufficient for the duty, a C200 jaw crusher may be the next best solution. The concern with the C200 is that its lowest Closed Side Setting (CSS) of 175 mm would produce P_{98}-P_{99} in the region of 350 mm top size, which would incur additional cost in belt width for the materials handling system. At higher throughput rates this specification may suit the application well if a wider belt is specified.

The 50-65 gyratory crusher would do the duty easily but is significantly underutilised at this throughput. It also requires the largest excavation envelope of the selection and is harder to maintain. It does, however, not require an apron feeder nor a grizzly feeder, as boggers/trucks can tip directly into the crusher feed pocket.

The Eccentric Roller Crusher (ERC) is the latest technology crusher on the market, that aims to straddle the capabilities of jaw and gyratory crusher offerings. It has a low form factor, with high throughput and high reduction factors for the feed material. It was deemed a plausible alternate option, particularly for this study and the possible underground application. The key risk with this option is the layout dimensions and configuration to which the voids would be required to be developed. Should this unit present problems with mechanical availability or utilisation and efficiency, it would be difficult to change to another option as it would require new equipment installation voids and underground infrastructure. This option has limited operational data to satisfy some key concerns at this stage, and warrants further analyses and reviews in the next study phase.

Finally, a sizer by MMD was also considered. Hard-rock applications are still limited, yet the technology has improved significantly in recent years. MMD provided a proposal for one of their more robust units and indicated that they had no concerns for this application. The design could also make provision for a scrolling function where large rocks are rejected to a reject pile and handled via rock breaker or blasting rather than backing up feed material. It is one of the smallest crushers considered and should handle the throughput with relative ease. It is worth visiting some current hard-rock installations that utilise the technology to assess reliability and throughput.

Table 18.3 indicates the breakdown in CAPEX and OPEX for the respective crusher options. Capital was considered by comparing the difference in cost for the units themselves, and consequent additional feeder mechanisms, as well as their expected final excavation dimensions. Budget estimates were received for each of the stated mechanical components. Only one estimate was obtained for an apron feeder and VGF.

Table 18.3 : Cost Breakdown for Respective Crushers

Company	**Unit**	**Metso: Outotec Group**			**TK**	**MMD**
Crusher Type		**C160 Jaw**	**C200 Jaw**	**Gyratory MKIII 50-65**	**ERC 22-20**	**1150 Twin-Shaft Sizer**
Capital Cost						
Cost	US$M	0.78	1.83	2.05	2.79	1.97
Additional Equipment						
Vibrating Grizzly Feeder (VGF)		Yes	Yes	No	No	No
VGF Cost	US$M	0.31	0.31	0.00	0.00	0.00
Apron Feeder		Yes	Yes	No	Yes	Yes
Apron Feeder Cost	$M	0.27	0.27	0.00	0.27	0.27
Total Mechanical Capex	**US$M**	**1.4**	**2.4**	**2.1**	**3.1**	**2.2**
Civil / Mechanical Install	$$	**$**	$$	$$$	$	$
Excavations						
Excavation Size (Estimate)	m^3	1 760	2 880	8 932	2 880	1 760
Cost of Excavation	$M	0.25	0.40	1.25	0.40	0.25
Total Relative CAPEX	US$M	**1.6**	**2.8**	**3.3**	**3.5**	2.5
Operating Cost	US$/t	**0.08**	0.10	0.10	0.10	0.095

The Metso C160 jaw crusher clearly stands -out on both the CAPEX and OPEX metrices for a system that is required to maintain 7.5 Mt/a. Two of these units could be installed for the block cave footprint, which would allow for some redundancy in crushing capacity, while significantly reducing congestion and tramming distance on the draw level.

18.3.5.2 20 Mt/a Crusher Selection

For the 20 Mt/a scenario a peak design throughput rate of 2900 t/h is considered. It should be noted that a Metso MKIII 50-65 could keep up with this instantaneous rate when set to its maximum OSS. The P_{98}-P_{99} produced in this setting should be roughly -280 mm to -300 mm and still aligns well with the conveyor and RopeCon specifications used in the Trade-off study.

With a centrally placed layout, a single gyratory crusher would be able to keep up with the required throughput, while providing a low risk proposition over the life of the asset, as proven by well over a thousand similar installations worldwide.

At the 20 Mt/a throughput rate, the Thyssenkrupp ERC 22-20 may be an attractive alternative, with its relatively small installed dimensions and comparable throughput range. The ERC is a relatively new technology and was being trialled at one operation during the time of the Scoping study.

Considering the cost implications shown in Table 18.4, it is clear that two C200 Jaw crushers would be required, with their consequent mechanical feeder assemblies, significantly increasing this solutions' capital intensity. A unique development layout would be required for each crusher option, with two apron feeders required for the C200 option, one apron feeder for the ERC option and direct feed to the Gyratory layout.

The sizer is not shown in this table but may be included for further analysis in the next stage of study.

Multiple tip arrangements feeding the same gyratory crusher is quite standard and seen as a robust solution.

Table 18.4 : Cost Breakdown for Respective Crushers – 20 Mt/a

Company	**Unit**	**Metso: Outotec Group**		**TK**
Crusher Type		**C200 Jaw**	**Gyratory MKIII 50-65**	**ERC 22-20**
Capital Cost				
Cost	US$M	**2 x 1.83**	2.05	2.79
Additional Equipment				
Vibrating Grizzly Feeder (VGF)		2 x Yes	No	No
VGF Cost	US$M	2 x 0.31	0.00	0.00
Apron Feeder		2 x Yes	No	2 x Yes?
Apron Feeder Cost	$M	2 x 0.27	0.00	2 x 0.27?
Total Mechanical Capex	**US$M**	**4.8**	**2.1**	**3.4**
Civil / Mechanical Install	$$	$$$$	$$	$$$
Excavations				
Excavation Size (Estimate)	m^3	2 x 2 880	8 932	1.3 x 2 880?
Cost of Excavation	$M	0.80	1.25	1.3 x 0.40?
Total Relative CAPEX	US$M	**5.6**	**3.3**	**3.9**
Operating Cost	US$/t	0.10	0.10	0.10

All three options have roughly the same expected OPEX cost of $0.10/t, with the gyratory solution possibly producing a lower cost solution per tonne at 20 Mt/a. The final selection criteria should thus come down to:

- Matching of crusher product with optimised MHS (conveyor selection)
- Risk profile associated with the crusher technology, including:
    - Excavation size and layouts
    - Benchmarked performance
    - Confirmed OPEX cost
    - Reliability (availability and utilisation).

With limited analysis completed, the Gyratory crusher is utilised for the PEA financial model's purposes.

18.4 Process Plant Infrastructure

18.4.1 Buildings

18.4.1.1 General Plant Buildings

The Project would include administrative and auxiliary buildings located in the proximity of the Sulphide Copper Plant. These buildings have been specified as transportable buildings to be installed for the Project, for the following offices:

- Administration office
- Safety and ERT (Emergency Response Team) building
- Training and Amenities building
- Change house facility
- Plant Mess.

18.4.1.2 Sulphide Copper Plant

Apart from the buildings allocated for the administration of the Project, transportable buildings to be installed for the Project include the following:

- Sulphide Plant Processing and Maintenance office
- Security gatehouse.

Steel framed buildings to be installed for the Project include the following:

- Sulphide Plant Maintenance Workshop and Warehouse
- Sulphide Plant Reagents Store
- Sulphide Light and Service Vehicle Workshop.

To control and operate the site the following buildings would be included:

- Crusher Control Room
- Sulphide Main Control Room.

18.4.1.3 Copper Oxide Plant

Apart from the buildings allocated for the administration of the Project, transportable buildings to be installed for the Project include the following:

- Security gatehouse.

Steel framed buildings to be installed for the Project include the following:

- Oxide Plant Reagents Store.

To control and operate the site the following buildings would be included:

- Crusher Control Room
- Oxide Plant SX/EW Control Room.

18.4.2 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.

18.4.3 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.

18.4.4 Switchrooms and Motor Control Centres (MCC)

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
  - Seawater supply
  - Seawater pumping station 1 MCC
  - Seawater pumping station 2 MCC
  - Seawater intake pumping station MCC.

18.4.5 Weighbridge

A weighbridge would be constructed near the entrance to the sulphide process plant. This weighbridge would be used for incoming product and for outgoing concentrate shipment.

18.5 Fuel Storage

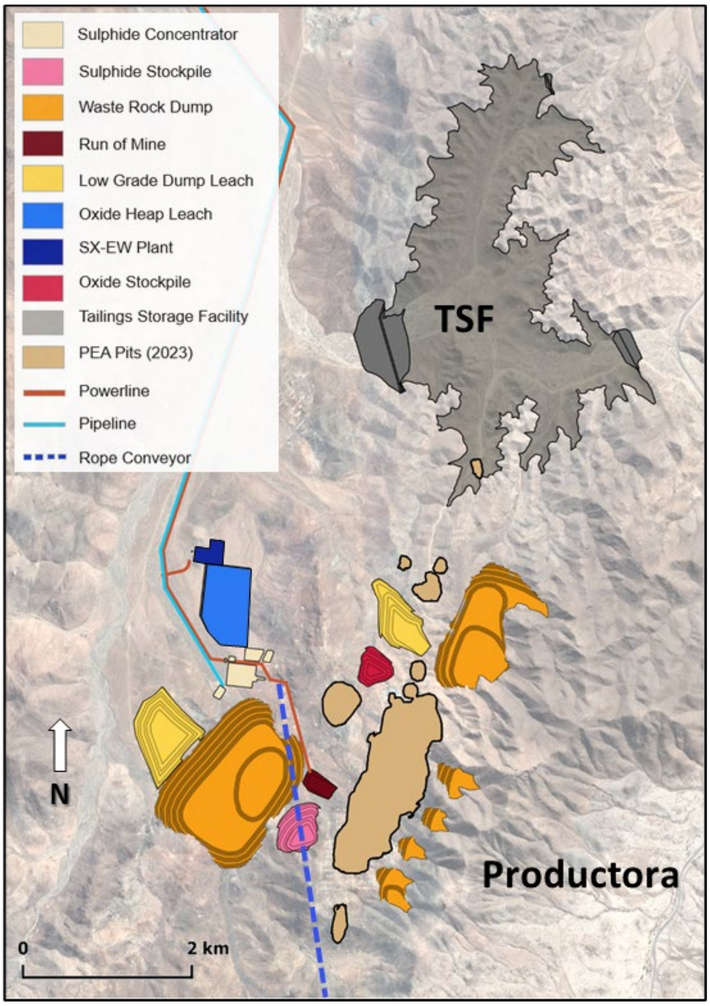
The sulphide plant area fuel farm would have storage capacity based on twenty days of site storage and annual consumption for administration and maintenance vehicles.

18.6 Tailing Storage Facilities

18.6.1 Introduction

The Tailings Storage Facility (TSF) is located approximately 5.4 km northeast of the Project plant site (Figure 18.9). The main embankment traverses the area of lowest elevation within the footprint of the TSF. The topography of the proposed TSF area is steep and hilly. The TSF perimeter is characterised by steep and undulating ridges. The basin is located in a valley that is bounded by ridges to the north and south. A second embankment would be constructed on the eastern side of the valley and a third, smaller saddle embankment would be constructed to the North 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 RL831 m above mean sea level.

Figure 18.9 : Location of Proposed TSF



The site is dry, with the average yearly rainfall for the Project area approximately 31 mm and the estimated annual potential evapo-transpiration (PET) 1700 mm/year.

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. Therefore, for the purpose of this study, indicative peak ground acceleration for the Operating Basis Earthquake is in the order of 0.35 g and the Maximum Design Earthquake (MDE) is likely to be in the order of 0.65 g.

A geotechnical investigation of the previously proposed TSF area was undertaken and indicates that the general site stratigraphy comprises alluvial soils (silts, sands, gravels and boulders) to 15 m overlying bedrock. It is assumed that the ground conditions in the new TSF area would generally be consistent with these findings, however they would be assessed at the TSF site with future site investigations.

A groundwater assessment of the previously proposed TSF area was undertaken and indicates that groundwater is present, generally at around 10 to 15 m below ground level. It is assumed that the groundwater in the new TSF area would generally be consistent with these findings, however this would also be confirmed with future site investigations.

18.6.2 TSF Type and Design

18.6.2.1 Design Data

The proposed facility is for valley storage with a multi-zoned embankment and in pit disposal within Productora Pit. The data used for design is summarised in Table 18.5.

Table 18.5 : TSF Design Data		
Item	**Value**	**Source**
Total tonnage (Mt)	500	HCH Ltd
-Tailings Storage Facility	300	
- In pit Disposal	200	
Throughput (Mt/a)	20	HCH Ltd
Discharge % solids	70	Wood
Estimated beach slope (70% solids)	1V:120H	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

A sample of representative tailings was previously tested by Knight Piésold in 2015. Samples for tailings testwork 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 of 150 µm.

Based on the tailings testwork, the design parameters provided in Table 18.6 were estimated by Knight Piésold. At 70% solids, it is expected that the tailings would exhibit highly thickened behaviour.

Table 18.6 : TSF Estimated Design Densities and Maximum Water Return	
Item Design Densities (t/m^3)	**Value**
Stage 1	1.49
Stage 2	1.54
Stage 3	1.56
Stage 4+	1.57 – 1.63
In Pit Disposal	1.20
Expected max. decant pump capacity	25% Return (75 L/s)

18.6.2.2 Design Concepts and Embankment Staging

A deposition model was run for the TSF; assuming deposition off the eastern and western embankments and end point discharge at two locations along the southern ridge of the facility. The nominated minimum freeboard for tailings was set at 1 m in accordance with the Chilean guidelines.

The climate at the site is very dry with average annual rainfall of about 31 mm, so water storage capacity would be a function of short-term storm events. The 24-hour storm events were estimated based on meteorological data from Vallenar. The quoted storm events are provided in Table 18.7.

Table 18.7 : Maximum 24-Hour Storm Events

Return Period (yr)	24-Hour Rainfall (mm)
2	14.5
10	37.0
50	55.2
100	62.6
200	69.7

The data sequence used to determine these events was relatively short (48 years of valid data). For the study, it was decided that the design requirement would be that the TSF would need to hold a 1 in 10 000 year / 24-hour storm event plus a freeboard allowance of 0.5 m. The calculation of the storage volume required is summarised in Table 18.8. An assessment of the upstream catchment was also completed.

Table 18.8 : TSF Storm Capacity

Item	Value
TSF catchment area (ha)	2 170
1 in 10 000 yr 24-hour storm depth (mm)	200
Catchment run-off coefficient	0.6
Design storm volume (Mm^3)	2.6
Upstream catchment area (ha)	5 945
1 in 10 000 yr 24-hour storm depth (mm)	200
Catchment run-off coefficient	0.6
Design storm volume (Mm^3)	7.2

The storm depth was determined by extrapolating the available 24-hour storm events out to a 1 in 10 000-year return interval. This gave a depth of between 120 mm and 180 mm, depending on the formula used for the extrapolation. A design value of 200 mm was nominated.

The catchment runoff coefficient was selected, based on an analysis of the Atacama flood of 2001 (Houston, 2006, pp 591-610).

In this analysis, runoff coefficients across a range of catchment areas were back-calculated as follows:

- Ephemeral streams 0.045 – 0.093
- Perennial stream 0.243 – 0.403.

Based on these values, a conservative runoff coefficient of 0.6 was selected for the design (however this value would likely be reduced for the next phase of design).

The causeway for the pipeline along the southern ridge would be constructed using a combination of cutting into the ridge and placing a rockfill wedge along the southern edge of the tailings surface.

A total of 14 TSF raises would be required over life of mine and would occur on an annual basis. Stage 1 would provide storage for 20 Mt of dry tailings and have a maximum embankment height of 30.7 m (RL 566.7 m). Stage Final would provide storage for 300 Mt of dry tailings and have a height of 80.4 m (RL 616.4 m). A total of 110 Mt of dry tailings would be stored within the Productora pit before an embankment would be required. An additional 90 Mt of tailings can be contained by constructing a 52 m high (RL 827.0 m) embankment around the Productora pit, this would be achieved in two stages.

It is noted that the water freeboard available is very large under normal operating conditions (much greater than one metre). However, Knight Piésold conducted a review of the water freeboard for the normal operating pond plus the extreme storm events and applied an additional minimum freeboard of 0.5 m for this case.

Due to the limited access to the embankment area and the requirement to raise all of the embankment zones simultaneously, it is anticipated that the embankment construction schedule might not align with annual lifts and therefore the staging would be rationalised to suit the mine waste production schedule.

18.6.2.3 TSF Decant Water System

In the initial years of operation, the upstream toe of the tailings moves up the valley by 3.0 km. Decant recovery in these initial stages would require a mobile decant pump system.

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 mounted decant pump would be placed on the causeway with the decant turret placed in the trench. The pump would be relocated at regular intervals along the causeway as the pond advances up the valley.

Decant water would be recovered from the TSF at a maximum value of 270 m^3/h. Decant water would be recovered via a two-stage pumping system. During the initial years of operation, a diesel-powered suction pump would recover decant water via the intake turret and transfer it to a fixed pump station on the eastern side of the TSF. The fixed station would be fitted with two diesel powered pumps to transfer the recovered water to the process water tank at the plant. A small storage tank would contain a seven-day diesel supply for the pumps.

18.6.2.4 Closure Concept

For closure, it is proposed that the final tailings profile is shaped to direct runoff to the northeastern corner of the facility, where a closure spillway would be excavated in the northern ridge so that any rainfall runoff would run over the tailings surface, to a sediment control area prior to discharge downstream. The tailings surface would be covered by a suitable cover of local borrow and waste rock to prevent erosion of the tailings surface.

18.6.2.5 Drainage Seepage Control

The drainage control for the TSF would focus on containment and recovery of seepage at the embankment location. The TSF embankment is located in the narrow section of the valley where the bedrock rises close to the surface (estimated to be 3 to 5 m deep). Previous geotechnical investigations have determined that the bedrock permeability is generally low. The system would include the following components:

- A cut-off trench under the embankment consisting of a 2 m wide bentonite treated zone plus a moisture conditioned and compacted local borrow zone. This zone would extend five metres below the basin surface into the underlying bedrock. The cut-off trench would be achieved using a bentofix layer keyed into the bentonite treated layer at the base and extending to the top of the embankment.
- On the upstream face of the embankment, seepage control would be achieved using a bentofix layer keyed into the bentonite treated layer at the base and extending to the top of the embankment.
- Downstream of the cut-off trench, a drainage system would be constructed under the embankment consisting of the following:
  - Finger drains – 100 mm draincoil in sand, wrapped geotextile, drains extending from the spine of the drainage system to the outside edge of the valley floor on either side (two pipes each nominally 330 m across width of valley base)
  - 160 mm draincoil surrounded by sand, wrapped in geotextile, as a collector drain along the spine of the valley to a downstream collection sump
  - Downstream of the collection sump would be three monitoring / collector bores drilled to a depth of 30 – 40 m which would intercept any seepage flow within the fractured bedrock zone.
- The geochemistry of the tailings indicates a potential issue with long term acid generation. Given the lack of rainfall at the site, it is proposed to incorporate a secondary downstream cut-off trench adjacent to the interception bores to enhance the seepage control system.

Beyond the drainage control system, a number of monitoring bore stations (with one shallow and one deep monitoring bore) would be installed to monitor any potential seepage. These bores would be constructed with sufficient diameter to be equipped with a pump if the seepage rates are high.

18.6.2.6 Quantities

The embankment would incorporate the following zones:

- Zone A1 – Low permeability zone consisting of selected local borrow materials (silty sandy gravel) with 5% bentonite added; moisture conditioned and compacted to achieve a target permeability of less than 5×10^{-9} m/s
- Zone A2 – Moderately low permeability zone consisting of selected local borrow materials (clayey or silty sandy gravel), moisture conditioned and compacted to achieve a target permeability of less than 5×10^{-8} m/s
- Zone C1 – Selected finer waste rock to form a suitable subgrade for placement of Zone A
- Zone C2 – Run of mine waste rock to form the downstream structural zone of the embankment
- Zone C3 – Free draining coarse waste rock to provide erosion protection on the upstream embankment face.

18.6.2.7 Risks

Beach Slope

The design is based on an average tailings beach slope of 0.833% (120H:1V). However, the beach slope is heavily dependent on the grind size and the production feed blend. Thus, small changes in plant performance or design, production feed type, or the production feed blend have the potential to change the tailings beach slope.

A number of approaches 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 a nominally 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 would be modified to bring the schedule back into line with the design, and the subsequent lifts would be on an annual basis essentially as per the design raise heights.

Steeper Beach Slope

If the measured beach slope is steeper than the design slope, the tailings rate of rise against the Stage 1 embankment would be faster than expected, and the Stage 1 TSF would reach capacity earlier than the design. The response to this would be to move Stage 2 construction of the TSF forward. Commencing the construction one or two months earlier should not have a significant impact.

It should be noted that for steeper beach slopes the potential tailings storage would be reduced, but the storm water storage capacity would 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 would be increased. The overall TSF stormwater storage capacity would not be affected, unless Stage 2 construction is deferred beyond the original construction schedule.

Achieved Densities

The staged TSF embankment crest elevations are based on the tailings characteristics and throughput. Changes in these characteristics and/or throughput would 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 Stage 2, either earlier or later than the design timing. It is recommended that monitoring of throughput, production feed blend, rate of rise and achieved densities be undertaken so that suitable planning and staging of the future embankment construction can occur.

Availability of Mine Waste for Construction

The proposed design of the TSF was based on fill material being delivered by the mining fleet. If waste is not readily available, additional borrows would be required and although this is possible, the cost per tonne of tailings stored would increase.

Life of Mine Planning

Any changes to the life of mine plan or throughput would impact upon the tailings management requirements for the site. Any significant increases in total throughput may require an expansion of the current TSF, or an additional facility to be considered. It should be noted that the current TSF location would allow additional tailings storage by raising the embankment above the current final height.

Tailings Geochemistry

Geochemical testing of the tailings should be continued at points throughout the life of the TSF to ensure that initial testing remains valid. Measurements would need to continue as part of ongoing operations to ensure information is available on the geochemical behaviour of the tailings.

Dam Break Assessment

The TSF is located to the north of the plant site. A significant failure of the TSF embankment (resulting in discharge of tailings from the TSF) would result in limited impact to site infrastructure, however a preliminary flow path assessment indicated that populated areas downstream may be at risk. A dam break assessment to model flow of tailings has not been undertaken for this phase of study but would be included in the subsequent PFS.

To reduce the likelihood of a failure of the TSF embankment, downstream raise construction methods have been included in the design, to promote embankment stability.

A preliminary TSF Emergency Plan would be developed as part of the operating guidelines.

18.6.2.8 Opportunities

Engineered Soil Cover

The current design for closure and decommissioning of the TSF includes an engineered fill cover constructed over the tailings beach as the most suitable long-term solution. On-going characterisation testing during construction may provide alternative cover design on the final (drained) tailings surface after compaction. If this is the case, the costs for rehabilitation of the tailings surface may be significantly reduced.

Embankment Slope Design Angle

For the next phase of design, the embankment slope angles should be optimised based on stability vs. cost vs. constructability. If it is practicable to steepen the slope, then some cost savings should be achievable.

Soil/Bentonite Mix

For this design, a low permeability soil is proposed by mixing naturally occurring sand/silt with 5% bentonite (based on published papers on the subject). This should be assessed in more detail during the PFS by laboratory testing in order to optimise the capital costs.

18.7 Waste Storage Facilities

18.7.1.1 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 by using approved suppliers of waste oils as required by the law.

18.7.1.2 Sewage

Three independent packaged sewage treatment plants (STP) would be installed to service the sulphide plant area buildings, the oxide 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.

18.8 Camps and Accommodation

Accommodation of temporary personnel during the construction phase and permanent mine site personnel during the operations phase would be provided in Vallenar and La Serena.

Personnel would be transported to the mine site by company bus, excluding where personnel have been allocated a company vehicle.

18.9 Port Concentrate Storage and Loadout

An existing port facility would be utilised for receival, storage, reclaim and ship loading of the Project copper concentrate. The facility would require upgrading (by others) to handle the volume of mine concentrate to be stored and shipped. The port facility would be available as required by HCH on a tariff basis.

HCH is advancing with discussions to secure port access and services for the Project. New facilities required to be constructed for the Project would include the following:

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

18.10 Comments on Section 18

The QPs note:

- The infrastructure design is in line with the planned process and mining rates, and is appropriate for a greenfields development in a remote area
- Further work is required on the Cortadera materials handling, noting that this is proposed future work
- Borrow pits would be required to provide the materials needed for construction through to mine closure
- The TSF design has a number of opportunities to be assessed in further stages of the Project and that process opportunities such as coarse ore floatation could also have an effect on the TSF design
- Water for process demands would be sourced from the seawater pumping and storage pond facility throughout the life of operations
- Port facilities for the Project are planned utilising an existing port, with discussions to secure port access and services. New facilities if they had to be constructed for the Project would include storage and loading facilities.
- No camp is planned for the accommodation of personnel.

19 Market Studies and Contracts

This section has been taken directly from the technical report titled, "Costa Fuego Copper Project: NI 43-101 Technical Report & Preliminary Economic Assessment (PEA)" dated August 14, 2023 with an effective date of June 28, 2023. The Mineral Resource estimate update, as described in this Report, does not materially affect the mineral resource inventory that formed the basis of the 2023 PEA and no new scientific or technical information has been generated that would materially affect the outcome of the 2023 PEA and, therefore, the results and conclusions of the 2023 PEA are considered current and have been restated in this Report.

QP Statement: It is the opinion of the Qualified Person that the metal prices and market assumptions are adequate for use in the preliminary economic analysis contained in this Report.

19.1 Metal Prices

Long-term metals prices assumed for the economic analysis are shown in Table 19.1. The long-term price guidance for copper, molybdenum, gold and silver are what Wood considers to be an industry consensus on the forecast and are supported by Wood's quarterly guidance for long-term metal prices.

Wood's guidance for metals pricing considers numerous sources to establish industry consensus metal prices including:

- Current market spot prices as of the start of 29 October 2022
- Three year moving average of metals prices
- Analyst consensus prices reflecting the average forecasted price from multiple reputable banks
- Median price used in most cashflow analyses posted in technical reports on SEDAR within the prior 12 months.

Table 19.1 : Metals Prices

Metal	Unit	Price ($)
Copper	US$/lb	3.85
Molybdenum	US$/lb	17
Gold	US$/oz t	1 750
Silver	US$/oz t	21

19.2 Market Analysis

A rapidly accelerating energy transition supporting a faster uptake of high copper intensive industries, like electric vehicles and renewable energy generation, would increase copper demand globally. The Project is well situated geographically to serve both Asian and North American markets, where China is the leading consumer of copper. Other Asian and American markets are also significant consumers of copper.

The Project copper-gold concentrate is expected to be a clean, high-value concentrate, with low levels of deleterious elements, thus incurring minimal quality penalties and making it sought after by smelters globally. Treatment charges and refining costs (TC/RC) are expected to be around $90/t and $0.09/lb, respectively.

The Project copper concentrate is expected to contain low levels of gold and silver. Two regional structures relating to payment terms exist for gold in copper-gold concentrates. Under a "European" structure, payment for gold and silver typically involves a minimum deduction followed by payment of the balance, while under an "Asian" structure, once the grade is above a specified minimum level, a percentage of all content will be paid. The "Asian" structure applies to the anticipated market for the Project and offers significant benefit to lower gold-grade concentrates, where the comparative payable terms are valued around 15% higher. Refining costs of $5/oz t for gold and $0.50/oz t for silver are expected.

Molybdenum concentrate will be produced as a secondary product by the Project. It is assumed that the copper concentrate and molybdenum concentrate will be sold separately.

19.3 Contracts

HCH has negotiated an offtake agreement with Glencore for early copper concentrate production from the Project in Chile. The Glencore offtake agreement covers 60% of copper concentrate from the Project for a period of eight years from commercial production and was negotiated on arms-length commercially competitive benchmark terms.

Glencore is the largest copper concentrate trader and one of the largest natural resource companies in the world. Glencore is also HCH's largest shareholder, and the offtake agreement rights are subject to a minimum shareholding requirement.

20 Environmental Studies, Permitting and Social or Community Impacts

Some sections, text and figures have been taken directly from the technical report titled, "Costa Fuego Copper Project: NI 43-101 Technical Report & Preliminary Economic Assessment (PEA)" dated August 14, 2023 with an effective date of June 28, 2023. The Mineral Resource estimate update, as described in this Report, does not materially affect the mineral resource inventory that formed the basis of the 2023 PEA and no new scientific or technical information has been generated on the Project that would materially affect the outcome of the 2023 PEA and therefore the results and conclusions of the 2023 PEA are considered current and therefore have been restated in this Report.

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 Environmental Impact Assessment (EIA) of the Project will be submitted for approval using the EIA System that is currently being applied in Chile. Internal study workstreams currently underway include metallurgical and geotechnical test work, as well as environmental baseline studies and financial scenario modelling. This work has outlined the potential for a large-scale, long-life, conventional open-cut and cave mining operation utilising conventional sulphide and oxide processing with strong environmental and social credentials.

The results of these studies are not yet final and will be completed in parallel for the Pre-Feasibility Study (PFS).

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. In addition, diverse environmental information was generated from HCH exploration projects.

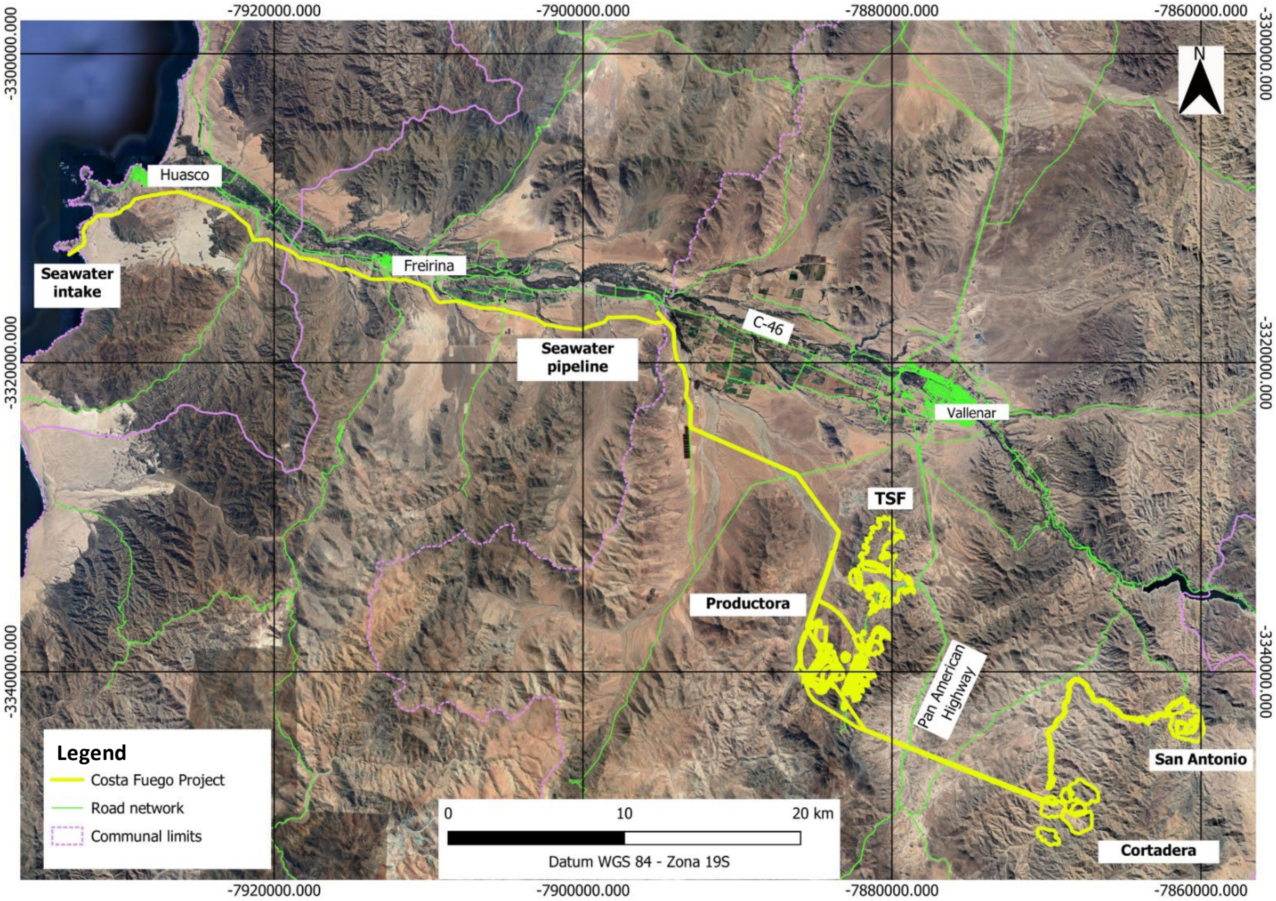
In parallel with the EIA work, three Environmental Impact Statements were submitted to the Authority in 2012, 2013 and 2018 to obtain the license needed for exploration at Productora and similarly in 2021 for Cortadera. These drilling campaigns occurred within mine areas, so the baseline information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

HCH interacts with many regional and local stakeholders who have an interest in the Project. The stakeholder engagement plan is divided in two sections to cover both the consultation of key stakeholders and community development issues. Key stakeholders were identified in the three concerned districts: Vallenar, Freirina and Huasco. The outcomes of this exercise were used as the basis to create a stakeholder engagement plan.

