Cañariaco Copper Project

NI 43-101 Technical Report & Preliminary Economic Assessment

Lambayeque Region, Peru

Effective Date: May 31, 2024
Report Date: Jun 10, 2024

Prepared for:
Alta Copper Corp.
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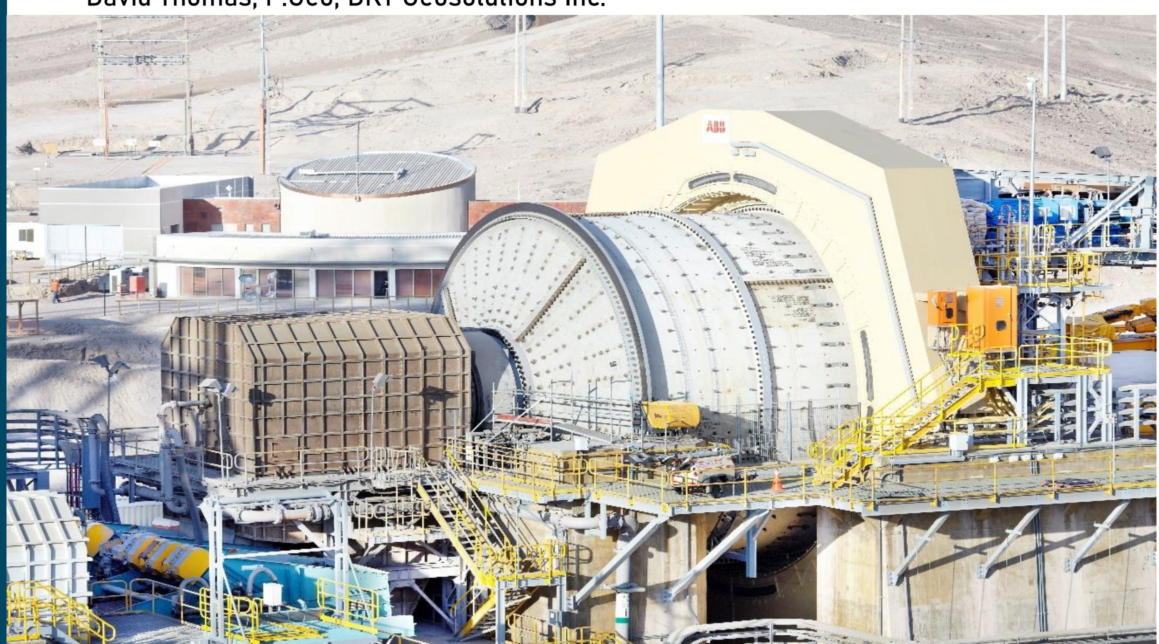

CERTIFICATE OF QUALIFIED PERSON

Kevin Murray, P.Eng.

I, Kevin Murray, P.Eng., certify that I am employed as a Principal Process Engineer with Ausenco Engineering Canada ULC ("Ausenco"), with an office address of 1050 West Pender Street, Suite 1200, Vancouver, BC, Canada, V6E 3S7.

1. This certificate applies to the technical report titled *"Cañariaco Copper Project NI 43-101 Technical Report & Preliminary Economic Assessment"* that has an effective date of May 31st, 2024 (the "Effective Date").
2. I graduated from the University of New Brunswick, Fredericton NB, in 1995 with a Bachelor of Science in Chemical Engineering.
3. I am a member of the Association of Professional Engineers and Geoscientists of the Province of British Columbia, Registration number# 32350 and the Northwest Territories Association of Professional Engineers and Geoscientists' Registration# L4940.
4. I have practiced my profession for 23 years. I have been directly involved in all levels of engineering studies from preliminary economic assessments (PEAs) to feasibility studies. I have led preliminary test work design, test work analysis and flowsheet development as well involvement in detailed design and commissioning. I have also developed operating cost estimates and contributed to and reviewed capital cost estimates. I have 14 years of experience in testwork, design and cost estimations for copper and nickel hydrometallurgical processes, as well as detail design and commissioning of a copper concentrate pressure oxidation, solvent extraction and copper electrowinning facility. I have 7 years of experience in testwork, design and cost estimations for gold processes, including whole ore leaching and flotation for gold leaching and concentrate sales as well as detail design, process plant optimization and commissioning. I have 7 years of experience in testwork, design and cost estimations for base metal flotation processes for concentrate sales.
5. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 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 those sections of the Technical Report that I am responsible for preparing.
6. I have not visited the Cañariaco Project.I am responsible for sections 1.1, 1.2, 1.9, 1.13, 1.14.1, 1.14.2, 1.14.4-1.14.6, 1.16-1.22, 2.1- 2.3, 2.5-2.7, 3.1, 3.4, 3.5, 13, 17, 18.1-18.5, 18.7-18.9, 19, 21.1, 21.2.1, 21.2.2, 21.2.4-21.2.10, 21.3.1, 21.3.2, 21.3.4-21.3.6, 22, 25.1, 25.5, 25.8, 25.9, 25.11-25.14, 25.15.1, 25.15.2.1-25.15.2.3, 25.15.4, 25.16.3, 25.16.4, 26.1, 26.3, 26.5, 27 of the Technical Report.
7. I am independent of Alta Copper Corp. as independence is described by Section 1.5 of the NI 43-101.
8. I have been previously involved with the Cañariaco Project as a QP on the NI 43-101 Technical Report on Preliminary Economic Assessment with the Effective Date: 8 February 2022.
9. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the 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: June 10, 2024

"*Signed and Sealed*"

Kevin Murray, P. Eng.

CERTIFICATE OF QUALIFIED PERSON
Scott Cameron Elfen, P.E.

I, Scott Cameron Elfen, P.E., certify that I am employed as the Global Lead Geotechnical and Civil Services within Ausenco Engineering Canada Inc. with ("Ausenco"), with an office address of 1050 West Pender Street, Suite 1200, Vancouver, BC V6E 3S7, Canada.

1. This certificate applies to the technical report titled "*Cañariaco Copper Project NI 43-101 Technical Report & Preliminary Economic Assessment*" that has an effective date of May 31st, 2024 (the "Effective Date").
2. I graduated from the University of California, Davis, California, in 1991 with Bachelor of Science degree in Civil Engineering (Geotechnical).
3. I am a Registered Civil Engineer in the State of California (license no. C56527) by exam since 1996 and I am also a member in good standing of the American Society of Civil Engineers (ASCE), and the Society for Mining, Metallurgy & Exploration (SME).
4. I have practiced my profession continuously for 26 years with experience in the development, design, construction and operations of mine waste storage facilities, such as waste rock storage facilities and tailings storage facilities ranging from slurry to dry stack facilities, focusing on precious and base metals, both domestic and international. In addition, I have developed geotechnical design parameters for pit slope design, plant foundation design, and other supporting infrastructure. Examples of projects I have worked on include: Skeena's Eskay Creek Project PEA, PFS and FS, O3 Mining's Marban Project PEA and PFS, First Mining Gold's Springpole PEA and PFS. SSR Mining's Puna Silver In-Pit Tailings Disposal PFS, and Detailing Engineering, and the Company's Cangrejos Project PEA.
5. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 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 those sections of the Technical Report that I am responsible for preparing.
6. I visited the Cañariaco Project from February 5, 2024, to February 6, 2024, for a visit duration of 2 days.
7. I am responsible for sections 1.14.3, 2.4.1, 18.6, 25.10, 25.11, 25.15.2.4, 26.6, 27 of the Technical Report.
8. I am independent of Alta Copper Corp. as independence is described by Section 1.5 of the NI 43-101.
9. I have had no previous involvement with the Cañariaco Project.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the 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: June 10, 2024

"Signed and sealed"

Scott Cameron Elfen, P.E.

Ausenco

CERTIFICATE OF QUALIFIED PERSON

James Millard, M. Sc, P. Geo.

I, James Millard, P. Geo., certify that I am employed as a Director, Strategic Projects with Ausenco Sustainability Inc., a wholly owned subsidiary of Ausenco Engineering Canada ULC (“Ausenco”), with an office address of Suite 100, 2 Ralston Avenue, Dartmouth, NS, B3B 1H7, Canada.

1. This certificate applies to the technical report titled “Cañariaco Copper Project NI 43-101 Technical Report & Preliminary Economic Assessment” that has an effective date of May 31st, 2024 (the “Effective Date”).
2. I graduated from Brock University in St. Catharines, Ontario in 1986 with a Bachelor of Science in Geological Sciences, and from Queen’s University in Kingston, Ontario in 1995 with a Master of Science in Environmental Engineering.
3. I am a member (P. Geo.) of the Association of Professional Geoscientists of Nova Scotia, Membership No. 021.
4. I have practiced my profession for 25 years. I have worked for mid- and large-size mining companies where I have acted in senior technical and management roles, in senior environmental consulting roles, and provided advise and/or expertise in a number of key subject areas. These key areas included: feasibility-level study reviews; NI 43-101 report writing and review; due diligence review of environmental, social, and governance areas for proposed mining operations and acquisitions, and directing environmental impact assessments and permitting applications to support construction, operations, and closure of mining projects. In addition to the above, I have been responsible for conducting baseline data assessments, surface and groundwater quantity and quality studies, mine rock geochemistry and water quality predictions, mine reclamation and closure plan development, and community stakeholder and Indigenous peoples’ engagement initiatives. Recently, I acted in the following project roles: Qualified Person for the environmental/sustainability aspects for “Puquios Project, Feasibility Study Report, La Higuera, Coquimbo Region, Chile”, “Volcan Project, NI 43-101 Technical Report on Preliminary Economic Assessment, Tierra Amarilla, Atacama Region, Chile” and, “Colomac Gold Project, NI 43-101 Technical Report and Preliminary Economic Assessment, Northwest Territories, Canada”; and principal author for the environmental/sustainability sections for the “Kwanika- Stardust Project, NI 43-101 Technical Report and, Preliminary Economic Assessment, British Columbia, Canada”.
5. I have read the definition of “Qualified Person” set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects (“NI 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 those sections of the Technical Report that I am responsible for preparing.
6. I have not visited the Cañariaco Project.
7. I am responsible for sections 1.15, 3.3, 20, 25.10, 25.15.3, 25.16.5, 26.7, 27 of the Technical Report.
8. I am independent of Alta Copper Corp. as independence is defined in Section 1.5 of NI 43-101.
9. I have had no previous involvement with the Cañariaco Project.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the 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: June 10, 2024

“Signed and sealed”

James Millard, M. Sc, P. Geo.

Ausenco

CERTIFICATE OF QUALIFIED PERSON
JONATHAN COOPER, P.ENG.

I, Jonathan Cooper, P. Eng., certify that I am employed as a Water Resources Engineer with Ausenco Sustainability ULC ("Ausenco"), with an office address of 11 King Street West, Suite 1500, Toronto, Ontario M5H 4C7.

1. This certificate applies to the technical report titled *"Cañariaco Copper Project NI 43-101 Technical Report & Preliminary Economic Assessment"* that has an effective date of May 31st, 2024 (the "Effective Date").
2. I graduated from the University of Western Ontario with a Bachelor of Engineering Science in Civil Engineering in 2008, and University of Edinburgh with a Master of Environmental Management in 2010.
3. I am a Professional Engineer registered and in good standing with Order of Engineers of Quebec (temporary engineer permit #6067376), Professional Engineers Ontario (registration #100191626), Engineers and Geoscientists British Columbia (registration #37864) and Northwest Territories and Nunavut Association of Professional Engineers and Geoscientists (registration # L4227).
4. I have practiced my profession for continuously for over 15 years with experience in the development, design, operation, and commissioning of surface water infrastructure. Previous projects that I have worked on that have similar features to the Novador Project are the Kwanika-Stardust for NorthWest Copper located in British Columbia, Colomac Gold Project located in the Northwest Territories and the Crawford Project located in Ontario.
5. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfil the requirements to be a "Qualified Person" for those sections of the Technical Report that I am responsible for preparing.
6. I have not visited the Cañariaco Project.
7. I am responsible for sections 1.14.7, 18.10, 27 of the Technical Report.
8. I am independent of Alta Copper Corp. as independence is defined in Section 1.5 of NI 43-101.
9. I have had no previous involvement with the Cañariaco Project.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the 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: June 10, 2024

"Signed and sealed"

Jonathan Cooper, P.Eng.

CERTIFICATE OF QUALIFIED PERSON
Gordon Zurowski, P.Eng.

I, Gordon Zurowski, P. Eng, certify that I am employed as a Principal Mine Engineer with AGP Mining Consultants Inc. with, an office address of #246-132 Unit K, Commerce Park Drive, Barrie, Ontario, L4N 0Z7, Canada. This certificate applies to the technical report titled Cañariaco Copper Project that has an effective date of May 31st, 2024, and a report date of June 10th, 2024 (the "Technical Report").

1. I graduated from the University of Saskatchewan with a B.Sc. in Geological Engineer in 1988.

2. I am a member in good standing of the Professional Engineers of Ontario (#100077750).

3. I have practiced my profession for over 35 years since graduation. I have been directly involved in mineral resource and reserve estimations, preliminary economic assessments, feasibility studies and involved in mine operations around the world. As a result of my experience and qualifications, I am a Qualified Person as defined in NI 43-101.

4. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 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 those sections of the Technical Report that I am responsible for preparing.

5. I visited the Cañariaco Copper Project site between February 5th and 6th, 2024 for a visit duration of 2 days.

6. I am responsible for sections 1.12, 2.4.2, 15, 16, 21.2.3, 21.3.3, 25.7, 25.16.2, 26.4, 27 of the Technical Report.

7. I am independent of Alta Copper Corp. as independence is defined in Section 1.5 of NI 43-101.

8. I have had no previous involvement with Cañariaco Copper Project.

9. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the 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: June 10, 2024

"Signed and sealed"

Gordon Zurowski, P.Eng.

CERTIFICATE OF QUALIFIED PERSON
David Gwilym Thomas, P. Geo.

I, *David Gwilym Thomas, P. Geo.*, certify that I am employed as a *Principal Geologist* with DKT Geosolutions Inc. ("DKT"), with an office address of *Suite 170 – 420 Richards Street, Vancouver, BC, Canada*. This certificate applies to the technical report titled *"Cañariaco Copper Project NI 43-101 Technical Report & Preliminary Economic Assessment"* that has an effective date of May 31st, 2024 (the "Effective Date").

I graduated from *the University of Durham, UK in 1993* with a *BSc. In Geology and from the University of London in 1995 with an MSc. In Mineral Exploration.* I am a *Professional Geoscientist* of Engineers and Geoscientists BC with membership number 149114. I have practiced my profession for *29 years*. I have been directly involved in *exploration, QAQC review, geological modelling and mineral resource estimation of porphyry copper-gold-moly deposits in Canada (GJ and Schaft Creek), USA (Rosemont), Greece (Skouries), Romania (Rovina), Bulgaria, Serbia and Argentina (Agua Rica and Lindero).*

I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 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 those sections of the Technical Report that I am responsible for preparing.

I visited the *Canariaco Project* between *February 14th and February 16th, 2022* for a visit duration of 3 days. I am responsible for Sections 1.3-1.8, 1.10,1.11, 2.4.3, 3.2, 4, 5, 6, 7, 8, 9, 10, 11, 12, 14, 23, 24, 25.2-25.4, 25.6, 25.16.1, 26.2, 27 of the Technical Report.

I am independent of *Candente Copper Corporation* as independence is defined in Section 1.5 of NI 43-101. I have been involved with the *Canariaco Project; specifically as co-author of the technical report for* Candente Copper Corporation, Cañariaco Norte Deposit, Lambayeque Department, Peru, NI 43-101 Technical Report on Pre-feasibility Study Progress Report: prepared by AMEC Americas Ltd for Candente Resource Corp., effective date 18 January 2011; co-author of the technical report for Candente Copper Corporation, Cañariaco Project, Lambayeque Department, Peru NI 43-101 Technical Report prepared by AMEC Americas Ltd for Candente Resource Corp., effective date 8 November 2010 and co-author of the technical report for Candente Copper Corporation, Canariaco Norte Project, Lambayeque Region, Peru NI 43-101 Technical Report on Preliminary Economic Assessment: prepared by Ausenco for Candente Resource Corp, effective date 8 February 2022.

I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the 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: June 10, 2024

"Signed and sealed"

David Gwilym Thomas, P. Geo.

Important Notice

This report was prepared as National Instrument 43-101 Technical Report for Alta Copper Corp. (Alta Copper*)* by Ausenco Engineering Canada ULC and Ausenco Sustainability ULC (collectively, Ausenco), AGP Mining Consultants Inc. (AGP), and DKT Geosolutions Inc. (DKT), collectively the Report Authors. The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in the Report Authors' services, 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 for use by Alta Copper subject to terms and conditions of its contracts with each of the Report Authors. Except for the purposed legislated under Canadian provincial and territorial securities law, any other uses of this report by any third party are at that party's sole risk. The cover page is a picture of the SAG mill at Centinela from Antofagasta Minerals and has been reproduced with permission from ABB.

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1 SUMMARY

1.1 Introduction

In 2023 Alta Copper Corp. (Alta Copper) commissioned Whittle Consulting (Whittle) to conduct a mining plan optimization based on the results of the PEA completed by Ausenco ULC (Ausenco) and AGP in 2022 (the "2022 PEA"). Furthermore, this PEA (the "2024 PEA") is to include mining of both the Sur deposit and the Norte deposit and use a significantly larger production rate from the start of production.

The objective of this approach was to realize the value of the combined resources and utilize the benefits of lower unit production costs to enhance overall project financial outcomes.

In coordination with the Whittle optimization work, Alta Copper engaged Ausenco Engineering Canada ULC (Ausenco), AGP Mining Consultants Inc (AGP) and DKT Geosolutions Inc (DKT Geoscience) to update the PEA to also include the Optimised Mine Plan as well as ESG enhancements identified in the 2022 PEA.

The 2024 PEA was prepared in accordance with the Canadian disclosure requirements of National Instrument 43-101 (NI 43-101) and the requirements of Form 43-101 F1.

The responsibilities of the engineering companies contracted by Alta Copper to prepare this report are as follows:

- Whittle provided comprehensive mine plan optimization for the Norte and Sur deposits. This included detailed assessment of metal grades metal prices, metal recoveries, mining and processing costs throughout the two deposits and by applying advanced computational analyses developed an optimized mine plan to maximize net economic value of the mining operation.
- Ausenco managed and coordinated the work related to the report. Ausenco developed the PEA-level design and cost estimate for the process plant, review of the metallurgical test program, general site infrastructure, site water management infrastructure, comingled waste rock and tailings facility and environmental studies and permitting. Ausenco also compiled the overall cost estimate and completed the economic analysis.
- AGP Mining Consultants Inc. (AGP) designed the open pit, mine production schedule, and mine capital and operating cost estimates.
- DKT Geosolutions Inc. updated the Mineral Resource Estimates and completed the work related to property description, accessibility, local resources, geological setting, deposit type, exploration work, drilling, sample preparation and analysis, and data verification.

1.2 Terms of Reference

The Report was prepared to support disclosures in Alta Copper's news release dated May 15th, 2024, entitled "Alta Copper Announces Robust Economics for Cañariaco with US$2.3 Billion After-Tax NPV and 24% IRR". The report is prepared for Alta Copper. Alta Copper changed their name from Candente Copper Corp. in May 2023, and any reference to Candente in this report refers to Alta Copper.

The Report was prepared to support the optimized mine plan by Whittle and the inclusion of the Sur deposit to complement the Norte deposit from the PEA study in 2022. It adopted project configurations that includes comingled waste rock and tailings facility and the usage of a single SAG and dual ball mill comminution circuit.

Units used in the report are metric units unless otherwise noted. Monetary units are in United States dollars (US$) unless otherwise stated. The Report uses Canadian English.

Mineral resources are estimated in accordance with the 2019 edition of the Canadian Institute of Mining, Metallurgy and Exploration (CIM) Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines (2019 CIM Best Practice Guidelines) and are reported using the 2014 CIM Definition Standards for Mineral Resources and Mineral Reserves (2014 CIM Definition Standards).

Readers are cautioned that the PEA is preliminary in nature. It includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the PEA will be realized.

1.3 Property Description and Location

The Cañariaco Project is situated within the Province of Ferreñafe, in the Region/Department of Lambayeque, in northwestern Peru, and is approximately 700 km northwest of Lima, the capital of Peru, and 102 km to the northeast of the city of Chiclayo (Figure 1-1).

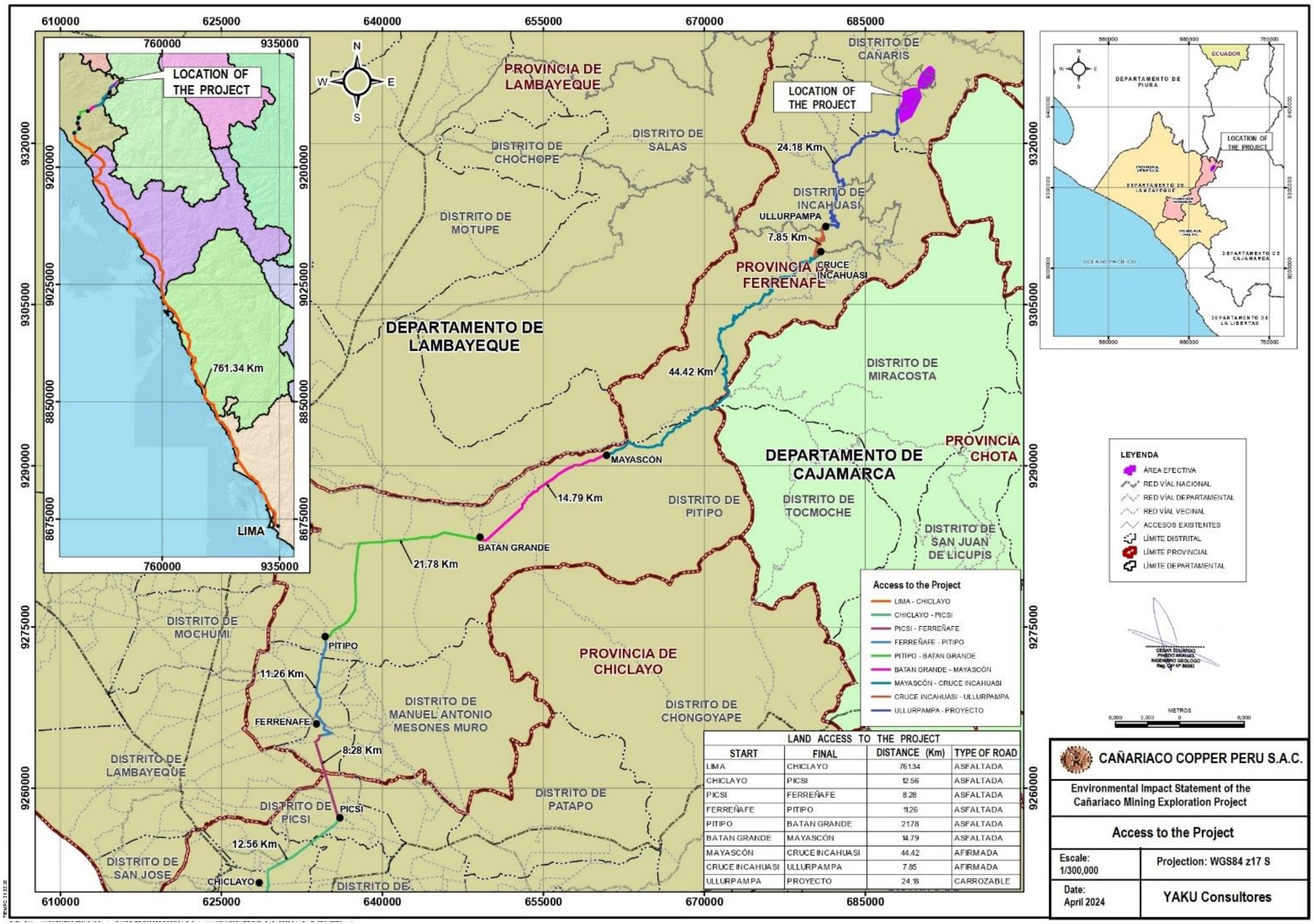
Either the Pan-American Highway (700 km, 11-hour trip) or one of the several daily commercial airline flights can be taken from Lima to Chiclayo. The route from Chiclayo to the Cañariaco Project is currently a 144 km five-hour trip along initially paved and then mostly secondary gravel roads via Uyurpampa and Incahuasi.

Temperatures in the Project area vary with altitude and range between 3°C and 20°C. The region receives between 830 mm and 1,700 mm of rainfall each year. The rainy season extends from November to March. It is expected that any future mining operations will be able to be conducted year-round.

The copper deposits are situated on the eastern side of the continental divide and infrastructure will be on the top as well as both western and eastern sides of the divide. The topography varies from steep incised valleys at lower elevations to open grassy highlands at upper elevations.

There is sufficient suitable land available within the concessions and close to the mining areas for the process plant, ancillary infrastructure and comingled waste rock and dry stack tailings facility.

Figure 1-1: Location and Access Map



Source: Yaku Consultants, 2024.