The Project layout covers multiple environment types. Mine areas are locally disturbed by activities such as herding, while the areas covered by the pipeline and powerline are disturbed through interaction with urban areas and other infrastructure projects. The existing plan to bury the pipeline underground avoids additional landscape impact. Impacts in the mine area will be managed via compensation measures, while other works will be managed with more mitigation-focused measures, especially for the power line. Coast area impacts are expected to be minimal, with no brine discharge, and the port operations are located in an area that is already disturbed.

Project layout, including the planned Costa Fuego Project and the existing road network is included in Figure 20.1.

Figure 20.1 : Project Layout and Territorial Information



20.1.1 Auditing

At Productora, five health and safety, and environment (HSE) external audits have been carried out – three on environment and two on health and safety.

At Cortadera, one external environment audit has been completed in 2022, which focused on drilling operations carried out with respect to environmental management on groundwater.

20.1.2 Regulators Inspections

Various regional regulators have visited Productora and Cortadera during HCH's ownership; the Mining Authority has visited the Project four times, focused on legal compliance and safety practices; any gaps identified have been addressed. The Forest Authority also visits frequently, with no issues from their site visits being brought to HCH's attention. The Water Authority has visited Productora and Cortadera on one occasion, focusing on groundwater. No issues were encountered. The Health Authority has also visited the Project twice and the Agriculture Inspection Service once.

20.1.3 Legislation

Since 1997, Chile has adopted an institutional system to prevent environmental degradation caused by project development. The EIA System ensures that investment projects incorporate environmental aspects into their design, construction, operation, and closure stages, to comply with the environmental requirements applicable to them.

There are specific Environmental Protection policies and strategies with environmental aspects contained in several legal bodies, as well as several specific Chilean laws called "Primary Policies", focused on the protection of the environment and human health, such as air quality, water, groundwater, noise pollution, flora and fauna, marine environments, and others. In addition, the National Strategy of Biodiversity focuses on promoting national biodiversity to benefit the wellbeing of future generations.

20.1.4 Permits

The specific environmental permits relevant to the Costa Fuego Project are listed in Table 20.1.

Table 20.1 : Environmental Specific Permits Applicable to the Project

EIA Standard Article (Number, Name of the Permit and Government Agency Delivering the Permit)	Main Requirements EIA Standard – D.S.40	Commentary
132. Permit for making archaeological, anthropological and paleontological excavations National Monuments Council (CMN)	This permit aims to ensure the protection and preservation of the cultural heritage including archaeological monuments and those with anthropological or paleontological value.	Twenty-one findings are classed with high archaeological value; this permit is needed firstly to extend the baseline information of particular sites with further excavations and other techniques, and secondly to allow any removal of sites before the Project is constructed.
135. Permit for the construction and operation of tailing storage facility (TSF) Mining Authority – Sernageomin	This permit aims to ensure the physical and chemical stability of the TSF and its surroundings, in order to protect the environment and people's health.	The Project considers a Tailings storage facility located at Productora.

Table 20.1 : Environmental Specific Permits Applicable to the Project		
EIA Standard Article (Number, Name of the Permit and Government Agency Delivering the Permit)	**Main Requirements EIA Standard – D.S.40**	**Commentary**
136. Permit for waste rock dump or mineral stockpile Mining Authority – Sernageomin	This permit aims to ensure the physical and chemical stability of the waste dump together with the highest safety measures during the construction phase and as long as the dump grows in size; the final aim is to protect the environment and the physical integrity of people.	The Project considers waste rock dumps or mineralised material stockpiles located at Productora, Cortadera and San Antonio.
137. Permit for a mining site closure plan Mining Authority – Sernageomin	This permit aims to ensure the physical and chemical stability of the mining area, so as to protect life, people's health and the environment.	Measures to ensure the protection of people's health and safety, as well as the environment, must be detailed in the closure plan. As there may be potential for acid generation in the waste material, it is important to demonstrate adequate neutralisation capacity through an appropriate waste rock management plan
151. Permit for cutting, destroying and grubbing xerophytic formations Forest Authority – CONAF	This permit aims to protect biodiversity.	Xerophytic plants communities are common in the Project area. In the mine area the associated infrastructure covers about 1 300 ha and xerophytic plants in this area need to be removed. Permit to remove those plants is critical to allow the construction of infrastructure.
157. Permit to build civil works to protect or arrange natural riverbeds Water Management Authority – DGA	The permit aims to not affect the life or people's health, by avoiding any significant changes in surface run-off and natural erosion processes of the riverbed as well as avoiding water pollution.	The Project infrastructure will cross approximately 45 riverbeds or intermittent watercourses, mostly related to the linear works (pipeline, powerline) as well as any channels to carry the tailings. Different civil works will need to be constructed to protect the riverbeds and the infrastructure. Those works can go underground or on surface, and this will depend on the physical situation of the intersection point crossing the riverbed. Well-developed engineering for several intersection points is needed to obtain permission approval.

20.1.5 Baseline and supporting studies

The main findings in the Project area refer to flora and fauna species under conservation status, archaeology findings with high value and the presence of some families living at the Project.

The outcome of the assessment process identified four significant impacts that require special attention. Those impacts are briefly described below:

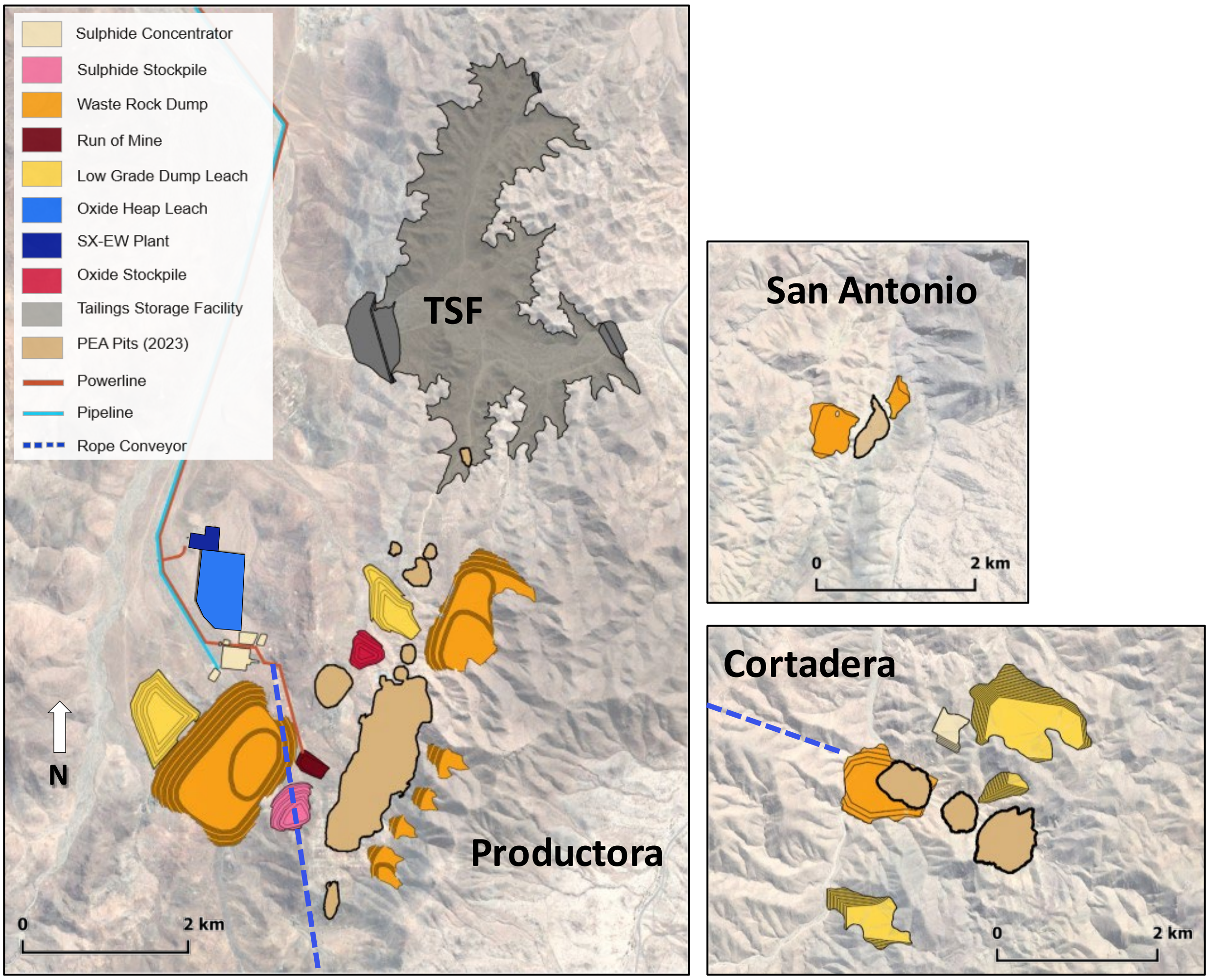
- Alteration and/or loss of terrestrial flora and vegetation
- Alteration and/or loss of habitat for fauna
- Significant changes in the livelihood of human groups
- Alteration of archaeological sites with high value.

20.2 Waste Management

Waste at the Project will be managed in accordance with national regulations (DS 594, DS 148).

Waste rock dump platforms will be located at Productora, Cortadera and San Antonio (Figure 20.2) and will not be required to be covered with any waterproof membranes, soil or other elements to prevent the entry of water, because the infrastructure will have systems to intercept and manage water run-off. The final platform of the dump will have a smooth slope to facilitate a controlled run-off of water.

Figure 20.2 : Planned and Existing Infrastructure Across the Costa Fuego Project, Showing the Location of the Waste Rock Dumps at Each Deposit



In terms of wind erosion, once the machinery and trucks have ceased working on the dump platform, the material will not be disturbed, so the surfaces will not be exposed to wind erosion. In addition, the average wind speed is very low at the area (10 km/hr) so there is not a high risk of wind erosion.

Initial testwork on the potential for acid mine drainage (AMD) was completed on select Productora drill holes in 2015. The results indicated almost 66% of samples had some neutralisation potential that is likely associated with the presence of carbonate minerals. An update to this AMD was completed during 2022 for Productora and Cortadera, testing 80 rock samples. Further tests will be executed on San Antonio during 2024.

Further discussion of waste and stockpile storage plans at a PEA stage of development are described in Section 18.5.

20.2.1 Tailings Storage Facility

Further discussion of the Tailings Storage Facility plans at a PEA stage of development are described in Section 18.8.

Studies evaluating wind erosion control measures from the basin and walls will be required. This will consider the dried condition of tailings and its material consolidation because of water evaporation.

The tailings surface could be capped with a suitable cover of local borrow and waste rock to prevent erosion of the tailings surface and reduce infiltration.

Studies evaluating seepage control measures from the basin will be required. Two hydrogeological monitoring wells have been constructed, one upstream and one downstream of the TSF embankments. A monthly monitoring program gathers groundwater data, including water level and water quality, with more than 30 elements tested.

20.2.1.1 Risk Management and Mitigation

A risk of tailings spill is present, due to breakage of the transport channel or tailings distribution system, dam collapse, saturation of dam walls, or damage of pumping system, all resulting either from operational and/or mechanical failures.

For the transport and deposition of tailings, a transport channel will be used to feed the discharge system of the reservoir. In case of a significant leakage in transporting the tailings from the plant to the TSF, the following general actions shall be taken:

- Stoppage of the entire tailings transportation system
- A specialised team will immediately address and clean the spill area
- All measures to confine the spill will be taken to ensure no contamination of water sources
- Any spilled tailings will be removed with heavy machinery and disposed in a storage area or directly at their original destination (TSF). The tailings accumulated in the storage area will be loaded into trucks and reincorporated into the process feed
- Any failure in the transportation system will be repaired and additional precautionary measures taken if necessary
- Situation will be communicated immediately to the authorities
- An investigation will be undertaken and monitoring conducted in the area to assess the effects that may arise over time.

Risk mitigation also considers preventive measures:

- Prior to the commencement of operations, tests with water or other innocuous fluids will be conducted to verify potential leakages
- Frequent inspections of the transportation system will be conducted, especially after an earthquake or landslide to assess the integrity of the works and retaining structures.

To prevent the collapse of the TSF wall, the following measures will be considered:

- During construction, a quality assurance/quality control (QA/QC) program will be put in place to meet design criteria for embankment.

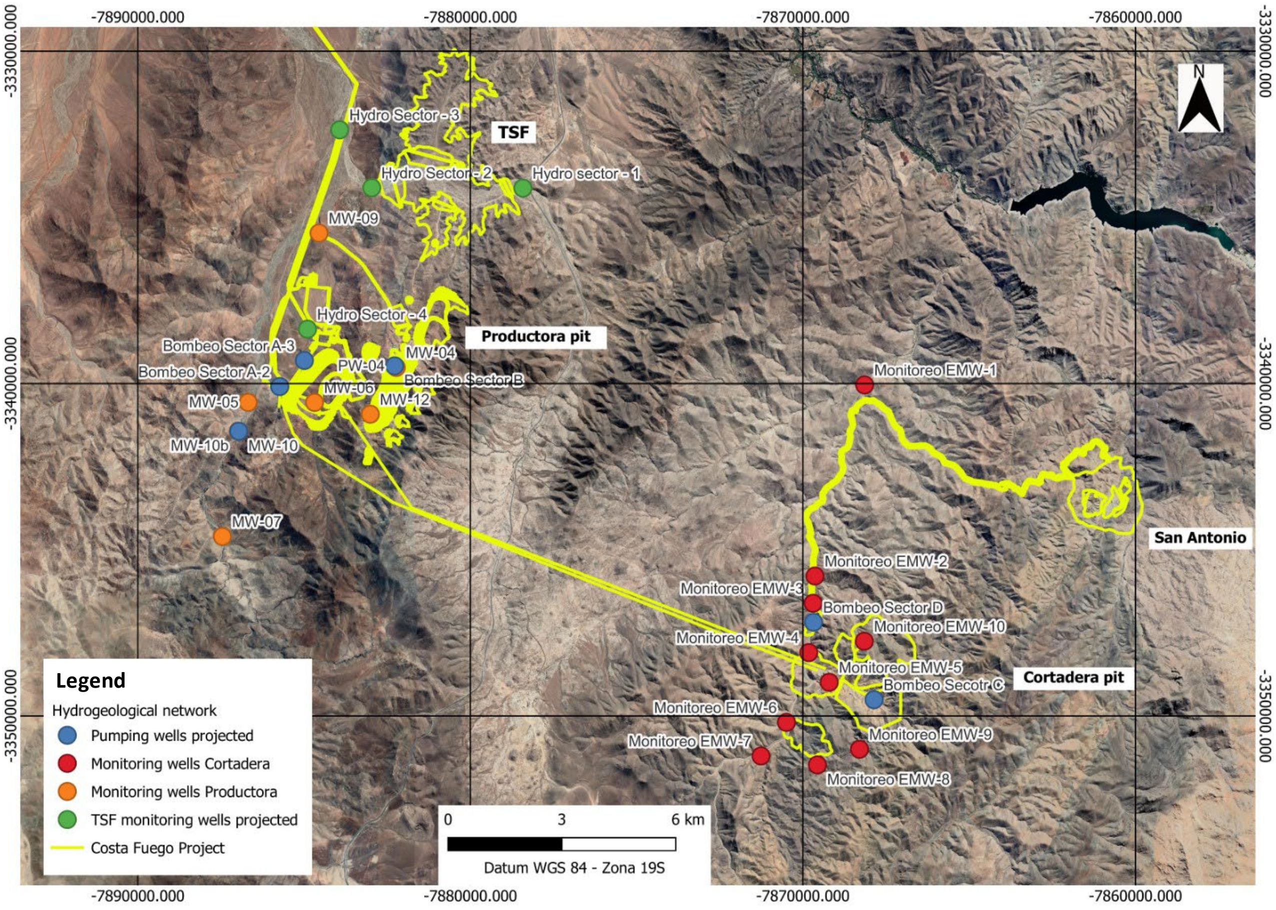
The static and pseudo-dynamic stability of the final design of the embankment will be analysed and earthquake resistance criteria will be applied.

20.3 Water Management

The operation will utilise raw seawater for processing, with water extraction (maritime concession) and coastal land access rights already secured. The use of seawater reduces the energy intensity of the Project (no large-scale desalination plant required) and preserves the limited groundwater resources available in the region.

The hydrogeological baseline study is being prepared for the PFS to address water management at the Project. A hydrogeological bore network has been in place at Productora gathering information since 2012, and development of a similar network at Cortadera and additional bore holes at Productora is complete, with monitoring ongoing.

Figure 20.3 : Network of Monitoring Wells Surrounding Costa Fuego

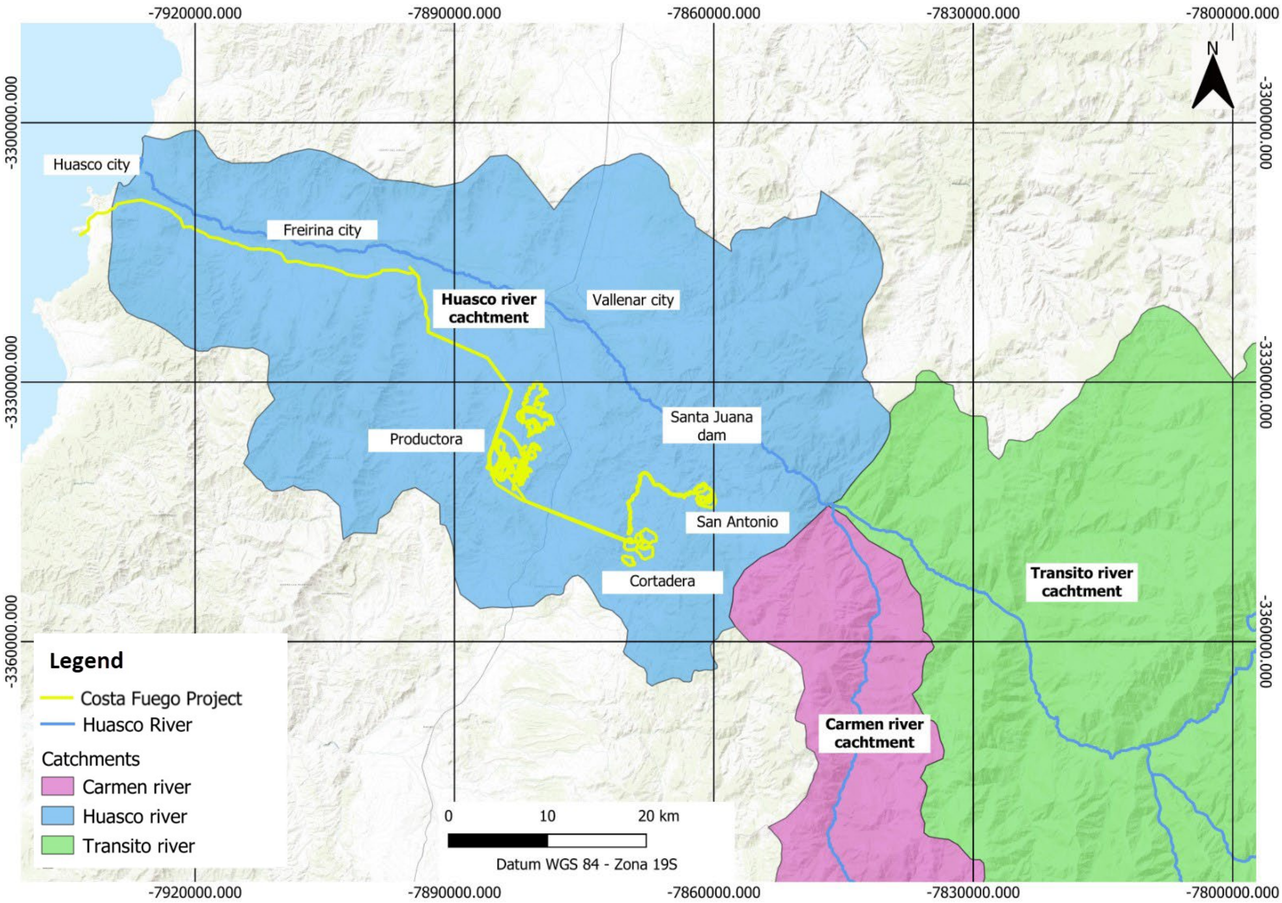


Information from the network at Productora and Cortadera suggests the potential to generate run-off or groundwater recharge in the catchment is very small. As a result, the valley floor remains dry all year-round with ephemeral (intermittent) stream flows only occurring following sporadic storm events.

The Productora Project area is located within the watershed of the Quebrada Maitencillo, while the Cortadera Project area lies within the Quebrada Camarones watershed, which joins the Rio Huasco downstream of the Santa Juana water dam.

The Cortadera and Productora Projects are located within the Río Huasco catchment, a regional watershed that stretches from the border with Argentina in the Andean Cordillera to the Pacific coastline. The two rivers which form this watershed have continuous flows due to the discharge of the snow melt during the spring and summer seasons. Their run-off is retained in the Santa Juana water dam located approximately 16 km upstream of Vallenar (Figure 20.4).

Figure 20.4 : Regional Río Huasco Watershed

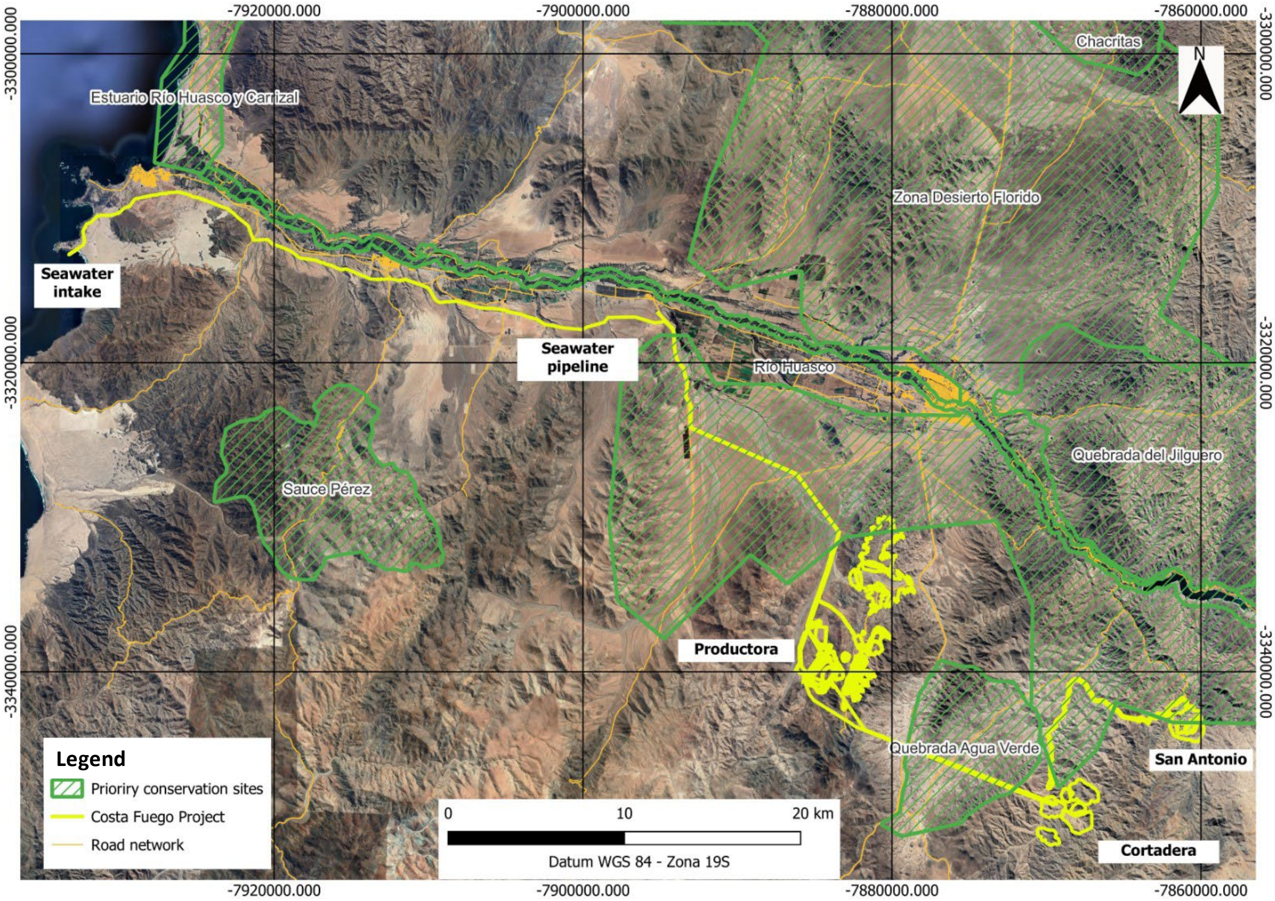


Further discussion of water usage and supply at a PEA stage of development are described in Section 18.7.

20.4 Priority Sites for Biodiversity Conservation

The Project is located within the boundaries of, and nearby, two priority sites for biodiversity conservation at the regional level, namely Desierto Florido, and Quebrada Agua Verde, as shown in Figure 20.5. These priority sites are not an issue for the development of the Project, as they are not high-level protected areas, such as National Parks.

Figure 20.5 : Biodiversity Sites

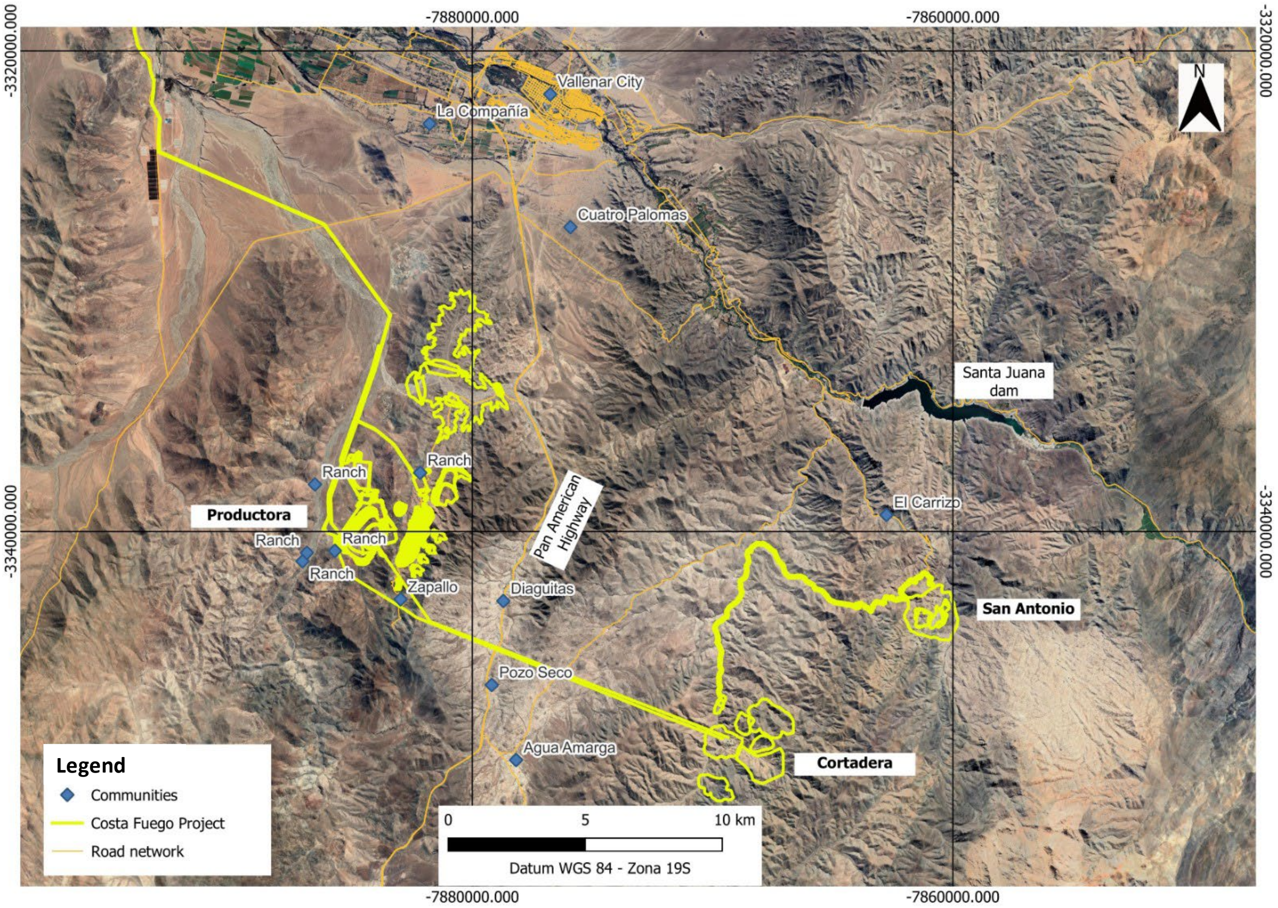


20.4.1 Communities

In relation to the non-indigenous population, there are some ranches located near Productora (see Figure 20.6), linked to livestock and small agriculture. Also, there are some villages near Cortadera such as Agua Amarga and Pozo Seco, both related to the abandoned train railway.

The indigenous communities of the study area are part of the Diaguita ethnic group, represented by a territorial indigenous council named Concejo Territorial Indígena Diaguita. A number of meetings have been carried out with indigenous council since 2022, with the main purpose to develop a strong long-term partnership with transparency and continuous information about the Project advance.

Figure 20.6 : Communities in the Costa Fuego Project Area



20.5 Closure Plan

All mining projects in Chile which have a production capacity above 10 000 t/month are required to present an official closure plan to be approved by the Mining Authority. Any associated infrastructure affected by a closure plan also needs to be developed to scoping engineering level.

The closure plan will specify the technical measures and activities at Costa Fuego designed to prevent, minimise, or control risks that may affect people and the environment. In Chile, the law governing the closure of mining sites (Law 20.551 and DS 41) does not require mining operators to rehabilitate the site where the mine infrastructure and facilities used to be located. However, a rehabilitation plan will be assessed during closure discussions.

The Costa Fuego Closure plan will primarily address:

- Risk assessment methodology used for the closure of site infrastructure
- Description of the environment
- Life of mine remediation and closure activities

- Description of mine infrastructure, including closure commitments
- Cost of closure measures
- Any required post-closure programs
- Financial guarantees
- Stakeholder communication program.

20.5.1 Closure Planning

The Closure Plan will be developed as part of the EIA, with the focus the return of the landscape to its pre-mining capability where possible.

The main closure items will include:

- Management of water as drawdown occurs within the heap leach facilities
- Placement of a cover on the heap leach facilities where possible
- Deconstruct aspects of the infrastructure that are not required post- mine closure
- Burying non-hazardous materials in the pit or waste storage facility
- Removal of hazardous waste to an acceptable off-site location
- Closing mine and TSF access and roads and install warning signs to restrict access of personnel and vehicles, and to prevent unsafe access
- Carrying out long term TSF, waste and site monitoring, and managing water post mine closure.

20.6 Social and Community Requirements

Over the past decade the Company has forged strong community engagement (including with indigenous communities) and contributed positively to the region (Figure 20.7). The Company has and will continue to:

- Provide ongoing support for the local regional communities through the Company's mental health program
- Recruit locally, wherever possible, to provide employment and training opportunities
- Preferentially procure local goods and services
- Provide ongoing support for two orphanages in Freirina and Vallenar
- Provide fresh water to local families in Agua Amarga for irrigation
- The Company is recognised as a leader in ongoing social support programs within the Vallenar and Huasco Valley region.

Figure 20.7 : Community Engagement in Huasco Valley - Meeting with Indigenous Community (Left) and Delivery of Water Tank to Indigenous Community (Right)

20.7 Stakeholder Engagement Plan

The key aspect of engagement at this stage of the Project is to make sure the EIA considers the opinion of stakeholders through a robust consultation process. Any issue that is indirectly related to the EIA will be addressed once the baselines, the impact assessment and the mitigation measures have been discussed with stakeholders.

Once the anticipated consultation activities with the relevant stakeholders are carried out, the Project will be well positioned for the EIA official and mandatory engagement process, which follows EIA submission.

20.8 Comments on Section 20

In Chile the environmental evaluation processes is lengthy and, on some occasions, challenging for non-technical reasons. The usual preparation duration is one year, followed by approximately two years of evaluation (depending on level of engagement carried out with communities prior to EIA submission).

Government agencies in the Atacama Region have experience dealing with mining projects, specifically the environmental and sectorial processes.

The use of seawater processing to preserve limited regional groundwater resources and minimisation of the environmental footprint by leveraging off existing infrastructure (port, power and roads) are significant positives for the Project.

Local and indigenous communities have experience negotiating with mining and energy projects on the Huasco valley, and they know which specific environmental impacts these types of projects will bring over the catchments and local landscape and biodiversity. The continuation of the early engagement program with local communities is highly recommended.

21 Capital and Operating Costs

This section has been taken directly from the technical report titled, "Costa Fuego Copper Project: NI 43-101 Technical Report & Preliminary Economic Assessment (PEA) dated August 14, 2023 with an effective date of June 28, 2023". The Mineral Resource estimate update, as described in this Report, does not materially affect the mineral resource inventory that formed the basis of the 2023 PEA and no new scientific or technical information has been generated that would materially affect the outcome of the 2023 PEA and, therefore the results and conclusions of the 2023 PEA are considered current and have been restated in this Report.

21.1 Introduction

As part of the 2016 study for HCH, Mintrex estimated the capital (Capex) and operating (Opex) costs for the proposed Productora operation. The 2016 estimate was based on a mixture of EPCM, EPC and EP approaches. The capital cost estimate was prepared to a level equivalent of that for a PEA study and was presented in US dollars as of the fourth quarter 2015 (4Q15) to an accuracy level of +/- 30%.

Wood was commissioned by HCH to update the previous study estimates to a base date of fourth quarter 2022 and to provide order of magnitude pricing for additional scope items and the process plant throughput increase from the 2016 study 14 Mt/a plant throughput to the current PEA 20 Mt/a plant throughput.

The indicative cost increase of the estimate from the study base date of 2016 is 20.4% which represents an average escalation rate of 2.69% per annum.

The estimates were updated by:

- Application of escalation factors by commodity
- Leveraging off Wood's Santiago database to understand local cost rates
- Updates in key input costs such as acid, power and diesel
- Updating of base estimates by application of scaling factors; reflecting the throughput increase from the 2016 study of 14 Mt/a to the current PEA 20 Mt/a plant throughput
- Developing order of magnitude costs from engineering reports developed for additional scope and process plant throughput
- Reviewing pricing assumptions with the Wood project team and HCH management.

The accuracy of the estimates contained within this PEA is estimated to be +/-40%

21.2 Capital Cost Estimates (2016 Study @ 14 Mt/a)

21.2.1 Introduction 2016 Study

The 2016 estimate was based on a mixture of EPCM, EPC and EP approaches. For the EPCM portions of work, the owner assumed the construction risk; therefore, the capital estimate did not include a constructor margin. For the EPC and EP portions of work, the contractor assumed the construction risk; therefore, the constructor's margin was assumed to be within the embedded contingency of the estimate. The willingness of contractors to assume the risk within this margin will depend upon the market at the time.

The capital cost estimate was presented in US dollars as of the fourth quarter 2015 (4Q15) to an accuracy level of +/- 30%.

21.2.2 Summary 2016 Study Capital Cost

The 2016 capital cost estimate for the Project, including contingency is summarised in Table 21.1.

Table 21.1 : 2016 Capital Cost Summary

Area	Sub-Total (US$M)	Contingency (US$M)	Total (US$M)
Area 01 Bulk Earthworks and Drainage	24.6	6.2	30.8
Area 02 Site Services	1.9	0.2	2.1
Area 03 Sulphide Process	179.2	20.5	199.6
Area 04 Oxide Process	61.9	6.3	68.2
Area 05 Molybdenum Process	7.7	0.8	8.5
Area 07 Infrastructure	115.7	16.1	131.8
Area 09 Mining	89.2	3.60	92.8
TOTAL DIRECT COSTS	**480.2**	**53.5**	**533.8**
Area 06 EPCM Costs	62.2	15.6	77.8
Area 08 Owners Costs	53.4	13.3	66.7
Area 10 Working Capital	50.00	-	50.00
TOTAL INDIRECT COSTS	**165.6**	**28.9**	**194.5**
TOTAL PROJECT COSTS	**645.8**	**82.4**	**728.2**

21.2.3 Basis of Estimate 2016 Study

21.2.3.1 Introduction

The estimate basis applies to the estimation of capital costs prepared by Mintrex for the process plants and associated infrastructure and notes where the inputs have been provided by others for incorporation into the estimate. The estimate has been prepared as described in the following sections.

21.2.3.2 Installation Unit Rates

HCH provided unit rates to be used in the capital estimate. The rates were developed by a combination of contractor quotes, benchmarking by HCH and international experience and were reviewed by Sedgman, an international engineering contractor with an office in Santiago.

All in contractor rates were developed for use in the 2016 study estimate. The all in “gang rates” are inclusive of all of the contractor costs, excluding mobilisation and demobilisation, including;

- Offsite corporate overheads
- Site facilities
- Project management
- Project supervision

- Training
- Indirect labour
- Direct Labour
- Plant and equipment
- Consumables
- Non-Productive time
- Accommodation and daily messing
- Contractors' overheads.

Gang rates were estimated for SMP, Piping and Electrical and are applied within the estimate with both a labour component and an equipment component. In addition, where pricing was provided by third parties as components of a fixed price the gang rates have been used to back calculate man-hours.

The unit rates provided included for

- Bulk earthworks
- Concrete supply and install
- Labour rate (part of the built-up all-inclusive gang rate)
- Equipment rate (part of the built-up all-inclusive gang rate)
- Local freight rates
- Structural steel installation hours/unit
- Platework installation hours/unit.

21.2.3.3 Freight

The inland freight estimate was based on information from HCH, quoted rates and estimated quantities. The estimate also included budget pricing from potential suppliers for freight in the following areas:

- Sea freight of 20 ft and 40 ft containers to Antofagasta port in Chile from Asia, Australia and Europe (including port charges). The ocean freight cost assumed 50% of mechanical equipment would be sourced from Europe and 50% would be sourced from Asia.
- An average all in freight cost of US$1300 /t was applied to the mechanical equipment on the basis of 5% of the purchase value.
- The ocean freight cost for structural steel and platework assumed the fabricated steel and platework would be sourced from either Asia or Europe.
- A unit cost per tonne for freight was developed and this was incorporated into the supply rate.

For all other items, freight was applied in the estimate on the following basis:

- Piping as a fixed proportion of the material supply costs
- Electrical as a fixed proportion of the electrical supply costs

- Freight costs for SAG mill and ball mills to site were quoted by Outotec.

21.2.3.4 Bulk Earthworks, Roads and Drainage

Earthworks costs for the Project were estimated as follows:

- Preliminary quantities (material take-offs, MTOs) were estimated based on the Project layout, client supplied survey data and preliminary designs for earthworks in the sulphide process plants, heap leach crushing plant and seawater storage ponds.
- Assumptions were made regarding geotechnical ground conditions based both on site visual inspection by Mintrex and a preliminary Geotechnical Report prepared by Ingeroc. The Ingeroc investigation program included a number of test pits at the heap leach pad area and at the original proposed plant site. The revised plant site location was assumed to have similar geotechnical ground conditions to the adjacent heap leach pad area.
- Heap leach pad earthworks MTOs were estimated by Knight Piésold, based on their preliminary design and the geotechnical report above.
- Access and internal road costs were estimated by the application of a US$/km applied to the road lengths and the type of road construction required. All roads are assumed to be unsealed.
- Unit rates for bulk earthworks provided by HCH and Knight Piésold were applied to the estimated quantities including an allowance for contractor overheads.

21.2.3.5 Sulphide Process Plant

21.2.3.5.1 Concrete Work - Supply and Install

Concrete quantities were established using a combination of the layout drawings prepared for the study, previous similar detailed designs and Mintrex's database of quantities from previous projects. Concrete installation rates were supplied by HCH as all-inclusive rates based on benchmark data of Chilean market for the various categories of concrete estimated.

The all-inclusive concrete unit rates include allowances for:

- Concrete supply and installation
- Blending or lean mix concrete
- Reinforcement
- Detailed excavation and backfill
- Holding down bolts and embedments
- Formwork
- Contractors direct and indirect costs.

The concrete quantities noted above include an additional seismic location allowance of 30% and the reinforcement quantities include an additional 100% relative to the low seismic design conditions used for design in Australia. The factors used were determined by taking a range of existing detailed designs and applying the seismic conditions associated with the Project location and local Chilean seismic loads. Concrete volumes and reinforcement requirements were then established for a range of structures i.e. columns, pedestals, walls etc. and an average established to apply across all quantities.

21.2.3.5.2 Structural Steel and Platework – Supply and Fabrication

Structural steel quantities including light, medium, heavy, grating, handrails, wear liners, etc. were established using a combination of the layout drawings prepared for the study, previous similar detailed designs and Mintrex's database of quantities from previous projects. Supply and fabrication rates were based on responses to requests for budget pricing received from three fabricators (one Asian, one European and one Chile based), HCH and Mintrex's database.

The structural steel quantities in the estimate included a seismic allowance of +5% for heavy steelwork (>75 kg/m) and +10% for medium steelwork (25 to 75 kg/m). The same approach adopted to establish seismic factors for concrete, see Section 21.2.3.3, were applied to establish factors to apply to steelwork quantities.

21.2.3.5.3 Mechanical – Supply

The mechanical equipment was sized to reflect the process design criteria and the plant throughput. The information was collated to prepare the mechanical equipment list for the study. Equipment data sheets (EDS) were prepared for major equipment, and these were issued to equipment vendors for budget pricing. Budget pricing was provided in a number of currencies, predominantly US$, AU$ and EUR, and have been input into the estimate in the currency they were quoted.

Approximately 90% of the mechanical equipment was subject to budget pricing for this study and includes the following:

- Primary crusher and pebble crusher
- Rock breaker
- Apron feeder
- Vibrating feeders and screens
- Belt feeders
- Grinding mills (SAG mill, ball mills and regrind mill)
- SAG mill discharge screen
- Ball charging magnet and magnet hoist
- Mill liner handlers
- Hydrocyclones
- Flotation cells
- Multi stream analyser
- Slurry pumps
- Thickeners
- Concentrate filter

- Air compressors
- Flocculant mixing plant
- Reverse osmosis plant.

The equipment pricing is all for new equipment except for the SAG and ball mills which were based on second hand unused equipment of which there are a number of mills currently available worldwide that would be suitable for the duty.

The balance of equipment pricing was based on recent project procurement pricing.

21.2.3.5.4 Structural and Mechanical Installation

The installation for the sulphide process plant, erection of all principal supplied structural steel, plate-work and mechanical equipment was calculated on benchmark man-hours for mechanical equipment and tonnes for structural steel and platework.

The study has adopted the following for the cost estimate:

- Structural steel man-hours as estimated by Mintrex from previous projects increased by use of a labour multiplier of 2.3 relative to the baseline hours
- Platework man-hours as estimated by Mintrex from previous projects increased by use of a labour multiplier of 2.3 relative to the baseline hours
- Mechanical installation man-hours as estimated by Mintrex and increased for local conditions by use of the same labour multiplier of 2.3 relative to the baseline hours
- The productivity factor of 2.3 has been established by direct comparison with quoted manhours for projects in Australia versus the quoted manhours in Chile.

21.2.3.5.5 Piping – Supply and Install

The supply and installation estimate for process plant piping was based on mechanical equipment supply price and compared with costs from previous projects in the Mintrex database. A fixed percentage of 30% of the mechanical supply costs was adopted for all areas of the process plant. This was distributed internally within the estimate to 40% supply, 52% installation and 8% freight.

The application of the same percentage across all areas of the plant is a reasonable approach for this 2016 study, a more detailed approach is required in subsequent studies.

The overland pipelines for tailings and decant return were estimated from first principles with engineered sizing, quantity take-offs and application of unit rates for supply and installation.

Installation hours were calculated from the installation costs and the piping gang rate for labour, equipment and overheads.

21.2.3.5.6 Electrical and Instrumentation

The low voltage electrical and instrumentation supply and installation estimate was prepared on a factored basis. A factor of 50% of mechanical supply costs was applied, benchmarked on previous capital estimates. This was distributed internally within the estimate at 50% supply, 45% installation and 5% freight.

Installation hours were calculated from the installation costs and the electrical gang rate for labour, equipment and overheads.

21.2.3.6 Oxide Plant

21.2.3.6.1 SX/EW Plant

A scope of work for the provision of pricing associated with the design, supply and construction of a Solvent Extraction and Electrowinning (SX/EW) plant including the required tank farm was developed and forwarded to three internationally recognised vendors to provide a lump sum design and construct (EPC) quote for the estimate. In addition, a quotation for the engineering design and equipment supply was received from a Chilean vendor.

The EPC quotations included design, supply, installation and commissioning of the complete SX/EW plant with the battery limits as follows:

- Bulk earthworks by others
- Flange on the incoming PLS pipeline at the SX plant
- Flange on the outgoing raffinate pipeline at the SX plant
- Medium Voltage (MV) and Low Voltage (LV) power at the outgoing terminals of the transformers
- Potable water supply pipeline.

The scope of supply specified a plant with a low level of automation.

Three complete budget quotes were received; the cost included in the estimate was based on the BGRIMM quotation. BGRIMM, a Chinese Engineering Contractor, has constructed a 10 000 t/a SX/EW plant for the Kounrad project in Kazakhstan and have an office in Santiago.

An additional allowance of US$2.0M was included for dedicated fire systems around the SX/EW plant.

21.2.3.6.2 Modular Crushing and Screening Plant

The budget estimate was based on a Turnkey Design and Supply quotation for a complete crushing and screening plant based on standard modules. Additional costs were included for agglomeration based on supplier quotations. Installation hours were estimated based on the equipment list, the Mintrex baseline installation hours and the labour multiplier of 2.0 adopted in the sulphide plant estimate.

Concrete quantities were based on a take-off for a plant design developed by Mintrex prior to the decision to proceed with a modular style plant.

21.2.3.6.3 Heap Leach Stacking Plant

The estimate for the heap leach conveying and stacking plant was developed based on vendor budget pricing and MTO quantities. Concrete, steel, platework quantities and installation hours has adopted the unit rates agreed for the Project.

21.2.3.6.4 Heap Leach Pads

The estimate for the heap leach pads was based on preliminary design including earthworks quantities by Knight Piésold, piping MTOs by Mintrex and an allowance for concrete.

21.2.3.7 Molybdenum Process Plant

21.2.3.7.1 Concrete Work - Supply and Install

Concrete quantities were established using a combination of the layout drawings prepared for the study, previous similar detailed designs and Mintrex's database of quantities from previous projects. Concrete installation rates were supplied by HCH as all-inclusive rates for the various categories of concrete estimated.

21.2.3.7.2 Structural Steel and Platework – Supply and Fabrication

Structural steel quantities including light, medium, heavy, grating, handrails, wear liners, etc. were established using a combination of the layout drawings prepared for the study, previous similar detailed designs and Mintrex's database of quantities from previous projects. Supply and fabrication rates were based on responses to requests for budget pricing received from three fabricators (one Asian, one European and one Chile based), HCH and Mintrex's database.

21.2.3.7.3 Mechanical – Supply

The mechanical equipment was sized to reflect the process design criteria and the plant throughput. The information was collated to prepare the mechanical equipment list for the study. Equipment data for major equipment were issued to equipment vendors for budget pricing. Approximately 90% of the molybdenum plant equipment was subject to budget pricing for this study and includes the following:

- Flotation cells and columns
- Thickeners.

21.2.3.7.4 Structural and Mechanical Installation

The installation for the molybdenum process plant, erection of all principal supplied structural steel, plate-work and mechanical equipment was calculated on benchmark man-hours for mechanical equipment and tonnes for structural steel and platework.

The study has adopted the following for the cost estimate:

- Structural steel man-hours as estimated by Mintrex from previous projects increased by use of a labour multiplier of 2.3 relative to the baseline hours
- Platework man-hours as estimated by Mintrex from previous projects increased by use of a labour multiplier of 2.3 relative to the baseline hours

- Mechanical installation man-hours as estimated by Mintrex and increased for local conditions by use of the same labour multiplier of 2.3 relative to the baseline hours.

21.2.3.7.5 Piping – Supply and Install

The supply and installation estimate for process plant piping was based on mechanical equipment supply price and compared with costs from previous projects in the Mintrex database. A fixed percentage of 30% of the mechanical supply costs was adopted for all areas of the process plant. This was distributed internally within the estimate to 40% supply, 52% installation and 8% freight.

Installation hours were calculated from the installation costs; and the project gang rate for labour, equipment and overheads.

21.2.3.7.6 Electrical and Instrumentation

The low voltage electrical and instrumentation supply and installation estimate was prepared on a factored basis. A factor of 50% of mechanical supply costs was applied benchmarked on previous capital estimates. This was distributed internally within the estimate at 50% supply, 45% installation and 5% freight.

Installation hours were calculated from the installation costs and the project gang rate for labour, equipment and overheads.

21.2.3.8 Project Infrastructure

21.2.3.8.1 Introduction

Project miscellaneous infrastructure estimates were prepared upon the following basis:

- Budget quotations based on rates received from suppliers and contractors for quantities specified for works detailed on drawings
- Quotations from single supplier quotations for specific items required
- MTOs on earthworks, concrete, structural steel, platework, mechanical, piping or electrical where required
- Allowances for items, for which no quotation was received, based on experience on similar projects.

21.2.3.8.2 Seawater Transfer System

A scope of work for the provision of pricing associated with the construction of the Seawater Transfer System was developed and issued to two experienced contractors to provide a lump sum price for the construction only of the pipeline and pumping stations. The installation scope provided the details of the preliminary pipeline design including pipeline quantities and pump station sizing.

- Seawater pipeline supply and transport pricing was based on a Chilean-based supplier quote based on design specifications.
- Seawater transfer pumps based on supplier quote as per EDS supplied by Mintrex.

Two quotations were received for the seawater and pump station installation with one being for a complete design, supply and installation of the entire system and the second being a heavily qualified quote for part of the installation component only. The quote for the complete system installation was adopted for the estimate and the independent price for supply of the pipeline materials and the pumps was not included. Subsequent to the receipt of the quotations, the design of the transfer system was modified to remove one pump station and to increase the pipeline size from 550 mm to 600 mm. The Acciona price was factored to reflect the changes.

The quotation from Acciona assumes 70% of the pipeline alignment can be free dug and 30% requires either drill or blast or rock breaker to allow trench excavation. Based on the recent Ingeroc geotechnical site investigation report, this is likely to be a conservative assumption. Acciona allowed an additional $79.50/m of pipeline, i.e. $333.90 vs. $254.41/m for the 30% that was estimated to require drill and blast.

Changes were made to the Acciona estimate due to the late design changes, i.e. the removal of one pump station and the shortening of the overall pipeline.

21.2.3.8.3 HV and MV Electrical Infrastructure

Following the development of the project electrical load list and an overall project single line diagram, a capital estimate was prepared for the following components;

- 220 kV and 23 kV substations
- 220 kV powerline
- 23 kV powerlines
- HV/MV switch rooms, switchboards and transformers.

The budget pricing was based on equipment quotations, MTOs and unit pricing for transmission lines.

21.2.3.8.4 Infrastructure Area Buildings

The design, supply and install estimate for plant buildings was based on budget pricing from Chilean transportable building suppliers based on preliminary specifications, including floor space.

All-inclusive unit rates per m^2 for the different building types were incorporated in the estimate.
Additional allowances were made for workshop tools and equipment, and office furniture and equipment.

21.2.3.8.5 Communications Systems

Pricing for site communications systems was based on a nominal allowance based on previous project experience.

21.2.3.8.6 Sewerage Systems

Pricing for site sewerage systems was based on budget pricing for the supply of the treatment plants and suitable allowances for installation based on prior project experience. System sizing was completed by vendors based on the personnel numbers provided.

21.2.3.8.7 Tailings Storage Facility

Knight Piésold conducted a detailed study into the requirements for a TSF at the Productora site. The cost associated with the construction of the TSF included:

- Preliminary costs
- Material supply
- Site preparation
- Embankment construction
- Embankment drainage system
- Decant return system
- Miscellaneous
- South embankment (for containment at the end of operations)
- Closure.

The construction cost estimate for each of the TSF components (listed above) was based on a preliminary design with earthworks quantities and unit rates provided by Knight Piésold. Knight Piésold estimated the initial capital cost and the ongoing cost of the lifts required on an annual or biannual basis. The ongoing cost (sustaining capital) has been incorporated into the financial modelling on the basis of a cost per tonne of process feed.

21.2.3.8.8 Pipelines

Pipeline costs include transport to Puerto Las Losas, supply, install and an allowance for miscellaneous items such as valves and fittings. The cost of pipelines was based on MTO quantities, vendor budget pricing and previous project rates, except for HDPE pipeline supply and install rates estimated by HCH. Major pipelines costed in the estimate include:

- Tailings slurry 600 mm diameter steel pipeline from tailings thickener to TSF
- Mine water, 250 mm diameter HDPE pipeline from sulphide plant to mine water storage tank
- Decant water return, combination 250 mm diameter steel pipeline and 225 mm diameter HDPE pipeline from decant tower to sulphide plant process water tank
- Raffinate solution transfer and ILS transfer 400 mm diameter HDPE pipelines
- Raffinate solution irrigation and ILS irrigation 110 mm diameter HDPE pipelines
- Potable water 225 mm diameter HDPE pipeline for SX/EW plant from the sulphide plant
- Potable water 63 mm diameter HDPE pipeline to the mine potable water storage tank from the sulphide plant.

21.2.3.9 Vehicles and Mobile Equipment

The cost of vehicles and mobile equipment to be used in the process plants was benchmarked on similar equipment purchased for another project in Chile in 2015.

All mobile equipment is allowed for as new other than the 250 t crawler crane which was included at a nominal 40% of new price.

21.2.3.10 EPCM Costs

The detailed engineering design estimate and engineering man-hours were factored based on past experience. The EPCM cost used in the capital cost estimate has been calculated as a variable percentage of the process plant and infrastructure direct cost. The EPCM estimate includes the following components:

- Engineering and procurement
- Construction management
- Commissioning
- Vendors
- Site investigation
- Testing
- Survey.

The total EPCM cost, including contingency, is approximately 15% of the Project direct costs including mining and 17% of the total direct costs excluding mining. Nominal percentages used for the process plant EPCM are 7% design, 8% construction management and 1% for commissioning. Variable percentages are used for infrastructure based on the assumed contracting method.

21.2.3.11 Spares

Pricing for plant mechanical, piping, electrical and instrumentation spares was factored based on a percentage of the overall mechanical and electrical supply costs. A nominal 5% was applied to both.

21.2.3.12 First Fills

Pricing for first fills was developed based on reagent usage rates (as per design criteria) and pricing (supplied by HCH) and details for the sulphide plant and oxide plant as follows.

21.2.3.12.1 Sulphide Plant

- Grinding media first fill quantities for the SAG mill, ball mills and regrind mill includes an initial ball charge plus one month of stock of SAG mill and ball mills media and regrind mill ceramic media.
- Sulphide plant flotation reagents first fill quantities were based on one month’s stock.
- Molybdenum sulphide plant flotation reagents first fill quantities were based on one month’s stock.
- Flocculant first fill quantities were based on one month’s stock.
- First fill quantities for other consumables (e.g. mill lubricants and RO plant chemicals) were based on one month consumption.

21.2.3.12.2 Oxide Plant

- Oxide plant front end lubricants first fill quantities were based on one month’s stock.
- Sulphuric acid addition first fill quantities were based on 1000 m^3 (1840 t) of storage and 170 t of first fills for the EW circuit. The 1840 t of storage was equivalent to two weeks consumption at the nominal 15 kg/t

consumption rate; and SX/EW plant reagents first fill quantities were based on initial fill requirements plus one month's stock.

21.2.3.13 Construction Overheads

21.2.3.13.1 Temporary Water Supply

An allowance was included for the supply of construction water. An approximate quantity of water was estimated at 1.9 Mm^3 of water at $1.83/$m^3$ to support the earthworks and pre-strip requirements of the Project. The unit rate per m^3 was provided from a recent project in Chile.

21.2.3.13.2 General

The construction overheads estimate for the Owner's components of the Project was estimated at 1.25% of total project direct costs, excluding mining, and represent cost items such as:

- Mobilisation and demobilisation of other than EPCM and EPC contractors and sub-contractors
- Travel costs for other than EPCM and EPC contractors and sub-contractors
- Accommodation and food costs for other than the EPCM and EPC contractors and sub-contractors
- Construction building and temporary facilities and services (water, power, communications, and sewerage).

21.2.3.14 Owner's Costs

Owner's costs were based on estimates and allowances (mostly time-based costs) supplied by HCH. Excluding the construction overheads estimate for the Owner's component of the estimate), Owner's costs include the following:

- Capital spares
- Insurances
- Pre-production labour
- Pre-production expenses
- Business systems
- Environmental and Social
- Legal.

Included in the Community Relations costs under Environmental is an allowance for community infrastructure as the Project assumes that all construction personnel will be housed in existing accommodation in Vallenar and Huasco. Consequently, no costs were included in the estimate for the installation and operation of construction camp facilities. Included in the estimate was a $3.0 M allowance for the supply of sustainability and other social license related programs to support the construction personnel during the construction phase.

21.2.3.15 Mining

The mining capital cost estimate was built through a combination of information contained on the mining contractor quote, fuel supplier quote, benchmark data or allowances for some other minor items.

21.2.3.15.1 Mining Pre-Strip

The mining pre-strip capital estimate was developed working through the mine pre-production plan and unit rates available on the mining contractor quote. In addition, an Owner´s team of up to 42 people was allocated to manage pre-stripping activities.

Mining contractor pre-stripping activities included drilling, blasting, loading, hauling, ancillary equipment, general administration and overheads for administration, safety and technical management. These activities were quoted on unit rate per tonne basis with the exception of overheads which was quoted on monthly basis. Indirect cost elements like transport, food, etc. were also factored on unit rates.

Mining contractor mobilisation, mine infrastructure and initial mine roads were estimated as separate items and allocated on mining infrastructure area.

HCH´s Owner team covered personnel for managing production, safety and technical planning activities during the entire pre-stripping period. This item also included allowances for indirect costs (food, transport, etc.) and temporary facilities.

21.2.3.15.2 Mining Infrastructure

Mining infrastructure will include the following elements:

- Heavy vehicle workshop
- Washdown pad
- Administration offices
- Dining room and change room
- Tyre management area
- Warehouse
- Oil waste management facility
- Diesel storage and supply facility
- Explosive magazine
- Seawater storage and water truck charge system
- Potable water and sewage treatment plant
- Pit dewatering and surface water management system.

From the list above, the heavy vehicle workshop, washdown pad, administration office, dining room and change room, tyre management area, warehouse and oil waste management facility were included in the mining contractor quote.

Diesel storage and supply facility capital estimate was based on a quote from a Chilean supplier. This included tanks to store a capacity of 100 m^3 approximately, separate heavy vehicle and light vehicle diesel supply stations and mobile trucks for delivering diesel supply for those machineries requiring diesel charge at the working areas.

Explosive magazine capital estimate was based on benchmark data for similar projects in Chile. Seawater storage and water truck system was based on a combination of quantity estimate and unit rate for earthworks plus allowances for piping and pumping. Potable water supply and sewage treatment plant were estimated based on escalated actual cost incurred to build existing facilities available at Productora site plus some allowances for minor items.

Capital estimate of infrastructure for pit dewatering and surface water management was based on infrastructure items recommended by Artois Consulting through an analysis of underground water and collection of natural and contact water. A high-level summary of the recommendations is as follows:

- Dewatering of Productora pit will commence only after the pit has reached 100 m deep. This should happen only after second year of production.
- Some infrastructure is required for the collection of contact water generated in the open pits, waste rock dumps and production feed stockpiles.
- Some infrastructure is required for the diversion of natural (non-contact) water to prevent flooding and minimise the impact on water resource in the catchment.

Based on the recommendations above, deep in-pit dewatering wells and sub-horizontal pit wall drainage holes for Productora pit dewatering capital were allocated to sustaining capital and it is not part of the initial capital estimate. Infrastructure for collection of contact water and natural water included water diversion channels and water ponds, the cost estimate for this infrastructure was allocated as part of the initial capital estimate.

21.2.3.16 Working Capital

The working capital estimate provides for funding to support the operation until production levels allow the operation to be cash flow positive. Based on the assumed project ramp up curves, this has been estimated at 4.0 months. The working capital estimate includes the following components:

- General and administration
- Sulphide processing
- Oxide processing
- Mining
- Production credit for months 1-3.

21.2.3.17 Exchange Rates 2016 Study

The exchange rates applied to the capital estimate were provided by HCH. They represent a 90- day average over the period August to October 2015.

Table 21.2 : Capital Cost Estimate Exchange Rates	
Currency	**United States Dollar (US$)**
Australian Dollar (AU$)	0.71800
Canadian Dollar (CA$)	0.76000
Euro (EUR)	1.12000
Chilean Peso (CLP)	0.00145

21.2.3.18 Contingency 2016 Study

The purpose of contingency is to make specific provision for uncertain elements of cost within the Project scope and thereby reduce the risk of cost overrun to a pre-determined acceptable level. Contingencies do not include allowances for scope changes, escalation, or exchange rate fluctuations but rather for assumptions, omissions and other uncertainties that affect the estimate accuracy.

Contingency reflects the measure of the level of uncertainties related to the scope of work. It is an integral part of an estimate and has been applied to all parts of the estimate, i.e. direct costs and indirect costs.

A variable level of contingency has been assigned by assessing the level of confidence in each of the inputs to the estimate, i.e. engineering, estimate basis, and vendor/contractor information.

The contingency allocation in the estimate of US$82.4 M (13%) has been built up as shown in Table 21.3.

Table 21.3 : Variable Contingency Distribution		
Contingency	**Materials**	**Installation**
Overheads	15%	25%
Bulk Earthworks	25%	25%
Concrete	10%	20%
Structural Steel	10%	10%
Platework	20%	20%
Mechanical Equipment	5%	20%
Electrical and Instrumentation	5%	20%
Piping	10%	20%
High Voltage Power Supply	10%	20%
EPCM	25%	25%
Owner's Costs	25%	25%
Mining (excluding pre-strip)	20%	15%
Solvent Extraction and Electrowinning (EPC)	5%	20%
Seawater Transfer System (EPC)	5%	25%
Heap Leach and Tailings Storage Facility	25%	25%
Working Capital	0%	0%
Freight	15%	15%

In establishing the variable contingency table as defined in Table 21.3, an assessment was made by the 2016 study team of the following;

- Level of testwork completed and variability of the results
- Level of engineering that has been completed in each area
- Confidence in both equipment sizing and pricing
- Whether estimate is based on quantities and rates or factors
- Current market conditions
- International construction experience
- Historical areas of cost growth i.e. earthworks, Owners' costs, EPCM etc.