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

The Project consists of 15 mining concessions totaling approximately 9,725.12 ha. Mining legislation in Peru does not require location of concession boundaries on the ground. To maintain the concessions in good standing, annual validity payments of $3.00 per hectare must be paid and when the minimum production or investment (exploration) has not been met by the 10th year calculated from the year following the concession grant, a penalty must be paid from the 11th year. In Alta Copper's case, the 10th year was 2008, and penalties have been paid since that date. Ausenco was advised by Alta Copper that all required property payments have been made, and the concessions are in good standing. Providing the annual property payments are made in a timely manner, the concessions will not expire.

It is a requirement of the Peruvian Government that any property developer either purchases the surface rights, or makes an appropriate agreement with the surface rights owner, for access to a property. Mining concession holders are protected under the Peruvian Constitution and Civil Code. Mineral concession rights do not, however, confer ownership of the land. Thus, the owner of a mining concession must deal with the registered landowner to obtain access rights. All transactions and contracts pertaining to a mining concession must be registered with the Public Mining Registry.

Alta Copper will need to obtain an authorization from the National Water Authority (ANA) to use water for exploration and/or mining purposes, including for domestic and industrial use.

There is a 0.5% royalty payable to Anglo Pacific, a third party, on the Cañariaco A, B, C, D and F1 concessions. Profit based taxes are due to the Government of Peru. It is expected that the Project will incur a net profits interest (NPI) royalty.

1.5 Geology and Mineralization

The Project covers a north-easterly 4–5 km long structural trend that hosts three porphyry centres, Cañariaco Norte, Cañariaco Sur and Quebrada Verde, as delineated by geophysics, geochemistry, geological mapping and drilling. In each of these centres various intrusive bodies have been identified. The host rocks to the mineralized intrusions are Calipuy Group andesite volcanic rocks. Drilling has ascertained that both Cañariaco Norte and Cañariaco Sur are porphyry copper-gold deposits, Quebrada Verde is a porphyry target.

The Cañariaco Norte deposit is hosted within a multiphase intrusive–breccia complex approximately 1.7 km in strike extent and 1.1 km wide. The deposit has been drill-tested to a depth of approximately 770 m in one hole and several holes were drilled to 400 or 500 m. The deepest hole and many others were stopped in mineralization so the deposit clearly remains open at depth. The majority of the copper–gold mineralization is hosted within the intrusive and breccia units, but locally extends for variable distances into the surrounding volcanic units. The intrusive units are nested and collectively roughly oval in shape, with older intrusive rocks being cut by successively younger intrusive bodies. In general, the intrusive units are north–south-trending, steeply-plunging bodies. The breccia units cut the intrusive units, are oval to circular in shape, and are also steeply plunging. The dykes generally strike northwest–southeast with a steep southwesterly dip. The shape and positioning of the intrusive, breccias, and dyke units was largely controlled by northwest–southeast and northeast–southwest trending faults. The main copper minerals are chalcopyrite, covellite, chalcocite, and enargite, with minor bornite, tennantite, and digenite, and trace tetrahedrite. Copper sulfide minerals are mainly associated with pyrite and other sulfide gangue minerals. In general, the highest copper grades (>0.5% Cu) occur in the biotite–feldspar porphyry and polymictic breccia units. Intermediate copper grades (0.35 to 0.5% Cu) are most common in the hydrothermal breccia and the coarse quartz porphyry, and the lowest grades (0.2 to 0.35% Cu) are found in the crowded quartz feldspar porphyry, tourmaline breccias, and volcanic host rocks.

The centre of the Cañariaco Sur deposit is located approximately 2 km southwest of the centre of the Cañariaco Norte deposit. Cañariaco Sur has extensive potassic, phyllic, silicic and propylitic alteration with a coincident copper–gold–molybdenum soil geochemical anomaly outcropping over 1,400 m x 900 m. Both the host rocks and the mineralized intrusions are similar as those found in Norte however the Cañariaco Sur intrusive complex is much less complicated than that hosting Cañariaco Norte. Three intrusive events include a CQFP intruded by the BFP and then followed by CQP and minor amounts of the IBXP breccia. Cañariaco Sur also lacks the later stage brecciation event that introduced arsenic into Cañariaco Norte and does not have the other breccias that complicated the copper grade controls at Cañariaco Norte. Cañariaco Sur also has a higher gold content than Cañariaco Norte.

Exposed on surface in the Quebrada Verde prospect is a Biotite Feldspar Poprhyry which covers an area of a 1 km x 750 m which, as in the Cañariaco Norte and Sur deposits, also intruded Calipuy Group andesite volcanic rocks.

The geological understanding of the settings, lithologies, and structural and alteration controls on mineralization in the different zones is sufficient to support estimation of Mineral Resources at Cañariaco Norte and Cañariaco Sur. The geological knowledge of the area is considered sufficiently acceptable to reliably inform conceptual mine planning at Cañariaco Norte. The Cañariaco Sur deposit was not used in the 2022 PEA, however an inferred resource has been developed for Cañariaco Sur and it has been included in this PEA.

1.6 History

Prior to Alta Copper's Project interest, the following entities or companies held an interest in the Project area: the Peruvian Servicio Nacional de Geología y Minería (INGEMMET), Placer Dome Exploration Inc. (Placer Dome), and Billiton Exploration and Mining Perú B.V. (Billiton). Work conducted included stream sediment sampling, geological mapping, rock chip and grab sampling, trenching and pitting, induced polarization (IP), resistivity, and ground magnetic geophysical surveys, petrographic studies, core drilling, mineral resource estimation, and very preliminary leach testwork.

Alta Copper acquired the Project in 2002. Work conducted since acquisition includes additional geological mapping, prospecting, ground magnetic, resistivity and magnetic geophysical surveys, rock chip sampling, petrographic studies, bulk sampling for metallurgical testing, re-logging and re-sampling of historic drill core, core drilling, and mineral resource estimation. An initial Preliminary Economic Assessment considering an SX-EW operation was completed in 2006 on the Cañariaco Norte deposit. Between 2007 and 2011, various updated PEA's were completed which considered various sizes of operations using open pit and flotation recovery process technology. These studies are not considered to be current; however, some of the data and information obtained during the 2008–2014 studies, such as engineering and environmental baseline data, were used to support an updated PEA considering a staged build and operation completed by Ausenco in 2022. The current 2024 PEA includes many of the ESG improvements identified in 2022 and reverts back to a single stage large scale operation.

1.7 Drilling and Sampling

Drilling on the Project consists of 289 core holes (85,183.16 m), including geotechnical, metallurgical, and hydrogeological drilling. Of this total, 272 core holes (81,708.85 m) were completed by Alta Copper.

Core from INGEMMET, Placer Dome and Billiton programs, referred to as legacy, was re-logged by Alta Copper. Core from the Alta Copper campaigns was logged for lithology, structure, veining, alteration, and mineralization.

Alta Copper drill collar locations were picked up by a surveyor using a total station instrument. All legacy drill collars were picked up by survey in 2006 and tied into the Project grid. Down-hole surveys were performed for the Alta Copper drill holes using Pajari, Sperry Sun, or Reflex EZ-Shot instruments.

The drill core generated by INGEMMET and Placer Dome was halved; there is no information as to the typical sample intervals. The Billiton drill core was halved and sampled on 2 m intervals. Alta Copper drill core was halved using a circular rock saw. Samples were 2 m in length unless a geological contact was present within the sample interval. In those instances, the sample interval was terminated at the contact. The subsequent sample interval terminated at the next meter depth mark that allowed a 1.50 m minimum sample length.

A total of 9,424 bulk density readings were taken by Alta Copper personnel during core logging using weight in air and weight in water methods.

1.8 Data Verification

All data in the field were recorded in written form in field books, logbooks, sample sheets, logging forms or shipping forms. All field data were hand-entered into Excel tables. Data from third parties such as laboratories or survey contractors were generally supplied in digital and printed form. All data was verified by Alta Copper personnel.

Drill data collected from the INGEMMET, Placer Dome, and Billiton campaigns were re-logged by Alta Copper personnel, and nine of the drill holes have been re assayed. Based on the correlations between the historical grades and the Alta Copper re-assay grades, all the historical data were accepted into the final database. Three pairs of twinned holes were drilled by Alta Copper to verify grade uniformity at short distances. In general, similar average grades were noted over the same depth intervals.

In the QP's opinion, the data collected from the Project adequately support the geological interpretations and constitute a database of sufficient quality to support the use of the data in Mineral Resource estimation.

1.9 Metallurgical Testwork

Initial metallurgical test work on the Cañariaco project was focused on the Norte deposit, however as the Project developed, the testing has expanded to include the Sur deposit in the 2023 test program. During the initial testing of the Norte deposit, the project focused on leachable copper. As the deposit definition increased the mineralization was confirmed as primarily sulfide and that conventional milling and flotation recovery would be a more appropriate process. The Sur mineralization has been confirmed to be predominantly sulfide, and, therefore, Sur testwork has also utilized conventional comminution and flotation recovery methods.

Three major phases of testwork were conducted for the Norte deposit, and one phase for the Sur deposit. The first phase of the Norte deposit consisted of process development to define the type of processing most applicable to mineralization. This was followed by more detailed work to optimize the process conditions. After a pause due to the financial crisis in 2008, definition work resumed in 2010 to allow for further development of the process parameters to allow primary equipment selection.

The third phase for the Norte Deposit was designed with extensive variability testwork in 2011 and 2012 to improve geometallurgical understanding of the deposit. The program is built upon the mineralogical characterization data, comminution tests, and roughing and cleaning flotation tests. The third phase of testing is the main driver for the design of the Project.

The testwork recommended key design parameters as follows:

- For a LOM feed grade of 0.35% copper, a LOM copper recovery of 88.2% is forecasted at the target grind;

- Recovery will vary with feed grade (based on a grade-recovery formula);
- Concentrate grade of 26% copper;
- Primary grind of 200 µm with a JKSimMet breakage parameter Axb value of 52.5; and
- Bond ball mill work index (BWi) of 12.2 kWh/t.

The most recent phase of testing assessed comminution and metal recoveries for the Sur deposit. The mineralized material in the Sur deposit exhibits similar characteristics to the Norte deposit and the Sur testwork confirms the mineralization is amenable to the same process flowsheet as for Norte. The Sur program included comprehensive chemical analysis, bond work index determination, and rougher flotation tests. Mineralogical characterization confirmed the Sur deposit shares the same rock types and composites as Norte. The grinding testwork on the Sur samples exhibit a bond work index of 11.12 kWh/t and resulted in higher copper recovery in testing with the same rougher conditions as applied for Norte testwork in 2011-2012.

Recovery models were developed using the test programs in phases 3 and 4, utilizing BFP composite samples at the target grind of 200 µm P_{80}. BFP represents the most prevalent mineralization type. The equation of each precious metal recovery relates to the feed grade and was used to estimate the recoveries in the annual mine plan.

Copper recoveries are expected to vary with feed grade, with LOM average of 88.2%. The metallurgical response of gold and silver has not been assessed to the same degree as copper; however, based on the testwork completed to date, LOM gold and silver recoveries are projected to be 63.3% and 55.3%, respectively.

1.10 Mineral Resource Estimation

1.10.1 Cañariaco Norte

Geological and alteration type interpretations were performed using north–south and east–west vertical sections that were spaced 100 m apart. Estimation domains were defined following evaluation of statistical distributions of lithological and alteration units. Seven domains were created for copper and four domains for gold and silver. The supergene layer in the Cañariaco Norte deposit is thin and laterally discontinuous, and the defined supergene-enriched domain was only used for copper estimation. Density values were assigned to blocks based upon the lithological codes.

Outlier grade values typically occur in the upper 1% of the distribution. Copper and gold values were capped at defined thresholds based on probability plots. For silver, outlier values were controlled during grade estimation by using a restricted search ellipse with a radius of 25 m x 25 m x 15 m.

Drill hole data were composited into 6 m lengths using no geological or domain boundaries.

Sage2001 software was used to construct down-hole and directional correlograms for the estimation domains for copper, gold, and silver.

Copper, gold and silver grades were estimated by estimation domains using ordinary kriging (OK) interpolation for the majority of domains. Inverse distance weighting to the second power (ID2) was used to interpolate gold and silver in two domains where the variography was not considered sufficiently robust. Grade estimation was completed in three passes, and the search orientations for all domains used the applicable variogram orientations. A minimum of 3–11 and a maximum of 9–15 drill hole composites were required for estimation; this varied by element and estimation pass.

Validation of the estimate included visual inspection, a comparison between OK and nearest-neighbour (NN) estimates, swath plots, and examination of change of support correction using Hermetian polynomials. Neither material biases nor material estimation issues were noted.

Resource blocks were classified as follows:

- Measured Mineral Resources: composites from a minimum of three drill holes within 75 m radius from a block centroid, or samples from two drill holes with the closest sample within 25 m of the block centroid.
- Indicated Mineral Resources: composites from a minimum of two drill holes within 110 m distance of the block centroid; and
- Inferred: a composite within 135 m of the block centroid.

1.10.2 Cañariaco Sur

A grade shell using a 0.1% Cu threshold was created by implicit modelling. Outlier grade values typically occur in the upper 1% of the distribution. Copper, gold, silver and molybdenum values were capped at defined thresholds based on probability plots. There were an insufficient number of composites to allow modelling of robust variograms. A density of 2.5 t/m^3 was assigned to all material below topography.

Assay intervals were composited into 15 m lengths using the grade shell boundary to split the composites.

Copper, gold, silver and molybdenum grades were estimated using ID2. Grade estimation was completed in two passes, and the grade shell was considered as a hard boundary. A minimum of 3–8 and a maximum of 2–8 drill hole composites were required for estimation; this varied by element and estimation pass.

Validation of the estimate included visual inspection, a comparison between ID2 and NN estimates, and swath plots.

All blocks falling within the grade shell were classified as Inferred Mineral Resources.

1.10.3 Reasonable Prospects of Eventual Economic Extraction

Reasonable prospects of eventual economic extraction were assessed by constraining the estimate within a conceptual pit shell. Mineral Resources for Cañariaco Norte are reported at a cut-off grade of 0.10% Cu; which is above the breakeven cut-off grade of 0.09% Cu. Mineral Resources for Cañariaco Sur are also reported at a cut-off grade of 0.10% Cu.

1.11 Mineral Resource Statement

Mineral Resources for the Project were classified using the 2014 CIM Definition Standards. The independent Qualified Person for the Mineral Resource estimates is David Thomas, P.Geo.

Mineral Resources for Cañariaco Norte are provided in Table 1-1 and for Cañariaco Sur Table 1-2. The estimates are updates (effective April 25, 2024) of the 27 January 2022 resources, based on the 2024 Optimized Mine Plan by Whittle. The updates give less than a 10% increase to the resources globally and therefore are not material.

Mineral Resources from both the Cañariaco Norte and Sur deposits support the 2024 PEA. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Table 1-1: Mineral Resource Statement for Cañariaco Norte at a 0.10% Cu Cut-off Grade

		Grade				Contained Metal		
Category	Tonnage (Mt)	Cu Eq. (%)	Cu (%)	Au g/t)	Ag (g/t)	Copper (Blb)	Gold (Moz)	Silver (Moz)
Measured	433.3	0.47	0.43	0.07	1.9	4.07	1.00	26.15
Indicated	693.3	0.38	0.34	0.05	1.5	5.26	1.17	34.22
Measured + Indicated	1,126.6	0.42	0.38	0.06	1.7	9.33	2.16	60.37
Inferred	416.3	0.29	0.26	0.04	1.3	2.41	0.52	16.90

Notes: **1.** The Mineral Resources estimate has an effective date of April 25, 2024. The Qualified Person for the estimate is David Thomas, P.Geo., of DKT Geosolutions Inc. **2.** The Mineral Resources were reported using the definitions set out in the 2014 CIM Definition Standards. **3.** Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. **4.** The Mineral Resources are reported within a constraining Lerchs Grossmann pit shell developed using Whittle software using: **a.** A copper price of US$4.25/lb; **b.** Mining cost of US$1.93/t; **c.** A combined processing, tailings management and G&A cost of US$7.25/t; **d.** Variable pit slope angles ranging from 40 to 48 degrees; **e.** An average copper process recovery of 88%; **f.** Copper concentrate smelter terms: US$75/dry metric tonne treatment charge, US$0.075/lb refining charge and 96.5% payable; **g.** Estimated concentrate shipping costs of US$138.95/wet metric tonne of concentrate; h. An average arsenic penalty of $0.60/t processed. **5.** Copper equivalent grades including contributions from gold and silver, were estimated using metal prices of copper: US$4.25/lb, gold: US$1, 950 /oz, and silver: US$23.00/oz), metallurgical recoveries of copper: 88%, gold: 63%; silver: 58%, and smelter payables of copper: 96.5%, gold: 93%, silver: 90%). Copper grade equivalent calculation: CuEq% = (Cu % + ((Au grade x Au price x Au recovery x Au smelter payable%) + (Ag grade x Ag price x Ag recovery x Ag smelter payable)/(22.0462 x Cu price x 31.1035 g/t x Cu recovery x Cu smelter payable%). **6.** All figures are rounded to reflect the relative accuracy of the estimate. Totals may not sum due to rounding as required by reporting guidelines. **7.** The contained metal figures shown are in situ.

Table 1-2: Mineral Resource Statement for Cañariaco Sur at a 0.10% Cu Cut-off Grade (base case in grey)

Cut-off Grade Cu (%)	Tonnes (Mt)	Grade					Contained Metal			
		Cu Eq. (%)	Cu (%)	Au (g/t)	Ag (g/t)	Mo (ppm)	Copper (Blb)	Gold (Moz)	Silver (Moz)	Molybdenum (Mlb)
Inferred										
0.1	474.1	0.29	0.24	0.09	1.2	23	2.52	1.34	17.61	24.04
0.15	418.0	0.31	0.26	0.09	1.20	24	2.37	1.25	16.08	22.12
0.20	311.0	0.34	0.28	0.10	1.26	23	1.95	1.02	12.57	15.77

Notes: **1.** The Mineral Resource estimate has an effective date of April 25, 2024. The Qualified Person for the estimate is David Thomas, P.Geo., of DKT Geosolutions Inc. **2.** The Mineral Resources were reported using the definitions set out in the 2014 CIM Definition Standards. **3.** A single 0.1% Cu grade shell domain was constructed using implicit modelling. **4.** Raw drill hole assays were composited to 15 m lengths broken at domain boundaries. **5.** Capping of high grades was considered necessary and was completed on assays prior to compositing. Copper assays were capped to a 0.8% threshold and gold assays were capped at a threshold of 1 g/t. **6.** Block grades for gold were estimated from the composites using ordinary kriging interpolation into 20 x 20 x 15 m blocks coded by the 0.1% Cu grade shell. **7.** The Mineral Resource is reported above a 0.10% Cu cut-off grade. Additional cut-off grades are shown for sensitivity purposes only. **8.** A dry bulk density of 2.5 g/cm^3 was used for all material. **9.** The Mineral Resources are reported within a constraining Lerchs Grossmann pit shell developed using Whittle software using: **a.** A copper price of US$4.25/lb; **b.** Mining cost of US$1.93/t; **c.** A combined processing, tailings management and G&A cost of US$7.25/t; **d.** 45-degree pit slope angles; **e.** An average copper process recovery of 87%, an average gold process recovery of 59%, an average silver recovery of 46%; **f.** Copper concentrate smelter terms: US$75/dry metric tonne treatment charge, US$0.075/lb refining charge and 96.5% payable; **g.** Estimated concentrate shipping costs of US$138.95/wet metric tonne of concentrate; arsenic grades are below the threshold that would incur a smelter penalty. **10.** Copper equivalent grades including contributions from gold, silver, and molybdenum, were estimated using metal prices of copper: US$4.25/lb, gold: US$1,950/oz, silver: US$23.00/oz and molybdenum: US$11.00/lb; metallurgical recoveries of copper: 88%, gold: 63%; silver: 58% and molybdenum: 60% and smelter payables of copper: 96.5%: gold: 93%; silver :90% and molybdenum: 100%. Copper grade equivalent calculation: Cu Eq% = (Cu % + ((Au grade x Au price x Au recovery x Au smelter payable%) + (Ag grade x Ag price x Ag recovery x Ag smelter payable%) + (Mo grade x Mo price x Mo recovery x Mo smelter payable%))/(22.0462 x Cu price x 31.1035 g/t x Cu recovery x Cu smelter payable%). **11.** Preliminary metallurgical test work was conducted to assess grindability and flotation recoveries on representative samples of predominant lithologies and alterations in Cañariaco Sur. Given the similarities with Cañariaco Norte, the same average recoveries were applied for conceptual pit shell generation and CuEq estimations in Cañariaco Sur. **12.** All figures are rounded to reflect the relative accuracy of the estimate. Totals may not sum due to rounding as required by reporting guidelines. **13.** The contained metal figures shown are in situ.

Factors that may affect the Mineral Resource estimate include: metal price and exchange rate assumptions; changes to the assumptions used to generate the copper grade cut-off grade; changes in local interpretations of mineralization geometry and continuity of mineralized zones; changes to geological and mineralization shape and geological and grade continuity assumptions; density and domain assignments; changes to geotechnical, mining and metallurgical recovery assumptions; changes to the input and design parameter assumptions that pertain to the conceptual pit constraining the estimates; and assumptions as to the continued ability to access the site, retain mineral rights, obtain surface rights to allow mine construction and operations, obtain environment and other regulatory permits, and obtain the social license to operate. In particular, any changes to the slope angle of the pit wall as a result of more detailed geotechnical information would affect the pit shell used to constrain the Mineral Resources.

1.12 Mining Methods

This 2024 PEA is preliminary in nature and is partly based on Measured, Indicated and Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the 2024 PEA based on these Mineral Resources will be realized.

Mining will be conducted using conventional open pit methods and equipment. The mine plan will deliver 120,000 t/d or 43.8 million tonnes per year of mineralized material to the process plant over a 28-year mine life with life of mine strip ratio of 1.33 waste to mineralized material. Mill feed and waste will be drilled, blasted, and loaded by electric hydraulic face shovels and front-end loaders from 15 m high benches. Haul trucks will haul above cut-off grade mineralized material to the crusher, or short-term stockpile as required. Waste will be hauled during pre-production to the planned Co-Placement Storage Facility (CPSF) directly, and during production to the waste crusher to be located near the mineralized material crusher north of the Norte pit. During production, crushed waste will be conveyed to the plant site where it will be combined with dewatered tails and stacked at the CPSF. The mine planning was performed using metal prices of US$3.85/lb Cu, US$1,850/oz Au and US$23.00/oz Ag. Measured, Indicated, and Inferred Mineral Resources were considered for processing.

Knight Piésold completed a preliminary level slope stability evaluation for the proposed Cañariaco Norte open pit (February 2012), which is considered suitable for PEA-level mine planning. Preliminary pit slope angles were determined in accordance with a review of selected geotechnical information and corresponding data analysis. Inter-ramp slope angles will range from 40–48º. The recommended pit slope angles are considered reasonable for the early-stage pit phases within the central area of the deposit. Slope configurations for the ultimate pit are largely extrapolated from limited data and experience with other large open pit operations. A detailed review of the existing drill core is required to assess geotechnical drilling requirements. Additional geotechnical analysis will be required to support feasibility level pit slope designs for the Cañariaco Norte pit.

The Sur pit has not undergone preliminary slope stability analysis. The assumption for the PEA was that the recommendations from the southwest quadrant proposed by Knight Piésold would be appropriate and were applied to the Sur pit design. Geotechnical analysis of Sur will be required to support feasibility level pit slope designs for the Sur pit.

Hydrological studies indicated that maximum pit inflows would be approximately 1,700 m^3/day after 16 years of mining. The significant pit high wall (>900 m) means that active dewatering will be required to stabilize the slope. An allowance for horizontal drains was included in the mine operating cost estimate. The possible need for vertical dewatering wells has not been considered at this time.

Dilution and mining loss adjustments were made in the block model. The mineralization is generally gradational across the mineralization/waste contacts. A diluted block density was calculated on a volume weighted basis. The diluted grades were calculated on a tonnage weighted basis. The resulting average percentage reduction in grades from the undiluted mineral resource grades were 0.5%, 0.22% and 0.19% for copper, gold and silver respectively. The diluted bulk density and grades were then used for all net smelter return (NSR) calculations and production reporting. No dilution was applied to Sur for the PEA study. Due to the gradational contact and analysis for Norte the assumption was that dilution would also be negligible at Sur. An additional 2% of mining loss was applied to account for carry back and mineralization routing errors for both Norte and Sur.

The open pits ultimate size and phasing shape guidance for Norte and Sur were determined by generating nested revenue factor (RF) Lerchs–Grossmann (LG) pit shells, utilizing various input parameters including estimates of the expected mining, processing and general and administrative (G&A) costs, as well as metallurgical recoveries, pit slopes and reasonable long-term metal price assumptions. The mining costs were estimated based on first principles cost buildup for bulk mining in 15-m benches using vendor provided equipment pricing, consumables costs and labour costs. Process and G&A costs were provided by Ausenco based on earlier internal studies. Whittle Consulting examined the deposits with these parameters in their Prober software to evaluate optimal phasing shapes and sequences. This work was used for the design of the Norte and Sur open pits.

Ten phase designs were developed for the Norte open pit. These multiple phases were designed to release mineralized material in a timely manner and to smooth out stripping requirements on an annual basis. The nested pit optimization shells used to determine the ultimate pit were also used to outline areas of higher value for targeted early mining and phase development. The phases split the pit into north and south segments on either side of the quebrada that bisects the Norte Pit. This splitting of the phases allows for better waste stripping sequencing while maintaining the 120 ktpd production rate to the process plant.