Site costs have a higher contingency applied than supply costs.

21.2.3.19 Clarifications 2016 Study

The following clarifications apply to the cost estimates:

- The capital cost estimate was based on the project implementation strategy
- The estimate was based on wage rates as provided by HCH and the expected site safety regulations and work practices
- The capital cost of the laboratory facility was excluded from the capital estimate because laboratory services were assumed to be provided by an external supplier
- The capital cost of the fuel storage system was assumed to be provided by the preferred fuel provider
- It was assumed that sufficient manpower resources will be available in Chile to undertake the Project in the timescale envisaged
- It was assumed that there was sufficient accommodation available in Vallenar and Huasco to meet the staffing requirements of the workforce during both the construction and operation phases of the Project
- It was assumed that there will be 10.5 productive hours per day in the normal 12 working hours per day shift.

21.2.3.20 Exclusions 2016 Study

The following exclusions apply to the cost estimates. No allowance was made for:

- Escalation of prices
- VAT or GST
- Financing costs or interest
- Import duty for capital items
- Currency exchange rate variations
- Owner's sunk costs prior to project implementation.

21.3 Operating Cost Estimates (2016 Study @ 14 Mt/a)

21.3.1 Introduction 2016 Study

21.3.1.1 Operating Cost Basis of Estimate

Operating costs were estimated for the mining, sulphide and oxide plant processing, administration, concentrate transport and seawater supply areas. The operating costs were estimated from a variety of sources, including:

- First principal estimates
- Mine plan developed by NCL for HCH
- Power ($0.095/kWh), diesel ($0.64/L) and labour unit rates provided by HCH
- Metallurgical testwork results
- Sulphide plant crushing circuit consumables cost provided by Metso
- SAG mill and ball mill liner cost and regrind ceramic media consumption rate and cost provided by Outotec
- Grinding media and liner consumption rates provided by DMCC Pty Ltd
- Grinding media and reagent costs provided by HCH
- Concentrate transport and handling and general transport costs provided by HCH
- Laboratory costs provided by HCH
- Heap leach pad sustaining capital costs provided by Knight Piésold
- Labour unit rates provided by HCH.

21.3.1.2 Operating Costs Summary and Details 2016 Study

The 2016 Study operating cost estimates are in the following sections of the report.

All costs are presented in US dollars and are based on prices for the third quarter of 2015.

Exchange rate assumptions used in the estimate are as follows:

- Australian dollar (AUD) to US dollar (US$) 0.71800
- Canadian dollar (CAD) to US$ 0.76000
- Euro (EUR) to US$ 1.12000
- Chilean peso (CLP) to US$ 0.00145 (1 US$: 689.7 CLP).

21.3.1.3 Qualifications and Exclusions

Qualifications and exclusions relating to the operating costs include the following:

- Rehabilitation costs have been excluded
- Corporate overheads/head office costs have been included in general and administration (G&A) costs
- Project financing or interest charges have been excluded

- Escalation of operating costs has been excluded
- Ongoing exploration costs have been excluded
- Concentrate and cathode storage and handling (including ship loading) costs at port and transport from site to port costs have been included
- Concentrate and cathode shipping costs from the port have been excluded, but they will be considered in the financial analysis as part of selling costs
- Reagent and consumable costs have been quoted as delivered to site.

21.3.2 Mining Operating Costs 2016 Study

21.3.2.1 Summary

Mining operation will be carried out by a mining contractor. HCH requested a quote from a suitable mining contractor with previous experience working on similar operations in Chile. The quote includes most of the operational activities, namely, drilling, blasting, loading, hauling, ancillary equipment, maintenance and general administration. Productora and Alice open pits mining cost is based on this quote plus some additional costs for minor items not included by the quote, e.g. HCH supervision, blast hole sampling, RC drilling for production feed control, seawater supply and pit dewatering.

Several scheduling scenarios were tested with peak material movement of between 80 Mt/a and 110 Mt/a, depending upon the low-grade production feed stockpiling capacity being used. The peak period duration ranged between 4 to 5 years and was followed by an evenly declining material movement over another 4 to 5 years, to a final stockpile production feed processing period of 0 to 2 years. The schedule with the best financial performance was selected from the scenarios tested. The selected mining schedule has a range of material movement between 43 Mt/a and 89 Mt/a excluding pre-stripping and stockpile production feed processing periods, the average material movement of this production schedule is 66 Mt/a. A summary of the operating cost estimate for the selected mining schedule is presented in Table 21.4. This cost estimate is based on the utilisation of a mining contractor to perform mining production activities.

Table 21.4 : Mining Cost Summary (Excluding Pre-Strip and Re-Handling)

Area	Material Mined ($/t)
Mining contractor	1.89
HCH mining staff	0.03
Blast hole sampling	0.01
RC drilling	0.01
Pit dewatering	0.00
Seawater supply	0.01
TOTAL	**1.95**

21.3.2.2 Mining Contractor Operating Cost

The mining contractor would carry out the majority of mining related activities. The scope of works considered by the quote includes drilling, blasting, loading, hauling, ancillary equipment, production feed re-handling and general administration costs. The operating cost also includes the component related to mobile equipment capital cost depreciation.

An establishment capital including mobilisation, construction of key mining infrastructure and initial mining roads was allocated as part of the capital estimate.

Diesel price variation had a significant effect on the mining cost, particularly the mining contractor rates. The mining contractor quote supplied an adjustment formula to identify which key cost inputs would affect the mining costs and the expected amount of influence of each input. The adjustment formula identified that the contract cost includes an 18% exposure to diesel price and an 8% exposure to the CLP exchange rate.

21.3.2.3 HCH Mining Staff

HCH allocated a mining staff team to conduct resource estimate, mine planning, production feed controlling, mining cost control, safety and production management. The total staff allocated is 43 people.

21.3.2.4 Blast Holes Sampling Cost

Blast hole sampling is necessary for grade control purposes. The blast hole will be sampled to test for copper, gold and molybdenum to define the sulphide process feed, transitional process feed, oxide process feed and waste.

HCH personnel would conduct the sample collection and deliver samples to an external company to do the sample preparation and laboratory analysis in Vallenar.

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.

The estimated operating cost of US$0.5 M per year is 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.

21.3.2.5 Reverse Circulation Drilling Cost for Process Feed Control

Due to the singularities of the Productora deposit, it is contemplated to use the assistance of RC drilling to support the modelling of resources for the short term. This activity is very important to identify the production feed to be mined up to three months in advance.

The estimated annual cost of US$0.8 M was based on a drilling pattern of 40 m x 80 m, excluding pre-stripping and stockpile process feed processing periods.

21.3.2.6 Pit Dewatering Cost

The Productora main pit will require pit dewatering to drain underground water and surface water coming from surrounding areas of the pit, as well as rainwater. The average annual cost of pit dewatering is US$0.1 M.

This estimate was based on an average inflow rate of 13 L/s from the second year of operations and excludes the stockpile process feed processing period.

21.3.2.7 Seawater Supply for Road Watering

Seawater supply will be used for road watering activities to reduce dust generation. In the mine area, the seawater will be stored in storage pond. The water available will be loaded into water trucks which will execute the road watering in the mine area.

The unit rate for the seawater supply is US$0.51/m^3, thus, the average annual cost estimated for seawater supply in the mine area is US$0.4 M.

21.3.3 Processing and Maintenance 2016 Study

21.3.3.1 Copper and Molybdenum Sulphide

Processing and maintenance cost estimates were developed using the plant parameters and production feed characteristics specified in the process design criteria. The sulphide treatment plant has a design capacity of 14 Mt/a of production feed and will produce nominally 264 kt/a of copper concentrate and 2240 t/a of molybdenum concentrate.

A sulphide plant availability of 91.3% or 8000 hours/year is used, taking into consideration downtime due to scheduled and unscheduled maintenance; this has been utilised in the process design criteria.

ROM pad costs, including rehandling production feed, have been included within the mining costs.

The average LOM copper sulphide plant processing and maintenance operating cost was $5.96/t material processed.

The average LOM molybdenum sulphide plant processing and maintenance operating costs was $5.68/t Mo throughput.

21.3.3.1.1 Power Costs

The cost of power for developing the plant operating costs was $0.095/kWh.

21.3.3.1.2 Reagents and Consumables Costs

Gyratory and pebble crusher liners (i.e. bowls, mantles, concaves) consumption rate was based on Metso simulations. Crusher consumables pricing has also been provided by the Metso (budget quote);

- SAG mill and ball mill grinding media and SAG mill and ball mill liners consumption rates were based on an estimate by DMCC Pty Ltd
- SAG mill and ball mill grinding media pricing was provided by supplier quotes
- SAG mill and ball mill wear liner pricing was provided by Outotec
- Regrind mill grinding media consumption rate and pricing were provided by Outotec
- Reagent consumptions in the copper and molybdenum flotation circuits were calculated from testwork results and from typical consumption rates from other operations. All reagent consumption rates were included in the process design criteria.

21.3.3.1.3 Mobile Vehicles Costs

Light vehicles will be provided and will include the following:

- Dual cab utilities (4WD)
- Land Cruisers (4WD).

Medium/heavy vehicles will be provided and will include the following:

- Ambulance
- 30-seater bus
- Forklifts
- Bobcat
- Integrated tool carrier
- Cranes
- Hiab trucks
- Tip trucks.

A summary of operating cost estimates for mobile vehicles in the copper and molybdenum sulphide plants is presented in Table 21.5.

Table 21.5 : Sulphide Plant Mobile Vehicles Cost Summary 2016

Area	Process Feed ($/t)
Light vehicles	0.01
Medium/heavy vehicles	0.05
TOTAL	**0.06**

21.3.3.1.4 Maintenance Costs

Various allowances per plant area have been estimated for maintenance and materials cost for the processing plants. This ranges from as low as 2.0% for the reagent area to a maximum of 5.0% for the milling and classification area.

A summary of cost estimates for maintenance in the copper and molybdenum sulphide plants is presented in Table 21.6.

Table 21.6 : Sulphide Plant Maintenance Cost Summary 2016

Area	Process Feed ($/t)	Mo Plant Throughput ($/t)
Copper sulphide plant maintenance	0.60	-
Infrastructure	0.07	-
SUBTOTAL – Copper sulphide plant	0.67	-
Molybdenum sulphide plant maintenance	-	1.01
TOTAL	**0.67**	**1.01**

21.3.3.1.5 Labour Costs

The personnel and salaries were based on HCH advice and experience and benchmarking of other mining companies with similar sized operations. Adjustments were made, where necessary, to reflect the availability of likely skills requirement and the operating environment.

A summary of cost estimates for labour in the copper sulphide plant is presented in Table 21.7.

Table 21.7 : Copper Sulphide Plant Labour Cost Summary 2016		
Area	**No. of Personnel**	**Copper Sulphide Plant Throughput ($/t)**
Processing	43	0.16
Plant Maintenance	49	0.14
Plant Technical Services	6	0.03
TOTAL	**98**	**0.33**

A summary of cost estimates for labour in the molybdenum sulphide plant is presented in Table 21.8.

Table 21.8 : Molybdenum Sulphide Plant Labour Cost Summary 2016		
Area	**No. of Personnel**	**Mo Plant Throughput ($/t)**
Processing	9	1.49
Plant Maintenance	6	0.84
TOTAL	**15**	**2.33**

21.3.3.1.6 Laboratory Costs

An estimate of the number and type of grade control, environmental, sulphide and oxide plant, metallurgical and concentrate dispatch samples was prepared and an annual estimate of $856k ($0.06/t processed) was made for laboratory costs in the copper and molybdenum sulphide plants, as shown in Table 21.9. The pricing was based on a quote from ALS for a laboratory based in Vallenar.

Table 21.9 : Sulphide Plant Laboratory Cost Summary 2016	
Area	**Material Processed ($/t)**
Laboratory	0.06
TOTAL	**0.06**

21.3.3.1.7 Concentrate Transport Costs

Concentrate transport costs were estimated based on the following:

- Copper concentrate haul distance from site to Las Losas port of 64 km
- Copper concentrate transport to port unit cost of $0.17 per tonne per km
- Port storage and handling of copper concentrate (including ship loading) unit cost of $10.90 per tonne
- Copper concentrate sampling unit cost of $0.39 per tonne of concentrate

- Molybdenum concentrate haul distance from site to smelter in Santiago region of 650 km
- Molybdenum concentrate transport to smelter unit cost of $0.20 per tonne per km (includes an allowance for concentrate sampling, and storage and handling at smelter).

A summary of cost estimates for concentrate transport from site to port (copper)/smelter (molybdenum) is presented in Table 21.10.

Table 21.10 : Concentrate Transport Labour Cost Summary 2016

Area	Copper Concentrate ($/t)	Molybdenum Plant Throughput ($/t)
Copper Concentrate	22.17	-
Molybdenum Concentrate	-	1.11
TOTAL	**22.17**	**1.11**

21.3.3.2 Copper Oxide

Processing and maintenance cost estimates were developed using the plant parameters and production feed characteristics specified in the process design criteria. The copper oxide treatment plant has a design capacity of 3.3 Mt/a of production feed and will produce 10 000 t/a of copper cathode.

Oxide crushing/agglomeration availability of 75% (6570 hours/year) and SX/EW plant availability of 95% (8322 hours/year) has taken into consideration downtime due to scheduled and unscheduled maintenance which has been utilised in the process design criteria.

ROM pad costs, including rehandling production feed, have been included within the mining costs.

A summary of the copper oxide treatment plant processing and maintenance operating costs is shown in Table 21.11

Table 21.11 : Copper Oxide Plant Operating Cost Summary 2016

Area	Material Processed ($/t)	Cu Cathode ($/t)
Power	0.34	-
Reagents and consumables	2.28	-
Maintenance	0.25	-
Water	0.21	-
Vehicles	0.06	-
Labour	1.01	-
SUBTOTAL – Front end	**4.16**	**-**
Power	-	239.10
Reagents and consumables	-	127.90
Maintenance	-	73.20
Cathode transport and port handling	-	23.70
Laboratory	-	7.45
SUBTOTAL – Back end	**-**	**471.20**
TOTAL	**4.16**	**471.20**

21.3.3.2.1 Power Costs

The cost of power, as advised by HCH, for developing plant operating costs was $0.095/kWh.

21.3.3.2.2 Reagents and Consumables Costs 2016 Study

In the plant front end, the consumption rates of crusher consumables (i.e. bowls, mantles) and screen, feeder and agglomerator consumables were based on allowances/assumptions by Mintrex. All consumables pricing were based on allowances/assumptions by Mintrex.

In the plant back end, typically reagent consumption rates were allowances/assumptions by Mintrex, with the exception of the sulphuric acid consumption rate which was based on testwork, and the bentonite consumption rate which was based on vendor information. All reagent pricing was provided by HCH, generally on the basis of in-country quotes.

Acid pricing was based on a benchmark price of $80/t FCA (Free Carrier) from four alternative points of supply and an allowance at $0.11/t/km for transport to Productora. The four potential points of supply are Chañaral, Salvador, Cachiyuyo and Paipote with transport distances ranging from 50 km to 386 km. An average distance of 230 km at $0.11/t/km was adopted giving an average transport price of $25.50/t and a total delivered acid price of $105.50/t.

All reagent consumption rates were included in the process design criteria. A summary of cost estimates for reagents and consumables in the copper oxide plant is presented in Table 21.12.

Table 21.12 : Copper Oxide Plant Reagents and Consumables Cost Summary 2016

Area	Material Processed ($/t)	Cu Cathode ($/t)
Front end consumables	0.47	-
Front end reagents	1.81	-
SUBTOTAL – Front end	2.28	-
Back-end reagents	-	127.90
SUBTOTAL – Back end	-	127.90
TOTAL	**2.28**	**127.90**

21.3.3.2.3 Mobile Vehicles Costs

Light vehicles for the copper oxide plant will be provided and will include the following:

- Dual cab utilities (4WD)
- Land Cruisers (4WD).

Medium/heavy vehicles will be provided and will include the following:

- 30-seater bus
- Bobcat
- Hiab truck

- Tip truck.

A summary of cost estimates for mobile vehicles in the copper oxide plant is presented in Table 21.13.

Table 21.13 : Copper Oxide Plant Mobile Vehicles Cost Summary 2016	
Area	**Material Processed ($/t)**
Light vehicles	0.03
Medium/heavy vehicles	0.03
TOTAL	**0.06**

21.3.3.2.4 Maintenance Costs

Various allowances per plant area were estimated for maintenance and materials cost for the copper oxide plant. An allowance of 2.4% of capital cost was made in the plant front end (crushing and agglomeration/stacking). An allowance of 2.0% of capital cost was made in the plant back end (SX/EW and infrastructure).

A summary of cost estimates for maintenance in the copper oxide plant is presented in Table 21.14.

Table 21.14 : Copper Oxide Plant Maintenance Cost Summary 2016		
Area	**Material Processed ($/t)**	**Cu Cathode ($/t)**
Front end	0.25	-
Back end	-	73.20
TOTAL	**-**	**-**

21.3.3.2.5 Labour Costs

The personnel and salaries were based on HCH advice and experience and benchmarking of other mining companies with similar sized operations. Adjustments were made, where necessary, to reflect the availability of likely skills requirement and the operating environment.

A summary of cost estimates for labour in the copper oxide plants is presented in Table 21.15.

Table 21.15 : Copper Oxide Plant Labour Cost Summary 2016		
Area	**No. of Personnel**	**Material Processed ($/t)**
Processing	44	0.64
Plant Maintenance	30	0.37
TOTAL	**74**	**1.01**

21.3.3.2.6 Cathode Transport Costs

Copper cathode transport costs were estimated based on the following:

- Haul distance from site to port of 64 km
- Copper cathode transport to port unit cost of $0.20 per tonne per km
- Port storage and handling unit cost of $10.90 per tonne.

A summary of cost estimates for copper cathode transport is presented in Table 21.16.

Table 21.16 : Copper Cathode Transport Cost Summary 2016	
Area	**Cu Cathode ($/t)**
Copper cathode	23.70
TOTAL	**23.70**

21.3.3.2.7 Laboratory Costs

An estimate of the number and type of grade control, environmental, sulphide and oxide plant, metallurgical and concentrate dispatch samples was prepared and an allowance of $7.45/t copper cathode for oxide production feed was made. This cost did not include the blast hole samples which was allocated to the mining cost.

21.3.3.3 Infrastructure Operating Cost 2016 Study

21.3.3.3.1 Seawater Supply 2016 Study

Operational and maintenance cost estimates were developed using the plant parameters specified in the process design criteria. The seawater supply system has a design annual consumption of 10.22 Mm^3 for usage in the sulphide plant, oxide plant and the mine. Seawater pipeline operational availability will be 95% (8322 hr/yr).

A summary of the seawater supply and storage operating costs is presented in Table 21.17.

Table 21.17 : Seawater Supply and Storage Cost Summary 2016	
Area	**Cost ($/m^3)**
Power	0.27
Mobile vehicles	0.01
Maintenance	0.16
Labour	0.07
TOTAL	**0.51**

21.3.3.3.2 Power Costs

The cost of power, as advised by HCH, for developing seawater supply operating costs is $0.095/kWh.
The following loadings were used as the basis of costing:

Seawater transfer system

- Installed power 6.56 MW
- Average 24-hour maximum demand 3.29 MW
- Energy consumption 28.9 GWh/yr.

A summary of the seawater system power costs is shown in Table 21.18.

Table 21.18 : Seawater Supply Power Cost Summary 2016	
Area	**Cost ($/m³)**
Seawater supply power	0.27
TOTAL	**0.27**

21.3.3.3.3 Mobile Vehicles Costs

Light vehicles for the Seawater Supply area will be provided and will include the following:

- Dual cab utilities (4WD).

There will be no additional heavy vehicles required.

A summary of cost estimates for the sea water supply mobile vehicles is presented in Table 21.19.

Table 21.19 : Seawater Supply Mobile Vehicles Cost Summary 2016	
Area	**Cost ($/m³)**
Light vehicles	0.01
TOTAL	**0.01**

21.3.3.3.4 Maintenance Costs

An allowance of 3.0% of capital cost was estimated for maintenance and materials cost for the seawater supply and storage infrastructure. A summary of cost estimates for seawater supply maintenance is presented in Table 21.20.

Table 21.20 : Seawater Supply Maintenance Cost Summary 2016	
Area	**Cost ($/m³)**
Seawater supply maintenance	0.16
TOTAL	**0.16**

21.3.3.3.5 Labour Costs

The personnel and salaries were based on HCH advice and experience and benchmarking of other mining companies with similar sized operations. Adjustments were made, where necessary, to reflect the availability of likely skills requirement and the operating environment.

A summary of cost estimates for the seawater supply and storage infrastructure labour is presented in Table 21.21.

Table 21.21 : Seawater Supply Labour Cost Summary 2016		
Area	**No. of personnel**	**Cost ($/m³)**
Seawater supply labour	16	0.07
TOTAL	**16**	**0.07**

21.3.3.4 General and Administrative Operating Cost 2016 Study

General and administration costs were estimated on an annual basis and have been prepared on the basis of both Mintrex and HCH experience and a detailed build up to labour numbers and costs. The cost summary is presented in Table 21.22.

Table 21.22 : General and Administration Cost Summary 2016

Area	Cost per Material Processed ($/t)
Staff	0.16
Corporate	0.09
Mine site offices	0.08
Maintenance (roads and buildings)	0.03
Insurances	0.11
Financial	0.01
Government charges	0.03
Consultants	0.01
Personnel	0.11
Contracts	0.04
Community	0.02
Environment	0.02
Power (buildings)	0.01
TOTAL	**0.77**

21.3.3.5 Sustaining Costs 2016 Study

Allowances as a percentage of the capital cost were made for sustaining costs in all areas of the processing plants, except for the TSF and the heap leach pad, where amounts have been provided by Knight Piésold. Overall sustaining costs are shown in Table 21.23.

Table 21.23 : Sustaining Cost Summary 2016

Area	Allowance (%)	Material Processed ($/t)	Cathode ($/t)	Mo Plant Throughput ($/t)
Copper Concentrator	1.0	0.18	-	-
Molybdenum Concentrator	1.0	-	-	0.39
Tailings Storage Facility	-	0.46	-	-
Copper Oxide – front end	1.0	0.12	-	-
Copper Oxide – back end	1.0	0.13	42.06	-
Copper Oxide – heap leach pad	-	0.94	-	-
TOTAL		**1.83**	**42.06**	**0.39**

21.3.4 Operating Cost Summary 2016

The total annual operating costs for the Project, with average mining of 66 Mt/a, treating 14 Mt/a copper/molybdenum sulphide production feed and 3.3 Mt/a copper oxide production feed, and producing 263 kt/a of copper concentrate, 2240 t/a of molybdenum concentrate and 10 kt/a of copper cathode, are shown in Table 21.24.

Table 21.24 : Total Operating Cost Summary 2016			
Area	**Material Processed ($/t)**	**Cu Cathode ($/t)**	**Mo Plant Throughput ($/t)**
Mining excluding production feed re-handling	9.14	-	-
Copper sulphide plant	5.96	-	-
Molybdenum concentrator	-	-	5.68
General and administration	0.77	-	-
Concentrate transport	0.42	-	-
Copper oxide – front end	4.16	-	-
Copper oxide – back end	-	471.20	-
TOTAL	**-**	**-**	**-**

21.4 Estimate Update for 2023 PEA

21.4.1 Estimate Updates for the PEA

21.4.1.1 Introduction

Wood has been commissioned by HCH to update the 2016 study estimates to a base date of fourth quarter 2022 and to provide order of magnitude pricing for additional scope items and the process plant throughput increase from the 2016 study 14 Mt/a sulphide plant throughput to the current PEA 20 Mt/a sulphide plant throughput, whilst the oxide plant throughput targets an output of 10 ktpa initially, increasing to 12 ktpa later in the mine life.

The Project processing plant will be fed from a number of deposits. The process feed will be supplied directly to the main plant at Productora, conveyed via rope conveyor from a crushing station located at Cortadera which is linked to the main processing plant at Productora (approximately 15 km from Cortadera), and trucked from San Antonio.

As part of the 2016 study for HCH, Mintrex estimated the capital (Capex) and operating (Opex) costs for the Productora operation at 14 Mt/a for the sulphide plant and 3.3 Mt/a for the oxide plant. The crushing station and associated rope conveyor system at Cortadera was not included in the 2016 study.

21.4.1.2 Methodology

The estimates were updated by:

- Application of escalation factors by commodity
- Leveraging off Wood's Santiago database to understand local cost rates
- Updates in key input costs such as acid, power and diesel
- Updating of the base estimates by application of escalation and throughput factors; reflecting the throughput increase from the 2016 study of 14 Mt/a to the current PEA 20 Mt/a plant throughput

- Developing order of magnitude costs from engineering reports developed for additional scope and process plant throughput
- Update to major mechanical equipment pricing due to increase of throughput for sulphide plant, minor mechanical equipment updated by scaling factor
- Reviewing pricing assumptions with the Wood project team and HCH management.

21.4.1.3 Exclusions and Qualifications

There have been no changes to the 2016 engineering basis for the main processing plant at Productora.

Estimates for Cortadera infrastructure (rope conveyor, crushing station and other infrastructure) are based on recent third-party engineering reports and quotations.

21.4.2 Updated Capital Cost Estimate

21.4.2.1 Basis of Estimate

The capital cost estimate meets the requirements consistent with AACE® International cost estimating guidelines for a Class 4 estimate. The estimate accuracy range of ±25% is defined by the level of project definition, amount of engineering inputs, the time available to prepare the estimate and the amount of project cost data available.

The capital cost estimate is based on Q4 2022 assumptions, in US dollars.

The estimate has been compiled by Wood with inputs from consultants for their responsible scope of work:

- ABGM Plus: Mine production, mine footprint and decline development, ventilation, dewatering and underground infrastructure, underground crushing and materials handling
- Knight Piésold: Heap leach, tailings storage, surface water management
- Wood: Process plants, surface infrastructure, tailings pipelines, power supply and distribution, services and utilities, and concentrate pipelines based on the 2016 Study by Mintrex
- HCH: Owners costs
- All: Indirect costs, EPCM
- Wood: Contingency allowance.

21.4.2.2 Updated Capex Estimate Summary

The objective of this review is to provide HCH with capital costs (+/-40% accuracy) to update the estimate base date to fourth quarter 2022 for the PEA.

The updated Capex estimate for the Project with a base date of fourth quarter 2022, is shown in Table 21.25.

Table 21.25 : Project Capex Summary PEA Update	
	US$M
Total Direct Construction Cost	725.3
Total Indirect Construction Costs (Includes EPCM and Owners Cost)	220.1
Total Construction Capex Cost	945.47
Total Capitalised Expenses	99.65
Total Pre-Start Capex	1 045.12
Expansion Capital	708.31
Sustaining Capital	965.94
Closure Costs	48.01
Total Capital Expenditure	2 768.77
Salvage Value	-48.01
Life of Project Capex	2 719.77

The cost increase of the estimate from the study base date of 2016 for the Project processing plant is 20.4% which represents an average escalation rate of 2.69% per annum.

21.4.2.3 Contingency

The purpose of contingency is to make specific provision for uncertain elements of cost within the Project scope and thereby reduce the risk of cost overrun to a pre-determined acceptable level. Contingencies do not include allowances for scope changes, escalation, or exchange rate fluctuations but rather for assumptions, omissions and other uncertainties that affect the estimate accuracy.

Contingency reflects the measure of the level of uncertainties related to the scope of work. It is an integral part of an estimate and has been applied to all parts of the estimate, i.e. direct costs and indirect costs.

The contingency allocation in the estimate has been assumed at 25% for all items.

21.4.2.4 Estimate Methodology

The 2016 study Capex estimate was prepared by Mintrex and was provided in detail to Wood. In accordance with the basis of estimate, the study estimate for the Productora process plant included a base date of fourth quarter 2015 and an expected accuracy of +/-30%. The inputs to the estimate are as described in Section 21.4.2.1.

The Wood office in Santiago was consulted to provide advice on the current rates for a number of commodities.

The current Chilean rates were used to calculate the average cost escalation/deflation from the original 2016 study by applying cost indices to each of the commodities that rates were returned.

To ensure the calculation reflected the impact of escalation only, the escalated rates for the various commodities were first calculated in 2015 US$ terms. The estimates were then adjusted to reflect the exchange rate difference in 2022 US$ terms which shows the combined effect of escalation and currency fluctuation. The equation used was:

2022 Estimated cost = A*E*(US$ 2015)*(US$ 2022/US$ 2015)

Where:

A is the cost of equipment in native currency

E is the escalation factor

US$ 2015 is the 2015 exchange rate in US$ terms

US$ 2022 is the 2022 exchange rate in US$ terms.

21.4.2.5 Currency Exchange Rates

The cost estimate is presented in US$. Forex rates for the indicative cost estimates were based on exchange rates quoted fourth quarter 2022.

The project foreign exchange rates for 1 US$ (second quarter 2022) are shown in Table 21.26.

Table 21.26 : Foreign Exchange Rates, 2Q2 2022	
$1 US$ =	1.33 AUD (Australian Dollar)
	0.86 EUR (European Euro)
	0.76 GBP (Great Britain Pound)
	800 CLP (Chilean Peso)

Table 21.27Table 21.28 shows the foreign exchange rates used for the study estimate in November 2015.

Table 21.27 : Foreign Exchange Rates, Q4 2015	
$1 US =	1.39 AU$ (Australian Dollar)
	0.89 EUR (European Euro)
	0.65 GBP (Great Britain Pound)
	689.7 CLP (Chilean Peso)

21.4.2.6 Escalation Rates

Table 21.28 shows the average escalation rates used in the cost estimate.

Table 21.28 : Average Escalation Rates Used	
Discipline	**(% /a)**
Mechanical Equipment – General (US$)	2.87
Equipment in AU$	3.19
Equipment in EUR	3.40
Equipment in CAD	3.61
Civils	2.51
Concrete	-2.3*
Steelwork	3.45
Platework	3.5
Piping	3.28

Table 21.28 : Average Escalation Rates Used	
Discipline	**(% /a)**
Electrical	2.57
Instruments and Controls	2.57

*Deflation evident in concrete rates.

21.4.2.7 Estimate Detail

The capital cost estimate is summarised in Table 1.9.