The Norte ultimate pit design was based on the RF 0.60 LG shell. The overall dimensions of the ultimate pit are approximately 2,700 m in the north–south direction and 2,600 m in the east–west direction. Due to the variable mountain topography the pit walls will vary from 480 m to a maximum height of 1125 m. It will have a single ramp exit point at the 2,710 m elevation, providing access to the mill feed and waste crushers, as well as the truck shop and fuel bay.

The Sur ultimate pit design has 6 phases based on the RF 0.636 LG shell. The overall dimensions of the Sur pit are 1,400 m in the north-south direction and 1,100 m in the east-west direction. The variable topography for Sur also results in pit slope heights ranging from 180 m to a maximum of 735 m. The Sur pit is designed to haul mill feed and waste to a portal located on the north-east side of the pit where a crusher and conveyor system will take this material to tie into the Norte system. The system uses a tunnel to connect to the Norte pit and then the conveyor runs along a berm to the north side of Norte. This avoids the longer haul from Sur that would result from a truck only system.

Ramps widths for both pits were based on the use of 290 t rigid-frame haul trucks. The operating width used for the truck is 9.0 m. Single-lane roads will be 27.1 m wide (twice the operating width plus berm and ditch) and double lane widths will be 36.1 m (three times the operating width plus berm and ditch). Ramp gradients planned to be 10% both in the pit and ex-pit for uphill gradients. Working benches were designed for a 90 m minimum mining width. Pioneering road development will be significant, and pit phase designs require internal ramps in their highwalls to access later pit phases. Descent rates were limited to 12 benches per year. The mine is scheduled to work 365 d/a, with five days' worth of delay time due to weather disruptions. The plant is scheduled to operate 365 d/a.

The steep topography in the upper elevations of one pit wall has resulted in designs and a mining schedule that has front-loaded significant quantities of waste stripping. Refinements from future iterations of road access and phase designs may be able to improve upon the current designs and reduce capitalized stripping costs.

Mine planning was performed based on marginal cut-offs applied to the NSR grade item. Three years of pre-production mining activities are required to develop approximately 8 km of cut-and-fill haul roads to connect the upper elevations of the Phase 1 and 2 pits to the truck shop area, the mineralized material and waste crushers, and to the CPSF, conduct road building/pioneering, and to strip 78 Mt of waste rock from the pit, stockpiling 2 Mt of mill feed, exposing initial mill feed material, and hauling the waste directly to the base of the CPSF. Pre-stripping will be performed in Years –3,

–2 and –1. The majority of the first two years of pre-stripping will be completed with smaller 40 t rigid body trucks common in Peru. These allow the preparation of the larger more efficient working faces for the large equipment in Year –1 and onwards.

Mill feed delivery to the crusher in the first production year is forecast at 32.8 Mt and will average 43.8 Mt per year for the life of mine.

The pit operations will work two 12-hour shifts per day with four crews on a standard rotation. Engineering, geology and some operations supervisory / support positions will be on day only 12-hour shifts which will rotate weekly.

The mining equipment fleet will include six 38 m^3 electric hydraulic face shovels, two 33 m^3 front-end loaders, thirty-eight 290 t electric-drive haul trucks, and nine blast hole drills that can drill 270 mm and 160 mm diameter drill holes. A fleet of smaller loaders and trucks will be utilized for early mine access development and initial pre-stripping.

A bulk loaded blended emulsion product will be used for blasting.

1.13 Recovery Methods

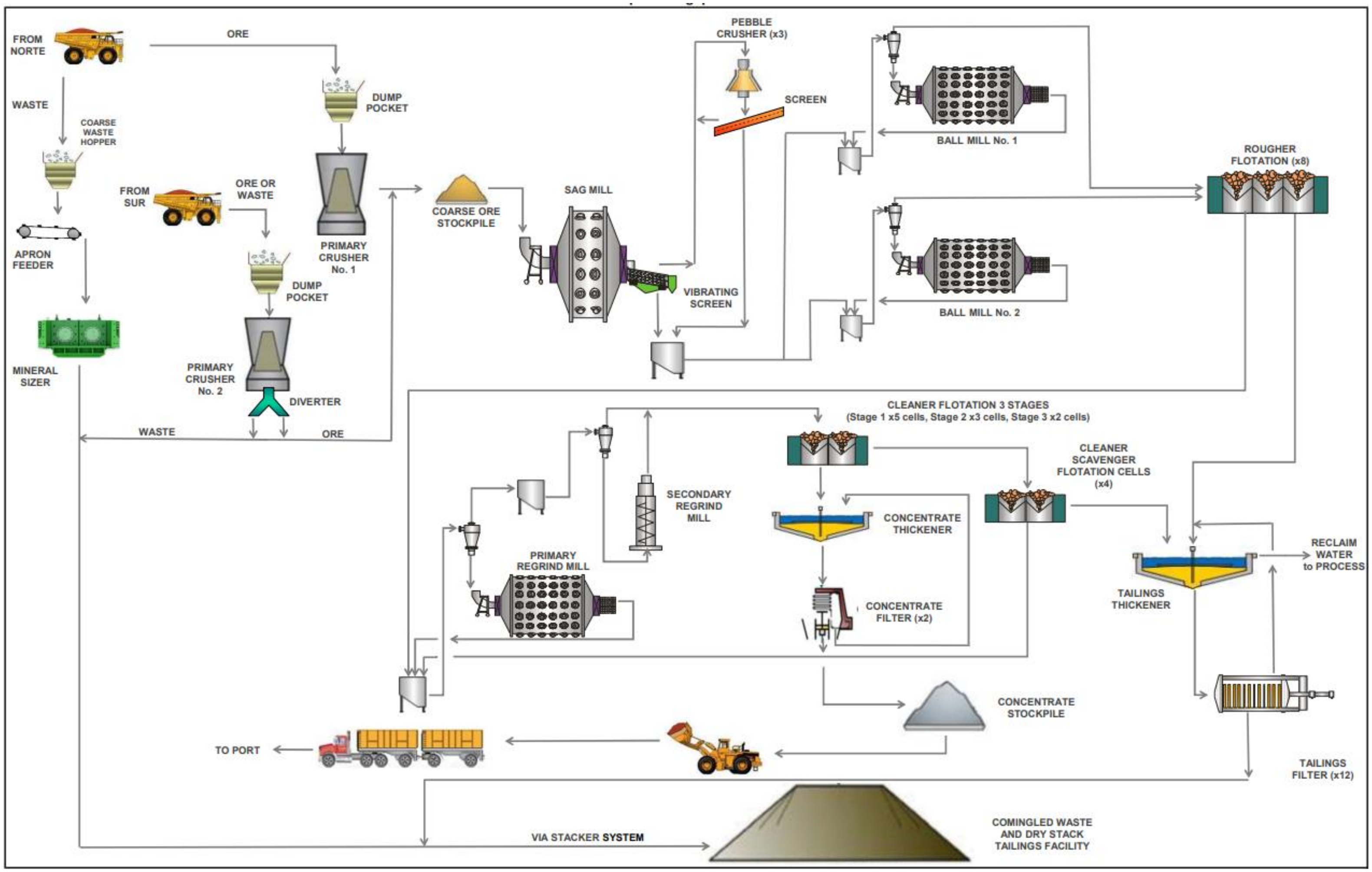
The plant is designed for a throughput of 120,000 t/d during its mine life. The mill will operate two shifts per day, 365 per year, operating at an availability of 91.3%, producing copper concentrate. The crusher plant preceding the mill is set at 75% availability, the tailings dewatering and filtering circuits following the mill are set to operate at 90% availability.

The proposed flowsheet features conventional processes for the following circuits:

- One primary gyratory crusher near the Norte pit to process mill feed with a dedicated overland mill feed conveyor to the crushed mill feed stockpile at the process plant.
- One waste rock mineral sizer near the Norte pit with a dedicated overland waste rock conveyor to the CPSF.
- One gyratory crusher near the Sur pit for both mill feed and waste rock with a single mill feed and waste conveyor to the Norte crushing plant and a diverter chute to direct mill feed and waste rock to their respective conveyors at the Norte crushing plant, installation deferred until Sur mining commences in year 17 of operation.
- Single Gearless Mill Drive (GMD) Semi-Autogenous Grinding (SAG) mill with pebble crushing and screen classification.
- Two lines of ball mills with cyclone classification.
- Rougher flotation followed by secondary regrind with cyclone classification.
- Three stages of cleaner flotation, with cleaner scavenger flotation cells, for copper concentrate production.
- Concentrate thickener, filtration, and loadout facilities.
- Tailing thickener and filtration for dry stack tailings disposal.

A simplified process flow diagram for the Cañariaco project is shown in Figure 1-2.

Figure 1-2: Process Flow Diagram



Source: Ausenco, 2024.

1.14 Project Infrastructure

1.14.1 Overview

Infrastructure of the Cañariaco project includes earthworks development, site facilities, buildings, on-site roads, water management systems, site electrical facilities, raw water supply, camp, and the comingled waste and tailings storage facility.

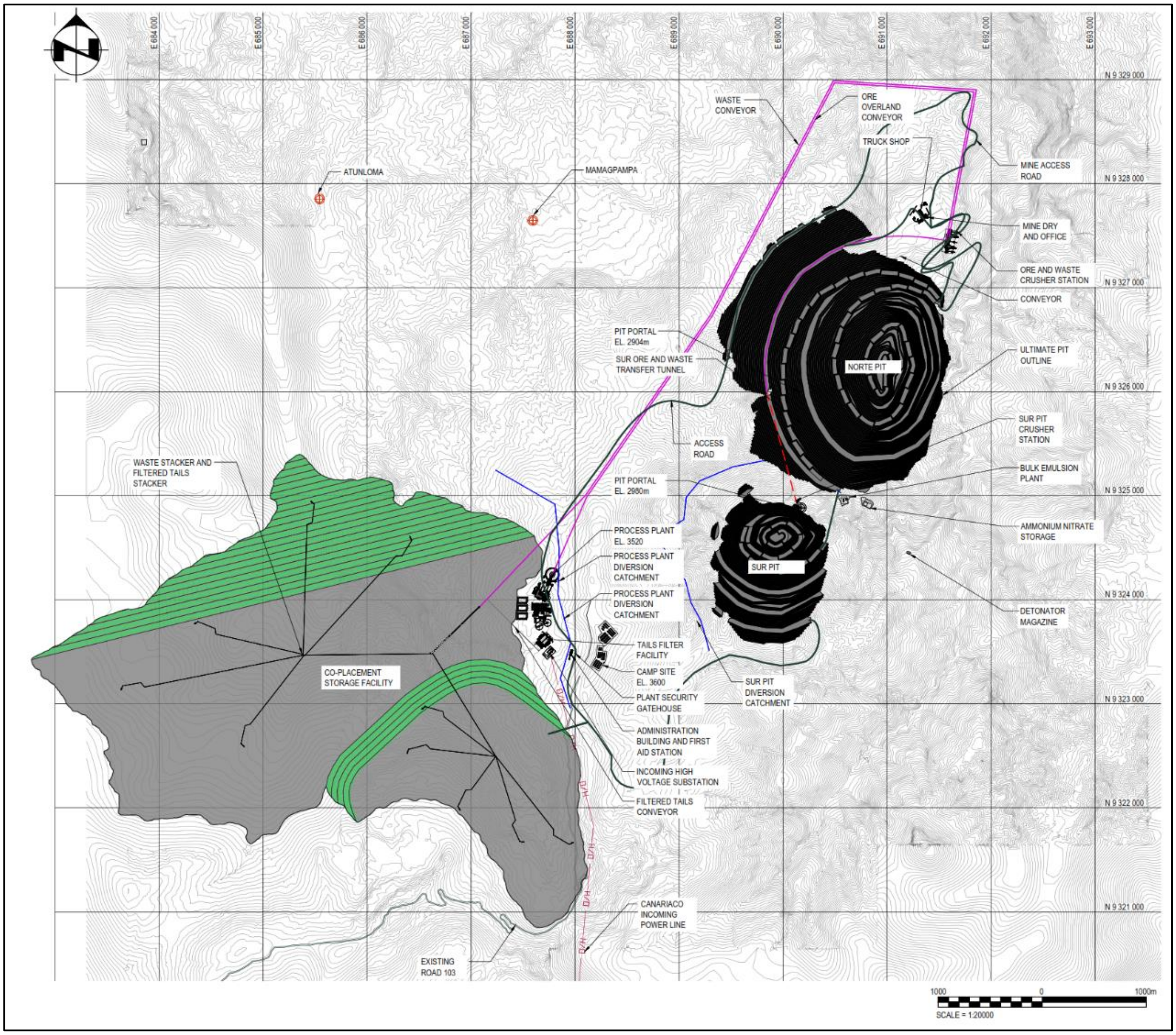
The site includes:

- Mine facilities including the truck shop and mine workshop;
- Common facilities such as the administrative building, medical clinic, helipad and security building. It is further supported with an incinerator, water storage tanks, potable water treatment plant (PWTP) and sewage treatment plant (STP);
- Process facilities housed in the process plant, including grinding, classification, flotation, thickening, filtration, and assay laboratory and process plant workshop and warehouse;
- On-site camp;
- Co-Placement waste rock and filtered (dry) tailings storage facility (CPSF);
- Overland conveyor to convey mineralized material from the pit to the process plant;
- Overland conveyor to convey waste rock from the pits to the CPSF.

Project infrastructure selection was determined based on the following considerations:

- Process plant location with suitable topography, access and proximity to the mining areas;
- Manned facilities located out of mine blasting radius;
- The location of the crusher/ROM pad near the open pit access points;
- Separation of heavy mine vehicle traffic from non-mining light-vehicle traffic;
- Locate permanent camp close to access road and process plant;
- Place administration building and medical clinic near process plant;
- Compliance with surface rights on tenured land with the registered landowner.

Table 1-3: General Site Layout



Source: Ausenco, 2024

The site was selected on a natural elevated area close to existing road and away from watersheds and so that the mine infrastructure area (MIA) and crusher/ROM pad are close to the mine pit to minimize the hauling distance, and to keep the ROM pad activities away from the administration area.

1.14.2 Roads and Access

Road access to the site will be via road 103, which runs along the Huancabamba Valley and through the Mamagpampa hamlet. Internal project site roads will be constructed to the process plant, and the Norte and Sur pits. The project access road will be suitable for the transport of concentrate, freight, and equipment, including all oversized equipment needed for construction and operation of the mine/plant facilities. The entrance to the road will be controlled through the plant security gatehouse, thereby restricting the road to authorized vehicles only. All users will need two-way radios, as the road is planned to be radio-controlled to increase safety and usability.

General vehicle access will be provided around facilities with service roads to remote structures and haul roads around the primary crushing plant, truck shop, and pit. During the life of the mine, roads and lay down areas will be modified and added as needed. The general access roads will be two-way and 8 m wide, and the service roads will be one-way and 5 m wide with pullouts. The haul roads will contain a running surface of 36 m wide, while the access road to the truck shop area will be 15 m wide.

1.14.3 Co-placement Waste and Tailings Storage Facility

A co-placement waste rock and tailings storage facility (CPSF) is envisaged to store this waste material in a single facility. Approximately 2,739 Mt of mine waste will be stored within the CPSF, which includes 1,176 Mt of filtered tailings and 1,563 Mt of waste rock. The CPSF is not expected to behave like a conventional mine waste rock facility because of the large proportion of stored filtered tailings behind waste rock embankments and buttresses. The facility will also consist of rock underdrains, and external berms to provide armoring to protect the filtered tailings from precipitation erosion. The CPSF was classified as high or moderate under both Canadian Dam Association and “Guidelines for Mine Waste Dumps and Stockpile Design” (2017), respectively.

During a portion of pre-production mine waste will be hauled to the CPSF by trucks while the crusher and overland conveyor systems are being constructed. The mine waste will then be crushed and conveyed to the CPSF. The mine waste will be utilized to construct embankments, benches, rock drains, and armor berms to contain the filtered tailings. The balance of the waste rock or potential acid generating (PAG) waste rock will be encapsulated within the filtered tailings. It is proposed that the pre-production waste rock will be non-acid generating (NAG), and any PAG waste rock will be mixed with NAG and encapsulated within the filtered tailings to mitigate metal leaching or acid rock drainage (ML/ARD). Based on the potential acid generation of materials, if required calcium carbonate will be added as part of the ARD mitigation measures. It is assumed that PAG materials will become more prevalent with depth. There is insufficient waste rock to “co-mingle” with the filtered tailings to take advantage of the waste rock strength. Therefore, the waste rock will be used to develop embankments, rock drains, armoring berms and the balance will be placed in the interior with the tailings near the front of the facility to improve physical stability.

The CPSF will be constructed in several phases over the 28-year operational mine life. The CPSF will include a contact water management pond (CWMP) downstream of the facility, surface water management structures, rock drains, access road, two overland waste conveyor systems (waste rock and filtered tailings), equipment for spreading and compacting materials, and a water reclaim pipeline from the CWMP.

Instrumentation and monitoring will be required to assess the performance of the facility. Vibrating wire piezometers will be installed to monitor pore pressure within the embankment fill materials and filtered tailings along with slope inclinometers and survey monuments installed in the embankments to monitor for potential slope movement and deformation.

1.14.4 Power Supply

Electrical power will be supplied to the Project site by a 220 kV overhead transmission line from the local utility substation at Carhuaquero, a distance of 57 km from the mine site. The incoming transmission line will terminate at a new main site substation where it will be stepped down from 220 kV to the site distribution/utilization level of 25 kV to be distributed for the different power requirements across the project site.

1.14.5 Fuel Supply

Tanker trucks will deliver fuel onto the site. The fuel storage and dispensing station will be located at the truck shop, and contain several above ground tanks, including diesel tanks, gasoline tanks and various propane tanks. The tanks will be contained in a lined containment berm to ensure no fuel can leak into the environment.

1.14.6 Water Supply

Raw fresh water will be sourced from the seepage collection pond, mine pits, and CWMP to be used to supply the main process in the form of process makeup water, gland seal water, cooling water and general facility use of fire and potable water.

1.14.7 Water Management Structures

Non-contact runoff waters are diverted through four diversion channels, while contact runoff water is collected from one collection channel. The collected water is captured in the in a collection pond northwest of the process plant. The collected contact water in the pond will be returned to the mill as make-up water, eliminating the need for additional water treatment process.

1.15 Environmental, Permitting and Social Considerations

The Project is located on the surface land of the Community of San Juan de Cañaris. Various baseline studies, investigations and field work have been carried out since 2003. Earlier studies supported the development of the 2012 EIAsd prepared by AMEC and approved by Directorial Resolution No. 177-2012-MEM/AAM in May 2012. In addition, AMEC conducted baseline studies for a detailed EIA from 2007 to 2014. Most recently baseline studies were conducted from 2021 through 2023 by Yaku Consultants for the development of a semi-detailed Environmental Impact Assessment (EIAsd 2021). This EIAsd work was used to support a Declaración de Impacto Ambiental ("DIA") drilling permit application submitted to the Ministry of Energy and Mines ("MINEM") in December 2023.

An overview of the setting of the Cañariaco Norte and Sur deposits within the Cañariaco Project is provided that outlines existing biological and physical baseline conditions, proposed new and ongoing baseline studies to support existing and future permitting and regulatory requirements, including water and waste management requirements. In addition, socio-economic baseline conditions, the status of community engagement, and mine closure and reclamation planning for the Project are outlined. This information was summarized in a previously published NI 43-101 report completed in 2022 (Ausenco). This report has been revised from 2022 to account for additional information made available since 2022, and changes to the scope of the Project including development of the nearby Sur deposit to complement the Norte deposit, and changes to infrastructure including processing facilities, roads, production rates, mine life and footprint of the CPSF.

1.15.1 Environmental Considerations

Completed baseline studies include:

- Physical components: climate and meteorology, air quality and environmental noise, vibrations, soils, geology, geomorphology, hydrology and hydrogeology, and water quality;
- Biological components: ecosystems, flora, fauna, and hydrobiology; and
- Socioeconomic and archaeological components.

The Surface Water Management Plan will preserve the "no contact" status of surface waters to the maximum extent practicable. Water that come into contact with Project facilities will be contained and treated if necessary, so that any water released to the environment will always meet applicable water quality regulatory guidelines. A system of impoundments, embankments, diversions and spillways will be developed immediately upon commencement of construction to manage runoff from construction-related activities. This system will continue to be developed throughout construction and operations to ensure that water discharged to the environment meets guidelines.

The Project will affect two main catchment areas: the Cañariaco valley, where the Norte and Sur open pits will be developed, as well as the mine truckshop, and the Quebrada Yerma valley, which is the proposed location of the process plant, CPSF camp and ancillary facilities. Design criteria for the various diversions and containment facilities will be consistent with Peruvian regulations and international best practices.

Water will be impounded upstream of the CPSF to supply the freshwater replenishment requirements of the process plant. A diversion channel will be constructed to direct excess water from the Cañariaco River around the CPSF. Contact water from the facilities in the well area, the process plant, and the CPSF will be collected immediately downstream of the CPSF and used as mill make-up water; the excess will be discharged to the Cañariaco River following water treatment. Because the mill is expected to consume most of this water, the site water balance indicates that such discharges are only expected towards the end of the mine life.

1.15.2 Closure and Reclamation Considerations

Site reclamation will comply with Peruvian environmental regulations and the International Finance Corporation (IFC) Environmental, Health and Safety Guidelines for Mining (IFC, 2007). The IFC guidelines state that closure and post-closure activities should be considered as early as possible in the planning and design stages.

The reclamation and closure plan will evolve hand-in-hand with the design as the Project progresses through feasibility and permitting. Site-specific knowledge will be acquired during mine development and operations, and the closure plan will be updated to incorporate this knowledge in addition to the environmental and social conditions and circumstances at the time of closure.

The economic analysis in the 2024 PEA assumes a closure cost of US$216M.

1.15.3 Permitting Considerations

In accordance with the requirements of the Peruvian Legal Framework and the Ministry of Energy and Mines, Alta Copper must initiate a process to obtain all environmental, construction and mining permits pertaining to the Project. The main permitting requirements identified to date for the commencement of construction and mining operations are summarized as follows:

- Environmental studies and permits;
- Environment Impact Assessment (EIA);
- Water authorization;
- Archaeological assessment;
- Closure plan;
- Mining operation certificate (MOC);
- Easement by agreement;
- Construction permit;
- Municipal permits to build;
- Labour permits;
- Surface water use license;
- Groundwater use license;
- Sanitary authorization for wastewater treatment;
- Permits to build roads;
- Deed of transportation of controlled substances and products;
- Beneficiation Concession (necessary to process the mineral); and
- Authorization to start operation.

1.15.4 Social Considerations and Community Engagement

The Cañariaco Project is located within, the district of Cañaris, in the province of Ferreñafe, in the Lambayeque region in northern Peru. The capital of the district is the village of Cañaris which is located 6 km north from the Project site.

The Peasant Community, San Juan de Cañaris (SJCC), holds the surface rights within most of the development area of the proposed Cañariaco Project. SJCC now comprises more than 40 hamlets and 3,000 families.

The community retains the main pre-Hispanic characteristics that have allowed it to organize and govern the underdeveloped territory in which it is located. These include:

- Partial collective management of the natural resources within its territory (land, water, forest, natural pastures).

- Its own sociopolitical organization in which authorities and representatives are democratically elected. The community board, which is elected every two years, is the body that represents the community consisting of all community members.
- Archaeological evaluations were carried out in 2007, 2010, May 2011 and June 2014, the results of which were presented as part of the EIAsd and the Modification of the EIAsd, approved by Directorial Resolution No. 177-2012-MEM/AAM and Directorial Resolution No. 462-2014-MEM/DGAAM, respectively. In 2021 and 2023, Yaku Consultants conducted additional archaeological assessments to complement the evaluation of the proposed study area. No archaeological sites of significance were identified in the surveys.
- The public consultation and engagement process for the preparation of the 2012 EIAsd was carried out in accordance with the provisions of D.S. N° 028-2008-EM (Regulations for Citizen Participation in the Mining Sub-Sector) and according to the rules approved by Ministerial Resolution (M.R.). N° 304-2008-MEM/DM, which details the participation mechanisms to be implemented at different stages of the development of a mining project.

Alta Copper continues to advance its involvement with the local community near to the Project by means of:

- Considerable investments in bringing local agricultural producers technical assistance and access to markets at fair prices.
- Articulation of social infrastructure improvements to be financed by other public and private sources.
- Four community offices which allow for ongoing opportunities to engage with the local community and to understand local concerns and needs.

1.16 Market Studies and Contracts

Project economics were estimated based on long-term flat metal prices of US$4.00/lb Cu, US$1,850 /oz Au, US$23.00/oz Ag. These metal prices are in accordance with consensus market forecasts from various financial institutions and are consistent with historic prices for these metals. Alta Copper retained an external specialist consultant for a review of metal payables (including penalty scales). The market terms for this study are based on the terms determined by the consultant as well as recently published terms from other similar studies. The QP is of the opinion that the marketing and commodity price information is suitable to be used in cashflow analyses to support this report. No contracts are in place for the development of the project, which is appropriate for a PEA study.

The metal payables and penalty scales are summarized in Section 19.

1.17 Capital Cost Estimates

The capital cost estimate conforms to Class 5 guidelines for a PEA-level estimate with +50%/-30% accuracy according to the Association for the Advancement of Cost Engineering International (AACE International). The capital cost estimate was developed in Q1 2024 United States dollars based on Ausenco's in-house database of projects and studies, as well as experience from similar operations.

The estimate includes open pit mining, processing, on-site infrastructure, and the comingled waste and tailings storage facility, off-site infrastructure, project indirect costs, project delivery, Owner's costs, and contingency. The capital cost summary is presented in Table 1-4. The total initial capital cost of Cañariaco Project is US$2,160.2 M and LOM sustaining costs are US$518.3 M. The sustaining costs include the expansion of the CPSF and the inclusion of the Sur deposit in year 17.

Table 1-4: Summary of Capital Costs

WBS	WBS Description	Initial Capital Cost (US$M)	Sustaining Capital Cost (US$M)	Total Capital Cost (US$M)
2000	Mine	429.0	186.0	615.0
3000	Process Plant	788.7	25.0	813.7
4000	Site Services and Utilities	105.9	-	105.9
5000	On-site Infrastructure	150.8	216.5	367.3
6000	Off-site Infrastructure	41.9	-	41.9
	Total Directs	**1,516.3**	**427.5**	**1,943.8**
7000	Common Construction Facilities and Services	245.1	13.7	258.8
8000	Owner's Cost	30.3	-	30.3
	Total Indirects	**275.4**	**13.7**	**289.1**
9000	Provisions	368.5	77.1	445.6
	Project Total	**2,160.2**	**518.3**	**2,678.5**

Note: Totals may not sum due to rounding.

1.18 Operation Cost Estimates

The operating cost estimate is presented in Q1 2024 US dollars. The estimate includes mining, processing, maintenance, power, and general and administration (G&A) costs. Table 1-5 provides a summary of the project operating costs. The overall life-of-mine operating cost is US$12,812.2 M over 28 years.

Common to all operating cost estimates are the following assumptions:

- Cost estimates are based on Q1 2024 pricing without allowances for inflation.
- Estimated cost for diesel is US$1.35/L.
- The annual power costs were calculated using a unit price of US$0.072/kWh. This is an average calculated using available power generation sources.
- AGP provided the mine operating costs from first principles applied to the mine production schedule.

Table 1-5: Summary of Operating Cost Estimate

Cost Area	Life-of-Mine Cost (US$M)	Unit Cost (US$/t milled)
Mining	6,317.5	5.37
Process	5,847.2	4.97
G&A	532.0	0.45
Co-Mingle Facility	115.8	0.10
Total	**12,812.2**	**10.89**

Note: Totals may not sum due to rounding.

1.19 Economic Analysis

The economic analysis was performed assuming an 8% discount rate. On a post-tax basis, the NPV8% is US$2,346M, the internal rate of return (IRR) is 24.1%, and the payback period is 3.1 years. A summary of project economics is tabulated in Table 1-6.

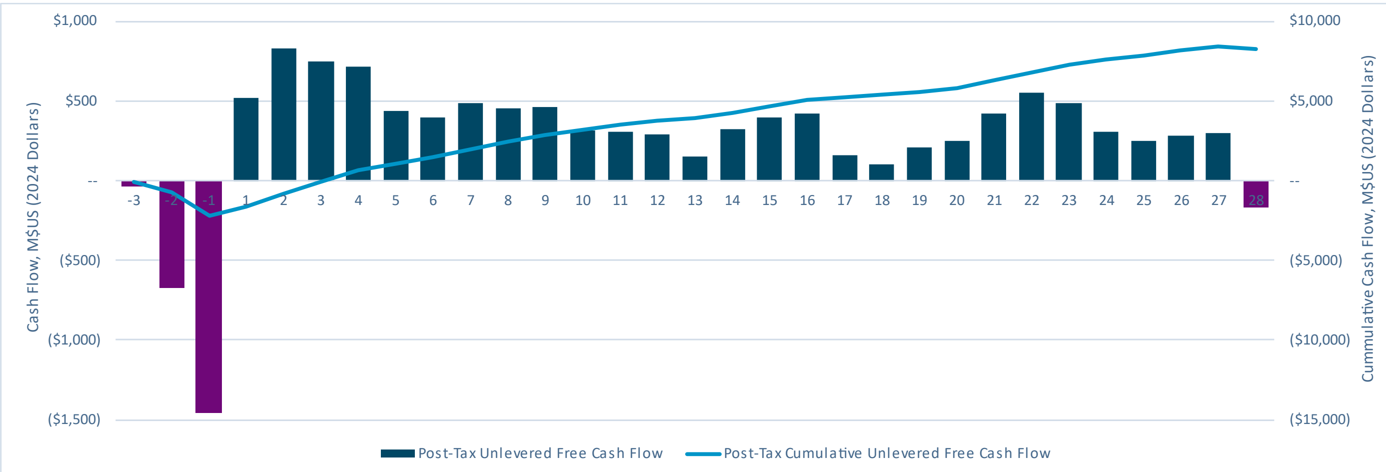
Table 1-6: Economic Analysis Summary Table

General	Units	LOM Total / Avg.	
Copper Price	US$/lb	4.00	
Gold Price	US$/oz	1,850	
Silver Price	US$/oz	23.00	
Mine Life	Years	27.3	
Total Mineralized Material Processed	Mt	1,176	
Total Waste	Mt	1,563	
Avg. CuEq Head Grade	%	0.41	
Production	**Units**	**LOM Total / Avg.**	
Avg. Head Grade – Cu	%	0.35	
Avg. Head Grade – Au	g/t	0.07	
Avg. Head Grade – Ag	g/t	1.59	
Avg. Recovery Rate – Cu	%	88.2	
Avg. Recovery Rate – Au	%	63.3	
Avg. Recovery Rate – Ag	%	55.3	
Total Payable Metal – Cu	M lbs	7,717	
Annual Payable Metal – Cu	M lbs/a	283	
Total Payable Metal – Au	k oz	1,538	
Annual Payable Metal – Au	k oz/a	56.3	
Total Payable Metal – Ag	k oz	29,867	
Annual Payable Metal – Ag	k oz/a	1,094	
Operating Costs	**Units**	**LOM Total / Avg.**	
Mining Cost	US$/t mined	2.38	
Mining Cost	US$/t milled	5.37	
Processing Cost	US$/t milled	4.97	
Co-mingle Tailings Cost	US$/t milled	0.10	
G&A Cost	US$/t milled	0.45	
Operating Cash Costs*, by-product basis	US$/lb Cu	1.20	
Total Cash Costs**, by-product basis	US$/lb Cu	1.82	
All-in Sustaining Costs (AISC)*** , by-product basis	US$/lb Cu	1.91	
Capital Costs	**Units**	**LOM Total / Avg.**	
Initial Capital (Incl. Capitalized Opex)	US$M	2,160	
Sustaining Capital	US$M	518	
Closure Costs	US$M	216	
Financials	**Units**	**Pre-Tax**	**Post-Tax**
NPV (Discounted at 8%)	US$M	4,169.2	2,345.9
IRR	%	32.4	24.1
Payback	Years	2.6	3.1

*Operating cash costs consist of mining costs, processing costs, co-mingle tailings operating costs, and G&A. **Total cash costs consist of operating cash costs plus transportation cost, off-site treatment & refining, transport costs, and non-governmental royalties. ***AISC consist of total cash costs plus sustaining capital, and closure cost.

A summary of the post-tax project cash flow is shown graphically in Figure 1-3.

Figure 1-3: Free Cash Flow – Post Tax



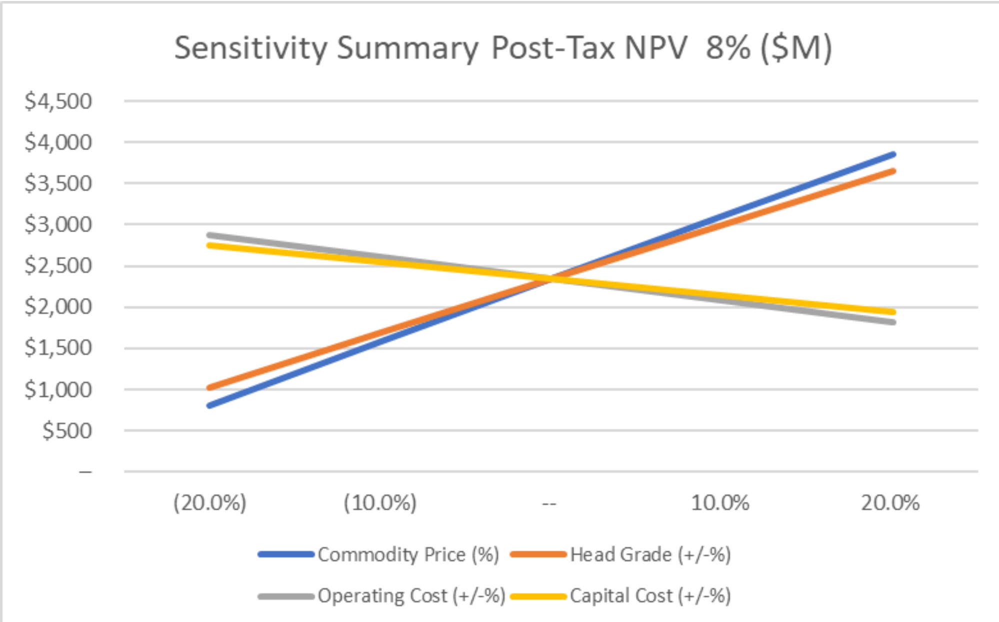
Source: Ausenco, 2024.

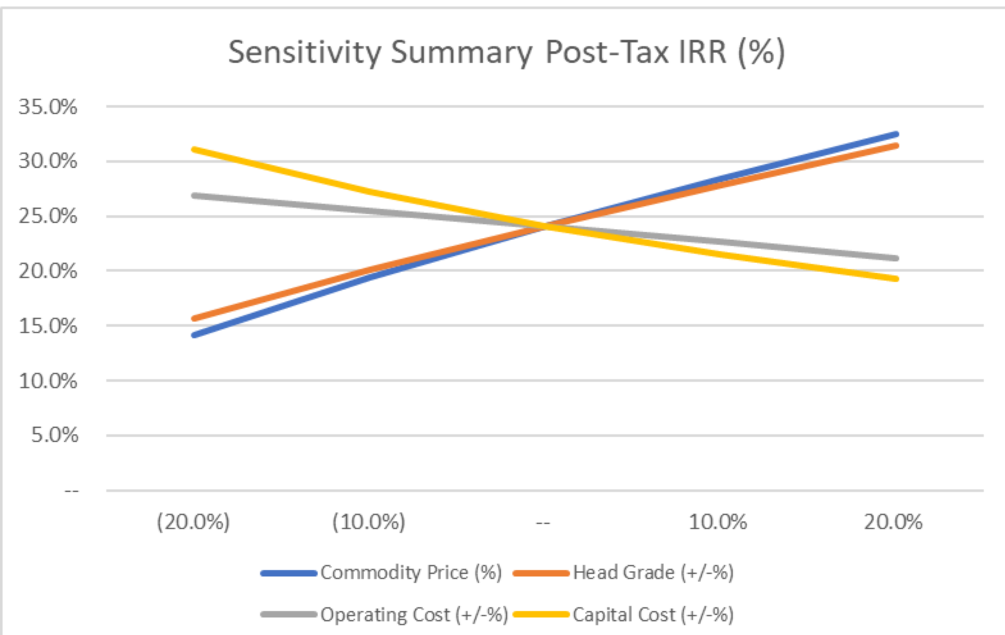
1.20 Sensitivity Analysis

A sensitivity analysis was conducted on the base case pre-tax and post-tax NPV8% and IRR of the Project using the following variables: metal price, discount rate, total operating cost, and initial capital cost.

The sensitivity analysis revealed that the Project is most sensitive to commodity price and head grade and less sensitive to initial capital cost and operating cost.

Figure 1-4: Post-Tax NPV and IRR Sensitivity Analysis





Source: Ausenco, 2024.

1.21 Conclusions and Interpretations

The mineral resource estimate (MRE) for Cañariaco Norte is comprised of Measured and Indicated Resources of 1,126.6 Mt at grades of 0.38% Cu, 0.06 g/t Au and 1.7 g/t Ag plus Inferred Resources of 416.3 Mt at 0.26% Cu, 0.04 g/t Au and 1.3 g/t Ag, all calculated at a cut-off grade of 0.10% Cu. The Measured and Indicated Resource has a copper equivalent grade (CuEq) of 0.42% and the Inferred Resource has a copper equivalent grade of 0.29%.

The Mineral Resource Estimate for Cañariaco Sur is an Inferred Resource of 474.1 Mt at grades of 0.24% Cu, 0.09% Au and 1.2 g/t Ag, calculated at cut-off grade of 0.10% Cu. The Inferred Mineral Resource has a copper equivalent grade of 0.29% Cu.

Based on the assumptions and parameters in this report, the preliminary economic assessment shows positive economics including a post-tax NPV8% of US$2,312M and 24.1% post-tax IRR). This PEA supports a decision to carry out additional detailed studies.

1.22 Recommendations

The Cañariaco Project demonstrates positive economics, as shown by the results presented in this technical report.

Continuing to develop the project through to pre-feasibility study is recommended. Table 1-7 summarizes the proposed budget to advance the project through the pre-feasibility stage.

Table 1-7: Cost Summary for the Recommended Future Work

Program Component	Estimated Budget (US$M)
Exploration, Drilling and Resource Estimation Updates	5
Metallurgical Testwork	0.3
Mining Methods	0.8
Process and Infrastructure Engineering	1.5
Site-wide Assessment and CPSF Geotechnical Field and Lab Program	1.5
Environmental, Permitting, Social, and Community	1.0
Total	**10.1**

2 INTRODUCTION

2.1 Introduction

In 2023 Alta Copper Corp. (Alta Copper) commissioned Whittle Consulting (Whittle) to conduct a mining plan optimization based on the results of the PEA completed by Ausenco ULC (Ausenco) and AGP in 2022 (the "2022 PEA"). Furthermore, this PEA (the "2024 PEA") is to include mining of both the Sur deposit and the Norte deposit and use a significantly larger production rate from the start of production.

The objective of this approach was to realize the value of the combined resources and utilize the benefits of lower unit production costs to enhance overall project financial outcomes.

In coordination with the Whittle optimization work, Alta Copper engaged Ausenco Engineering Canada ULC (Ausenco), AGP Mining Consultants Inc (AGP) and DKT Geosolutions Inc (DKT Geoscience) to update the PEA to also include the Optimised Mine Plan as well as ESG enhancements identified in the 2022 PEA.

The 2024 PEA was prepared in accordance with the Canadian disclosure requirements of National Instrument 43-101 (NI 43-101) and the requirements of Form 43-101 F1.

The responsibilities of the engineering companies contracted by Alta Copper to prepare this report are as follows:

- Whittle provided comprehensive mine plan optimization for the Norte and Sur deposits. This included detailed assessment of metal grades metal prices, metal recoveries, mining and processing costs throughout the two deposits and by applying advanced computational analyses developed an optimized mine plan to maximize net economic value of the mining operation.
- Ausenco managed and coordinated the work related to the report. Ausenco developed the PEA-level design and cost estimate for the process plant, review of the metallurgical test program, general site infrastructure, site water management infrastructure, comingled waste rock and tailings facility and environmental studies and permitting. Ausenco also compiled the overall cost estimate and completed the economic analysis.
- AGP Mining Consultants Inc. (AGP) designed the open pit, mine production schedule, and mine capital and operating cost estimates.
- DKT Geosolutions Inc. updated the Mineral Resource Estimates and completed the work related to property description, accessibility, local resources, geological setting, deposit type, exploration work, drilling, sample preparation and analysis, and data verification.

2.2 Terms of Reference

The Report was prepared to support disclosures in Alta Copper's news release dated May 15th, 2024, entitled "Alta Copper Announces Robust Economics for Cañariaco with US$2.3 Billion After-Tax NPV and 24% IRR". The report is prepared for Alta Copper. Alta Copper changed their name from Candente Copper Corp. in May 2023, and any reference to Candente in this report refers to Alta Copper.

The Report was prepared to support the optimized mine plan by Whittle and the inclusion of the Sur deposit to complement the Norte deposit from the PEA study in 2022. It adopted project configurations that includes comingled waste rock and tailings facility and the usage of a single SAG and dual ball mill comminution circuit.

Units used in the report are metric units unless otherwise noted. Monetary units are in United States dollars (US$) unless otherwise stated. The Report uses Canadian English.

Mineral resources are estimated in accordance with the 2019 edition of the Canadian Institute of Mining, Metallurgy and Exploration (CIM) Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines (2019 CIM Best Practice Guidelines) and are reported using the 2014 CIM Definition Standards for Mineral Resources and Mineral Reserves (2014 CIM Definition Standards).

Readers are cautioned that the PEA is preliminary in nature. It includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the PEA will be realized.

2.3 Qualified Persons

The individuals presented in Table 2-1 serve as the qualified persons for this technical report as defined in National Instrument 43-101, Standards of Disclosure for Mineral Projects, and in compliance with Form 43-101.

Table 2-1: Report Contributors

Qualified Person	Professional Designation	Position	Employer	Independent of Alta Copper	Report Section
Kevin Murray	P. Eng.	Principal Process Engineering	Ausenco Engineering Canada ULC	Yes	1.1, 1.2, 1.9, 1.13, 1.14.1, 1.14.2, 1.14.4-1.14.6, 1.16-1.22, 2.1- 2.3, 2.5-2.7, 3.1, 3.4, 3.5, 13, 17, 18.1-18.5, 18.7-18.9, 19, 21.1, 21.2.1,21.2.2, 21.2.4-21.2.10, 21.3.1, 21.3.2, 21.3.4-21.3.6, 22, 25.1, 25.5, 25.8, 25.9, 25.11-25.14, 25.15.1, 25.15.2.1-25.15.2.3, 25.15.4, 25.16.3, 25.16.4, 26.1, 26.3, 26.5, 27
Scott C. Elfen	P. E.	Global Lead Geotechnical Services	Ausenco Engineering Canada ULC	Yes	1.14.3, 2.4.1, 18.6, 25.10, 25.11, 25.15.2.4, 26.6, 27
James Millard	P. Geo.	Director, Strategic Projects	Ausenco Sustainability ULC	Yes	1.15, 3.3, 20, 25.10, 25.15.3, 25.16.5, 26.7, 27
Jonathan Cooper	P.Eng.	Water Resources Engineer	Ausenco Sustainability ULC	Yes	1.14.7, 18.10, 27
Gordon Zurowski	P. Eng.	Principal/ Partner	AGP Mining Consultants Inc	Yes	1.12, 2.4.2, 15, 16, 21.2.3,21.3.3, 25.7, 25.16.2, 26.4, 27
David Thomas	P. Geo.	Principal	DKT Geosolutions Inc.	Yes	1.3-1.8, 1.10,1.11, 2.4.3, 3.2, 4, 5, 6, 7, 8, 9, 10, 11, 12, 14, 23, 24, 25.2-25.4, 25.6, 25.16.1, 26.2, 27

2.4 Site Visits and Scope of Personal Inspection

A summary of the site visits completed by the QPs is presented in Table 2-2.

Table 2-2: Site Visits

Qualified Person	Date of Site Visit(s)
Kevin Murray, P. Eng.	Has not visited site
Scott C. Elfen, P. E.	February 5 – 6, 2024
James Millard, P. Geo.	Has not visited site
Jonathan Cooper, P.Eng.	Has not visited site
Gordon Zurowski, P. Eng.	February 5 – 6, 2024
David Thomas, P. Geo.	June 7 – 11, 2010, February 14 – 16, 2022

2.4.1 Site Inspection by Scott C. Elfen, P.E.

Scott C. Elfen visited the property on February 5-6, 2024, and was able to review the general topography of the project site and site access options.

2.4.2 Site Inspection by Gordon Zurowski, P. Eng.

Gordon Zurowski visited the property on February 5-6, 2024, and was able to review the general topography of the project site and site access options.

2.4.3 Site Inspection by David Thomas, P. Geo.

Mr. David Thomas most recently visited the Project area from February 14 to 16, 2022. He visited the Cañariaco Sur deposit area including two drill platforms which were located in the field to verify the collar coordinates of drillholes CS12-005 and CS12-006. The QP also reviewed drill core from three drillholes (CS08-001, CS08-002 and CS 13-009). Observations of drill-core confirmed the lithological, alteration and mineralization models for Cañariaco Sur. The QP visually inspected the Cañariaco Norte area.

Mr. Thomas also visited the Project area from 7 to 11 June 2010. During that site visit, Mr. Thomas visited the Project area, where he conducted an inspection of core and surface outcrops, viewed drill platforms and sample cutting and logging areas; discussed geology and mineralization interpretations with Alta Copper staff and reviewed geological interpretations in sections and plans.

2.5 Effective Dates

The effective date of the Mineral Resource estimate: 25 April 2024.

The effective date of the overall report is 31 May 2024.

2.6 Information Sources and References

Reports and documents listed in Section 3 and Section 27 of this technical report were used to support preparation of the technical report. The authors are not experts with respect to legal, socio-economic, land title, or political issues, and are therefore not qualified to comment on issues related to the status of permitting, legal agreements, and royalties. The sources of information include historical data and reports compiled by previous consultants and researchers of the project and supplied by Alta Copper personnel, as well as other documents cited throughout the report and referenced in Section 27 previously completed reports filed on System for Electronic Document Analysis and Retrieval (SEDAR) by previous owners. The QP's opinions contained herein are based on information provided to the QPs by Alta Copper throughout the course of the investigations.

The QPs have relied on Alta Copper's internal experts and legal counsel for details on project history, regional geology, geological interpretations, and information related to ownership and environmental permitting status. This report has been prepared using the documents noted in Section 27 References. The QPs used their experience to determine if the information from previous reports was suitable for inclusion in this technical report and adjusted information that required amending.

This report includes technical information that 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.

2.6.1 Previous Technical Reports

Alta Copper has previously filed the following technical reports on the Project under Candente Copper Corp.:

- Murray, K., Elfen, S., Weston, S., Melnyk, J., Thomas, D. 2022: Cañariaco Norte Project Lambayeque Region, Peru NI 43-101 Technical Report on Preliminary Economic Assessment: technical report prepared by Ausenco Engineering Canada Inc for Candente Copper Corp, effective date 8 February 2022.
- Thomas, D., Melnyk, J., Lipiec, T., and Kozak, A., 2011: Candente Copper Corporation, Cañariaco Norte Deposit, Lambayeque Department, Peru, NI 43-101 Technical Report on Pre-feasibility Study Progress Report: technical report prepared by AMEC Americas Ltd for Candente Resource Corp., effective date 18 January 2011.
- Thomas, D., and Lipiec, T., 2010: Candente Copper Corporation, Cañariaco Project, Lambayeque Department, Peru NI 43-101 Technical Report: technical report prepared by AMEC Americas Ltd for Candente Resource Corp., effective date 8 November 2010.
- Bonson, C., Campbell, R., Bender, M., Doerksen, G., Johnston, A., Meyer, T., Nowak, M., Pilotto, D., Van Egmond, R., Critikos, P., Ostolaza, R., and Huanani, A., 2008: Revised Preliminary Economic Assessment Technical Report, Cañariaco Norte Project, Peru: technical report prepared by SRK Consulting Ltd for Candente Resource Corp., effective date 30 November 2008.
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- McCrea, J.A., 2006: Technical Report on the Cañariaco Copper Porphyry Prospect Department of Lambayeque Northwest Peru: technical report prepared for Candente Resource Corp., effective date 27 April 2006.
- McCrea, J.A., 2005: Technical Report On The Cañariaco Copper Porphyry Prospect Department of Lambayeque Northwest Peru: technical report prepared for Candente Resource Corp., effective date 22 April 2005.
- Currie, J.A., 2004: Technical Report On The Cañariaco Copper Porphyry Prospect Department Of Lambayeque Northern Perú Latitude 6° 05’ S Longitude 70° 17’ E, technical report prepared for Candente Resource Corp., August 2004.
- Huanqui, F., Freeze, J.C., and Coder, J.M., 2002: Geological Report On The Cañariaco Copper-Gold Porphyry Prospect Department Of Lambayeque Northwest Perú NTS 13-E 70° 17’ E 6° 05’ S: technical report prepared for Candente Resource Corp., effective date 9 August 2002.