Table 21.29 : Capital Cost Summary Update – Q4 2022			
	Sub-Total (US$ M)	**Contingency (US$ M)**	**Total (US$ M)**
Stage 1 Capital			
Area 01 Bulk Earthworks and Drainage	37.10	9.27	46.37
Area 02 Site Services	2.51	0.63	3.13
Area 03 Sulphide Process	266.13	66.53	332.66
Area 04 Oxide Process	67.02	16.75	83.77
Area 05 Molybdenum Process	10.42	2.61	13.03
Area 07 Infrastructure (Excluding TSF)	145.46	36.36	181.82
Area 07 Infrastructure – TSF Only	25.26	6.31	31.57
Area 09 Mining (Excluding Prestrip)	26.38	6.59	32.97
Total Direct Construction Costs	**580.27**	**145.07**	**725.34**
Area 06 EPCM Construction Stage 1 Costs	94.67	23.50	118.2
Area 08 Owners Costs	81.58	20.39	102.97
Total Indirect Construction Costs	**176.24**	**43.89**	**220.13**
Total Construction Project Costs – Stage 1	**756.51**	**188.96**	**945.47**
Capitalised Expenses			
Area 10 Preproduction Mining Cost	79.72	19.93	99.65
Total Capitalised Expenses	**79.72**	**19.93**	99.65
Total Pre-Start Capex			**1 045.11**
Expansion CAPEX			
Plant Upgrade			-
Stage 2 - Cortadera Infrastructure	56.79	14.20	70.99
Stage 2 - Rope Conveyor	132.29	33.07	165.37
Stage 3 - Block Cave Development	324.72	81.18	405.90
Stage 3 - Block Cave Infrastructure	52.84	13.21	66.05
EPCM for Expansion			
Total Expansion CAPEX	**566.65**	**141.66**	**708.31**
Total Project Costs	**1 402.88**	**350.55**	**1 753.42**
Sustaining Capital			
Tailings			59.1
Sulphide Process			183.6
Molybdenum Process			1.8

Table 21.29 : Capital Cost Summary Update – Q4 2022			
	Sub-Total (US$ M)	**Contingency (US$ M)**	**Total (US$ M)**
Oxide Process			44.4
LG Leach Process			47.4
Waste Stripping			629.6
Total Sustaining Capex			**965.94**
Closure			48.1
Total Capital Expenditure			**2 768.77.**
Salvage Value			-48.01
Life of Project Capex			**2 719.77**

21.4.2.8 Block Cave PEA Estimate Methodology

A PEA level capital cost estimate was prepared for a 20 Mt/a block cave. Capital costs were estimated using a benchmark average footprint establishment cost of US$ 2 250/m^2 cave footprint area, plus an estimate for access, materials handling and infrastructure capital required to service the block cave footprint. Development capital costs of US$ 3 850/m lateral and US$ 2 450/m vertical were applied to the underground design metrics.

Geovia's PCBC Footprint Finder™ was used to determine a potentially economic cave footprint, with a resulting area of 132 600 m^2 for a total footprint establishment cost of US$ 300 million.

A PEA level design was produced in Deswik.CAD and used to estimate required mining physicals for declines and level access development, services and materials handling. Cost estimates were prepared using the physicals from Deswik.CAD and benchmark unit rate costs based on the size and type of excavation, amounting to a further US$ 107.5 million to support the block cave footprint. Infrastructure estimates for the block cave to link the cave to the open pit infrastructure amounted to a further US$ 66.0 million for a total block cave capital estimate of US$ 472.0 million.

21.4.2.9 Cortadera Infrastructure Direct Costs

The capital costs for the infrastructure at Cortadera have been estimated from third party budget prices and reports.

It includes the costs of a primary crushing station, a 15 km overland rope conveyor to the process plant at Productora, a powerline and a pipeline from Productora.

The Cortadera Rope Conveyor (RopeCon®) is a bulk material transport system that successfully combines conveyor technology with cableway features. RopeCon® transports material on a flat belt with corrugated side walls. As with conventional belt conveyors, the belt performs the haulage function and is driven and deflected by a drum in the head or tail station.

Based on the selection of RopeCon as the primary material handling method between Productora and Cortadera, Doppelmayr (RopeCon supplier) proposed a three-section RopeCon® system:

- Section 1 – Loading Section: From Loading Station to Transfer 1 – Length: 5 499 m
- Section 2 – Intermediate Section: from Transfer 1 to Transfer 2 – Length: 6 161 m

- Section 3 – Unloading Section: from Transfer 2 to ROM Pad – Length: 3 085 m.

Loading of all RopeCon sections is done through a chute. Unloading of the RopeCon is made through a discharge chute, except for RopeCon on Section 3, which discharges on the ROM pad.

In March 2022, Doppelmayr provided CAPEX and OPEX estimates for a 15 Mtpa and 25 Mtpa RopeCon system, providing the following estimates:

Table 21.30 : Rope Conveyor Vendor Cost Estimate – March 2022			
Area	**Unit**	**Unit Cost**	
Exchange Rate = 0.91	**EUR:USD**	**15Mt/a**	**25Mt/a**
Rate (nominal*)	t/hour	1 875	2 800
Capital (incl. shipping and install)	US$M	94.8	126.4
OPEX	US$/t	0.0847	0.0719
Site Operational Labour (5%)*	US$/t	0.0050	0.0050
Total OPEX*	US$/t	0.0897	0.0769

* - *Standardised based on latest agreed availability and utilisation, plus site labour allowance*

An updated OPEX quotation for the 20 Mt/a case was provided in March 2023. This throughput rate has a peak design throughput rate of 2 900 t/h, and makes allowance for 15% more capacity than the nominal required rate of 2 520 t/h. This nominal throughput rate is calculated based on a 91.3% system availability, also making allowance for planned maintenance activities.

The 20 Mt/a cost estimate is outlined in Table 21.31 and was used to support the MHS cost allowance in the financial model.

Table 21.31 : Rope Conveyor Vendor 20 Mt/a Cost Estimate – March 2023		
Area	**Unit**	**Unit Cost**
Exchange Rate = 0.92461218	**EUR:USD**	
Rate (nominal)	t/hour	2 520
Capital	US$M	110
Concrete, EW, Spares, Sub-Station	US$M	22.5
Contingency	US$M	20
Total CAPEX	US$M	152.5
OPEX		
Energy	US$/t	0.0478
Spares	US$/t	0.0492
Labour (Doppelmayr – direct)	US$/t	0.0030
Sub-Total	US$/t	0.1000
Site Operational Labour (5%)	US$/t	0.0050
Total OPEX	US$/t	0.1050

During the initial trade-off study, it was highlighted that a 7.5 Mt/a RopeCon installation would be the largest of its kind in the world, when compared to current installations. Scaling this up to 20 Mtpa amplifies the risk associated with this technology for the stated throughput. In contrast, OLCs have many examples within this throughput range and become cheaper to operate per tonne of production feed. The stated $0.007/tkm is still a third of the cost of the OLC and thus requires further investigation during the next phase of study to determine which MHS will be best suited to transport production feed between Cortadera and Productora.

21.4.2.10 Seawater Infrastructure and Cost Update

As part of the current PEA update, the required volume of seawater was reviewed, and an additional pumping capacity identified as being required for the increase in throughput from 14 Mt/a to 20 Mt/a. The capital cost estimate was escalated based on both the time elapsed between the PEA and the 2016 Study and also to allow for increase in capacity.

21.4.2.11 Sustaining Capex

The sustaining capital is for planned future capital works for the Project, including:

- Tailings dam wall raises and expansion
- Process Plant upgrades (Sulphide, Molybdenum and Oxide)
- Low Grade Sulphide Leach
- Waste Stripping.

21.4.2.12 Closure Costs

An allowance for closure has been allocated as part of the overall Capex by HCH, this allowance is expected to cover the following types of activities:

- Operations phase (portions of infrastructure related to the first tailings storage facility, to be closed prior the end of the Life of Mine)
- Active Closure phase (closure of the remaining mine infrastructure)
- Post Closure phase (monitoring of the mine infrastructure).

21.4.3 Updated Operating Cost Estimate

21.4.3.1 Introduction and Basis of Estimate

The operating cost estimate updates are based on:

- Update to the sulphide process plant throughput from 14 Mt/a to 20 Mt/a
- Oxide process plant throughput is planned with cathode production of 10 kt/a output increasing to 12 kt/a in yr 8
- Changes in rates and costs as detailed below against each operating cost category

- Only the average Operating Cost estimate was updated in each case, representing a snapshot of the processing costs associated with a composite production feed sample being treated.

The methodology used for updating the operating cost estimates is described below:

- The 2016 OPEX for oxide, sulphide/molybdenum and seawater operating estimate calculations were reviewed
- Inputs such as diesel, electricity were update to current pricing (refer to Table 21.37) below
- Exchange rates were updated as per Section 21.4.2.5.

The operating cost basis is outlined in Table 21.32 (for both the 2016 and 2022 estimates).

Table 21.32 : Operating Cost Basis

Operating Cost Item	Units	Year	
		2016	2022
Concentrator Plant Sulphide Throughput	Mt/a	14	20
Concentrator Plant - Molybdenum Throughput	Mt/a	0.2638	0.37584
Molybdenum Concentrate Production	t/a	2 240	3 200
Crushing Plant Operating Availability	hr/a	6 570	6 570
Milling Circuit Operating Availability	hr/a	8 000	8 000
Power Cost	$/kWh	0.095	0.065
Diesel Fuel Cost	$/L	0.64	0.73
Oxide Throughput - Crushing Plant	t/a	3 300 000	3 300 000
Oxide Throughput - Heap Leach	t/a	3 300 000	3 300 000
Copper Cathode Production	t/a	10 000	10 000
Crushing Plant Operating Availability	hr/a	6 570	6 570
SX/EW Plant Operating Availability	hr/a	8 322	8 322
Crushing Plant Operating Availability	hr/a	6 570	6 570

The estimate has been compiled by Wood with inputs from consultants for their responsible scope of work:

- ABGM Plus: Mining, rehandling
- Knight Piésold: Heap leach, tailings storage, surface water management
- Wood: Process plants, surface infrastructure, tailings pipelines, power supply and distribution, services and utilities, and concentrate pipelines based on the 2016 Study by Mintrex
- Wood: General and Administration (G&A) based on escalation of the 2016 Study by Mintrex
- HCH: Low Grade Sulphide Leach.

Operating costs were estimated for mining, sulphide and oxide plant processing, administration, concentrate transport and seawater supply areas as part of the 2016 study. These costs have been escalated by 20.4% and are presented in US dollars as of the second quarter 2022 (2Q22) to an accuracy level of +/- 40%.

Exchange rate assumptions used in the estimate are as follows:

- Australian dollar (A$) to US dollar (US$) 0.75000
- Canadian dollar (C$) to US$ 0.80000
- Euro (EUR) to US$ 1.16000
- Chilean peso (CLP) to US$ 0.00125.

Financial analysis utilised a power cost of $0.065/kWh based on current power supply negotiations and a diesel cost of $0.73/L in line with independent recommendations and current long-term forecasts, respectively.

The mining operating cost estimate is based on a mining contractor quote. The quote excludes costs for minor items such as contractor supervision, blast hole sampling, RC drilling for process feed control, seawater supply and pit dewatering. For these activities not included in the mining contractor scope, HCH prepared the operating cost estimate.

Average project operating costs are summarised in Table 21.33 and were benchmarked for comparable projects and are within the range of the comparable projects.

Table 21.33 : Operating Costs Summary

Operating Costs	Unit	Life of Mine
Mining		
Open Pit	US$/t mined	2.21
Underground	US$/ t mined	6.55
Processing		
Sulphide Concentrator – Cu/Au/Ag Concentrate	US$/t Process Feed	6.04
Sulphide Concentrator – Mo Concentrate	US$/t Mo in Conc	0.56
Sulphide Leach – Front End Processing	US$/t	1.03
Sulphide Leach – Back End Processing	US$/lb Cu	0.26
Oxide Leach – Front End Processing	US$/t	4.62
Oxide Leach – Back End Processing	US$/lb Cu	0.26
G&A	US$M/quarter	3.28

The operating cost estimate is based on Q4 2022 assumptions, in US dollars. The operating cost was prepared on the following basis:

- All equipment and materials will be new
- The labour rate build-up will be based on the statutory laws governing benefits to workers that were in effect at the time of the estimate.

The following items were excluded from the operating cost estimate:

- Sales and marketing
- Foreign exchange, finance and interest charges
- Costs due to extraordinary currency fluctuations (e.g. materials sourced from overseas)
- Changes in Chilean law
- Other duties and taxes (except as identified)
- Force majeure events
- Pre-operations training of personnel
- Freight estimates are based on vendor supplied freight quotations or in-house data
- No contingency is assumed
- No cost escalation (or de-escalation) is assumed.

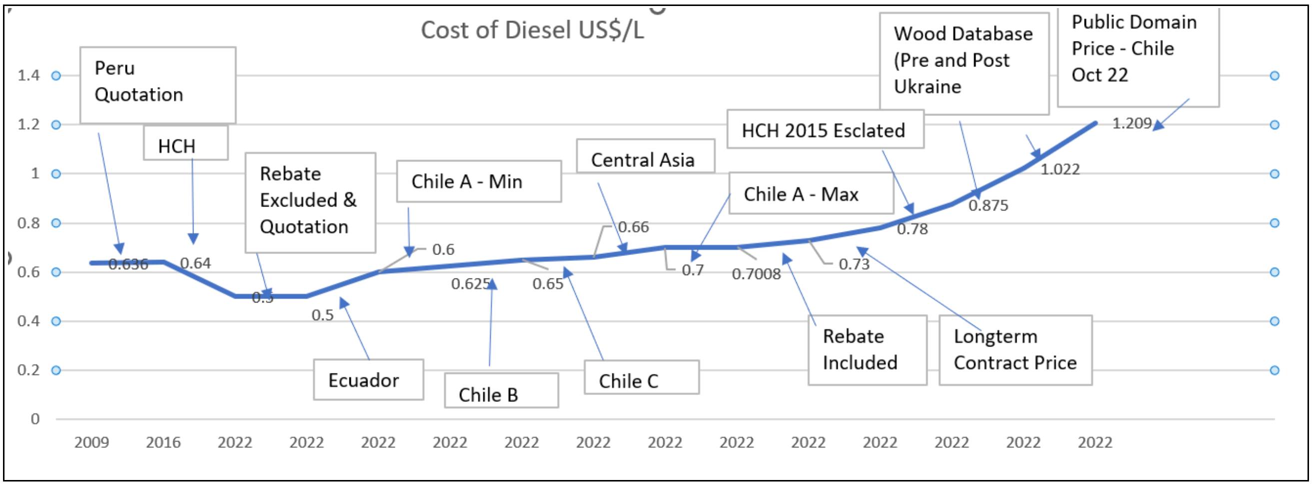
21.4.3.2 Diesel Cost Update

Long term diesel fuel prices have been sourced from other comparable mining operations in South America.

A 23% excise applies to diesel and gasoline sold in Chile, the "Impuesto Especifico a los Combustibles". Most mining companies can rebate 100% of this fuel tax.

The final fuel price will be defined based on the commercial agreement with the main supplier and will depend on market conditions and service costs contracted with the main supplier.

Figure 21.1 : Benchmarked Cost of Diesel US$/L



21.4.3.3 Power Cost Update

As part of the current studies for the Project, HCH has engaged with several electrical market providers and advisors and several non-binding long-term power quotes have been received. All providers have committed to sourcing power from renewable energy sources.

The current PEA power cost of $0.065/kWh is based on the outcome of this process.

Chile´s Central Authority Electrical Regulator has approved HCH's application for connection to the Maitencillo sub-electrical power station. Connection to Maitencillo is a key step forward for the development of the Project, providing access to Chile's national energy grid and its multiple renewable energy providers.

HCH expects to start a binding process to select one or more electrical providers for the Project in parallel with delivery of the PFS.

21.4.3.4 Mining Operating Cost

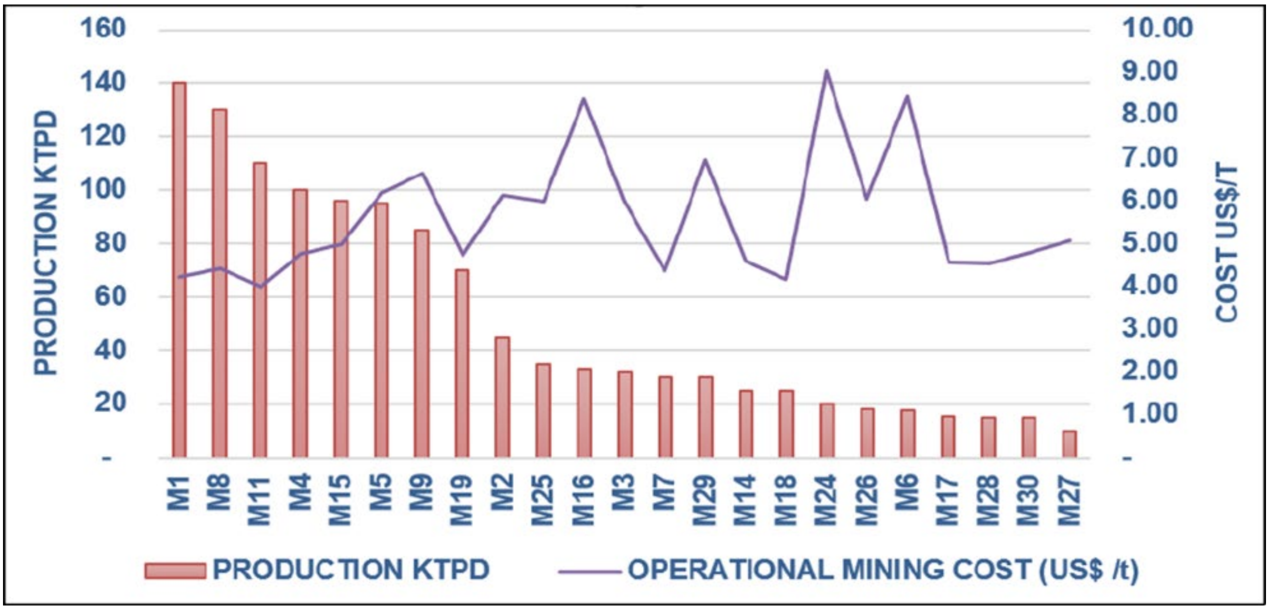
The mining contractor who supplied the 2016 quote was reengaged to supply an updated contract mining budget quote based on the current PEA mining schedule for the life of the Project and a defined responsibility matrix as outlined in Section 16.4.1. An Open Pit reference mining cost of US$2.03/t material mined was applied in the study, along with vertical mining cost adjustments of US$0.0125/t mined/5 m below pushback exit RL and US$0.00425/t mined/5 m above pushback exit RL.

The mining contractor will carry out the majority of mining related activities. The scope of works considered by the quote includes drilling, blasting, loading, hauling, ancillary equipment, production feed rehandling and general administration costs. The operating cost also includes the ownership component related to mobile equipment capital cost depreciation. Establishment capital, including mobilisation, construction of key mining infrastructure and initial mining roads, was allocated as part of the capital estimate and is not part of the operating cost estimate.

The mining contractor delivered a quote, based on material haul distances for the different operations, which was then used to develop the final mining cost model. The quote obtained excludes some additional costs for minor items, e.g. HCH supervision, blast hole sampling, RC drilling for process feed control, seawater supply and pit dewatering. For these activities not included by the scope allocated to the mining contractor, HCH conducted the operating cost estimate. Refer to Section 16.3.3 for further details.

A PEA level operating cost estimate was used for the Stage 3 - block cave operation. The cave is considered to be in operation with the open pit, attracting the same processing and general and administration cost. A typical benchmarking mining operating cost was used for an operating cave of 20 Mt/a (54.8 kt/d), considering factors such as the footprint layout, proposed mining method and depth below surface (See Figure 21.2). An underground block cave operating cost of US$6.55/t was applied to the study.

Figure 21.2 : Operational Mining Cost vs. Production Feed Production



Wood Block Caving Benchmarking, 17 September 2020

21.4.3.5 Cortadera Production Feed Transport Costs

The operating costs of rope conveying the process feed 15 km to the process plant at Productora have been estimated from third party budget prices and reports.

Table 21.34 : Cortadera Production Transport Costs 20 Mt/a

	Unit	Value
Capital	$M	110
Rate	t/hr	2 520
Energy	$/t	0.0478
Spares	$/t	0.04492
DDT Labour	$/t	0.003
Site Labour	$/t	0.005
Total (Nominal)	**US$/t**	**0.105**

21.4.3.6 Copper and Molybdenum Sulphide Process Plant Operating Cost Update

Processing and maintenance cost estimates were developed using the plant parameters and process feed characteristics specified in the process design criteria. The sulphide treatment plant has a design capacity of 20 Mt/a of production feed and will produce nominally 375.84 kt/a of copper concentrate and 3200 t/a of molybdenum concentrate. As part of the increase of throughput, the manning of the copper sulphide plant was reviewed and based on benchmarked process plants, the number of personnel was considered sufficient and hence remains constant. However, salaries were adjusted for CPI.

Due to the increased throughput, reagents and consumables usage was increased and hence costs of reagents and consumables recalculated.

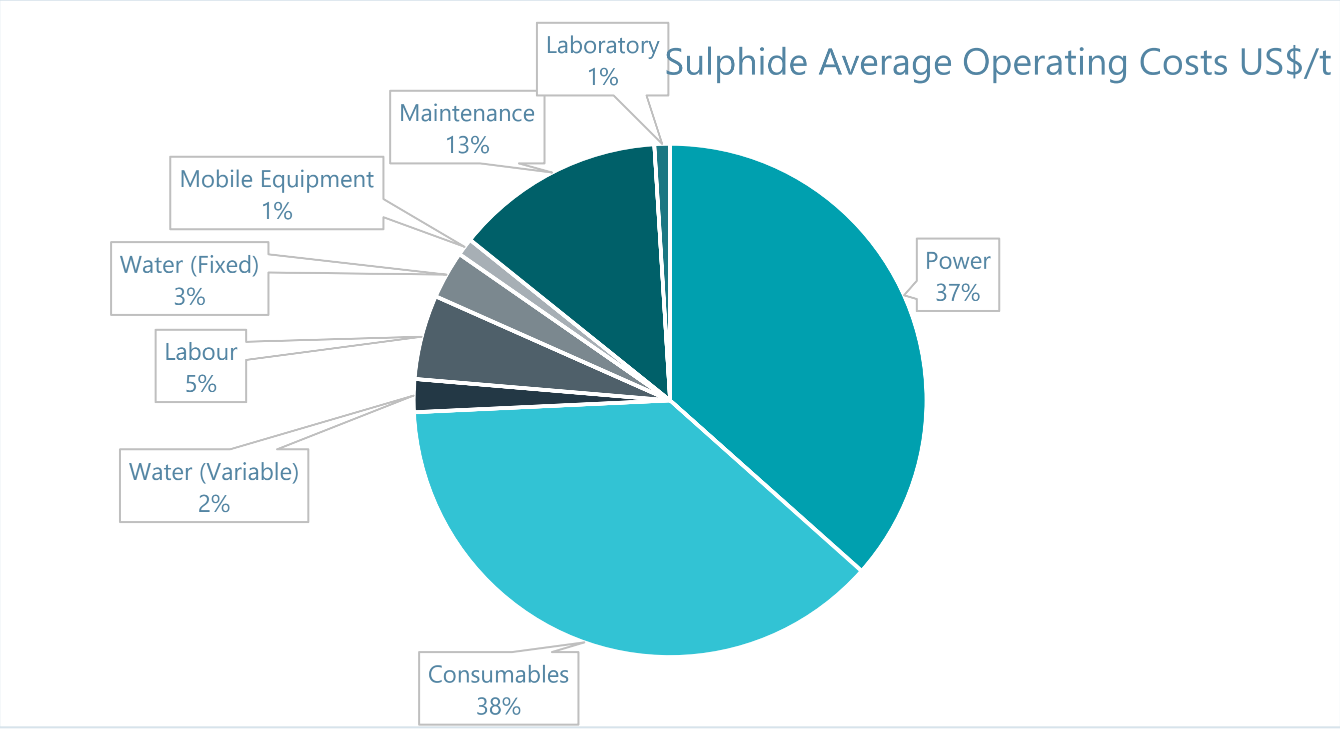
A summary of the copper sulphide plant processing and maintenance operating costs is shown in Table 21.35, the operating cost has been assessed for each of the main production feeds and LOM.

Table 21.35 : Copper Sulphide Plant Operating Cost Summary

Area	Productora	Alice	Cortadera OP	Cortadera UG	Average LOM Processed ($/t)
Average Throughput (Mt/a)	22.3	23.2	24.2	19.4	
Processing Cost ($/t)	5.21	5.07	4.93	5.74	6.04
Additional Mining Cost (Waste) ($/t)_	0.12	0.12	0.12	0.12	
Additional Rehandling (Average) ($/t)	0.47	0.47	0.47	0.47	

Note for Cortadera the Rope Conveyor operating costs are calculated separately.

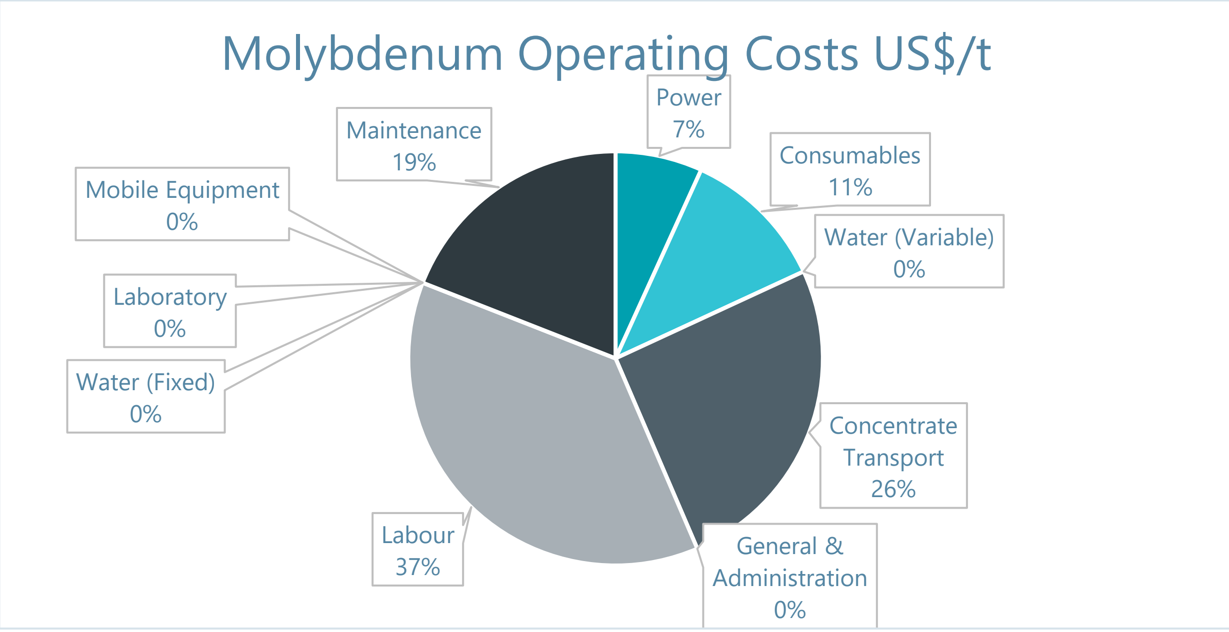
Figure 21.3 :Distribution of Operating Costs US$/t



A summary of the molybdenum sulphide plant processing and maintenance operating costs is shown in Table 21.36.

Table 21.36 : Molybdenum Sulphide Plant Operating Cost Summary					
Area	**Productora**	**Alice**	**Cortadera OP**	**Cortadera UG**	**Average LOM Processed ($/t)**
Processing Cost (US$/dry t of Cu Concentrate)	5.30	5.30	5.30	5.30	
Molybdenum Processing Cost (US$/lb Mo in concentrate)	**0.18**	**0.81**	**0.63**	**0.26**	**0.56**

Figure 21.4 :Distribution of Operating Costs US$/t



21.4.3.7 Copper Oxide Processing Plant Operating Cost Update

Processing and maintenance cost estimates were developed using the plant parameters and process feed characteristics specified in the process design criteria. The copper oxide treatment plant has a design capacity of 3.3 Mt/a of process feed and will produce 10 k t/a of copper cathode increasing to 12k t/a in year 8.

Oxide crushing/agglomeration availability of 75% (6570 hr/yr) and SX/EW plant availability of 95% (8322 hr/yr) has taken into consideration downtime due to scheduled and unscheduled maintenance which reflects 2016 Study process design criteria.

Acid is supplied through annual contracts with a diversified supplier base comprising smelters (Potrerillos, Paipote) in the Atacama Region, smelters in the Antofagasta Region (Altonorte) and direct supply from international suppliers (Peru, Europe, other) and traders. Annual contracts are typically secured for between 90% and 100% of annual requirements. Forward looking assumptions for sulphuric acid supply, demand and price included in this report were obtained from independent expert reports (CRU and Cochilco). Currently the long term price averages around 90 - 95 US$/ton CIF by 2025.

The manning of the copper oxide plant was reviewed, and based on benchmarked process plants, the number of personnel was considered sufficient and hence remains constant. However, salaries were adjusted for CPI.

A summary of the copper oxide treatment plant processing and maintenance operating costs is shown in Table 21.37.

Table 21.37 : Copper Oxide Plant Operating Cost Summary

Area	Productora	Alice	Cortadera OP	Cortadera UG	Average LOM Production Feed Processed ($/t)
Front End Processing Cost (US$/t)	4.57	4.57	4.57	4.57	4.62
Back End (US$/tonne of cathode	584.05	584.05	584.05	584.05	
Back End US$/lb Cu in cathode	0.34	0.34	0.34	0.34	0.26

21.4.3.8 Low Grade Sulphide Leach Operating Cost

The operating costs allocated for the PEA in terms of the low-grade sulphide leach operating cost by HCH is US $1.03/t for the front end and US$ 0.26/lb for the back end based on benchmark data. These costs were benchmarked and are reasonable for the PEA.

21.4.3.9 Seawater Infrastructure Operating Costs

Seawater Infrastructure operating costs were escalated and scaled to allow for the increase in production capacity for the PEA. The cost is $0.43/m^3 which takes into account updated power costs, mobile vehicles, maintenance and labour.

21.4.3.10 General and Administration Operating Costs

The General and Administration (G&A) Costs, as escalated by Wood based on HCH costs, encompass the following costs:

- G&A Costs: HCH provided the basis for estimating the G&A Costs. US$0.66 per tonne throughput (average) which equates to $3.28M per quarter.

The G&A costs were benchmarked with comparable projects and are reasonable for the PEA.

21.5 Comments on Section 21

The initial capital cost is estimated to be US$1046M, with a further US$708M allocated for planned expansion. The sustaining capital is estimated to be US$966M and a further $48M for post-production, and closure costs.

The overall life of mine operating cost is US$6400M.

Checks were undertaken in Q4 2022 as part of the PEA to confirm the cost estimate. The review considered escalation, scaling due to increase in production capacity and a review of power and diesel pricing.

The QPs consider that this review supports the cost estimates presented in this section.

22 Economic Analysis

This section has been taken directly from the technical report titled, "Costa Fuego Copper Project: NI 43-101 Technical Report & Preliminary Economic Assessment (PEA)" dated August 14, 2023 with an effective date of June 28, 2023. The Mineral Resource estimate update, as described in this Report, does not materially affect the mineral resource inventory that formed the basis of the 2023 PEA and no new scientific or technical information has been developed that would materially affect the outcome of the 2023 PEA and, therefore, the results and conclusions of the 2023 PEA are considered current and have been restated in this Report.

22.1 Cautionary Statement

The results of the economic analysis in the PEA 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:

- 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 Costa Fuego mining operation
- Estimation of the Mineral Resources and the realisation of the Mineral Resource estimates within the PEA mine plans
- Time required to develop the mine based on the PEA mine design
- Assumptions regarding mine dilution
- Assumptions regarding losses
- Expected grade of the material delivered to the mill
- Metallurgical recovery rates
- 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, mine closure
- Assumptions regarding geotechnical and hydrogeological factors.

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 PEA is preliminary in nature, and a portion of the Mineral Resources in the mine plans, production schedules, and cashflows include Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorised as Mineral Reserves, and there is no certainty that the PEA will be realised. Due to the conceptual nature of the PEA, Mineral Resources cannot be converted to Mineral Reserves and therefore do not have demonstrated economic viability.

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 PEA.

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, grade, discount rate and metal price.

The financial model assumes 100% equity.

All pricing and costs are stated in constant (Real) fourth quarter 2022 United States dollars (US$).

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$/lb	3.85
Molybdenum	US$/lb	17.00
Gold	US$/oz t	1 750.00
Silver	US$/oz t	21.00

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 55%, silver recovery averages 35% and molybdenum averages 56%.

Recoverable copper from leach processing is assumed to be 55% for oxide material over a one-year leach period, and 40% for low-grade sulphide material over a four-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 1.11.

Table 22.2 : Exchange Rates	
Currency	**Exchange Rate**
US$/AU	1.33
US$/CLP	800

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 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 (26%)	%	25
Au Grade in Concentrate (5 ppm)	g/t	1-135
Ag Grade in Concentrate (24 ppm)	g/t	+30
Mo Concentrate Moisture	%	5
Mo Grade in Concentrate (7 411 ppm)	%	50
Payables		
Cu Payable (Cu-Au Con.)	%	96.5
Cu Cathode Payable	%	100
Mo Payable (Mo Con.)	%	98
Au Payable (Cu-Au Con.)	%	90
Ag Payable (Cu-Au Con.)	%	90
Selling Cost		
Sulphide Copper	US$/lb Cu	0.42
Cathode/Leach Copper	US$/lb Cu	0.07
Molybdenum	US$/lb Mo	0.82
Gold	US$/oz Au	5.00
Silver	US$/oz Ag	0.50

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 outcome of the review found that the financial model tax assumptions were correct and reasonable. HCH subsequently provided a final reliance letter titled "Hot Chile Costa Fuego Reliance on Other Experts for Tax Information" dated 5 May 2023 for Wood to rely on.

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$260 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%.

The financial model assumes that the Project is subject to the Chilean Specific Mining Tax (SMT) in effect as of the date of this report. Annual sales of less than 12 kt CuEq are exempt from SMT. For operations with annual sales ranging between 12 – 50 kt CuEq, a marginal rate of 0.5% to 4.5% is applied over the taxable operating income. Annual sales over 50 kt Cu are subject to a marginal rate ranging from 5% and 14% over the taxable operating income based on the operating margin.

The Project will produce more than 50 kt Cu per year in all years except the final year of production. The applicable marginal rates are shown in Table 22.4 and Table 22.5 below.

Table 22.4 : SMT Tax Schedule, Copper Equivalent Production <=50,000 tonnes/year

Copper Equivalent Production (kt/year)		SMT (%)
Min	Max	App. Rate
0	12	0.0%
12	15	0.5%
15	20	1.0%
20	25	1.5%
25	30	2.0%
30	35	2.5%
35	40	3.0%
40	50	4.5%

Table 22.5 : SMT Tax Schedule, Copper Equivalent Production >50,000 tonnes/year

Mining Operating Margin (%)		SMT (%)
Min	Max	App. Rate
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%

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.

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 22.6 : Depreciation Schedule		
Asset description	**Straight-Line Depreciation Years**	**Accelerated Depreciation Years**
Bulk earthworks and drainage	9	3
Site services	5	1
Plant for sulphide and oxide processing	9	3
Tailings storage facilities (TSF)	10	3
Mining fleet and associated infrastructure	9	3
EPCM and owner's costs	5	1
Sustaining capital for the two plants	5	1
TSF extensions	10	3
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

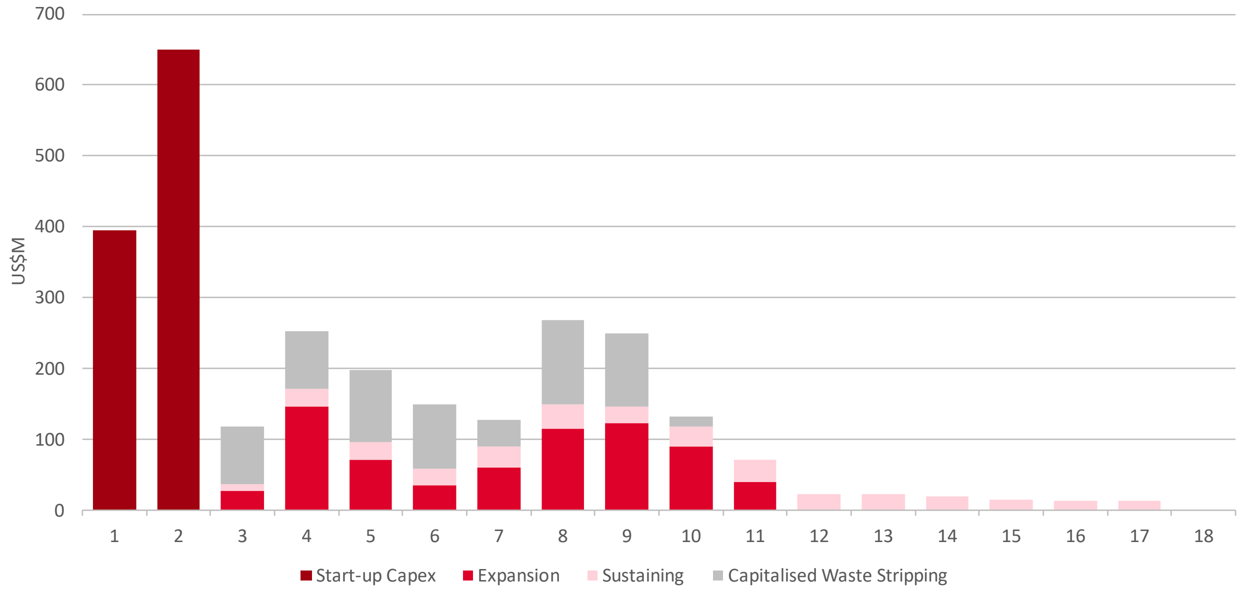
22.10 Capital Costs

Total capital expenditures over the life of the Project total US$2.77 B. A summary of capital expenditures is shown in Table 22.7.

Table 22.7 : Capital Expenditure Summary	
Capital Expenditure	**Amount US$ M**
Initial Construction Capital	945.47
Capitalised Expenses	99.65
Total Pre-Start Capex	1 045.12
Expansion Capital	708.31
Sustaining Capital	965.94
Closure Costs	48.01
Total Capital Expenditure	2 768.77
Salvage Value	(48.01)
Life of Project Capex	2 719.77

Closure costs in the financial model are estimated at $48 M, or 5% of direct construction capital costs plus expansion capital for Cortadera infrastructure and rope conveyor, and are assumed to be offset by salvage 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. These costs are included in the life-of-mine average below.

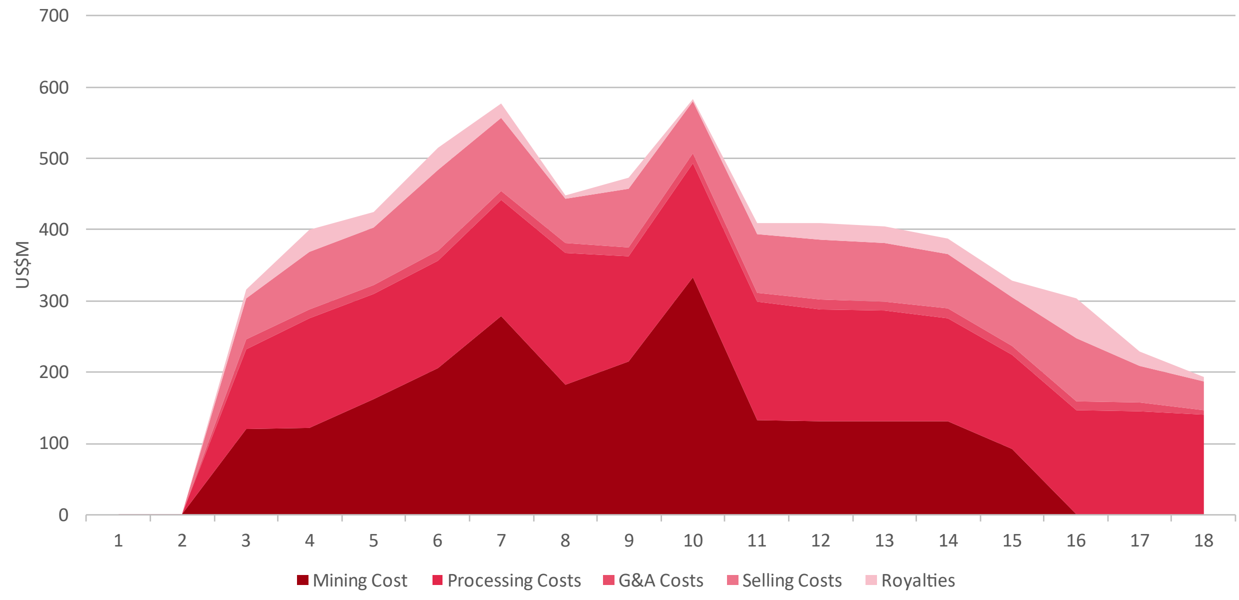
During operations, the cost of mining waste in each period in excess of the average life-of-mine strip-ratio is also capitalised. This capitalised stripping is not included in the life-of-mine averages below.

Table 22.8 : Operating Cost Input Summary

Operating Costs	Unit	Life of Mine
Mining Cost Average	US$/t mined	2.87
Open Pit	US$/t mined	2.21
Underground	US$/t mined	6.55
Processing Costs		
Sulphide Concentrator		
Cu/Au/Ag Concentrate	US$/t Process Feed	6.04
Mo Concentrate	US$/lb Mo in Conc.	0.56
Sulphide Leach		
Front End Processing	US$/t	1.03
Back End Processing	US$/lb Cu	0.26
Oxide Leach		
Front End Processing	US$/t	4.62
Back End Processing	US$/lb Cu	0.26
G&A	US$M/quarter	3.28

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$1 100 M with 21% IRR and 3.5-year payback period. Table 1.12 below presents a summary of the financial analysis results.

Table 22.9 : Summary of the Financial Analysis

Project Metric	**Units**		**Value**	
Financial Measures				
Pre-Tax	Cu US$3.85/lb	NPV8%	US$M	1 540
		IRR	%	24
Post-Tax	Cu US$3.85/lb	NPV8%	US$M	1 100
		IRR	%	21
Payback period (from start of operations)			yr	3.5
Open Pit Strip Ratio			W/P	1.8
NPV/Capex			Ratio	1.1
Capital Costs				
Total Pre-production Capital Expenditure			US$M	1 086.1
Expansion			US$M	707.7
Sustaining			US$M	1 014.02
Total			US$M	2 768

Table 22.9 : Summary of the Financial Analysis		
Project Metric	**Units**	**Value**
Operating Costs		
C1	$/lb Cu	1.33
Total Cash Cost (net by-products and including royalties)	$/lb Cu	1.43
All-in-Sustaining Cost	$/lb Cu	1.74
All-In Cost LOM	$/lb Cu	2.31
Mine Life and Metal Production		
Primary Mine Production Including Ramp-up	yr	14
Mine Life (Life of Mine Processing)	yr	16
Primary Mine Production – Average Annual Copper Equivalent Metal	kt	112
Primary Mine Production – Average Annual Copper Metal	kt	95
Primary Mine Production – Average Annual Gold Metal	Koz t	49

Noting:

- C1 Costs consist of mining costs, processing costs, G&A, selling costs
- 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 PEA mine plan are shown in Table 22.10.

Table 22.10 : Financial Model Output

		Total	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18
Mining																				
Production Feed Tonnes Mined	kt	471 525	4	4 558	31 221	23 430	41 314	59 271	70 368	19 556	48 135	80 960	20 233	19 580	19 576	19 503	13 817	-	-	-
Waste Tonnes Mined (excl. capitalised)	kt	344 259	2 397	41 042	25 885	33 879	34 309	34 007	48 702	63 497	31 465	26 816	604	421	434	493	308	-	-	-
Rehandle	kt	132 135	-	-	3 242	9 546	706	2 859	2 614	17 393	5 207	5 053	3 498	3 413	3 380	1 515	6 511	22 400	22 400	22 400
Capitalised Waste	kt	282 466	-	-	38 894	38 691	47 541	41 451	16 909	54 404	39 578	4 998	-	-	-	-	-	-	-	-
Processing Sulphide Concentrator																				
Production Feed Processed	kt	334 131	-	-	16 700	22 300	22 625	22 397	22 558	22 280	20 476	19 465	19 477	19 490	19 477	19 478	20 210	22 400	22 400	22 400
Processed Grade																				
Copper	%	0.44%	-	-	0.49%	0.43%	0.45%	0.60%	0.54%	0.34%	0.46%	0.43%	0.48%	0.48%	0.48%	0.45%	0.41%	0.47%	0.29%	0.26%
Gold	g/t	0.12	-	-	0.11	0.10	0.09	0.10	0.11	0.07	0.12	0.14	0.17	0.18	0.18	0.17	0.16	0.11	0.07	0.08
Silver	g/t	0.45	-	-	-	-	0.18	0.23	0.34	0.17	0.51	0.83	0.92	0.94	0.94	0.87	0.73	0.08	0.12	0.53
Molybdenum	ppm	117.18	-	-	138.26	142.13	135.77	145.44	144.57	109.31	139.23	137.10	140.36	117.24	91.30	71.64	47.93	147.98	131.31	31.83
Contained Metal																				
Copper	kt	1 468	-	-	83	97	101	134	123	75	95	83	93	94	93	89	83	106	64	57
Gold	000 oz	1 314	-	-	56	71	68	74	81	54	77	86	109	116	116	109	103	80	53	60
Silver	000 oz	4 811	-	-	-	-	131	167	249	121	334	518	577	589	589	543	475	56	83	379
Molybdenum	kt	39	-	-	2	3	3	3	3	2	3	3	3	2	2	1	1	3	3	1
Low Grade Sulphide Leach																				
Production Feed Processed	kt	100 104	-	-	-	-	-	-	11 200	22 400	-	11 200	19 600	11 982	10 787	10 787	2 147	-	-	-
Processed Cu Grade	%	0.14%	-	-	-	-	-	-	0.17%	0.15%	-	0.14%	0.14%	0.14%	0.14%	0.14%	0.17%	-	-	-
Contained Cu	kt	144	-	-	-	-	-	-	19	33	-	15	27	17	15	15	4	-	-	-
Oxide Leach																				
Production Feed Processed	kt	37 290	-	-	2 859	3 300	3 300	3 300	3 300	3 300	3 300	3 300	3 300	3 300	3 300	1 431	-	-	-	-
Processed Cu Grade	%	0.42%	-	-	0.62%	0.56%	0.56%	0.53%	0.44%	0.44%	0.37%	0.36%	0.29%	0.29%	0.26%	0.25%	-	-	-	-
Contained Cu	kt	157	-	-	18	18	18	18	15	15	12	12	10	10	9	4	-	-	-	-
Production																				
Copper Concentrate	dmkt	5 050	-	-	245	332	344	466	429	251	337	297	334	338	333	317	290	358	214	167

			1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18
Copper Concentrate	wmkt	5 490	-	-	267	361	374	506	466	272	366	323	363	368	362	344	315	389	233	182
Copper in concentrate	kt	1 263	-	-	61	83	86	116	107	63	84	74	83	85	83	79	72	89	54	42
Gold in concentrate	000 oz	718	-	-	29	41	38	42	46	26	44	49	63	67	67	62	55	43	24	23
Silver in concentrate	000 oz	1 699	-	-	-	-	36	48	72	35	124	204	219	223	223	206	167	15	22	102
Molybdenum Concentrate	dmkt	44	-	-	1	3	3	3	4	2	4	4	4	3	2	2	1	3	3	1
Molybdenum Concentrate	wmkt	46	-	-	1	3	3	4	4	2	4	4	4	3	3	2	1	3	3	1
Molybdenum in concentrate	kt	22	-	-	1	2	2	2	2	1	2	2	2	2	1	1	1	2	1	0
Sulphide Copper Cathode	kt	58	-	-	-	-	-	-	1	4	5	5	6	7	7	9	7	4	2	1
Oxide Copper Cathode	kt	86	-	-	6	10	10	10	9	7	7	7	6	5	5	3	0	-	-	-
Metal Sold																				
Copper in Concentrate	000 lbs	2 783 545	-	-	133 865	184 064	184 064	262 151	234 263	139 442	184 064	161 753	184 064	189 641	184 064	172 908	156 175	200 797	117 131	95 099
Gold	000 oz	718	-	-	29	41	37	43	46	27	43	48	62	68	67	62	54	44	24	24
Silver	000 oz	1 699	-	-	-	-	35	50	72	36	123	201	219	227	224	204	165	19	21	104
Molybdenum	000 lbs	47 981	-	-	1 539	3 330	3 581	3 770	3 927	2 482	4 367	4 210	4 304	3 519	2 639	2 011	1 257	3 424	2 827	795
Copper Cathode	000 lbs	316 352	-	-	13 228	22 046	22 046	22 046	22 046	24 868	26 455	26 455	26 455	26 455	26 455	26 455	15 785	8 642	5 115	1 797
Year			1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18
		Total																		
Revenue																				
Copper in Concentrate	$'000	10 341 566	-	-	497 340	683 843	683 843	973 958	870 345	518 063	683 843	600 953	683 843	704 565	683 843	642 398	580 230	746 010	435 173	353 318
Gold	$'000	1 131 582	-	-	45 019	64 155	58 599	67 428	71 846	41 913	68 405	76 139	98 421	107 587	105 319	97 388	85 114	69 219	37 947	37 084
Silver	$'000	32 118	-	-	-	-	655	946	1 352	675	2 321	3 791	4 147	4 294	4 240	3 858	3 116	352	405	1 965
Molybdenum	$'000	799 369	-	-	25 646	55 479	59 666	62 807	65 423	41 348	72 751	70 134	71 704	58 619	43 965	33 497	20 936	57 049	47 105	13 240
Copper Cathode	$'000	1 217 957	-	-	50 927	84 878	84 878	84 878	84 878	95 742	101 853	101 853	101 853	101 853	101 853	101 853	60 773	33 272	19 692	6 920
Total Revenue	**$'000**	**13 522 592**	**-**	**-**	**618 931**	**888 354**	**887 64**	**1 190 016**	**1 093 845**	**697 740**	**929 174**	**852 870**	**959 968**	**976 918**	**939 220**	**878 993**	**750 169**	**905 903**	**540 321**	**412 527**
Expensed Operating Costs																				
Mining Cost	$'000	(2 239 768)	-	-	(120 896)	(122 066)	(162 854)	(205 103)	(278 015)	(182 784)	(215 557)	(333 565)	(133 375)	(131 001)	(131 062)	(130 972)	(92 519)	-	-	-
Processing Costs	$'000	(2 403 654)	-	-	(111 594)	(152 844)	(146 210)	(151 590)	(163 231)	(185 227)	(146 782)	(159 865)	(165 235)	(157 485)	(155 557)	(145 249)	(131 428)	(146 489)	(144 512)	(140 355)
G&A Costs	$'000	(203 307)	-	-	(13 114)	(13 114)	(13 114)	(13 114)	(13 114)	(13 114)	(13 114)	(13 114)	(13 114)	(13 114)	(13 114)	(13 114)	(13 114)	(13 114)	(13 114)	(6 600)
Selling Costs	$'000	(1 229 269)	-	-	(58 277)	(81 405)	(81 610)	(114 429)	(102 929)	(62 214)	(82 641)	(73 253)	(82 733)	(84 455)	(81 396)	(76 186)	(67 760)	(87 546)	(51 750)	(40 684)
CCHEN Royalties	$'000	(45 584)	-	-	(3 910)	(12 351)	(5 569)	(3 816)	(3 528)	(2 072)	(4 059)	(356)	(473)	(523)	(280)	-	(12)	(6 099)	(2 536)	-
Purisima Royalties	$'000	(13 345)	-	-	-	-	(1 678)	(1 444)	(1 608)	(633)	(3)	(1)	(59)	(343)	(68)	(41)	(2 523)	(925)	(1 221)	(2 798)

			1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18
Zapa Royalties	$'000	(1 136)	-	-	-	-	-	-	(1)	(15)	(274)	(750)	(96)	-	-	-	-	-	-	-
SMT	$'000	(264 176)	-	-	(8 488)	(18 444)	(13 696)	(25 003)	(13 940)	(1 378)	(11 016)	(1 690)	(14 599)	(22 382)	(22 340)	(21 875)	(20 448)	(49 050)	(16 582)	(3 246)
Total Operating Costs	**$'000**	**(6 400 238)**	**-**	**-**	**(316 278)**	**(400 224)**	**(424 730)**	**(514 498)**	**(576 365)**	**(447 437)**	**(473 444)**	**(582 595)**	**(409 684)**	**(409 303)**	**(403 817)**	**(387 437)**	**(327 805)**	**(303 223)**	**(229 714)**	**(193 684)**
Capital																				
Construction	$'000	945 874	390 451	555 424	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Capitalised Expenses	$'000	99 649	5 134	94 515	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Expansion	$'000	708 306	-	-	27 560	146 407	71 383	34 941	60 743	114 505	123 069	89 198	40 500	-	-	-	-	-	-	-
Sustaining	$'000	965 937	-	-	90 974	106 515	126 391	114 177	67 412	153 373	126 407	43 579	31 165	22 850	22 284	20 060	14 262	13 244	13 244	-
Total Capital	**$'000**	**2 719 766**	**395 584**	**649 939**	**118 534**	**252 921**	**197 774**	**149 118**	**128 155**	**267 878**	**249 476**	**132 777**	**71 665**	**22 850**	**22 284**	**20 060**	**14 262**	**13 244**	**13 244**	**-**
EBITDA	**$'000**	**7 122 354**	**-**	**-**	**302 654**	**488 131**	**462 911**	**675 518**	**517 480**	**250 304**	**455 729**	**270 275**	**550 285**	**567 615**	**535 403**	**491 556**	**422 364**	**602 680**	**310 607**	**218 843**
Depreciation (accelerated)	$'000	(2 719 766)	-	-	(573 299)	(323 267)	(399 455)	(240 779)	(199 474)	(185 391)	(236 351)	(192 549)	(140 188)	(95 138)	(54 078)	(26 971)	(17 809)	(13 498)	(13 244)	(8 277)
Tax Loss Carry Forward	$'000	(681 795)	-	-	(96 328)	(211 950)	(63 456)	(302 325)	-	(7 737)	-	-	-	-	-	-	-	-	-	-
Taxable Income	**$'000**	**4 142 588**	**-**	**-**	**-**	**-**	**-**	**132 415**	**318 007**	**64 913**	**219 378**	**77 727**	**410 097**	**472 477**	**481 325**	**464 586**	**404 555**	**589 181**	**297 363**	**210 566**
Corporate Income Tax Rate		27%																		
Corporate Income Tax	**$'000**	**(1 118 499)**	**-**	**-**	**-**	**-**	**-**	**(35 752)**	**(85 862)**	**(17 526)**	**(59 232)**	**(20 986)**	**(110 726)**	**(127 569)**	**(129 958)**	**(125 438)**	**(109 230)**	**(159 079)**	**(80 288)**	**(56 853)**
Free Cashflow																				
Revenue	$'000	13 522 592	-	-	618 931	888 354	887 641	1 190 016	1 093 845	697 740	929 174	852 870	959 968	976 918	939 220	878 993	750 169	905 903	540 321	412 527
Operating Costs	$'000	(6 400 238)	-	-	(316 278)	(400 224)	(424 730)	(514 498)	(576 365)	(447 437)	(473 444)	(582 595)	(409 684)	(409 303)	(403 817)	(387 437)	(327 805)	(303 223)	(229 714)	(193 684)
Capital	$'000	(2 719 766)	(395 584)	(649 939)	(118 534)	(252 921)	(197 774)	(149 118)	(128 155)	(267 878)	(249 476)	(132 777)	(71 665)	(22 850)	(22 284)	(20 060)	(14 262)	(13 244)	(13 244)	-
Pre-Tax FCF	**$'000**	**4 402 588**	**(395 584)**	**(649 939)**	**184 119**	**235 209**	**265 136**	**526 400**	**389 326**	**(17 574)**	**206 253**	**137 498**	**478 619**	**544 765**	**513 119**	**471 497**	**408 102**	**589 436**	**297 363**	**218 843**
Corporate Income Tax	$'000	(1 118 499)	-	-	-	-	-	(35 752)	(85 862)	(17 526)	(59 232)	(20 986)	(110 726)	(127 569)	(129 958)	(125 438)	(109 230)	(159 079)	(80 288)	(56 853)
Post-Tax FCF	**$'000**	**3 284 089**	**(395 584)**	**(649 939)**	**184 119**	**235 209**	**265 136**	**490 648**	**303 464**	**(35 101)**	**147 021**	**116 512**	**367 893**	**417 196**	**383 161**	**346 058**	**298 872**	**430 357**	**217 075**	**161 990**

Table 22.11 and Table 22.12 summarise the before and after-tax valuation indicators for the Project. Figure 22.3 shows the annual and cumulative pre-financing project free cashflows. Figure 22.4 shows the breakdown of the annual Project cashflows.

Table 22.11 : Before Tax Financial Results		
Before-Tax Valuation Indicators	**Unit**	**Value**
Undiscounted cumulative cashflow	US$M	4 403
NPV5	US$M	2 289
NPV8	US$M	1 540
NPV10	US$M	1 172
Payback period (from start of operations)	years	3.5
IRR before tax	%	24%

Table 22.12 : After Tax Financial Results		
After-Tax Valuation Indicators	**Unit**	**Value**
Undiscounted cumulative cashflow	US$M	3 284
NPV5	US$M	1 675
NPV8	US$M	1 100
NPV10	US$M	817
Payback period (from start of operations)	years	3.5
IRR	%	21%

Figure 22.3 : Pre-Financing Project Cashflows

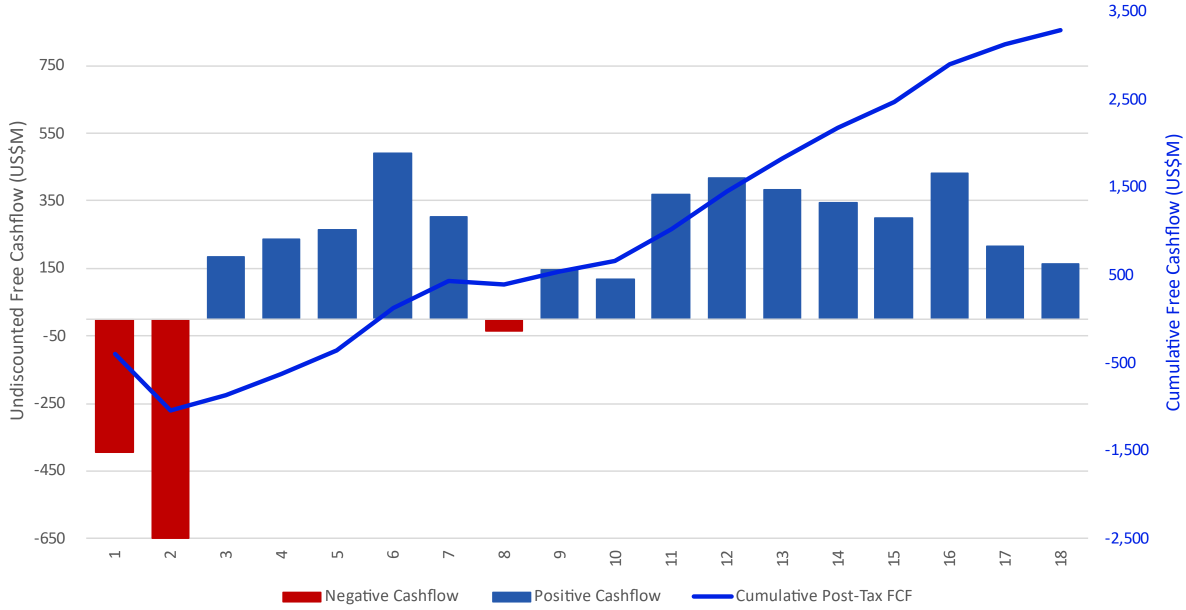
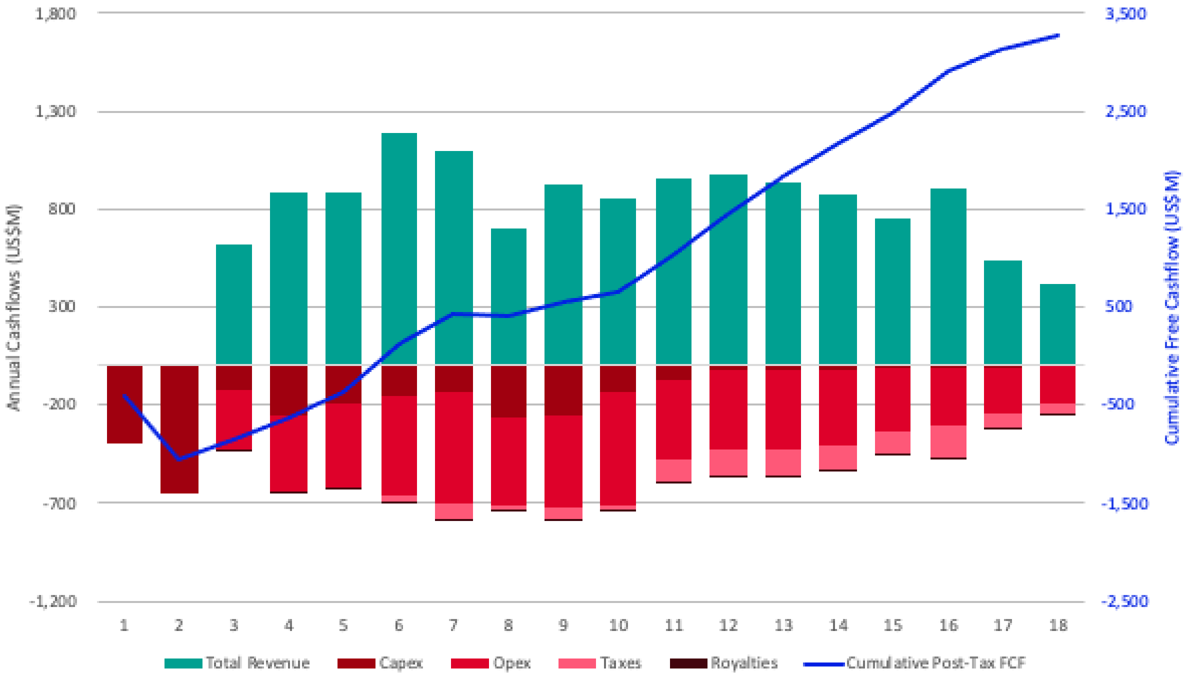


Figure 22.4 : Project Cashflows Breakdown



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 22.5 to Figure 22.10.