2.7 Name and Unit Abbreviations

Table 2-3: Abbreviations

Abbreviation	Description
Ai	abrasion index
ANA	Autoridad Nacional de Agua (National Water Authority, Peru)
BWi	Bond ball mill work index
BXWL	BXWL core size
CM	contract management
CWi	crusher work index
CWMP	contact water management pond
CPSF	Co-Placement storage facility
DMT	dry metric tonne
DWT	drop weight test
EA	environmental assessment
EDGM	earthquake design ground motion
EIAsd	semi-detailed Environmental Impact Assessment
GEMA	Servicios Geográficos y Medio Ambiente S.A.C.
GMD	Gearless Mill Drive
IDF	inflow design flood
INGEMMET	Peruvian Instituto Geológico, Minero y Metalúrgico
IP	induced polarization
LG	Lerchs-Grossmann

Abbreviation	Description
LOM	life of mine
MAP	mean annual precipitation
ML/ARD	metal leaching and acid rock drainage
NCD	NCD core size
NPI	net profits interest
NSR	Net Smelter Return
NWA	National Water Authority Peru (Autoridad Nacional de Agua)
NXWL	NXWL core size
PLS	programmable logic controllers
PMA	particle mineralogical analysis
PWPT	potable water treatment plant
QP	Qualified Person
RF	revenue factor
ROM	run of mine
RQD	rock quality designation
SUNARP	Superintendencia Nacional de los Registros Públicos (Peru)
UIT	*Unidad Impositiva Tributaria* (imposed fiscal unit)
WMT	wet metric tonne
WTP	water treatment plant

Table 2-4: Unit of Measurement

Unit	Description
PEN S/	Peruvian nuevo sol
US$	United States dollars
°C	degrees Celsius
cm	centimetre
g	gram
g/L	grams per litre
h/d	hours/day
ha	hectare
hr	hour
K	hydraulic conductivity
km	kilometres
km/h	kilometres per hour
kt	kilo tonnes
Kv	kilovolts
L	litre
L/s	litres per second
m	metre
masl	metres above sea level
Min	minute

Unit	Description
Mm^3	Million cubic metres
Mt/a	million tonnes per annum
t	tonnes
t/a	tonnes per annum
t/h	Tonnes per hour
µm	micron

3 RELIANCE ON OTHER EXPERTS

3.1 Introduction

The QPs have relied upon the following other expert reports, which provided information regarding mineral rights, surface rights, property agreements, royalties, environmental, permitting, social license, closure, taxation, and marketing for sections of this Report.

3.2 Property Agreements, Mineral Tenure, Surface Rights and Royalties

The QPs have not independently reviewed the legal status and ownership of the Project area and any underlying property agreements, mineral tenure, surface rights, or royalties. The QPs have fully relied upon, and disclaim responsibility for, information derived from Alta Copper and legal experts retained by Alta Copper for this information through the following document:

- Mario Chirinos Dentons Peru, 2022: Legal Report: report prepared for Cañariaco Copper, March 11, 2022.

This information is relied upon in Section 1.4, 4 of the Report and is used to support the Mineral Resource estimate in Section 14 and the economic analysis in Section 22.

3.3 Environmental, Permitting, Closure, and Social and Community Impacts

The QPs have fully relied upon, and disclaim responsibility for, information supplied by Alta Copper through the following document:

- Yaku Consultores, 2021. Estudio de Impacto Ambiental semidetallado Proyecto Cañariaco. CapÍtulo 3: Línea de Base. Prepared for Cañariaco Copper Perú S.A. (Semi-detailed Environmental and Social Baseline).

This information is relied upon in Section 1.15, 20 of the Report and in support of the Mineral Resource estimate in Section 14 and the economic analysis in Section 22.

3.4 Markets

The QPs have fully relied upon, and disclaim responsibility for, information derived from Alta Copper and experts retained by Alta Copper for this information, including the following:

- Open Mineral 2024, "Marketing Study Update for the Cañariaco Project"; report prepared for Alta Copper; April 2024.

This information is relied upon in Sections 1.16, 1.19, 19 and 22.

3.5 Taxation

The QPs have not independently reviewed the taxation information. The QPs have fully relied upon, and disclaim responsibility for, taxation information and guidance supplied by Alta Copper's tax consultant, Ernst and Young Ltd. Who have reviewed and approved the tax model as confirmed via email:

- Mary Seminario Cisneros, 2024. EY – Subject: RE: 107967-01 Alta Copper Fin Model Rev E. Email received 10-05-2024 at 6:40AM PDT.

This information is relied upon in Sections 1.19 and 22.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Property and Title (Jurisdiction)

The Cañariaco Project is situated within the Province of Ferreñafe, in the Region/Department of Lambayeque, in northwestern Peru, and is approximately 700 km northwest of Lima, the capital of Peru, and approximately 102 km to the northeast of the city of Chiclayo. Project centroids are latitude 06° 05' south and longitude 79° 17' west.

4.2 Project Ownership

The Cañariaco Project comprises a total area of 9,725.12 ha in 15 concessions. Two porphyry copper deposits and one porphyry copper prospect, Cañariaco Norte, Cañariaco Sur, and Quebrada Verde, have been identified. Alta Copper, through its Peruvian subsidiary Cañariaco Copper Perú S.A (CCPSA), and others described below holds a 100% interest in the Cañariaco Project.

4.3 Mineral Tenure

The government of Peru auctioned the mineral rights to the Project in 2001 under "Public Auction (Bid) PRI-51-2000, Private Investment Promotion – Mining Prospects". Alta Copper, through its wholly-owned Peruvian subsidiary, Exploraciones Milenio S.A. (name subsequently changed to Candente Copper Peru S.A. (CCPSA), was the sole bidder, and made a one-time payment of US$75,880 for an unencumbered 100% interest in the Project. Alta Copper has since acquired additional claims. Some of the newer claims are held by Alta Copper's other subsidiary, Cobriza Metals Peru S.A. ("CZA") while the most recent seven claims were acquired by Peru Fortescue S.A.C. ("F") and were subsequently transferred to CCPSA in March 2022. The current mineral tenure is summarized in Table 4-1 and the claims are shown in Figure 4-1.

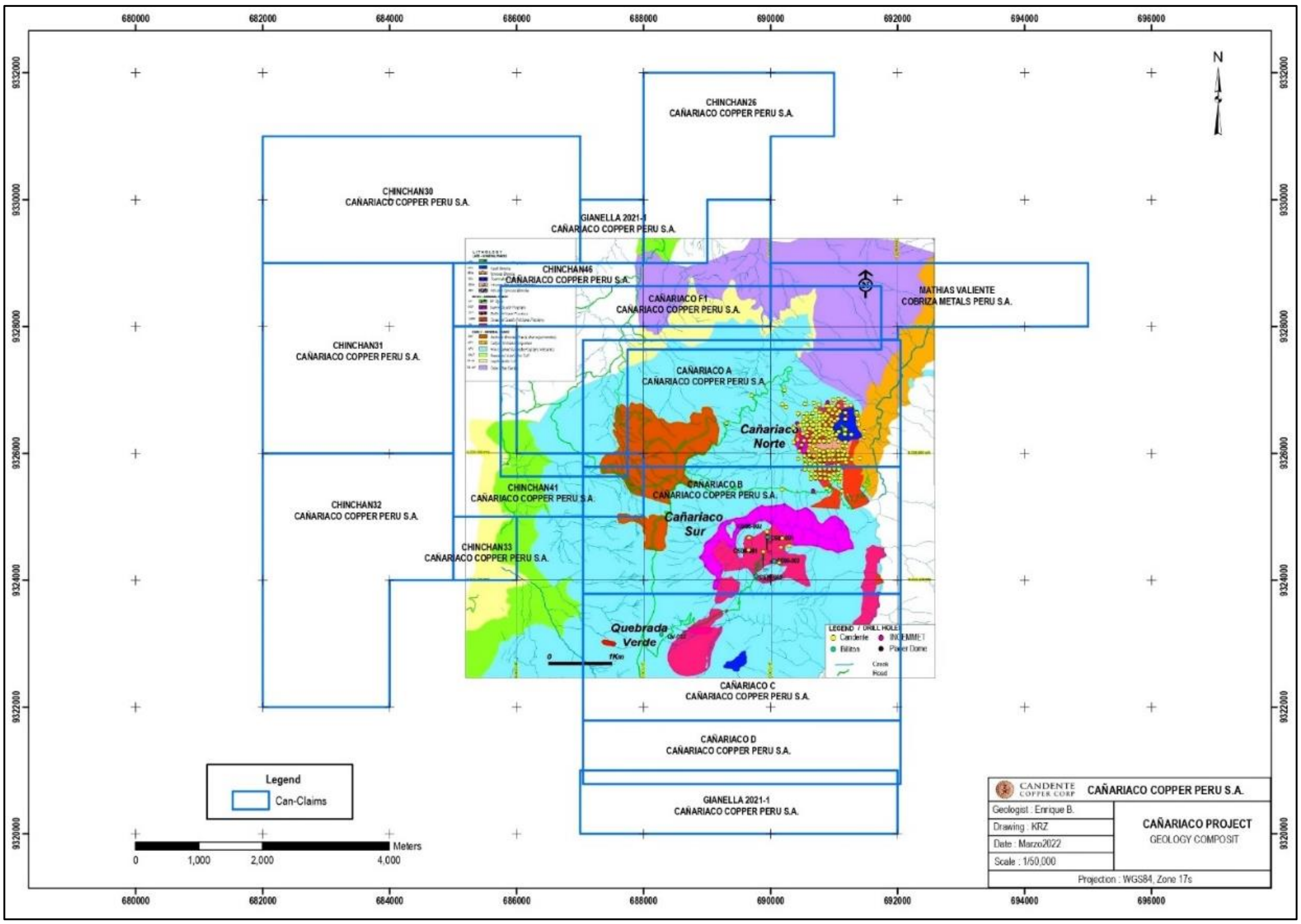
Table 4-1: Mineral Tenure Table

No.	Claim Name	Code	Holder	Granted	Hectares	Validity Fee* (US$)	Penalty (US$ or Peru Sole)	Years Penalty Owed
1	Cañariaco A	15000004Y02	CCPSA	1992	1,000	3000	20,000	2023
2	Cañariaco B	15000005Y02	CCPSA	1992	1,000	3,000	20,000	2023
3	Cañariaco C	15000006Y01	CCPSA	1992	1,000	3,000	20,000	2023
4	Cañariaco D	15000007Y01	CCPSA	1992	500	1,500	10,000	2023
5	Cañariaco F1	01-00312-04	CCPSA	2004	789.5	2,368.5	S/ 72,634	2023
6	Mathias Valiente	01-01981-20	CZA	2024	600	1,800	N/A	2023
7	Gianella 2021-1	01-02100-21	CCPSA	2022	100	300	N/A	2023
8	Gianella 2021-2	01-02101-21	CCPSA	2023	500	1,500	N/A	2023
9	Chinchan26	01-00234-20	CCPSA	2022	600	1,800	N/A	2023
10	Chinchan30	01-00205-20	CCPSA	2022	1,000	3,000	N/A	2023

No.	Claim Name	Code	Holder	Granted	Hectares	Validity Fee* (US$)	Penalty (US$ or Peru Sole)	Years Penalty Owed
11	Chinchan31	01-00217-20	CCPSA	2021	900	2,700	N/A	2023
12	Chinchan32	01-00204-20	CCPSA	2022	1,000	3,000	N/A	2023
13	Chinchan33	01-00233-20	CCPSA	2021	100	300	N/A	2023
14	Chinchan41	01-00188-20	CCPSA	2021	306.17	318.52	N/A	2023
15	Chinchan46	01-00219-20	CCPSA	2021	329.45	988.34	N/A	2023

Notes: ** The validity fee is currently US$3 per year and per hectare. Non-payment of the fee for two consecutive years results in cancellation of the mining concession.

Figure 4-1: Mineral Tenure Plan



Source: Figure prepared by Alta Copper, 2022.

Mining legislation in Peru does not require location of concession boundaries on the ground. The boundary limits are defined by UTM coordinates in the mineral concession title filed on record in the registrar of the Instituto Geológico, Minero y Metalúrigico (INGEMMET) and the Superintendencia Nacional de los Registros Públicos (SUNARP).

To maintain the concessions in good standing, annual validity payments of $3.00 per hectare must be paid and when the minimum production or investment (exploration) has not been met by the 10th year calculated from the year following the concession grant, a penalty must be paid starting in the 11th year. In Alta Copper's case, the 10th year was 2008, and penalties have been paid since that date when exploration investments were not sufficient (minimum of .10 times the penalty per year and per hectare). Only penalties from 2022 are currently owing and are to be paid June 30, 2024.

The penalty rate is calculated as 2% of the minimum production, which is derived by multiplying the number of hectares by 1 UIT (*Unidad Impositiva Tributaria*). The 2% rate increases to 5% in the 16th year and 10% in the 21st year.

If the minimum production is not obtained by the expiration of the 30th year, the mining concession expires.

Minimum Production is defined as the equivalent of one UIT per year (PEN S/4,600 soles or US$1,250 approx.) per hectare granted, in the case of metallic substances.

Ausenco was advised by Alta Copper that all required property payments have been made, and the concessions are in good standing. Providing the annual property payments are made in a timely manner, the concessions will not expire.

4.4 Property Agreements

There are no agreements in place that are relevant to the Project.

4.5 Surface Rights

Mineral concession holders are protected under the Peruvian Constitution and Civil Code. Mineral concession rights do not, however, confer ownership of the land nor surface rights, thus, the owner of a mining concession must deal with the registered landowner to obtain access or use of land rights. It is a requirement of the Peruvian Government that any mineral property developer either rents or purchases the surface rights, or makes an appropriate agreement with the surface rights owner, for the use of their land.

The agreements for access rights (servidumbre) are all negotiable, there are no fixed terms country wide and may involve payments and/or benefits including jobs and/or social/development programs. The level of remuneration generally increases with both the level of impact on the land and/or current use of the land. As such agreements for non-invasive activities such as geological mapping, sampling rocks or soils or doing EIA baseline studies have much lighter obligations than drilling, trenching and exploitation. Agreements related to the latter are required as part of the permitting process and therefore must be registered with the Public Mining Registry (MINEM).

In the case of the Cañariaco property, the Community of San Juan de Cañaris is the owner of most of the land covered by the mineral concessions. However, in addition, individual community members ("posesionarios") have been granted rights from the community to use certain lands for agriculture or other purposes.

Given the two-level ownership situation it is the practice in the San Juan de Cañaris community that CCPSA enters servidumbre agreements with the posesionarios that involve direct payments which are minimal for non-invasive activities but increase with invasive activities (drilling, trenching etc.). The agreement with the community as a whole generally includes benefits such as social and development programs.

The Company currently has agreements with the *posesionarios* holding rights to lands where activities have been conducted since 2020 such as EIA baseline studies and technical visits. One of these agreements includes a monthly fee for rental of the area used by the Cañariaco Norte camp. In addition, funds have been provided since 2021 for community projects such as road works and irrigation canals. While most of these agreements are documents in written actas or formal agreements, some are verbal as many posesionarios are not literate and the activity on their land is very occasional and minimal. In the latter situations, the agreements may be evidenced by the receipts for the payments accepted by the posesionarios.

The last agreement for with the community for access rights for drilling was in effect for the period of July 2012 to July 2015 and included a commitment of 1,500,000 Peruvian soles (approximately US$400,000) to be provided for sustainable development projects over the three years. An administrative committee involving the Company, the Community and a local institution (presiding member) oversaw the implementation of the funding and the projects.

A new ESIA baseline study was recently completed in 2023 in support of a DIA drilling permit applications which was submitted to MINEM in late 2023. Given this, new servidumbre agreements for drilling are under discussion with the community and the posesionarios. There are no other requirements, other agreements with the community and the posesionarios for the Company to have legal access to the property.

4.6 Water Rights

Alta Copper will need to obtain an authorization from the National Water Authority (ANA) to use water for exploration and/or mining purposes, including for domestic and industrial use.

4.7 Royalties and Encumbrances

There is a 0.5% royalty payable to Anglo Pacific, a third party, on the Cañariaco A, B, C, D and F1 concessions.

Profit based taxes are due to the Government of Peru. It is expected that the Project will incur a net profits interest (NPI) royalty. The QP notes that if changes to the royalty legislation currently being considered by the Peruvian Government are made, then the royalty burden could increase.

4.8 Environmental Considerations

Refer to Section 20.1.

4.9 Permitting Considerations

Exploration activities to date have been undertaken in accordance with the appropriate Peruvian regulations. A Class B environmental assessment (EA) was conducted by AMEC S.A. (Peru) (AMEC) and/or Servicios Geográficos y Medio Ambiente S.A.C. (GEMA) for Alta Copper as required by Peruvian law when trenching, drilling, or other major work was performed from 2012 through 2024. GEMA also conducted a Class C EA and obtained a Class C permit from the Peruvian Government for advanced drilling and exploration work at the Cañariaco Project in 2014. Yaku Environmental Consultants were engaged in 2021 to conduct an EIAsd ("semi detailed EIA" under the new category regime) and this was reduced to a DIA and submitted in December 2023 to support a new drilling application needed to support the planned geotechnical and exploration and resource definition drilling programs discussed in Section 21.

A number of permits will be required to support project development and operation and are discussed in Section 20.

Alta Copper has received certificates of confirmation from the Peruvian National Institute of Culture (INC) that no archaeological remains exist within the area of the proposed Cañariaco Norte open pit as well as the current known area of mineralization at Cañariaco Sur and projected for Quebrada Verde.

4.10 Social License Considerations

The Project footprint is located within the Community of San Juan de Cañaris. The community holds the surface rights such that under Peruvian Laws Alta Copper will be required to have a valid agreement with the community to undertake exploration and/or exploitation activities. Alta Copper has operated under community agreements since exploration started in 2004.

Development of the Cañariaco Project is unlikely to require resettlement of families residing within the immediate Project area; however, a baseline study will be required prior to Project development to confirm this assumption.

4.11 Comments on Property Description and Location

In the opinion of the QP, the information discussed in this section supports the declaration of Mineral Resources, based on the following:

- As of the effective date of the report, there are no other significant factors, risks that may affect access, title or the right or ability to perform work on the Property;
- Information from legal experts support that the mining tenure held is valid and is sufficient to support declaration of Mineral Resources;
- Surface rights are held by the Peasant Community of San Juan de Cañaris. Access has been negotiated with the community as required for exploration to date and is underway for the upcoming drill program;
- There is a 0.5% net smelter return (NSR) royalty due to a third party, Anglo Pacific;
- Profit based taxes are payable to the Peruvian Government, at a variable scale. The current profit-based tax rates are under review by the Government of Peru, and may increase;

- Permits obtained by Alta Copper to the Report effective date to support exploration activities were sufficient to ensure that activities are conducted within the regulatory framework required by the Peruvian Government;
- Additional permits will be required for Project development and operations; preliminary discussions have been held with the relevant statutory authorities;
- Development of the Cañariaco Project is unlikely to require resettlement of families residing within the immediate Project area; however, a baseline study will be required prior to Project development to confirm this assumption. If relocation is required, this will require additional studies and negotiations;
- At the effective date of this Report, environmental liabilities are limited to those expected for an exploration-stage project, and include drill pads and access roads; and
- The current state of knowledge on environmental and permit status for the Project supports the estimation of Mineral Resources (refer to discussions in Section 20).
- As of the effective date of the report, there are no other significant factors and risks that may affect access, title right or ability to perform work on the Property.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

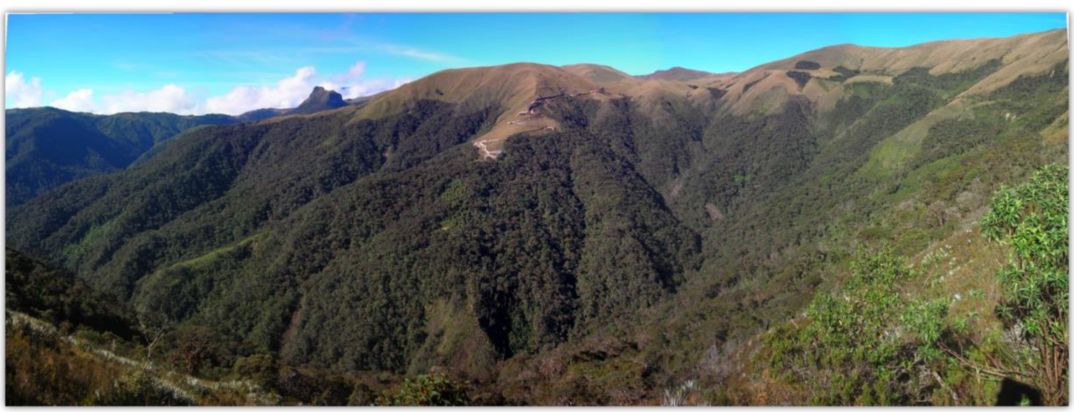
5.1 Physiography

The Cañariaco Norte Project area covers moderate elevations ranging from 2,600 to 3,600 metres ("m") above sea level. The copper deposits are situated on the eastern side of the continental divide and infrastructure will be on the top as well as both western and eastern sides of the divide. The topography varies from steep incised valleys at lower elevations to open grassy highlands at upper elevations (Figure 5-1 and Figure 5-2). Within the mineral resource estimate area, fingers of sub-tropical forests extend up small valleys. These forests transition into open grasslands and broad valleys as the elevation increases (>3,000 masl) to the west of the main mineralized zones.

Figure 5-1: View of Cañariaco Norte Project Mineral Resource and Pit Area – Looking West

Source: Alta Copper, 2022.

Figure 5-2: View of Cañariaco Sur Deposit and Quebrada Verde Prospect – Looking South from Cañariaco Norte



Source: Alta Copper, 2022.

Vegetation at higher elevations consists mainly of ichu and other types of natural grasses used for livestock grazing. Localized agriculture plots for subsistence farming are maintained at lower elevations.

5.2 Accessibility

The Cañariaco Project is accessible by road from Lima via the city of Chiclayo as shown in Figure 1-1 and Figure 5-3.

Figure 5-3: Existing Cañariaco Property Access Routes and Regional Infrastructure

Source: Alta Copper, 2022.

Either the Pan-American Highway (700 km, 11-hour trip) or one of the several daily commercial airline flights can be taken from Lima to Chiclayo. The route from Chiclayo to the Cañariaco Project is currently a 150 km six-hour trip along mostly unimproved roads via Incahuasi. The road is not currently suitable for heavy trucks, and there is no landing strip within the Project area. Road travel time from Chiclayo to the Cañariaco site is summarized in Table 5-1.

Table 5-1: Existing Routes from Chiclayo to Cañariaco

Route	Distance (km)	Road/Trail/Type
Chiclayo – Motupe – Pucara	217	Paved road
Pucara – Cañaris	59	Unimproved road
Cañaris – Cañariaco	6	New Mountain Road
Total	**281**	-
Chiclayo – H. da. Batan Grande	51	Paved road
H. da. Batan Grande – Incahuasi – Ullurpampa	68	Secondary gravel road
Uyllurpampa – Road Terminus	23	Drivable track
Road Terminus – Cañariaco Norte Camp	2	4x4 access road to camp
Total	**144**	-

5.3 Climate

The Project spans several climate zones and temperatures may vary by up to 10°C with altitude from the lower forested valleys to the upper grassy ridges. Annual summer and winter temperatures range from 3°C to 25°C. The region receives between 830 mm and 1,700 mm of rain each year, with the wet season extending from November to March. Periods of precipitation are generally of short duration, from 1½ to 2 hours. The prevailing winds are typically from the southeast and are generally light to moderate, with maximum gusts up to 90 km/h. Climate conditions are not expected to affect ground-based operations, although helicopter-supported activities and four-wheel drive roads could be affected during the rainy season. The proposed mine operations envisaged in the 2024 PEA have been scheduled year-round, 24 h/d, with no allowance for shutdowns due to weather-related events.

5.4 Local Resources and Infrastructure

Peru has a long history of mining. Consequently, mining professionals and machine operators are generally available in most population centres. Alta Copper is expected to be able to source Peruvian personnel for virtually all mine administration and operation requirements.

Supplies and other normal services are available in Chiclayo and other towns to the north such as Pucara and Jaen. Specialist exploration services, such as drilling and geophysical and geochemical analyses, are contracted out of Lima.

At present, Alta Copper has a fully established camp and catering facility for 100 people at the Project. Core-handling facilities are also in place on site. Core storage and detailed logging are conducted at Alta Copper's secure facility in Chiclayo.

The main camp is situated near the Río Cañariaco and has ample water supply from the river and several tributaries. Electricity is generated on site. Before proposed Project operations begin, a 220 kV overhead transmission line will be installed from the local utility substation at Carhuaquero, 57 km to the south of the planned mine site.

Peruvian freight forwarders have recommended three ports that could be used for construction purposes: Callao, Paita, and Salaverry.

Information on infrastructure that will be required to support the assumptions in the 2024 PEA is provided in Section 18.

5.5 Comments on Accessibility, Climate, Location Resources, Infrastructure and Physiography

In the opinion of the QP:

- There is sufficient suitable land available within the concessions for tailings disposal, mine waste disposal, and installations such as a process plant and related mine infrastructure.
- A review of the existing and likely power and water sources, manpower availability, and transport options indicate that there are reasonable expectations that sufficient labour and infrastructure is available to support estimation of Mineral Resources.
- Mining operations are envisaged to be year-round.

6 HISTORY

6.1 Property Exploration History

Copper was first discovered at Cañariaco Norte in the period from 1967 to 1970, during a regional stream sediment sampling program carried out by the Peruvian Instituto Geológico, Minero y Metalúrgico (INGEMMET), in cooperation with a British Geological Survey team. This work identified copper anomaly A-2. Subsequently, from 1971 to 1974, INGEMMET undertook a detailed geochemical study of the Río Cañariaco valley, which can delineate three centres of alteration and copper mineralization, designated Cañariaco Norte, Cañariaco Sur, and Quebrada Verde. The geochemistry was supported by geological mapping, rock chip and soil sampling, induced polarization/resistivity (IP) and ground magnetic geophysical surveys, and core drilling of five drill holes (1,500 m) in Cañariaco Norte. Thin section and petrographic studies were also performed. A mineral resource estimate was undertaken at the conclusion of this work.

Placer Dome Exploration Inc. (Placer Dome) optioned the area from the Peruvian Government in 1994. Work completed included geological mapping, rock chip sampling, trenching (2,200 m) and pitting (80 test pits), petrographic studies, re-interpretation of the available INGEMMET IP/resistivity data, three core holes (853.91 m), and a very preliminary estimate of tonnage and grade. This work identified a porphyry copper system related to a Late Tertiary intrusive-breccia complex emplaced into Early Tertiary, Calipuy Group volcanic basement of andesite to rhyolite pyroclastics and flows. However, in 1997, the option was not exercised and reverted to the Peruvian Government.

During 1999, Billiton Exploration and Mining Perú B.V. (Billiton) optioned the property from the Peruvian Government. The work program comprised geological mapping, soil and rock chip sampling, IP/resistivity and ground magnetic geophysical surveys, and seven core holes (1,128.7 m). Petrographic samples were collected and studied, and sequential leach copper analysis was carried out on some of the drill core. A resource estimate was completed in 1999. The option was dropped in 2000, with the property returning to the Peruvian Government.

The property was subsequently put up for auction. Alta Copper, through its wholly owned Peruvian subsidiary, Exploraciones Milenio (subsequently CCPSA), acquired 100% ownership of the Project in February 2002. Since that date, Alta Copper has completed geological mapping, prospecting, IP/resistivity and ground magnetic geophysical surveys, rock chip sampling, petrographic studies, resource definition drilling, drilling for metallurgical testwork and geotechnical assessment, and re-logging of existing drill core. A total of 289 core holes (85,183.16 m) were drilled by the end of 2014. Mineral resources were first estimated on behalf of Alta Copper in 2004 and were updated in 2006, 2008, 2010, and again in 2022.

A preliminary economic assessment (PEA) was undertaken in 2006. The study envisaged conventional open pit mining of the Cañariaco Norte deposit, with mineralization being crushed, then placed on a heap leach pad. Copper recovery was planned through a solvent extraction-electrowinning (SX–EW) plant to produce copper cathode.

In 2008, additional leach testing had indicated that heap leaching recoveries were variable, and a substantial increase in the extent of the sulfide mineralization warranted a metallurgical process suitable for sulfide type copper mineralogy. Therefore, a conventional process plant consists of primary crushing, semi-autogenous grinding (SAG), ball mills, and a flotation circuit to produce copper concentrate. In late 2010, AMEC completed an updated resource upon which a new PEA was completed in 2011.

A pre-feasibility study commenced in March 2010 and in January 2011 a Pre-feasibility Progress Report Update was completed. Alta Copper engaged AMEC and Knight Piésold to prepare aspects of a more detailed mining study from April 2011 to June 2013. During that time, metallurgical drilling was completed; 50% of a planned geotechnical drilling program was conducted; site layout, process plant, infrastructure and ancillary facilities design was well advanced; preliminary mine design was completed; and the environmental and social impact study was approximately 80% completed. The study was put on hold in 2014 due to a downturn in global metal mining investments and lack of interest in the equity markets which in turn significantly reduced Alta Copper's ability to raise funding.

Table 6-1 summarizes the Project exploration history.

Table 6-1: Exploration History

Year	Operator	Work Undertaken
1967–1970	INGEMMET	Regional stream sediment sampling. Identified anomalous copper in drainages surrounding the Cañariaco Norte deposit.
1971–1974	INGEMMET	Geological mapping at 1:25,000 scale defined 9 km^2 of alteration and mineralization, with three distinct centres identified.
		Limited soil and rock chip sampling; numbers of samples unknown.
		Nine trenches; locations unknown. A total of 23 rock chip samples taken from the trenches and analysed for Cu and Mo.
		Infill geological mapping at 1:5,000 scale; this detailed mapping was hampered by dense vegetation and steep topography.
		IP/resistivity ground geophysics over Cañariaco Norte.
		Five vertical core drill holes completed at Cañariaco Norte to depths of 300 m for 1,500 m total drilling. A total of 579 core samples were collected along 3 m or 1.5 m lengths and analysed for Cu and Mo. A total of 66 composite geological samples were analysed for total copper (TCu), Mo, Ag, and Au.
		Thin section and petrographic studies on selected drill core samples.
		Mineral resource estimate.
1994–1997	Placer Dome	Geological mapping at 1:1,000 scale at Cañariaco Norte; preliminary geological mapping, scale not known, at Cañariaco Sur.
		Soil and rock chip sampling over an area of 1.3 km x 1.5 km at Cañariaco Norte. A total of 715 rock chip samples assayed for Au, Cu, Mo, (and some for Ag and As). Reconnaissance rock chip and grab sampling at Cañariaco Sur.
		Trenching (2,200 m) and pitting (80 test pits) at Cañariaco Norte and Cañariaco Sur.
		Re-interpretation of INGEMMET IP/resistivity data; the re-interpretation noted some correlation between copper grades and chargeability and/or resistivity. It was also noted that some of the strongest IP targets had not been drilled by INGEMMET.

Year	Operator	Work Undertaken
		Three core holes (853.91 m) at Cañariaco Norte.
		Petrographic studies on selected drill core samples.
		Mineral resource estimate.
1999–2000	Billiton	Geological mapping; scale unknown.
		Soil and rock chip sampling. Sampled outcrops in the streams south of the drilled area at Cañariaco Norte. Low-level Cu anomalies (<500 ppm) were collected from phyllically-altered quartz porphyry at Cañariaco Norte. The northern side of the quartz porphyry generated additional surface copper anomalies (500 to 2000 ppm Cu) in outcrops of basaltic andesite. In the outcrops around the drill holes in the northern part of the quartz porphyry, gold graded in the 100 to 200 ppb Au range. In the southern half of the system, the grades were generally in the 40 to 100 ppb Au range. Soil samples were only taken at Cañariaco Sur and Quebrada Verde, and outlined co-incident Cu, Au and Mo anomalies over a 750 m diameter area with smaller anomalies of copper and gold adjacent to this target. Soil sample line spacing was typically 200 m with 100 m infill where initial results were favourable. Sample spacing was 50 m along the lines.
		IP/resistivity and ground magnetic geophysical surveys on four lines at Cañariaco Norte. For the IP survey, a dipole–dipole array, with electrode spacing of 100 m, was used. The results were not useful because only a very weak current was received at the potential electrode. The ground magnetic data used the same lines and spacings; no domains of magnetite destruction could be defined.
		Seven core holes (1,128.7 m) drilled at Cañariaco Norte, Cañariaco Sur, and Quebrada Verde.
		Petrographic studies on selected drill core samples.
		Sequential leach copper analysis on 12 samples from two intervals of core; results indicated that the mineralization was potentially bio-heap leachable.
		Mineral resource estimate.
2002 to present	Alta Copper	Geological mapping, including 1:2,000 scale at Cañariaco Norte and Cañariaco Sur. Preliminary mapping of the Cañariaco Sur and Quebrada Verde zones was also carried out to assess soil geochemistry anomalies outside of the main mineralized zones. Zones of silicification and quartz vein stockwork exist peripheral to the main circular feature at Cañariaco Sur.
		Structural measurements. A total of 311 structural measurements including faults, veins, fractures etc. were collected.
		Prospecting, rock chip and grab sampling. A total of 148 rock samples were collected in 2002 to 2004 from surface for gold, total copper and 35-element inductively-coupled plasma (ICP) analyses. In addition, a number of PIMA and petrology samples were collected to define favourable alteration including alunite and sericite and to define the various igneous phases. A total of 392 soil geochemistry samples and 355 rock samples were collected over the geophysical grid at Cañariaco Sur and Quebrada Verde in 2008. Soils with elevated levels of copper of up to 5,720 ppm (0.57%) and gold up to 330 ppb covering over an area of 0.9 km x 1.4 km and potassic alteration mapped over a length of 2.3 km at Cañariaco Sur. Anomalous levels of copper of up to 2,200 ppm (0.22%) and up to 497 ppb gold in soils cover an area of approximately 0.7 km x 0.9 km at Quebrada Verde.
		Re-logging of existing drill core.
		20-line km of IP/resistivity and ground magnetic geophysics. Outlined areas of magnetite destruction and delineated the various alteration zones and helped confirm and/or identify old/new fault structures at Cañariaco Norte. A coincident magnetic high; IP/resistivity low and chargeability high identified at Cañariaco

Year	Operator	Work Undertaken
		Sur. A large IP chargeability high centered on a resistivity anomaly and covering an area of 0.8 km x 2.0 km identified at Quebrada Verde.
		Petrographic studies on selected drill core samples.
		Four bulk samples for leach testing collected in 2004.
		As of 2009, 72,189,7 m had been drilled in 248 core holes at Cañariaco Norte and Cañariaco Sur;
		EIAsd 2012 obtained and drilling commenced Dec 2012;
		In 2012–2013, 2,761 metres were drilled in 10 metallurgical holes and 2,553 metres were drilled in 6 geotechnical holes in Cañariaco Norte;
		2012-2013, 4,205.15 m were drilled in 10 drill holes for exploration in Cañariaco Sur, to date 15 holes have been drilled in Cañariaco Sur which indicate a copper-gold porphyry deposit that is still open to the south and west;
		By 2014 a total of 85,183.16 m had been drilled in 289 drill holes in Cañariaco Norte and Cañariaco Sur;
		2020 to 2023 - Yaku conducted environmental work for a new baseline for drilling permits;
		December 2023 submitted DIA application for permits for additional exploration drilling in Cañariaco Norte and Sur and initial drilling in Quebrada Verde
		2022 PEA completed for improved ESG aspects and a smaller start-up by Ausenco.
		Current 2024 PEA by Ausenco goes to 120,000 tpd throughput, adds Cañariaco Sur to an Optimized Mine Plan by Whittle which also continues with improved ESG aspects.

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

The Cañariaco deposits are located in a prolific mineral belt that hosts many well-known copper and gold mines and deposits including La Granja, Cerro Corona and Yanacocha (see Figure 7-1).

Basement rocks comprise pelitic schists of the Precambrian to Early Paleozoic Olmos Complex that are unconformably overlain by Late Triassic–Early Jurassic La Leche Formation marine sediments that have minor intercalated volcanic units. An Early to Late Jurassic volcano-sedimentary sequence, the Oyotún Formation, overlies the earlier units. A regional geological plan for the project area is shown in Figure 7-2.

Following regional uplift, erosion, and subsequent subsidence, the lower portion of the Goyllarisquizga Group, a regionally extensive quartz arenite, was unconformably deposited on the earlier lithologies. In turn, the arenite was overlain by Early Cretaceous to mid-Late Cretaceous marls, shales, and limestone. A stratigraphic column is provided in Figure 7-3.

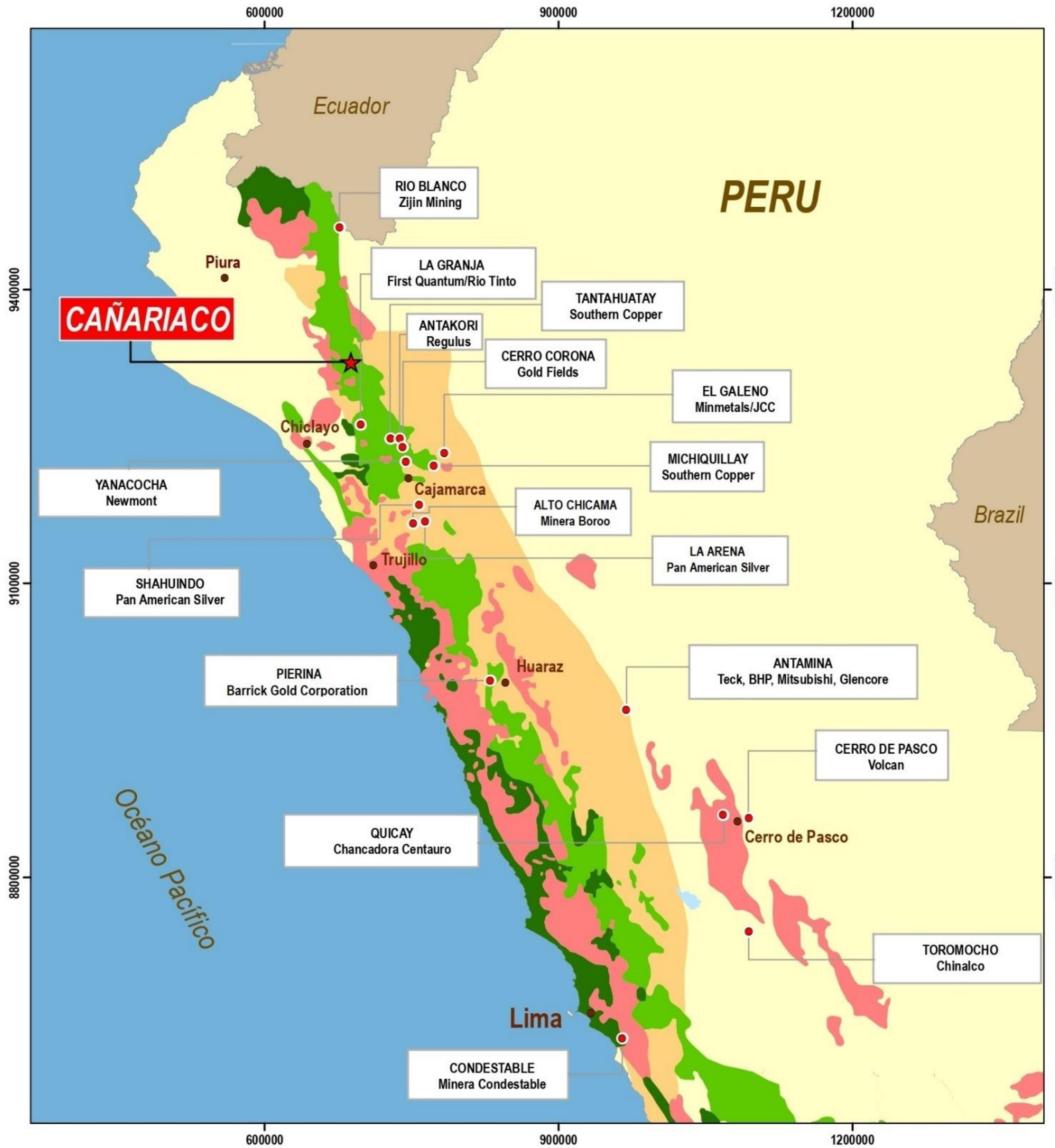
In the Early Tertiary, volcanic units of the Llama and Porculla Formations of the Calipuy Group had erupted, followed by uplift and erosion. Renewed volcanic activity commenced with the deposition of volcanic rocks of the Huambos Formation during the Late Tertiary. Porphyry stocks, breccias, and dykes that are also Late Tertiary in age intrude the Cretaceous and Early Tertiary units (Figure 7-3).

Age dating using K/Ar and Re–Os on intrusive rocks, breccias, and alteration minerals in the Cañariaco Norte deposit returned dates ranging from 15.8 Ma to 17.9 Ma (Mathur, 2008; Casselman et al., 2008).

A number of circular features/intrusions have been identified district wide. A single circular feature that measures 8 km x 10 km encompasses all three mineralized centres in the Project area. This feature is centered on, or close to, major fault intersections, suggesting that the emplacement of the intrusive complex was localized by fault intersections (Murphy, 2004).

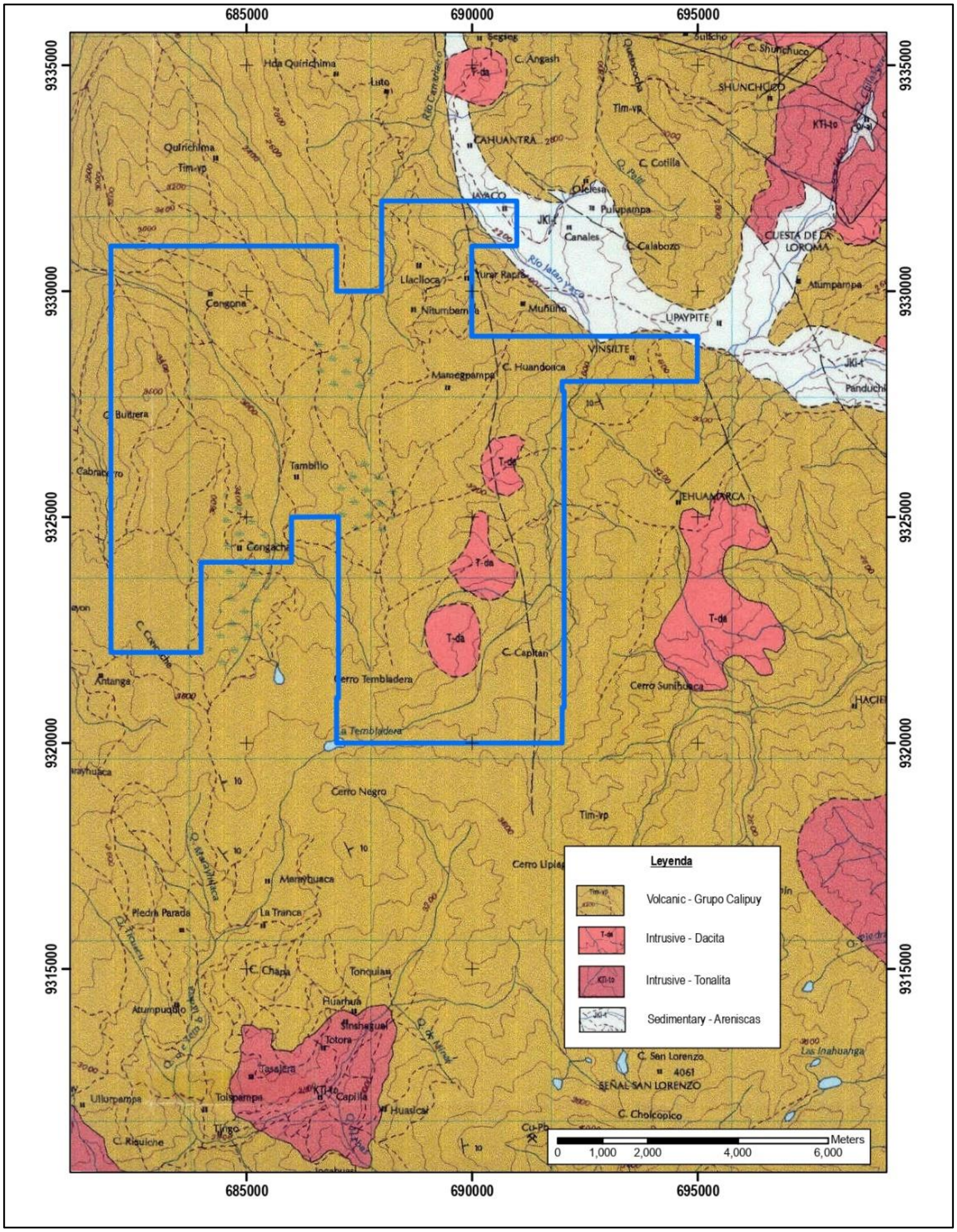
Faults that have been identified at a regional scale consist of two parallel, long range, northwest–southeast-trending district-scale faults. One bisects the Project area, and the second fault is approximately 7 km to the northeast. The northwest–southeast faults have a probable conjugate northeast–southwest fault set that tends to be of medium range. These faults appear to control the location and development of the intrusive–breccia complexes and related mineralization and alteration in the Project area.

Figure 7-1: Copper and Gold Mineral Belts with Mines and Deposits in Central and Northern Peru



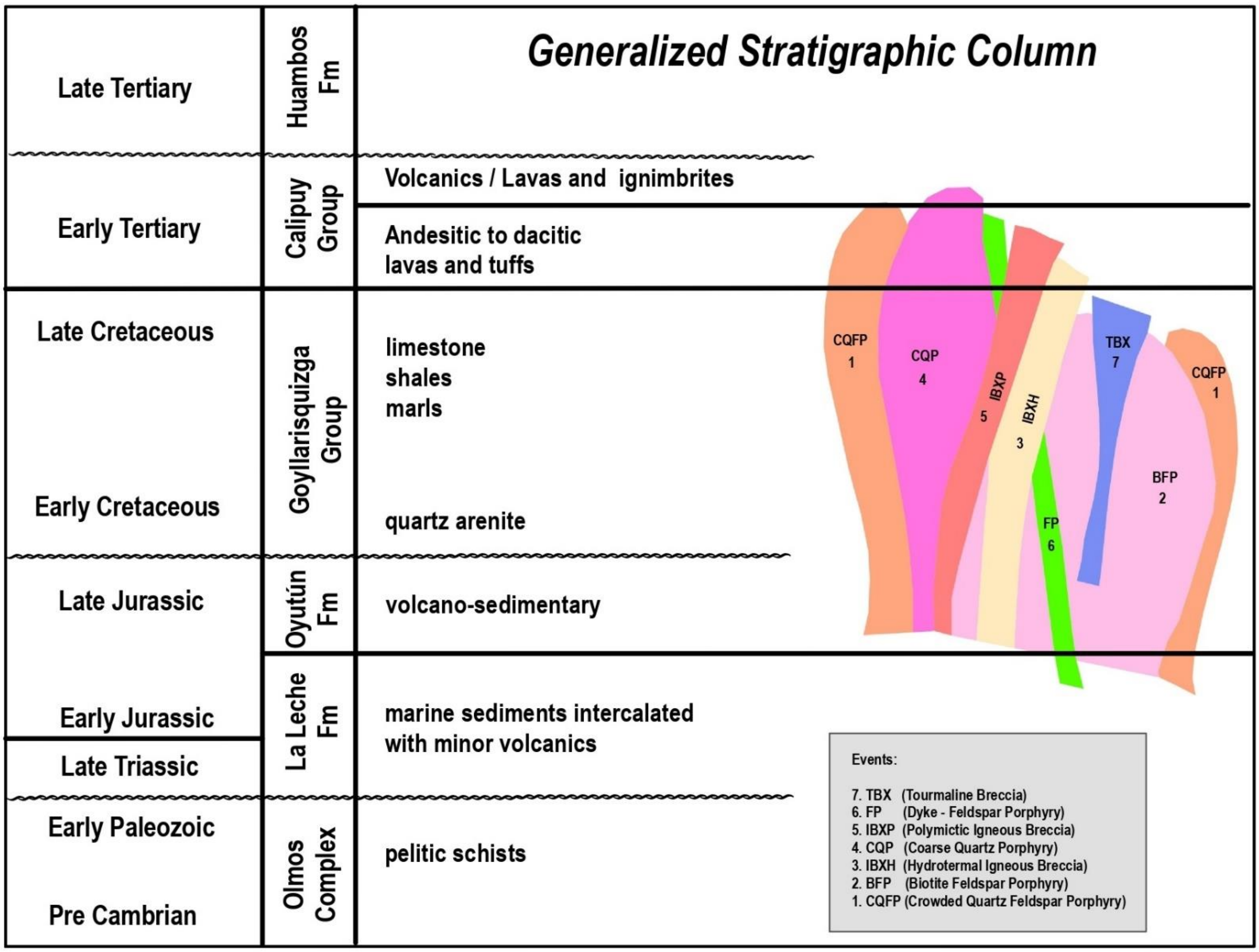
Source: Alta Copper Corp., 2024.

Figure 7-2: Regional Map Cañariaco Project



Source: Alta Copper Corp., 2011. Notes: Tenure outlines shown on the plan are superseded by the tenure outlines in Figure 4-1. Grid squares on the plan are 4 km x 4 km.

Figure 7-3: Regional Stratigraphic Column with Cañariaco Intrusive Complex



Source: Alta Copper, 2011.