Figure 22.5 : Project NPV Sensitivity Spider Graph – Metal Price and Discount Rate

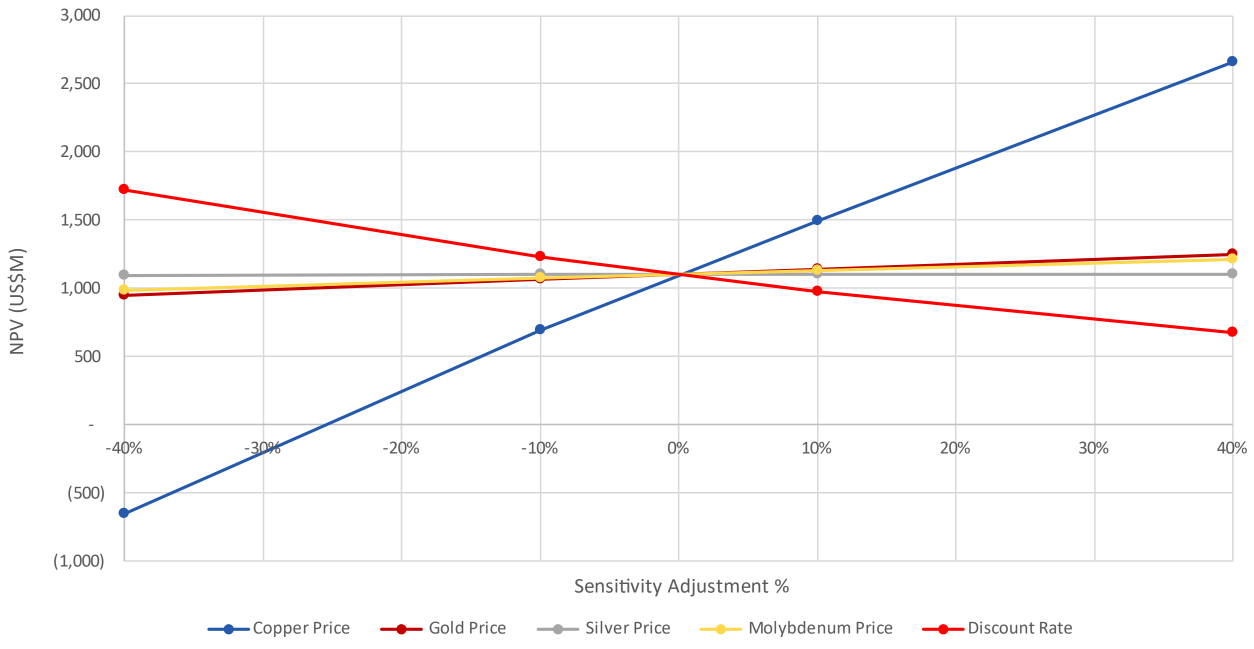


Figure 22.6 : Project NPV Sensitivity Spider Graph – Process Grade or Recovery

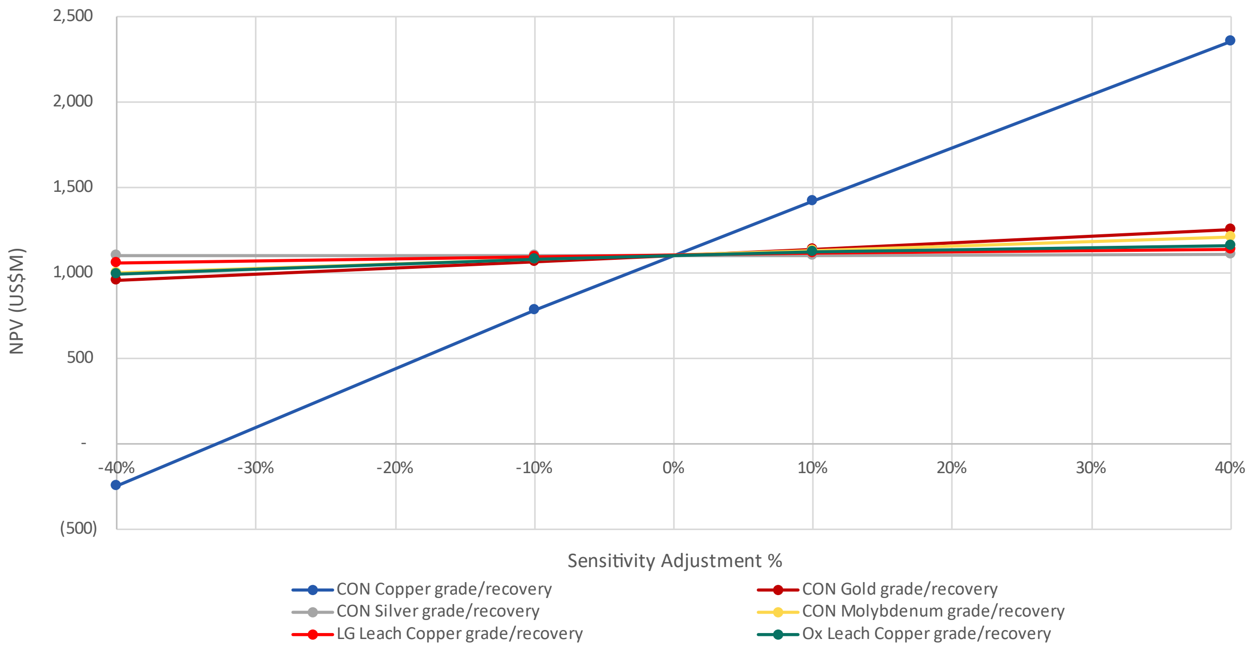


Figure 22.7 : Project NPV Sensitivity Spider Graph – Capital and Operating Costs

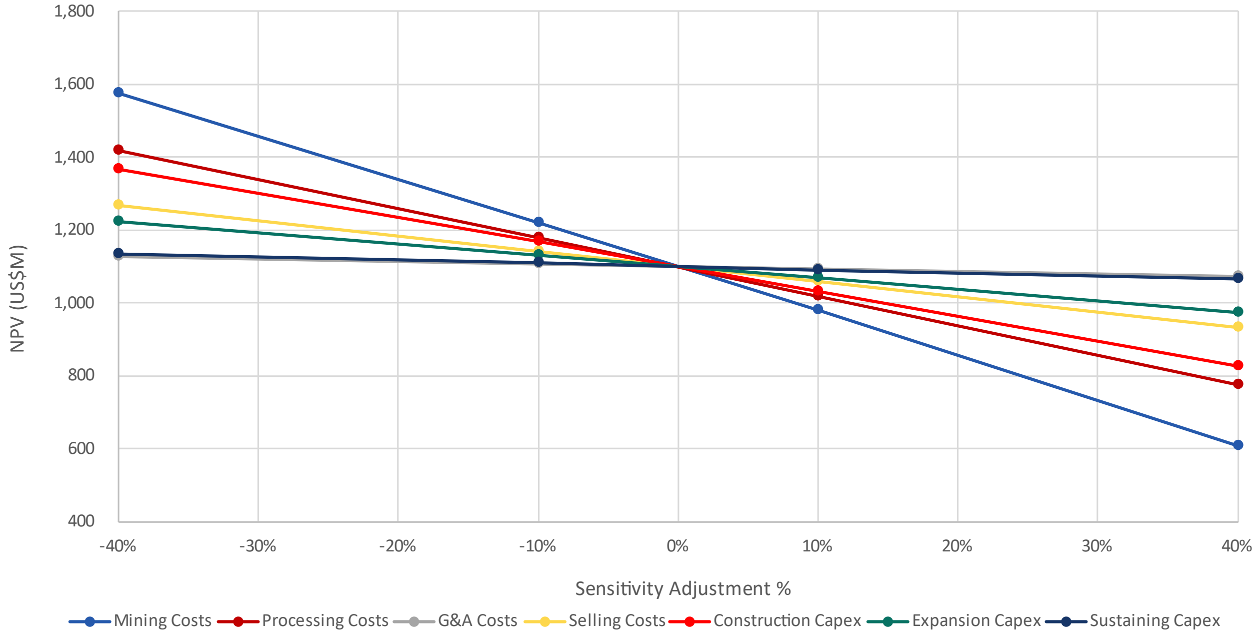


Figure 22.8 : Project IRR Sensitivity Spider Graph – Metal Prices

Figure 22.9 : Project IRR Sensitivity Spider Graph – Process Grade or Recovery

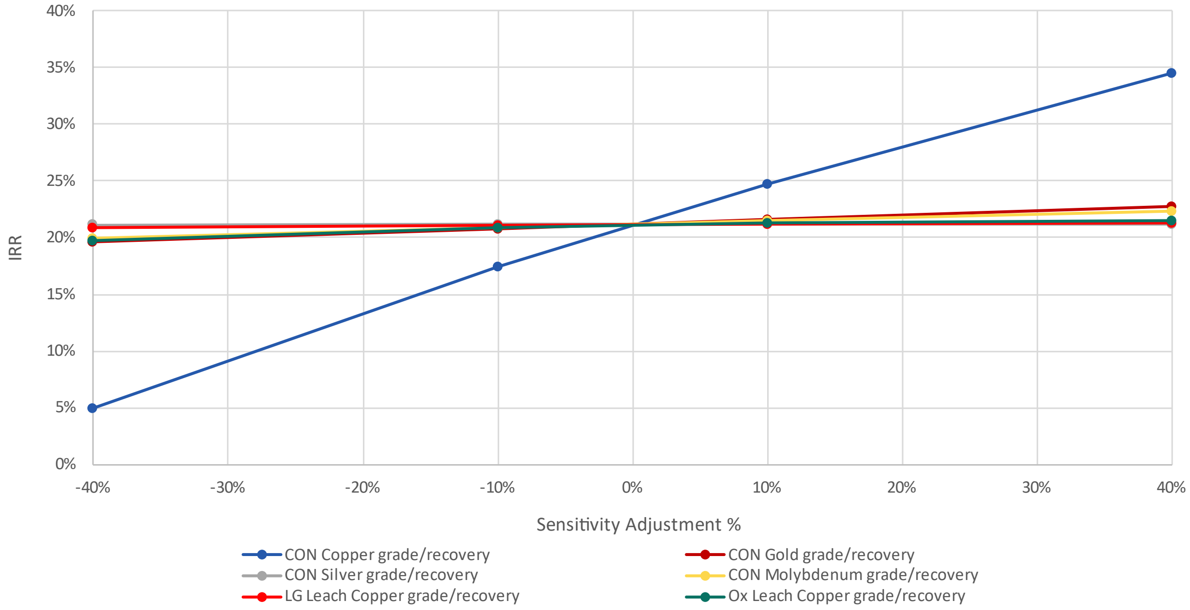
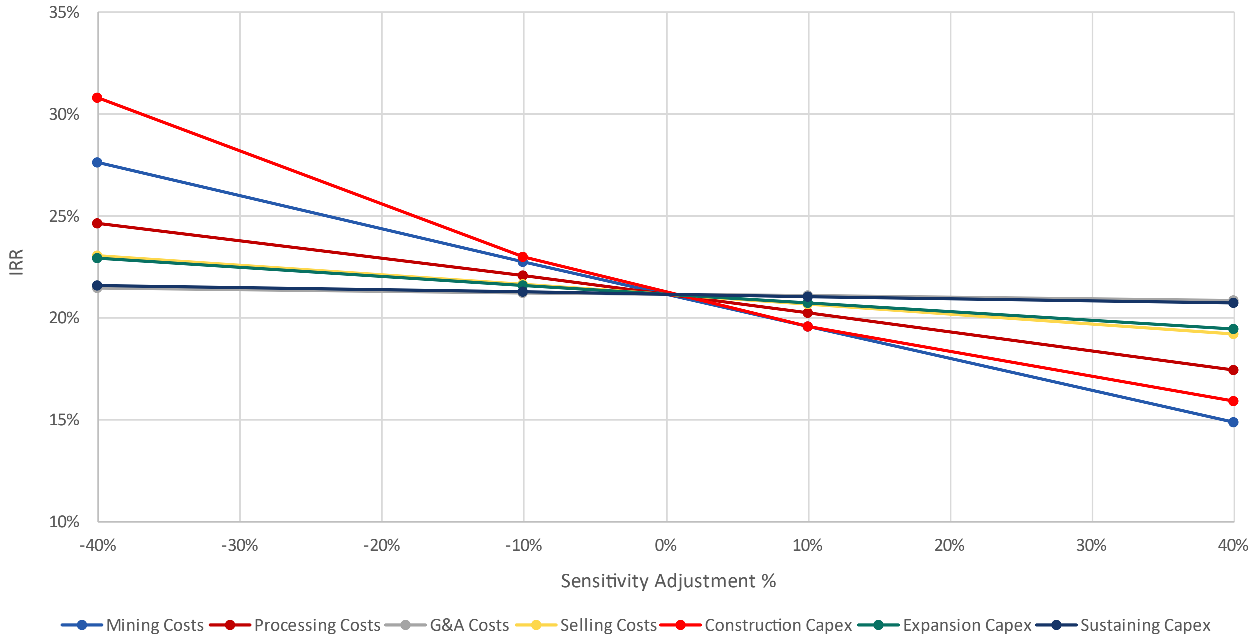
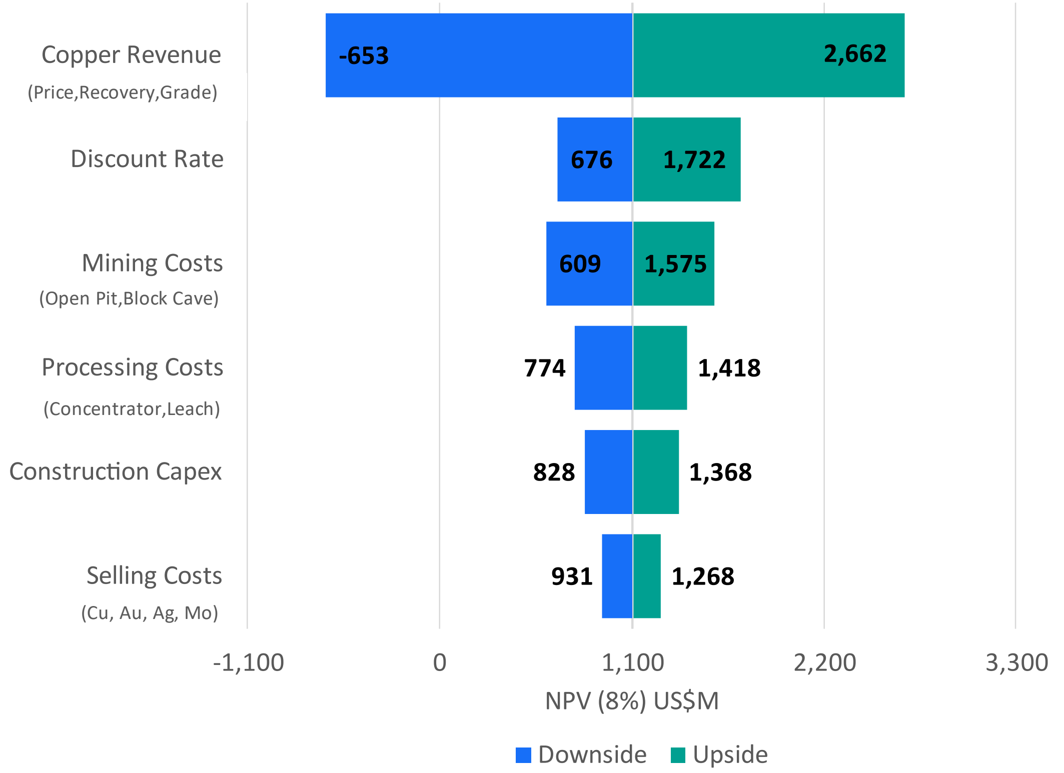


Figure 22.10 : Project IRR Sensitivity Spider Graph – Capital and Operating Costs



The tornado chart in Figure 22.11 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 22.11 : 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 22.12 shows the Project's post-tax NPV8 and IRR at a range of copper prices.

Figure 22.12 : Project Post-Tax NPV (8%) and IRR as a Function of Copper Price

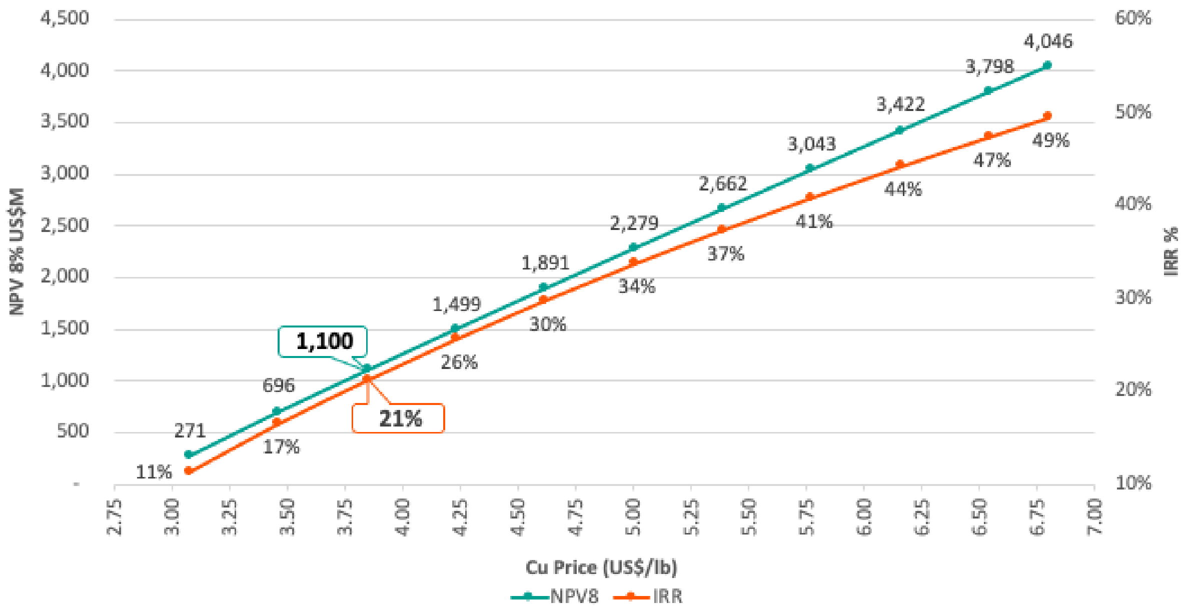


Table 1.13 shows financial metrics at lower and upper copper price sensitivity ranges (±10%).

Table 22.13 : Copper Price Ranges – Lower-, Base-, and Upper- Case Sensitivity Scenarios

Project Metric		Units	Copper Price		
			Lower (-10%) (US$3.50/lb)	Base (US$3.85/lb)	Upper (+10%) (US$4.20/lb)
Pre-Tax	NPV8%	US$M	1 040	1 540	2 030
	IRR	%	19%	24%	29%
Post-Tax	NPV8%	US$M	731	1 100	1 460
	IRR	%	17%	21%	25%
Annual Average EBITDA		US$M	384	445	506
Annual Average Free Cash Flow		US$M	161	205	250
Payback Period (From First Production)		yr	4.25	3.50	3.25
Post-Tax $NPV_{8\%}$ /Start-up Capital			0.7	1.1	1.4

23 Adjacent Properties

HCH knows of no immediately adjacent properties which might materially affect the interpretation or evaluation of the mineralisation or exploration targets of the Project.

24 Other Relevant Data and Information

This section has been taken directly from the technical report titled, "Costa Fuego Copper Project: NI 43-101 Technical Report & Preliminary Economic Assessment (PEA)" dated August 14, 2023 with an effective date of June 28, 2023. The Mineral Resource estimate update, as described in this Report, does not materially affect the mineral resource inventory that formed the basis of the 2023 PEA and no new scientific or technical information has been developed that would materially affect the outcome of the 2023 PEA and, therefore, the results and conclusions of the 2023 PEA are considered current and have been restated in this Report.

24.1 Cortadera Large Open Pit Option

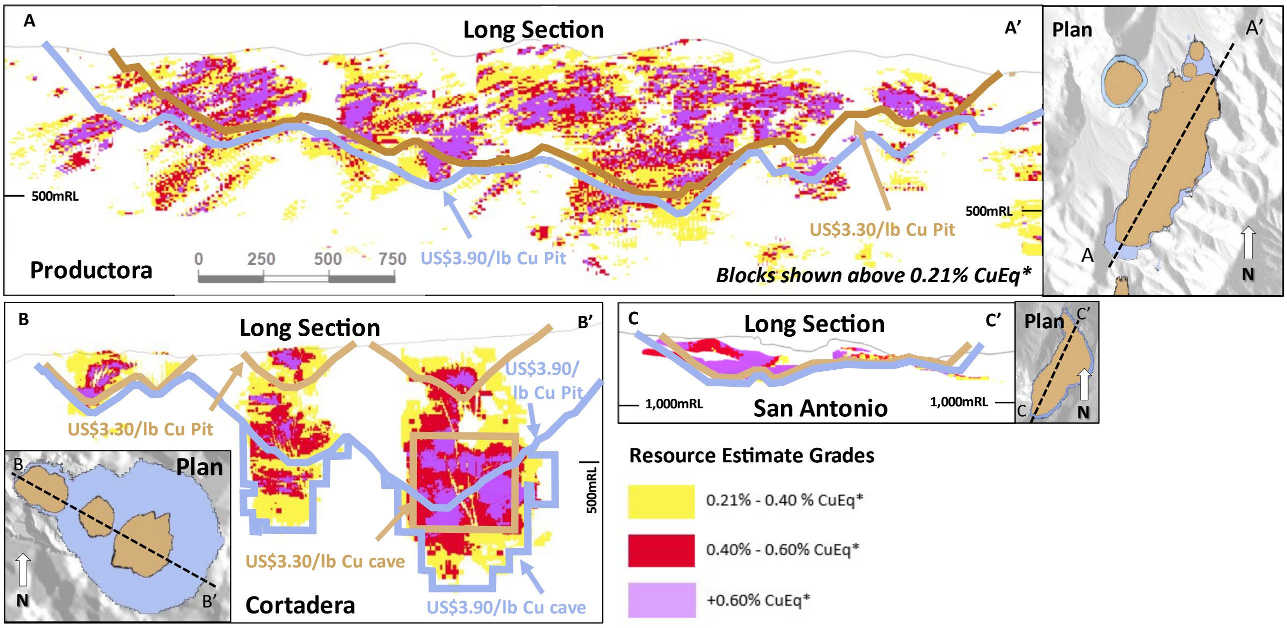
The Project currently considers a combination of open pit and underground block cave mining at Cortadera. With increasing copper price in mind, an alternative scenario would be the option to replace the block cave with a single large scale open pit mine.

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 24.1) the growth potential at Costa Fuego when a higher copper price is applied, resulting in a larger pit surface volume and greater quantity of production feed.

Figure 24.1 : Comparison Shown Between US$3.30/lb and US$3.90/lb Copper Price Pit Shells (Note 2022 Resource Model Blocks shown).



25 Interpretations and Conclusions

Section 25 (except 25.1, 25.2, 25.3) have been taken directly from the technical report titled, "Costa Fuego Copper Project: NI 43-101 Technical Report & Preliminary Economic Assessment (PEA)" dated August 14, 2023 with an effective date of June 28, 2023. The Mineral Resource estimate update, as described in this Report, does not materially affect the mineral resource inventory that formed the basis of the 2023 PEA and no new scientific or technical information has been developed that would materially affect the outcome of the 2023 PEA and, therefore, the results and conclusions of the 2023 PEA are considered current and have been restated in this Report.

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 20 000 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 (~4727 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 agreed upon payment. An Option Agreement was also negotiated with Bastion Minerals for the right to acquire 100% of Bastion's Cometa Project, located approximately 15 km southeast of Cortadera. The exploration and mining concessions cover approximately 5 600 ha. An Option Agreement to acquire 100%

with private parties for the historical copper mine areas Marsellesa and Cordillera, located approximately 10 km southwest of Productora was also completed by HCH.

As per the community engagement strategy, 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. This 2023 PEA was the first Study to be completed on the combined Mineral Resource Estimates, incorporating Cortadera and Alice as copper-gold porphyry deposits, Productora as a copper-gold-molybdenum breccia with porphyry and IOCG characteristics, and San Antonio as a skarn-hosted copper deposit.

Drill spacing across Costa Fuego 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 2022 Mineral Resource Estimates (MRE) for the Cortadera, Productora, Alice, and San Antonio deposits informing the 2023 Preliminary Economic Assessment (PEA) 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. Additional drilling would be required to potentially convert Inferred and unclassified material within the constraining surfaces to Indicated or Measured Mineral Resources for consideration in future studies. The 2024 MRE update, as described in this Report, does not materially affect the mineral resource inventory that formed the basis of the 2023 PEA and no new scientific or technical information has been developed that would materially affect the outcome of the 2023 PEA and, therefore, the outcome of the 2023 PEA is current.

25.4 Mine Plan

QP Statement:

As this is a PEA Study, the mining engineering and mine planning work is indicative and conceptual in nature. Caution should be exercised when reading the mining sections of the PEA as the level of study accuracy is deemed relatively low and further engineering work and further studies could modify the future economic results.

The geotechnical parameters are based on some reasonable levels of technical work/studies, however, further geotechnical and geohydrology studies are required to further improve the mine planning parameters. The next level of mining engineering involves pit designs. Although modified open pit shells were developed, allowing for slopes to be modified for potential access ramps, pit designs will potentially contain more waste rock and production feed loss, when compared to an open pit shell.

The block cave assessment is at a concept level of accuracy and was conducted to support the PEA only. The block cave portion of the PEA mine schedule is not underpinned by detailed draw-bell designs and schedules and therefore, the production ramp-up schedule could be deemed optimistic and practically challenging to realise per the current mine schedule. Additional detailed block cave undercut, and draw-bell design work is required whilst developing more detailed mine scheduling, to refine this part of the overall mine schedule. There are other potential opportunities, like changing the development start dates, to ensure that a slightly longer block cave production ramp-up period won't impact plant feed with similar feed and cost results. Based on the PEA level of assessment it was concluded that the block cave makes a positive contribution to the overall mine and processing plan and should therefore be considered in the overall decision-making process, such as processing, infrastructure, mine life, etc.

The mining schedules is based on the best estimates and assumptions available at the time of its creation and will require the Project team to minimise disruptions and implement mitigation measures during mining. However, the effectiveness of these measures in avoiding delays cannot be guaranteed.

Open pit mining methods have been selected as the key exploitation technique for the Costa Fuego deposits, these methods are supported by near surface mineralisation which allows for low waste/mineral strip ratios and associated cashflow. There is additional underground block cave exploitation potential at the Cortadera Cuerpo 3 deposit.

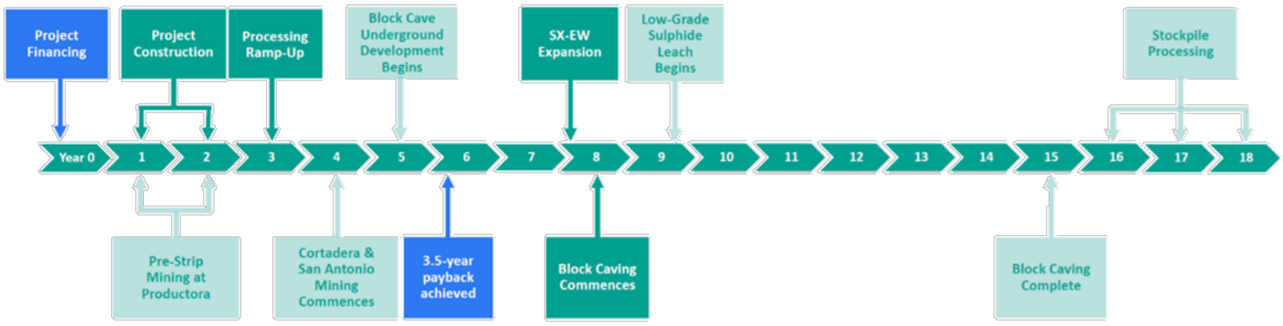
A 20 Mt/a sulphide concentrate processing and 10-12 kt/a cathode SX-EW production throughput rate was considered during the PEA mine scheduling.

The mine plan contemplates mining a total of 471 Mt of production feed material and 627 Mt of waste (1.8:1 open pit strip ratio mined) over an 18-year mine production life, including stockpile reclamation. The current LOM plan focuses on achieving steady plant feed production rates, and mining of higher-grade material early in schedule, as well as balancing grade and strip ratios. An elevated cut-off grade is applied throughout the mine life.

The process feed cut-off strategy, defined by the mining schedule, requires deferring any low-grade production feed produced to maximise the plant's throughput rate and the Project NPV. The low-grade production feed would be placed on independent sulphide heap leach dumps, bringing the leached product forward in the production schedule. Figure 25.1 illustrates the Costa Fuego planned production timeline.

The schedule produces an annual copper equivalent metal production profile of over 100 kt for a 16-year mine life (including over 110 kt for the first 14 years).

Figure 25.1 : Costa Fuego Production Timeline



25.5 Metallurgical Testwork

Extensive relevant comminution, flotation and column leach metallurgical testwork was conducted on the copper-gold-molybdenum sulphide production feed as well as the copper oxide production feed, with the aim of providing design criteria for the copper sulphide process plant and copper oxide process plant and to provide metallurgical recoveries for use in the Project financial modelling.

The quantity and quality of metallurgical testwork data developed from the Project drill core samples for both the oxide and sulphide deposit, has led to the development of optimized energy efficient comminution circuits for both plants.

The oxide comminution circuit is followed by a standard acid copper heap leach recovery process and solvent extraction and electrowinning process to produce sheet copper.

The sulphide comminution circuit is followed by a standard copper sulphide flotation circuit followed by a regrind and molybdenum flotation recovery circuit to produce good quality copper and molybdenum concentrates.

QP Statement: It is the opinion of the Qualified Person that the metallurgical testwork completed on the Cortadera, Productora and San Antonio deposits adequate for use in the preliminary economic analysis contained in this Report. Additional supporting metallurgical testwork such as filtration and thickening, is planned to complete the testwork program for the PFS.

25.6 Infrastructure

The Project has the benefit of being able to utilise and/or tie into the considerable existing infrastructure and services in the Vallenar/Huasco region.

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.

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. Water, power and road access will also have to be established from Productora to the Cortadera deposit.

The Tailings Storage Facility (TSF) will be located to the northwest of the sulphide process plant. The TSF embankments are largely contained by natural topography and three embankments constructed from open pit mine waste.

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.

25.7 Environmental, Permitting and Social Considerations

A number of environmental studies have been and are currently being conducted at the Project site in support of development of the Environmental Impact Assessment System and as required for environmental and operational permits.

Studies conducted have investigated soils, climate and meteorology, geology, geochemistry, biological resources, cultural and anthropological sites, socio-economics, hydrogeology, and water quality.

The QP is not aware of political, environmental, or other risks that could materially affect the potential development of the Mineral Resources other than those detailed in Section 25.12.

25.8 Markets and Contracts

Long-term metals prices assumed for the economic analysis for copper, molybdenum, gold and silver are what Wood considers to be an industry consensus on the forecast and are supported by Wood's quarterly guidance for long-term metal prices.

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

The estimate is a Type 4 estimate according to Wood and ACCE International standards, with an accuracy of -40% to +40%.

All capital cost estimates have been escalated to fourth quarter 2022 US dollars.

The estimates are based on a combination of direct quotes, benchmarking and Mintrex base data which has been escalated and scaled accordingly.

The initial capital cost is estimated to be US$1046 M, with a further US$708 M allocated for planned expansion. The sustaining capital is estimated to be US$966 M and a further $48 M for post-production, and closure costs. The total capital cost for the Project is therefore US$2768 M.

25.10 Operating Cost Estimates

The estimate is considered to be PEA study level with an accuracy of -40% to +40%.

The estimates are based on a combination of direct quotes, benchmarking and HCH supplied data.

The operating cost estimated has been escalated to second quarter 2022 US dollars and factored for the plant throughput increase for the sulphide plant.

The overall life of mine operating cost is US$6 400 M.

25.11 Economic Analysis

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$1100 M with 21% IRR and 3.5-year payback period.

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.

Closure costs in the financial model are estimated at $48 M, or 5% of direct construction capital costs plus expansion capital for Cortadera infrastructure and rope conveyor and are assumed to be offset by salvage value.

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.

During the PFS phase, identified risks will need to be investigated further and possibly reduced or eliminated. Similarly, further investigation and evaluation of opportunities may allow their incorporation into the 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.

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.

The most significant risks were evaluated in a risk review in 2022 and 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 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	Continued development of good relationships with the local stakeholders, particularly with the nearby communities is critical to the success of the Project. A number of environmental studies have been and are currently being conducted at the Project site in support of development of the Environmental Impact Assessment System and as required for environmental and operational permits. Studies conducted have investigated soils, climate and meteorology, geology, geochemistry, biological resources, cultural and anthropological sites, socioeconomics, hydrogeology, and water quality. The Mining and Society Department of the Ministry of Mining helps to strengthen the relations between mining companies and local communities. That department

Table 25.1 : Key Risks		
Risk	**Explanation**	**Possible Mitigation**
	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 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	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. HCH is actively progressing EIA studies and the approvals process.
Cost Risk	Capital and operating cost escalation as Project plans and parameters change or are refined.	

Table 25.1 : Key Risks		
Risk	**Explanation**	**Possible Mitigation**
	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.	
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 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.	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. 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	Unexpected variations in the quantity of process feed material, grade or recovery rates.	Process plant design is based on current operating schedule.
Infrastructure	Tailings management facility (TMF) reaching design capacity; Investment may be required to increase TMF capacity, which could lead to delays in mine production.	Address the specific TMF studies needed and conduct the necessary work.

Table 25.1 : Key Risks		
Risk	**Explanation**	**Possible Mitigation**
	Tailings Deposition and geochemistry of tailings.	In further studies undertake testing of expected tailings composition to confirm geochemistry properties and any impact(s) on tailing design.