7.2 Project Geology

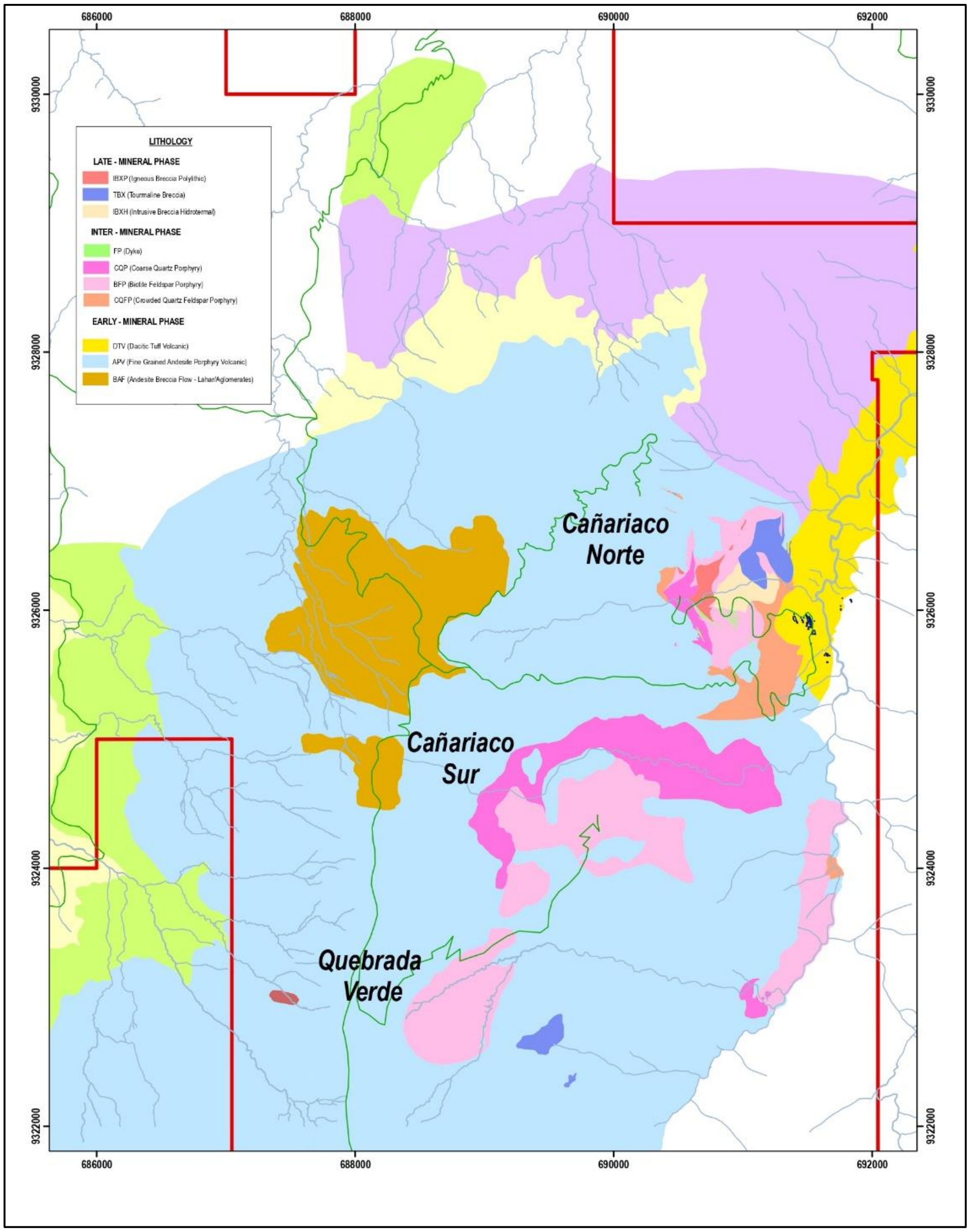
The Project covers a north-easterly 4–5 km long structural trend that hosts three porphyry centres, Cañariaco Norte, Cañariaco Sur and Quebrada Verde, as delineated by geophysics, geochemistry and geological mapping and drilling.

In each of these centres various intrusive bodies have been mapped. The host rocks to the mineralized intrusions are Calipuy Group andesite volcanic rocks.

Large, coincident magnetic highs, IP/resistivity lows, and chargeability highs are identified coincident with stream sediment and soil geochemical anomalies and areas of altered rocks that are typical of porphyry deposits.

Property Geology map is provided in Figure 7-4.

Figure 7-4: Project Geology Plan – Cañariaco Norte, Sur & Quebrada Verde



Source: Alta Copper, 2013.

7.2.1 Cañariaco Norte

The Cañariaco Norte deposit is hosted within a multiphase intrusive–breccia complex approximately 1.7 km in strike extent and 1.1 km wide. The deposit has been drill-tested to a depth of approximately 770 m and remains open at depth. The majority of the copper–gold mineralization is hosted within the intrusive and breccia units, but locally extends for variable distances into the enclosing volcanic units. Intrusive units comprise approximately 55 to 60% of the deposit, breccias approximately 30 to 35%, and pre-mineral volcanic rocks approximately 5 to 10%.

The intrusive units are nested and collectively oval in shape, with older intrusive rocks being cut by successively younger intrusive bodies. In general, the intrusive units are north–south-trending, steeply-dipping bodies. The breccia units cut the intrusive units, are oval to circular in shape, and are steeply plunging. The dykes generally strike northwest–southeast with a steep southwesterly dip. The shape and positioning of the intrusive, breccias, and dyke units was largely controlled by northwest–southeast- and northeast–southwest-trending faults (Casselman et al., 2008).

7.2.1.1 Lithologies

Table 7-1 presents the Cañariaco Norte stratigraphy. Please refer to Figure 7-3: Regional Stratigraphic Column with Cañariaco Intrusive Complex.

Table 7-1: Cañariaco Norte Stratigraphy

Lithology	Rock Unit Identification	Relative Age	Rock Unit Description
Breccias	IBxP	Breccia (youngest)	Polymictic breccia
	TBx	Breccia (middle)	Tourmaline breccia
	IBxH	Breccia (oldest)	Hydrothermal breccia
Intrusive Units	FP	Dykes	Feldspar porphyry dykes
	CQP	Intrusive (youngest)	Coarse quartz porphyry
	BFP	Intrusive (middle)	Biotite feldspar porphyry
	CQFP	Intrusive (oldest)	Crowded quartz feldspar porphyry
Volcanic Units	VC	Calipuy Group	Pre-mineral andesitic, dacitic and rhyolitic volcanics

The following sections on lithologies have been taken from Casselman et al. (2008).

The oldest rocks, ascribed to the Early Tertiary Calipuy Group, are a series of dacite tuffs with lesser, bedded, rhyolite tuffs overlain by andesite porphyry flows and pyroclastic rocks. Andesite pyroclastic rocks and flows dominate on the western, northern and southern sides of the intrusive complex where elevations are higher. The eastern side of the intrusive complex is lower in elevation, and thus the dacite and rhyolite volcanic rocks that underlie the andesite volcanic rocks are exposed adjacent to the intrusive complex.

Three major intrusive generations have been identified. The oldest intrusive unit is a crowded quartz–feldspar porphyry (CQFP), which consists of 1 to 3% quartz eyes, 35% feldspar phenocrysts, and 3 to 5% hornblende–biotite. Grain sizes of the constituents range from 1 to 2 mm. The unit is interpreted as dioritic in composition.

The middle intrusive unit is a biotite–feldspar porphyry (BFP) interpreted to be granodioritic in composition, with 3 to 5% quartz eyes, 15 to 20% feldspar phenocrysts, 2 to 5% euhedral biotite, and traces of hornblende. Grain sizes typically range from 3 to 5 mm.

The youngest intrusive unit, a coarse quartz porphyry (CQP), consists of 5 to 10% quartz eyes (grain size range from 3 to 5 mm), 15 to 20% euhedral feldspar crystals (<3 mm) and traces of biotite (1 to 2 mm). The CQP is interpreted to be of quartz monzonitic composition.

The youngest intrusive stage is a set of feldspar porphyry dykes/breccias (FP) that range in thickness from 2 to 30 m and have a northwest–southeast strike, with a steep, southwesterly dip. The dykes have a fine-grained ground mass with 10% to 20% feldspar phenocrysts (3 to 10 mm) and 5% to 10% hornblende phenocrysts (2 to 8 mm). The dykes commonly display cooling contacts. Where the dykes intersect the breccias, dyke fragments occur as large (>10 m), rotated, and weakly fractured blocks within the breccias. The dykes have been only weakly altered and contain minor copper mineralization where they have been fractured, brecciated, and crackle-brecciated near the south margin of the hydrothermal breccia unit.

Three breccia bodies post-date the intrusive rocks. The oldest unit, hydrothermal breccia (IBxH), consists of matrix-supported angular to sub-angular biotite–feldspar porphyry and crowded quartz–feldspar porphyry fragments that display little or no evidence of transport. Fragment sizes within the central part of the breccia are generally 1 to 5 cm in a fine-grained matrix. Near the southern margin of the breccia, there is a high component of feldspar porphyry dyke fragments up to tens of metres in size. The unit is not well mineralized, with copper grades related to the inclusion of mineralized porphyry fragments. The margins of the hydrothermal breccia can show crackle brecciation.

The middle tourmaline breccia (TBx) has a fine-grained matrix consisting of quartz and tourmaline. The unit is extensive and was emplaced along the northeastern margin of the intrusive bodies. Breccia fragments are angular to sub-angular, 1 to 10 cm in size, and include clasts of the biotite–feldspar porphyry, crowded quartz–feldspar porphyry, and the hydrothermal breccia. The margin of the breccia shows crackle brecciation. The breccia is not mineralized, apart from copper grades related to the inclusion of mineralized porphyry fragments.

The youngest breccia unit, polymictic breccia (IBxP), is a late-stage breccia with an erratic shape, cross-cutting all earlier units. Sub-rounded to rounded clast fragments include vein quartz, all three intrusive units, and the two earlier breccia phases. Fragments range from 0.5 to 10 cm with the breccia margins often grading into a crackle breccia. The unit shows multiple breccia pulses, the last of which is a fluidized micro-breccia with rounded fragments that are typically <3 mm in size, cross-cutting all other pulses. Copper mineralization occurs in both the matrix and the fragments.

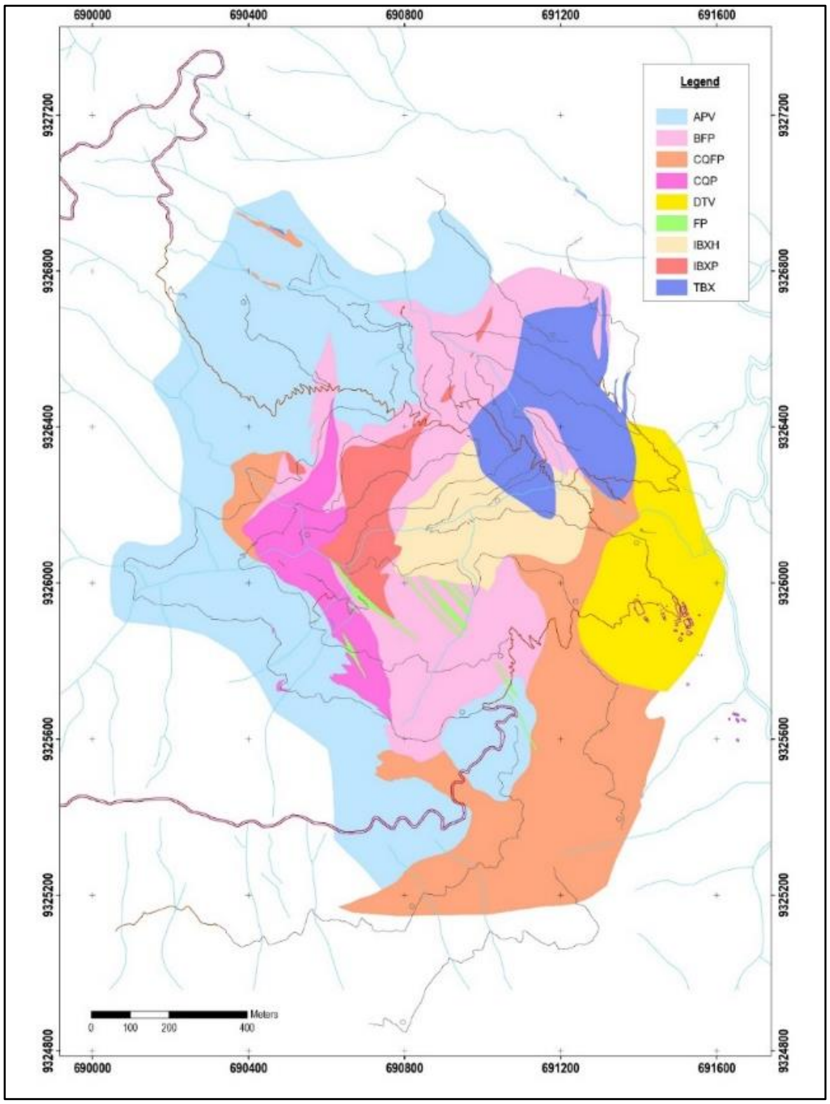
Mineralization occurs primarily as sulfide disseminations and in fractures, sulfide and quartz veins, faults, and breccias. Fracture density is the single most important factor influencing copper grades and alteration intensity, although breccias and faults can locally be important.

Copper mineralization was introduced as a series of events closely following the emplacement of each of the intrusive units and the polymictic breccia unit. Initially, copper mineralization comprised chalcopyrite and minor bornite, introduced following emplacement of each of the crowded quartz–feldspar porphyry and biotite–feldspar porphyry units, with the greatest amount introduced following emplacement of the biotite–feldspar porphyry unit. This event was followed by introduction of chalcocite, covellite, minor tennantite–digenite, following emplacement of the coarse quartz porphyry unit. The mineralizing process terminated with enargite, chalcocite and covellite, minor tennantite–digenite, introduced concurrently with emplacement of the polymictic breccia unit (Casselman et al., 2008).

Near surface, the deposit has been intensely weathered, resulting in the formation of a leached cap that contains less than 0.05% Cu, trace pyrite and tenorite, and variable concentrations of limonite, goethite, jarosite, and hematite. The leached cap varies significantly in thickness, ranging from less than a metre to as much as 120 m, averaging approximately 40 to 50 m in thickness. The water table is at, or near, surface.

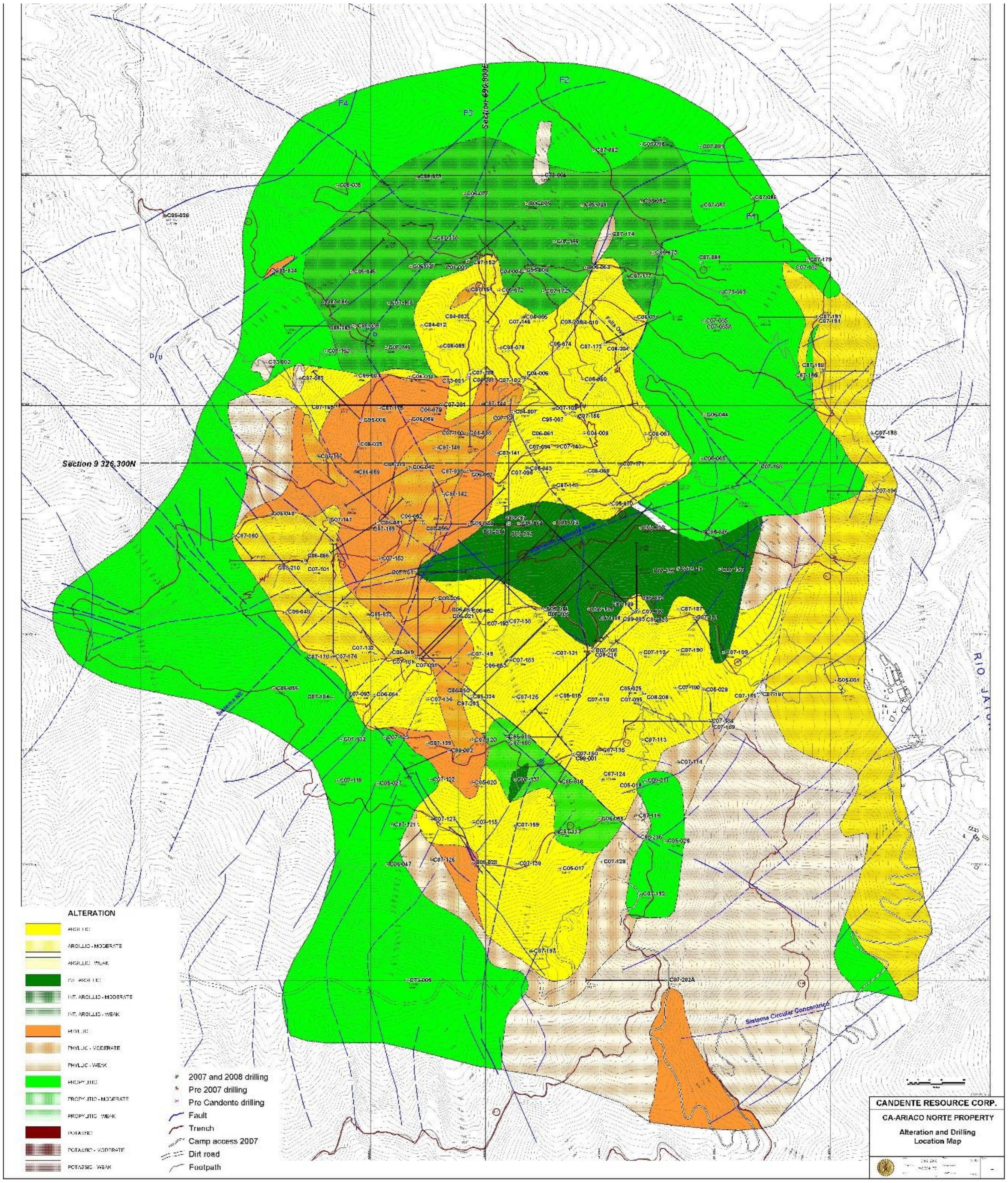
A geological map for the deposit is presented in Figure 7-5 and alteration map in Figure 7-6.

Figure 7-5: Geology Map Cañariaco Norte



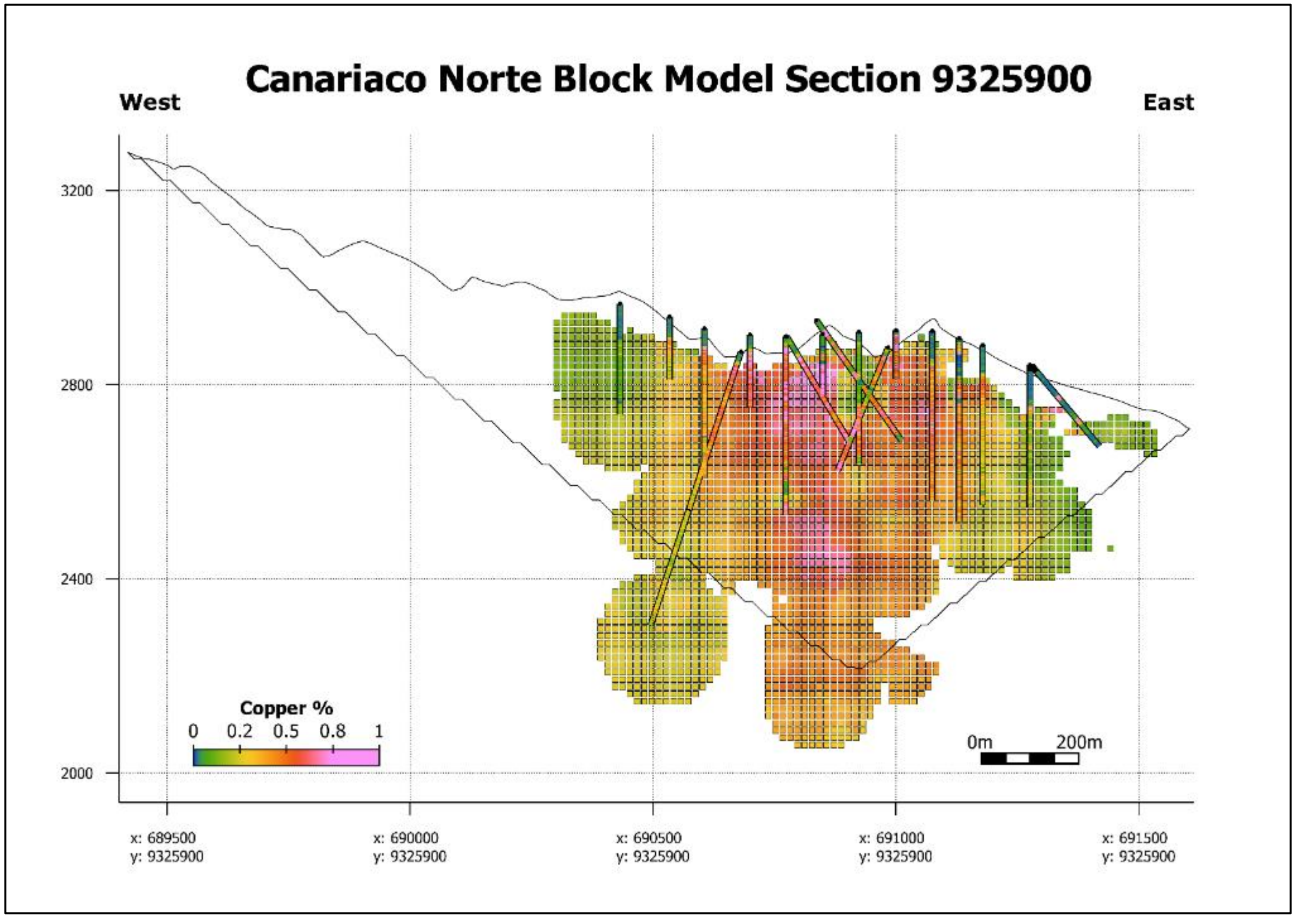
Source: Alta Copper, 2013.

Figure 7-6: Alteration Map Cañariaco Norte



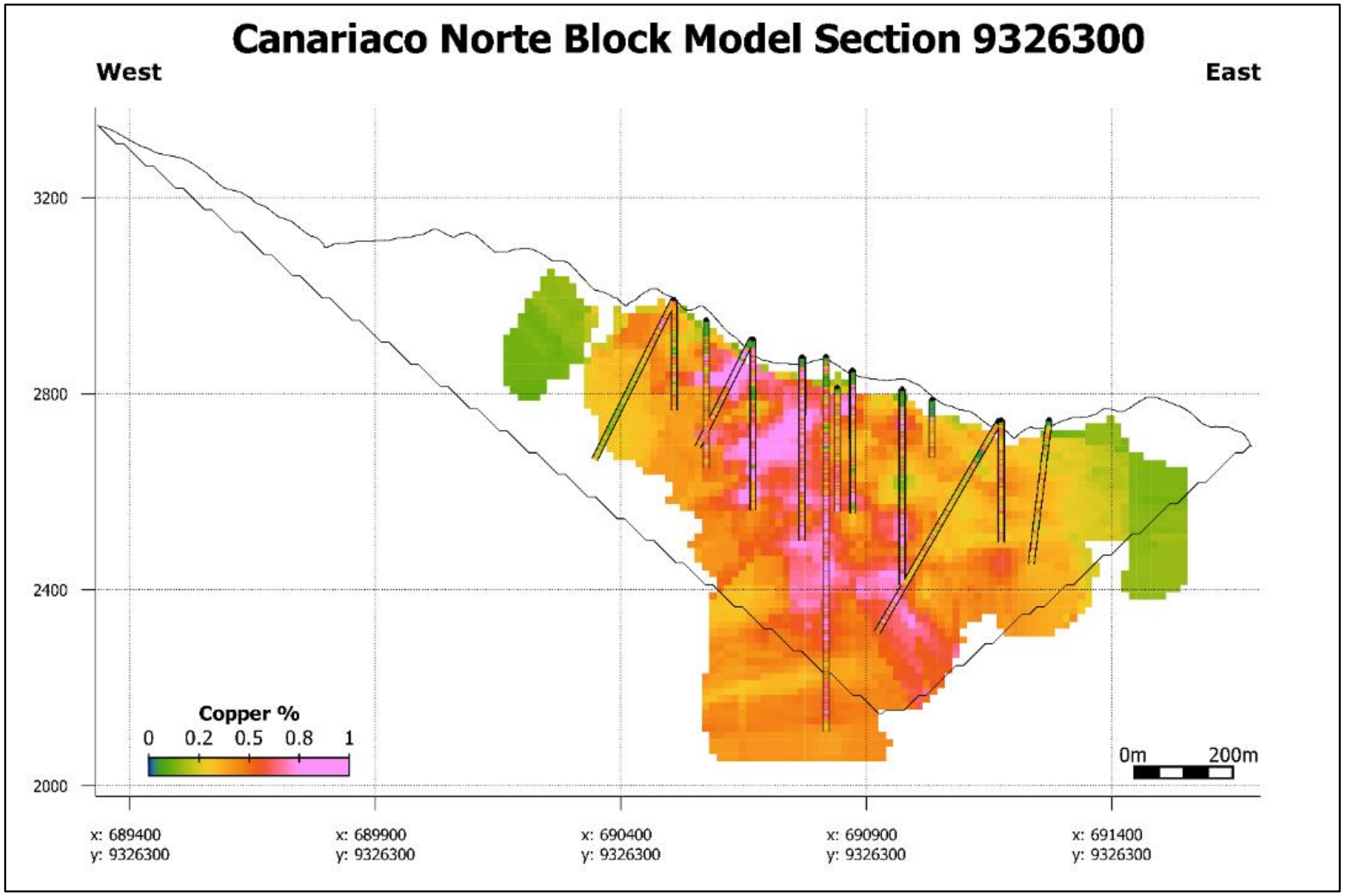
Source: Alta Copper, 2013.