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 in a pre-feasibility study 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 (assuming positive results) there is opportunity to optimise Capex and Opex.
Additional revenue streams	Include cobalt in the process feed inventory.	After completion of initial test work, 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.
	Consider further recovery of acid via pyrite roasting.	After completion of initial test work, 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.
Optimise materials handling systems for Cortadera.	RopeCon vs. conveyor vs. autonomous haulage options.	Opportunity to optimise Capex and Opex
Geotechnical considerations being better than those currently assumed	Cortadera and San Antonio will require further mining geotechnical investigations compared to Productora.	Improve slope angles in open pits which could potentially result in reduced stripping ratios.
Block cave mining strategy	Undertake a study to assess the Block cave vs. open pit trade-off. If the Block cave proves to be more viable, then Footprint layout design optimisation and further detailed planning.	Opportunity to optimise Capex, Opex and production schedule.
Optimise process flowsheet	Optimise grinding mill sizes and SAG/ball mill split. Investigate use of HPGRs as alternative technology. Investigate elimination of pebble crusher from the grinding circuit as comminution parameters suggest that the process feed may be SAG-millable. Investigate possibility of a volume reduction in the flotation feed conditioning tank by reducing the residence time from the current five minutes to one or two minutes. Investigate to eliminate the requirement for three trash screens ahead of the flotation circuit. Investigate use of flotation column cells instead of conventional flotation cells.	Optimisation of flowsheet reflecting in Capex and Opex.
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 during the PFS.	

25.13 Conclusions

The results of this 2024 Mineral Resource Estimate (MRE) update at Costa Fuego further supports the Project's intended Prefeasibility Study (PFS) completion.

The results of this Report indicated that the Project demonstrates favourable economic potential that warrants further work toward the completion of pre-feasibility studies.

- 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 metallurgical testwork undertaken is reasonably extensive and suitable for this level of study. The design of the processing circuit is based on this testwork data in conjunction with assumptions based on typical industry values.
- The mineralised material is of moderate competency and hardness, and amenable to grinding in a conventional circuit. The mineralogy is fine grained and testwork indicates a requirement to re-grind to a fine particle size to achieve adequate liberation for flotation, as is common within the industry.
- Overall recoveries are estimated at 91% for copper and 75% for molybdenum, which are contained in metals concentrates.
- The Project has been designed to meet current social and environmental management practices.

26 Recommendations

Section 26 (except 26.1, 26.2, 26.6, and 26.8) has been taken directly from the technical report titled, "Costa Fuego Copper Project: NI 43-101 Technical Report & Preliminary Economic Assessment (PEA)" dated August 14, 2023 with an effective date of June 28, 2023. The Mineral Resource estimate update, as described in this Report, does not materially affect the mineral resource inventory that formed the basis of the 2023 PEA and no new scientific or technical information has been developed that would materially affect the outcome of the 2023 PEA and, therefore, the results and conclusions of the 2023 PEA are considered current and have been restated in this Report.

26.1 Pre-Feasibility Study

It is recommended to advance the Project to a complete Prefeasibility Study (PFS) to incorporate the recent, current and planned drilling and exploration, testwork activities into the process plant and associated infrastructure for the Project.

The PFS is already well advanced, with minimal study expenditure required to finalise the report. The completion of the PFS is recommended to advance the Project to a place where any additional detailed information necessary provides support of capital and operating cost estimates which lead to a potential Project development decision.

The PFS would be expected to solidify the base case through additional field work, testwork, engineering studies, capital and operating cost estimates and financial analysis.

In addition, the PFS should also explore the following options:

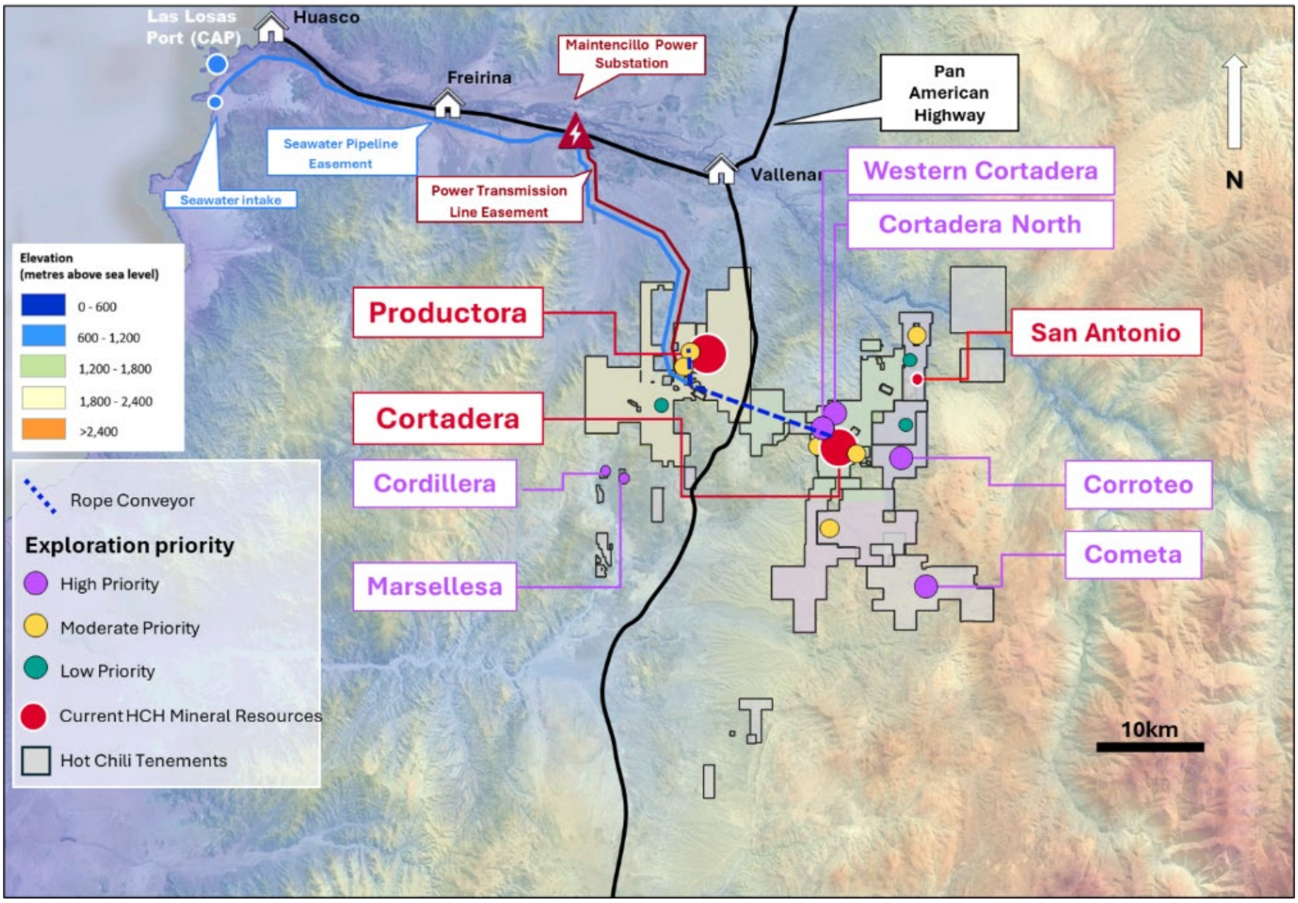
- Potential recovery of cobalt for an additional source of revenue
- Potential recovery and roasting of pyrite to enable the production of acid and generation of electricity
- Coarse particle flotation to reduce costs and improve tailings deposition
- Other metallurgical optimizations to increase recovery, such as chalcocite recovery
- Continuation of development study program, which will further refine metallurgical, geotechnical, and hydrogeological model inputs
- Investigate a large single open pit scenario for Cortadera (no underground block cave) with the potential to materially increase processing feed inventory and mine life
- The incorporation of further potential Mineral Resource increases and upgrades provides an opportunity to potentially increase the mine life and scale of production.

Figure 26.1 : Costa Fuego Project Roadmap

26.2 Exploration and Resources

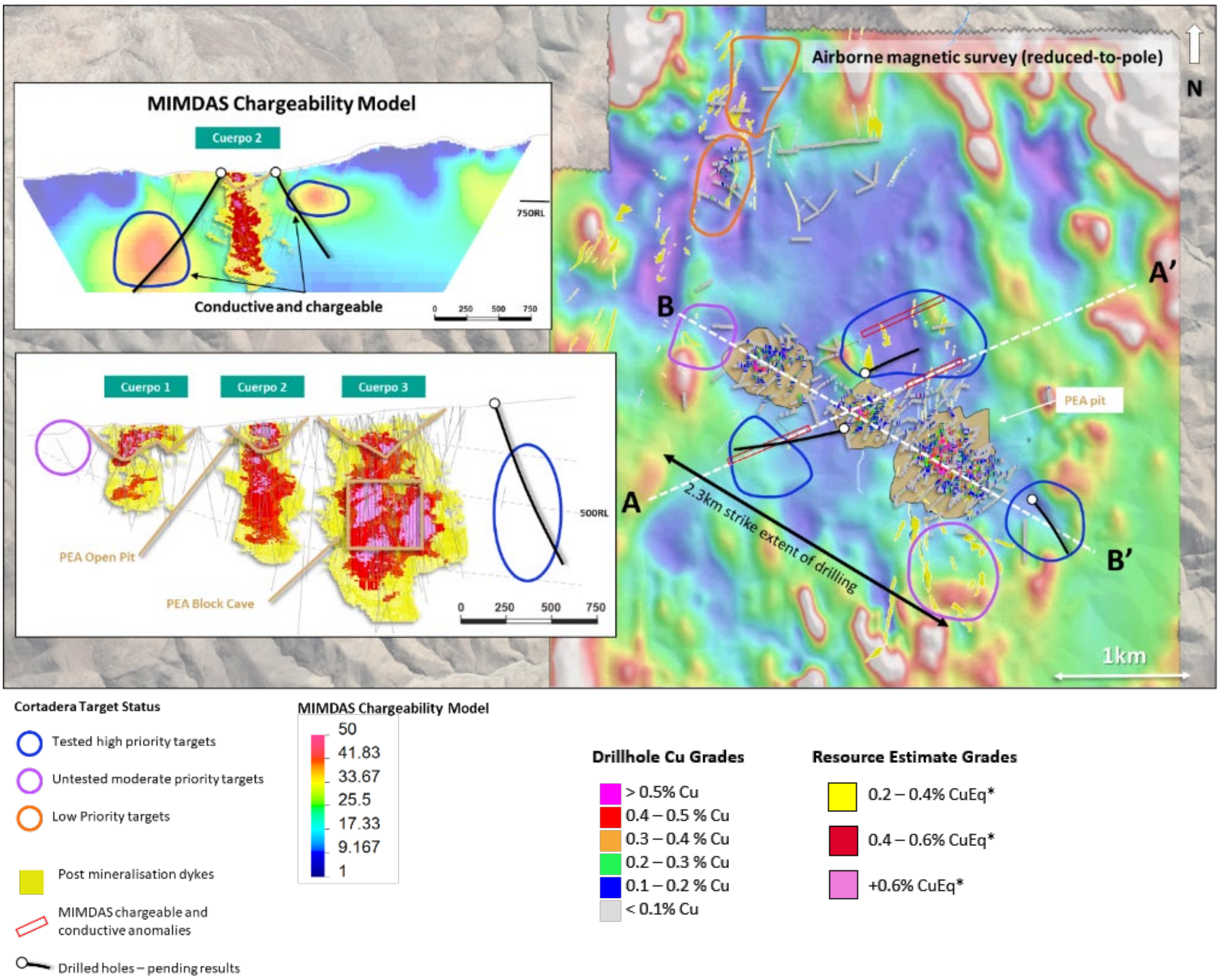
Recommendations for advancement of the Exploration and Mineral Resource aspects of the Costa Fuego Project focus on resource expansion, development drilling such as hydrogeological and geotechnical drilling, particularly around regional exploration targets (Figure 26.2). Considering the PEA completion and the results of this Report, the exploration target criteria are focused on large scale copper deposits proximal to current planned infrastructure.

Figure 26.2 : Plan Map Showing the Costa Fuego Growth Pipeline



Resource development drilling completed at Cortadera through 2023 and Q1 of 2024 aimed to test targets remaining following the pause in drilling in early 2021. Drilling provided critical information on target areas, which are now being reviewed by the HCH geology team. Planned geophysical surveys will be used in conjunction with completed drilling, surface mapping and soil geochemistry to determine additional targets across the highly prospective Cortadera tenements. Figure 26.3 below shows planned exploration activity at Cortadera.

Figure 26.3 : Plan to Test Large Porphyry Targets at Cortadera

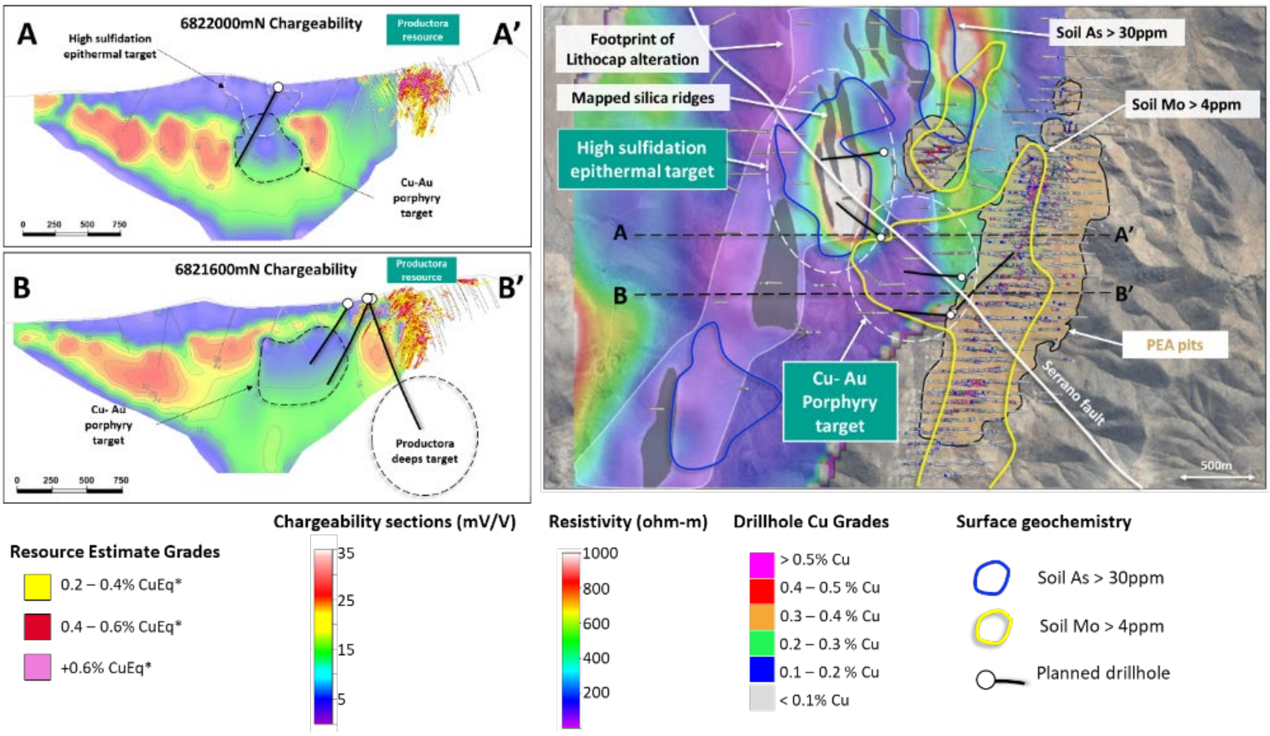


The Productora Resource and mineralisation remains open at depth in several places as well as laterally. While it is understood the width of mineralisation at Productora decreases with depth, additional drilling may add incremental mineralised tonnage, and will also aid in the upgrade of existing Inferred resource to Indicated classification.

Additional exploration drilling around the Alice area is still very prospective. The updated 2024 Alice resource estimate has identified multiple target areas for potential resource extension.

Figure 26.4 outlines planned exploration activity across Productora and Alice.

Figure 26.4 : Large-scale Targets Adjacent to Productora and Alice Resources



Work at San Antonio will focus on improving confidence in the depletion shapes used for the San Antonio MRE. With the spatial uncertainty present, both the volume of material depleted as well as the resource classification surrounding workings can be considered conservative.

26.3 Mining

The mine plan in this Report is conceptual in nature and some elements require further study to refine concepts and increase accuracy of the estimates.

A mining study will be required, including pit and underground optimisation, mine designs, mining schedule/sequence, waste dump designs, mine fleet analysis and operating cost and capital cost estimates. The mining study will cover open pit and underground mining, geotechnical and ground and surface water studies.

The next stage of study will include:

- Investigate a large single open pit scenario for Cortadera, trade-off investigations for large open pit vs. underground block cave for the Cortadera area
- Detailed studies and costings to support a PFS.

Future Geotech work required to support further mining studies includes:

- Additional Geotech drilling to support increasing study detail
- Pit and waste dump slope stability studies based on final waste dump designs and capacities for all areas

- Dump leach location and stability investigations.

26.4 Metallurgy and Processing

A specific metallurgical drilling program has been completed to provide sufficient core and variability samples across all deposits for the PFS. This was based on discussions with relevant project geology, mining and metallurgy personnel, and has obtained targeted representative samples of the materials that are likely to be processed by the Project. The sampling was designed to provide both spatial variability data and typical life of mine (LOM) representative production feed samples.

Further comminution, flotation, filtration and thickening testwork is currently underway on sulphide ores, based on sample availability and representivity.

Following completion of the testwork program noted above, the current process plant design criteria (PDC) will be updated for the PFS. Once the PDC has been updated, the engineering of the process plants will be completed to a sufficient level of detail to allow preparation of capital estimates to a +/-30% accuracy.

26.4.1 Further Sulphide Testwork

Testwork recommended to be undertaken during the PFS includes the following:

- Confirm the comminution power requirements for each deposit to be used to design the common comminution circuit for all deposits
- The design is to include reagent feed capability to copper scavengers for two additional water-soluble copper collectors, for SMBS and for diesel
- The copper rougher concentrate pumping is to have alternative destinations such as to regrind and/or copper rougher cleaning
- Copper regrind is to have capacity to grind all concentrates (rougher plus scavenger) and have turndown capability to grind only scavenger concentrates
- Molybdenum flotation is to be designed to operate continuously but may not be required with some feeds
- Alice and Cortadera OP are low grade molybdenum deposits while Productora and Cortadera UG are high grade molybdenum deposits.
- It is possible that the blend being treated at any given time could have a very low grade (<50 g/t Mo) or a very high grade (>500 g/t Mo). To smooth the feed to the molybdenum circuit it is recommended that some significant storage (up to 48 h) be incorporated in the design for the molybdenum circuit rougher concentrate. This will allow first stage molybdenum concentrate from rich feeds to be accumulated and then for the accumulated material to be used when grades are low. The 48 h maximum is to allow the mine to be directed to produce lower molybdenum grade plant feed in an extended high grade event.
- An investigation is required into the gold being lost to rougher tails, especially in the Cortadera underground deposit samples.
- An extended QEMScan rare phase search is recommended for Cortadera UG rougher tails, using the sample that has been already provided and analysed in a single mounted section. In the extended analysis

multiple mounts will be made specifically for this purpose (many more particles will be visible in each mount because efforts will not be made to separate particles with graphite). The number of mounts will be a matter for recommendation from ALS Mineralogy based on the grade of the sample

- It is also recommended that a sample of Cortadera UG feed and a sample of rougher tailings be leached with cyanide under normal conditions to establish how much of the gold is easily leached from the whole sample and from what remains after the sulphides are removed by flotation
- A program of flotation tests assessing copper collectors that are also good for free gold should also be performed
- An investigation is required into the use of additional copper collectors in the scavenger flotation stage of the Cortadera OP flowsheet
- Various supplemental copper collectors will be tested in this program, and they will be evaluated on ability to increase copper recovery while remaining selective against pyrite
- Recoveries of molybdenum and gold will be monitored during this work
- The optimal collector and addition rate will be tested in the scavenger flotation stage with samples from the other three deposits.

26.4.2 Further Oxide Testwork

Testwork recommended to be undertaken during the PFS includes the following:

- Additional metallurgical drilling/sampling of all areas, especially early sources of process feed in the mine schedule
- Additional copper solubility and acid consumption testwork (sequential analysis) at Productora to be able to directly estimate and inform the block model
- Additional bottle roll and 6 to 8 m column tests to add additional data for scale up, optimisation and to further understand spatial variability
- Further column tests are performed on oxide samples from Productora and Cortadera as part of the ongoing works with Nova Mineralis.

This work is included in the budget for the PFS.

26.5 Infrastructure

Major site infrastructure engineering and design has been performed to PEA level.

In most areas, this will be the basis for the commencement of PFS engineering. This work is included in the budget for the PFS.

As part of the PFS development of the infrastructure the tailings facility will be developed.

26.6 Port

In March 2024, Hot Chili executed a Memorandum of understanding (MOU) with Puerto Las Losas SA (PLL) for the right to negotiate a binding Port Services Agreement for 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, 50km west of Costa Fuego.

In consultation with Hot Chili, 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, Hot Chili 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 Feasibility Study will include bulk loading alternatives for copper concentrates from existing facilities, potentially with or without modifying the existing infrastructure for the operating port.

26.7 Environment and Social

The environmental and social study to be included as part of the PFS will be limited to summarising the Costa Fuego Environmental Impact Assessment (EIA) study. HCH has planned to complete the EIA prior to finalising the PFS, therefore, valuable information related to base lines, impact assessment and management plans for environmental and social elements will be integrated into the PFS. The key issues determining the EIA timing are the development of the hydrogeology model, getting preliminary agreements with families to be resettled and running the stakeholder engagement plan.

The Project EIA will be managed as a separate study and it will be submitted to Chilean authorities for their consideration prior to the finalisation of the PFS.

After receiving the EIA, the SEA, after receiving has 120 days to communicate a resolution on the application. However, the SEA, on average, approves projects EIAs in approximately 16 months; therefore, its completion before finalising the PFS is of vital importance.

After an administrative process, the SEA issues a resolution allowing the construction, operation or closure of the Project and certifies that it complies with the applicable environmental regulations (Resolución de Calificación Ambiental or RCA).

26.8 Work Plan

The estimated work program budget for the completion of Project PFS is summarised below (Table 26.1).

Table 26.1 : Future Work Program	
Work Program	**Cost (US$M)**
G&A	1.5
Exploration and Resources	3.0
Development Studies	2.7
Contingency and Opportunity	0.3
Total	**7.5**

The Qualified Persons (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
min	Minutes/minimum
mm	Millimetre
mRL	Mean Reduced Level (Mean Sea Level)
Mt/a	Million tonnes per annum
Mo	Molybdenum
No.	Number

Table 27.1 : Abbreviations and Units	
Abbreviation	**Definition**
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**
CMP	100% subsidiary of Compañía Minera del Pacífico S.A
BHOD	Best Height of Draw
CCHEN	The Chilean Nuclear Energy Commission
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CMP	Compañía Minera del Pacífico S.A
CuEq	Copper Equivalent
DD	Diamond Drill
DD	Diamond Drilling
DL 600	Decree-Law 600
EIA	Environmental Impact Assessment
EIA	Environmental Impact Assessment
FCFM	Universidad de Chile laboratory
FF	Footprint Finder
FoS	Factors of Safety
Frontera	Sociedad Minera Frontera SpA
G&A	General and Administration
GAC	Gestión Ambiental Consultores
HCH	Hot Chili Limited
HCH/Company	Hot Chili Limited
Ingeroc	Ingeniería de Rocas Ltda
JV	joint venture
KP	Knight Piésold Consulting Engineers
LOM	Life of Mine ()
MAO	Material Allocation Optimiser
MMS	Multimine Software
Mo	Molybdenum
MRE	Mineral Resource Estimate

Table 27.2 : Glossary of Terms	
Abbreviation	**Definition**
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
OCG	iron-oxide-copper-gold
OP	Open Pit
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
QQ	
RC	Reverse Circulation
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
ST	Stacking Test
SX/EW	Solvent extraction and electro-winning
TC/RC	Treatment and refining costs
the Company	Hot Chili Limited
the Project	Costa Fuego Copper Project
TSF	Tailings Storage Facility

Appendix A - Qualified Person Disclosures

Certificate of Qualified Person

I, Elizabeth Haren, B.Sc, FAusIMM(CPGeo), MAIG, as co-author of the NI 43-101 Technical Report, "Costa Fuego Copper Project, NI 43-101 Technical Report Mineral Resource Estimate Update", Chile, effective date 26th February 2024 (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 28 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 1.6, 1.9, 1.10, 1.12, 1.23, 1.24, 1.25, 7, 8, 9, 10, 11, 12.1 – 12.6, 12.9, 14, 25.3. 25.12, 25.13 and 26.2 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 08 April 2024 at Scarborough, Australia.

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, "Costa Fuego Copper Project, NI 43-101 Technical Report Mineral Resource Estimate Update", Chile, effective date 26th February 2024 (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, 12.8, 15, 16, 23, 24, 25.4 and 26.3. I 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 28 March 2024, Perth, Australia.

Anton von Wielligh FAusIMM

Director/Principal Mining Engineer

ABGM PTY LTD

Certificate of Qualified Person

I, Edmundo J. Laporte, P.E., P.Eng., C.P.Eng., do hereby certify that:

1. I am a Principal Consultant and Director with High River Services, LLC, 4440 Walnut Creek Drive, Lexington, KY, 40509 and we are acting as consultants for Gestión Ambiental Consultores S.A. (GAC), from Santiago, Chile.
2. This certificate applies to the Technical Report titled "Costa Fuego Copper Project, NI 43-101 Technical Report Mineral Resource Estimate Update" with an Effective Date of 26th February 2024 (the "Report").
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 member in good standing of the Association of Professional Engineers and Geoscientists of Saskatchewan (Member #37445). I am also a Professional Engineer in Alberta, Nova Scotia and Ontario, as well as in multiple jurisdictions in the United States, and a Chartered Professional Engineer in Australia. I have worked as an Engineer for a total of 35 years since my graduation from university.
4. My relevant experience includes exploration, mine planning, mine design, rock mechanics, ground control 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 1.17, 12.10, 20, 25.7, and 26.7 of the technical report titled "Costa Fuego Copper Project, NI 43-101 Technical Report Mineral Resource Estimate Update" and dated 8 April, 2024 (the "Technical Report") relating to the Costa Fuego Project.
7. I have not visited the Costa Fuego Project property and relied on documentation provided to me by GAC, as well as the firsthand impressions of and reports by Mr. Cristobal Julia, Environmental Manager of Hot Chili Limited, who has visited the project multiple times and is coordinating all the environmental component of the ongoing studies at the project site. If have satisfied myself that it is reasonable to rely on Mr. Julia.
8. I have not had prior involvement with the property that is the subject of the Technical Report.
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 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, Form 43-101F1 and the Technical Report, and the Technical Report has been prepared in compliance with that instrument and form.

Dated this 27th Day of March, 2024

SME
Society for Mining, Metallurgy & Exploration
Edmundo J. Laporte
SME Registered Member No. 4150038
Signature__________
Date Signed March 27, 2024
Expiration date December 31, 2024

Certificate of Qualified Person

I, David Morgan, M.Sc, MAusIMM, CPEng, as co-author of the NI 43-101 Technical Report, "Costa Fuego Copper Project, NI 43-101 Technical Report Mineral Resource Estimate Update", Chile, effective date 26th February 2024 (the "Report") do hereby certify that:

1. I am a full-time employee of Knight Piésold Pty Limited with an office at 184 Adelaide Terrace, East Perth, Western Australia, 6004, tel. +61 (08) 9223 6300, email dmorgan@knightpiesold.com.
2. I graduated from the University of Southampton, UK, with a Master of Science (Irrigation Engineering) Degree, and I have continually practiced my profession since 1982 as a civil engineer. I have worked in the mining industry for over 40 years with experience in tailings management and design.
3. I am a member of The Australasian Institute of Mining and Metallurgy, membership number 202216. I am a Chartered Professional member of the Institution of Engineers Australia, membership number 974219.
4. I have personally inspected the Costa Fuego project site during October 2015.
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 Section 18.6 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 memoranda 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 a preliminary design for Tailings Management at the Costa Fuego Project. The report section is based on a review of project files and preliminary designs.
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 Section 18.6 contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated this day 28 March 2024 at Perth, Australia.

David Morgan, MIEAust CPEng APEC Engineer IntPE(Aus)

Managing Director

Knight Piésold Pty Limited

Certificate of Qualified Person

Wood Group USA, Inc.
17325 Park Row
Houston, TX 77084 USA

I, Piers Wendlandt, PE am employed as a Principal Mining Engineer with Wood Group USA, Inc.

This certificate applies to the technical report titled "Costa Fuego Copper Project, NI 43-101 Technical Report Mineral Resource Update", Chile, effective date 26th February 2024 (the "Technical Report").

1. I am a Registered Professional Engineer in the States of Colorado (PE.0047235), Arizona (77120), and Alaska (199933). 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.
2. I have practiced my profession for 17 years since graduation. I have been directly involved in the financial modelling and economic analysis of numerous mining projects.
3. As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 Standards of Disclosure for Mineral Projects (NI 43–101), for those sections and sub-sections of the technical report that I take responsibility.
4. I have not visited the Costa Fuego project site.
5. I am responsible for sections 1.18, 1.21, 1.22, 12.7, 19, 22, 24, 25.8, and 25.11 of the technical report.
6. I am independent of Hot Chili, Limited as independence is described by Section 1.5 of NI 43–101.
7. I have been involved with the Costa Fuego project since August 2020. I have authored private reports and technical memos for Hot Chili Limited on the Costa Fuego property.
8. I have read NI 43–101 and the sections and sub-sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.
9. As of the effective date of the technical report, to the best of my knowledge, information and belief, the sections and sub-sections of the 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: 26 March, 2024

"Signed and sealed"

Piers Wendlandt, PE

Certificate of Qualified Person

I, Jeffrey Peter Stevens, BSc (Chem Eng), as co-author of the NI 43-101 Technical Report, "Costa Fuego Copper Project, NI 43-101 Technical Report Mineral Resource Estimate Update", Chile, effective date 26th February 2024 (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, co-ordination 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 not 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 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 1.19, 21.1, 21.2, 21.5, 25.9. I accept professional responsibility for those sections of this technical report.
9. I have had prior involvement with the subject property as co-author of the NI 43-101 Technical Report, "Costa Fuego Copper Project, NI 43-101 Technical Report Preliminary Economic Analysis", Chile, effective date 28th June 2023 (the "Report").
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 05 April 2024, Perth, Australia

Jeffrey Stevens BSc (Chem Eng)

Senior Study Manager

Wood Minerals & Metals Australia

Certificate of Qualified Person

I, Dean David, B.App.Sc, Metallurgy, South Australian Institute of Technology, as co-author of the NI 43-101 Technical Report, "Costa Fuego Copper Project, NI 43-101 Technical Report Mineral Resource Estimate Update", Hot Chili Limited, effective date 26th February 2024 (the "Report") do hereby certify that:

1. I am a consultant engaged by Hot Chili Limited with an office at 240 St Georges Terrace, Perth WA, 6100, tel. +61 6314 2400, dean.david@woodplc.com.au.
2. I graduated from the South Australian Institute of Technology, Australia with a Bachelor of Applied Science - Metallurgy (B.App.Sc[Met]) in 1981. I have worked in the mining industry for 43 years with experience in mineral processing flowsheet development, operations, management and consulting.
3. I am a Fellow and Chartered Professional of The Australasian Institute of Mining and Metallurgy, membership number 102351.
4. I have not 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. As a Qualified Person, I am independent of the issuer as defined in Section 1.5 of National Instrument 43-101.
7. I am a co-author of this report and responsible for sections 1.1 to 1.5, 1.7, 1.8, 1.11, 1.15, 1.16, 1.20, 2, 3, 4, 5, 6, 12.7, 13, 14.7, 17, 18, 21.3, 21.4, 25.5, 25.6, 25.10, 25.12, 25.13, 26.4, 26.5 and 27. I accept professional responsibility for those sections, or relevant parts of the sections, of this technical report.
8. I have had prior involvement with the subject property. Since August 2020 I have overseen components of the metallurgical testwork, authored private reports and authored technical memos for Hot Chili Limited.
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 3 April 2024, Perth, Australia

Dean David, FAusIMM CP (met)
Senior Process Consultant
Wood