Figure 7-7: West-East Cross-Section of Block Model with Cu Grades above 0.1% Cu in Drill Holes



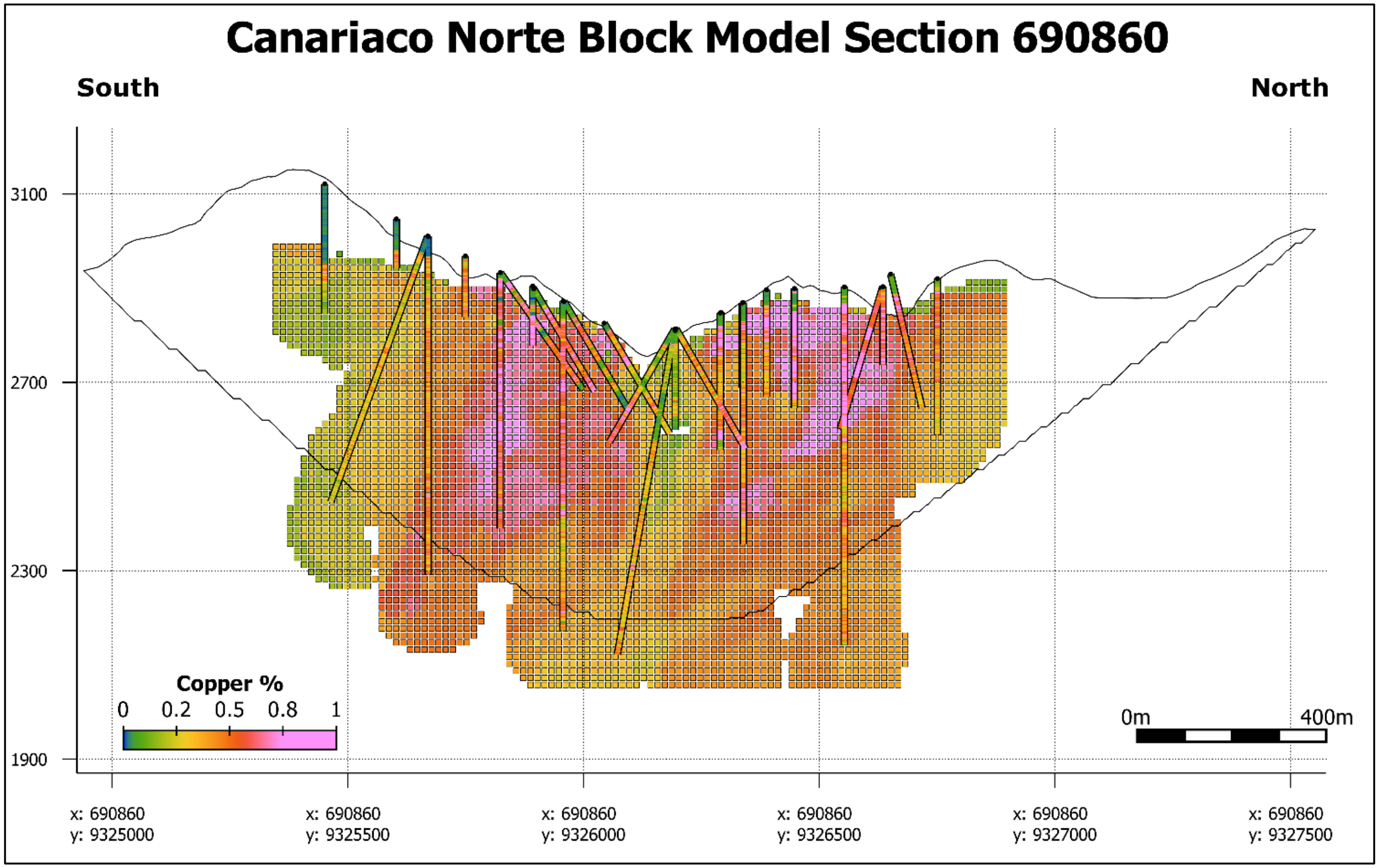
Source: Alta Copper, 2024.

Figure 7-8: West-East Cross-Section 9326300 of Block Model with Cu Grades above 0.1% Cu in Drill Holes



Source: Alta Copper, 2024.

Figure 7-9: South-North Cross-Section of Block Model with Cu Grades above 0.1% Cu in Drill Holes



Source: Alta Copper, 2024.

7.2.1.2 Alteration

Alteration forms distinct concentric zones with a central potassic alteration; central to intermediate, overlapping, and partly overprinting phyllic, argillic, and advanced argillic alteration; and fringing propylitic and minor silicic alteration. Alteration intensity is directly related to the intensity of fracturing in the hosting lithologies and brecciation in the polymictic breccia unit. Locally, alteration distribution and intensity are controlled by northeast and northwest-trending faults. The following sections on alteration are summarized from Casselman et al., (2008).

Phyllic and argillic alteration dominate in the upper 50 to 300 m in the southern half of the deposit and the upper 100 m to locally 150 m in the northern half of the deposit (excluding overburden and the leached cap). At depth, most of the northern and southern halves of the deposit under the layer of phyllic and argillic alteration is dominated by potassic alteration.

Alteration types include:

- Potassic alteration consists of variable proportions of secondary potassic feldspar and biotite, chlorite, and magnetite. Associated with deposition of chalcopyrite and minor bornite.
- Propylitic alteration comprises illite, chlorite, epidote, and smectite. Associated with deposition of chalcopyrite and minor bornite. Found mainly outside the deposit margin but can locally occur within the deposit. Propylitic alteration can extend for significant distances outside of the intrusive–breccia complex into the bordering volcanic rocks. It is often associated with silicic alteration near the margins of the deposit.
- Phyllic alteration includes sericite and quartz with variable concentrations of quartz veinlets and stockworks (0.5 to 1.5 cm thick). Associated with deposition of chalcocite, covellite, and minor tennantite and digenite. Commonly intermixed with argillic alteration. Affected parts of the crowded quartz–feldspar porphyry and biotite–feldspar porphyry units and all of the coarse quartz porphyry and polymictic breccia units. Late-stage phyllic alteration has extended along northeast- and northwest-trending faults outwards from the polymictic breccia unit and affected units distal from the polymictic breccia unit.
- Argillic alteration consists of kaolinite and illite. Associated with deposition of chalcocite, covellite, and minor tennantite and digenite. Commonly associated with phyllic alteration. Affected parts of the crowded quartz–feldspar porphyry and biotite–feldspar porphyry units and all of the coarse quartz porphyry and polymictic breccia units.
- Advanced argillic alteration comprises alunite, kaolinite, pyrophyllite, and dickite. Associated with deposition of enargite, chalcocite, and covellite, and minor tennantite and digenite. Commonly associated with late-stage phyllic alteration. Primarily associated with the polymictic breccia unit but has also variably affected the bordering coarse quartz porphyry and biotite–feldspar porphyry units. Has extended along northeast- and northwest-trending faults outwards from the polymictic breccia unit and affected units distal from the polymictic breccia unit.
- Silicic alteration comprises very fine-grained pervasive silica. Associated with chalcopyrite and minor bornite. Found mainly on the deposit margins but can locally occur within the deposit. Silicic alteration can extend outside of the intrusive–breccia complex into the bordering volcanic rocks.

7.2.1.3 Structure

The northwest–southeast- and the northeast–southwest-trending fault sets appear to control the emplacement of the intrusive phases at Cañariaco Norte, as well as the distribution of copper mineralization and alteration. In addition, the faults form structural corridors for late-stage polymictic breccias and associated alteration and mineralization. The smaller set of northwest–southeast-oriented faults control dyke emplacement (Casselman et al., 2008).

7.2.1.4 Mineralization

Although all of the lithologic units listed in Table 7-1 host mineralization, the average grades vary. In general, the highest copper grades (>0.5% Cu) occur in the biotite–feldspar porphyry and polymictic breccia units. Intermediate copper grades (0.35 to 0.5% Cu) are most common in the hydrothermal breccia and the coarse quartz porphyry, and the lowest grades (0.2 to 0.35% Cu) are found in the crowded quartz feldspar porphyry, tourmaline breccias, and volcanic host rocks. The feldspar porphyry dykes typically do not contain copper grades unless the dykes have been fractured, brecciated, or crackle-brecciated near the south margin of the hydrothermal breccia, where they have been weakly mineralized (average copper grade range of 0.1 to 0.2% Cu).

A QEMSCAN examination of core samples selected for metallurgical testwork in 2008, performed by SGS Metallurgical Laboratories in Santiago, Chile, indicated the predominant minerals and mineral associations present in the Cañariaco Norte samples are:

- Copper mineral species – dominantly chalcopyrite, covellite, chalcocite, and enargite, with minor bornite, tennantite, and digenite, and trace tetrahedrite. Copper sulfide minerals are mainly associated with pyrite and other sulfide gangue minerals;
- Non-metallic gangue minerals – quartz and micas/biotite/chlorite/clay group;
- Sulfide gangue mineral – pyrite, which is present as liberated pyrite or in association with other sulfide minerals.

Copper mineralization paragenesis (Casselman et al., 2008) comprised the following stages:

- Initially, chalcopyrite (bornite) + pyrite;
- Intermediate, chalcocite and covellite (tennantite, digenite) ± pyrite;
- Lastly, enargite, chalcocite, and covellite (tennantite, digenite) + pyrite.

Copper minerals and pyrite are disseminated, veined, and fracture-hosted, with copper grades directly related to the intensity of fracturing and alteration type and intensity. Higher grades are associated with potassic, phyllic, and argillic alteration and less commonly with propylitic and silicic alteration.

Casselman et al., (2008) interpreted that chalcopyrite (bornite) was deposited initially with the coarse quartz–feldspar porphyry and biotite–feldspar porphyry units, and with potassic and propylitic alteration. As the system evolved, chalcocite and covellite (tennantite, digenite) developed in association with the coarse quartz porphyry and with phyllic and argillic alteration and overprinted and replaced chalcopyrite (bornite) and potassic and propylitic alteration in the central and upper parts of the deposit. Enargite, chalcocite, and covellite (tennantite, digenite) associated with advanced argillic and phyllic alteration was the last mineralizing phase to form. These minerals are typically restricted in distribution to the polymictic breccia unit and the adjacent coarse quartz porphyry and biotite–feldspar porphyry units, where they occur as disseminations, clots, veins, and fracture coatings cutting earlier-formed chalcopyrite, chalcocite, covellite, potassic, propylitic, phyllic, and argillic alteration.

Chalcopyrite (bornite), chalcocite, covellite, and enargite (tennantite, digenite) are primarily hypogene in origin, with only limited development of supergene chalcocite and covellite (Mathur, 2008). In the opinion of Alta Copper staff, between the water table being at or near the present topographic surface and the active erosional environment, any supergene chalcocite and covellite that formed is being rapidly removed. Locally thin (<30 m) discontinuous layers of supergene chalcocite and covellite occur immediately under the leach cap.

Pyrite is common in all alteration types, averaging approximately 5% throughout the deposit as disseminations, veins, and fracture coatings. Magnetite is less common and primarily associated with chalcopyrite (bornite) in potassic alteration. Tourmaline is restricted to the matrix of the tourmaline breccia unit.

Gold and silver values are anomalous throughout the deposit; however, higher gold grades only occur with higher copper grades. Gold grades range, on average, between 0.04 g/t Au and 0.11 g/t Au and silver grades average 1.3 g/t Ag to 2.5 g/t Ag throughout the different rock types in the deposit (Bonson et al., 2008).

Molybdenum grades are low, averaging <40 ppm, and are slightly higher on the margins and at depth in the deposit (Bonson et al., 2008). Molybdenum grades are unlikely to be economically recoverable based on current information.

7.2.2 Cañariaco Sur

The Cañariaco Sur deposit is at an earlier stage of exploration than Cañariaco Norte.

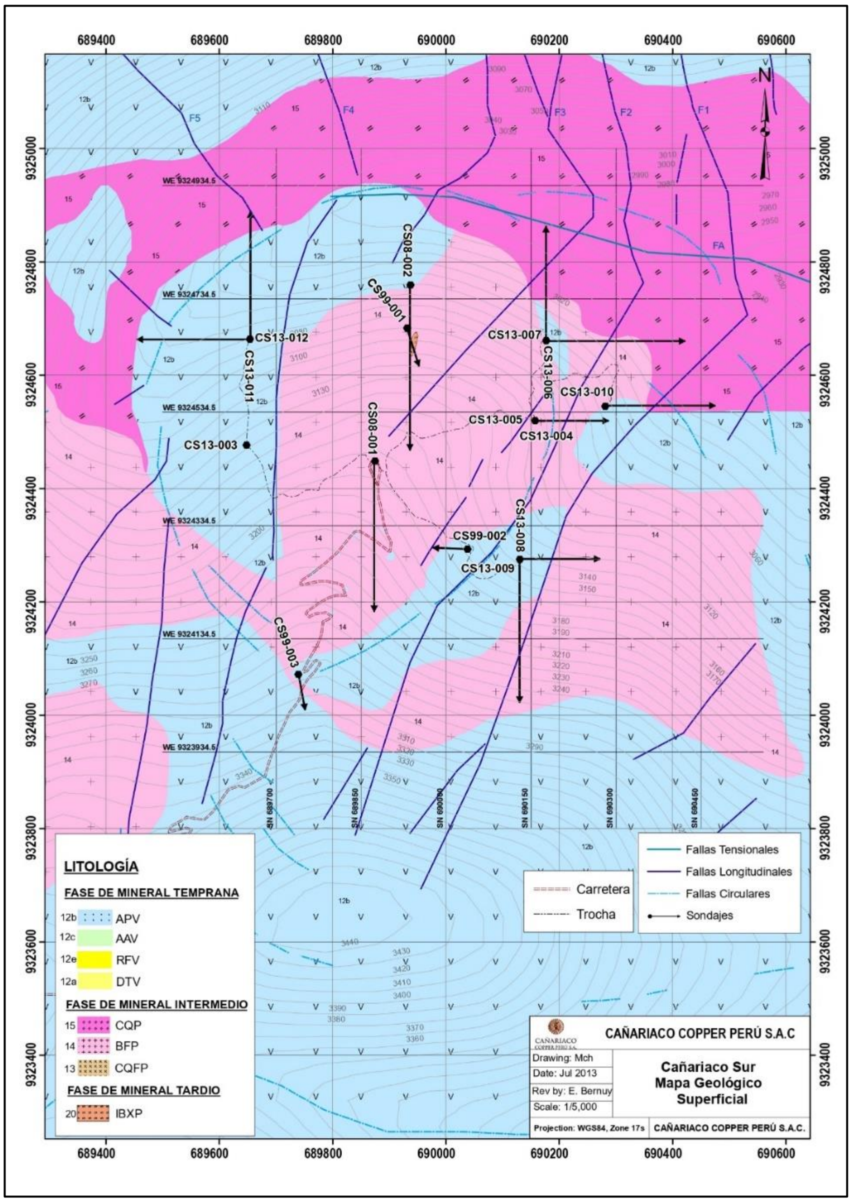
Fifteen holes have been drilled into Cañariaco Sur and although it is only partially drilled off it is understood to be a simpler porphyry system than Cañariaco Norte. Cañariaco Sur lacks the later stage brecciation event that introduced arsenic into Cañariaco Norte and does not have the other breccias that complicated the copper grade controls at Cañariaco Norte. Cañariaco Sur also has a higher gold content than Cañariaco Norte. A large, coincident magnetic high, IP/resistivity low, and chargeability high was identified coincident with the soil geochemical anomaly and area of potassic alteration. Alta Copper has interpreted the prospect as a porphyry-copper-gold centre dominated by potassic alteration with chalcopyrite–bornite–molybdenite mineralization and anomalous copper–gold–molybdenum grades.

The Cañariaco Sur deposit is centred approximately 2 km southwest of Cañariaco Norte.

The deposit has a potassic core surrounded by phyllic, silicic and propylitic alteration with a coincident copper–gold–molybdenum soil geochemical anomaly outcropping over 1,400 m x 900 m. The host rocks to the mineralized intrusions are the same as with Norte. Andesite to dacite tuffs and flows belonging to the Calipuy Group volcanic rocks were intruded by an intrusive complex which has similar but less intrusive events than those hosting Cañariaco Norte. Only four intrusive events have been identified which include the CQFP intruded by BFP and followed by CQP then minor amounts of the IBXP breccia. Cañariaco Sur also lacks the later stage brecciation event that introduced arsenic into Cañariaco Norte and does not have the other breccias that complicated the copper grade controls at Cañariaco Norte. Overall drilling to date indicates that the Cañariaco Sur deposit has a lower copper grade, a higher gold content and lower arsenic content than Cañariaco Norte. Mineralization at Cañariaco Sur is predominantly chalcopyrite and bornite with minor pyrite. The depth of oxidation averages 40 to 50 m and contains copper oxides in the form of copper wads. Meter-scale supergene enrichment is present in some drillholes.

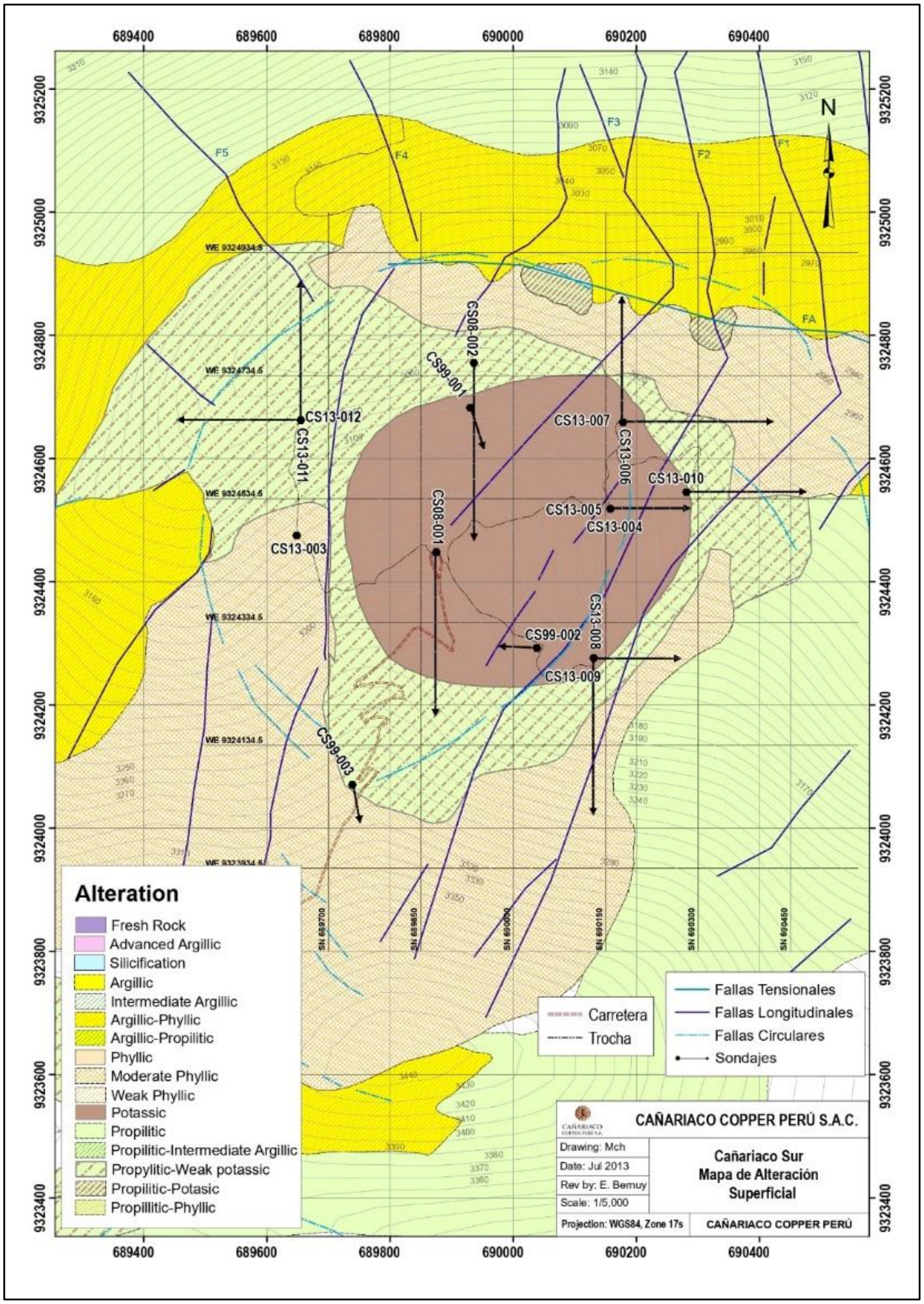
Figure 7-10 is a geology map of the Cañariaco Sur deposit. Figure 7-11 shows the alteration types in the deposit area. Figure 7-12 and Figure 7-13 are cross sections of the Block Model showing Cu grades above 0.1% in Drill Holes.

Figure 7-10: Cañariaco Sur



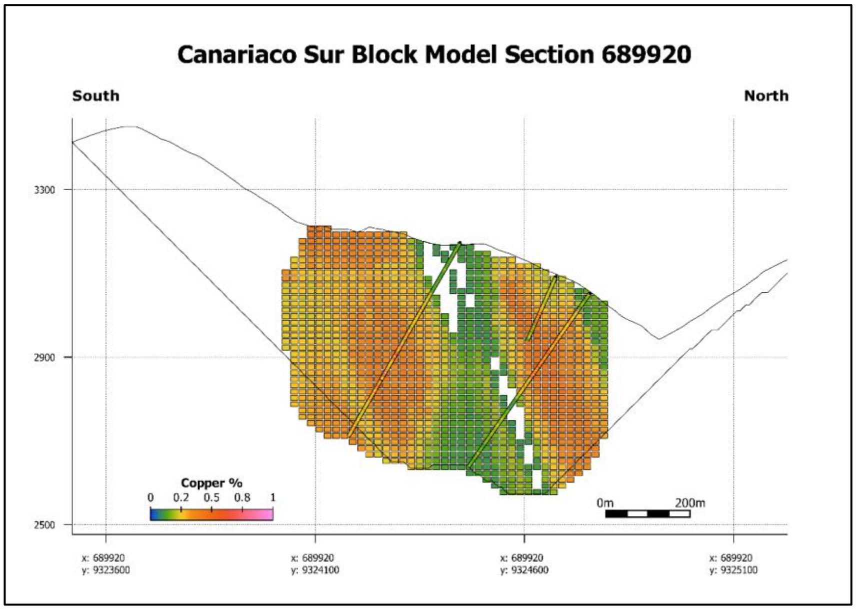
Source: Alta Copper 2024.

Figure 7-11: Cañariaco Sur



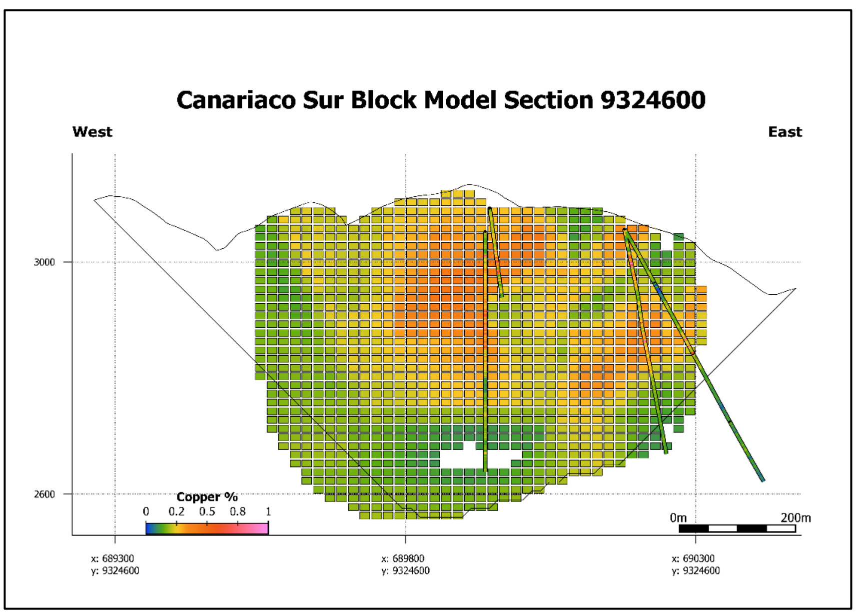
Source: Alta Copper 2024.

Figure 7-12: South-West Cross Section 689920 of the Block Model showing Cu above 0.1% in Drill Holes



Source: Alta Copper 2024.

Figure 7-13: North-South Cross Section 9324600 of the Block Model showing Cu Mineralization above 0.1% in Drill Holes



Source: Alta Copper 2024.

7.2.3 Quebrada Verde Prospect

The Quebrada Verde prospect is 3.2 km south of Cañariaco Norte and 1.5 km south of Cañariaco Sur (Figure 7-4). Strongly anomalous levels of copper and gold in soil samples cover an area of 0.7 km x 0.9 km. A large IP chargeability high centred on a resistivity anomaly and magnetic high, and covering an area of 0.8 x 2.0 km, occurs coincident with the soil geochemical anomalies. Alta Copper staff have interpreted the prospect as a porphyry copper–gold centre. The one historic drill hole did not test the geochemical/geophysical target.

Quebrada Verde host rocks known to date are predominated by a diorite porphyry (BFP) intruding Calipuy Group andesite volcanic rocks (Wilson, 1999). The BFP is exposed on surface in two areas, the largest extending over an area of 1 km x 750 m and a second over 400 m x 400 m approximately 200 m northeast of the larger exposure. In addition, an east–west striking, post-mineralization granodiorite dyke, with dimensions of approximately 1 km x 100 m, intruded the volcanic rocks and the larger BFP diorite porphyry stock.

7.3 Comments on Geological Setting and Mineralization

Knowledge of the deposit settings, lithologies, and structural and alteration controls on mineralization within the Cañariaco Norte and Cañariaco Sur deposits are sufficient to support Mineral Resource estimation. The mineralization style and setting of the Cañariaco Norte and Cañariaco Sur deposits are sufficiently well understood to support Mineral Resource estimation.

The geological knowledge of the area is also considered sufficiently acceptable to reliably inform conceptual mine planning for the Cañariaco Norte deposit. The Cañariaco Sur deposit was not used in the 2022 PEA however an Inferred Mineral Resource estimate has been completed for Cañariaco Sur and it has been used in this PEA.

Quebrada Verde, has not been drill tested. The lithologies, structural, and alteration controls on mineralization are currently insufficiently understood to support estimation of Mineral Resources.

8 DEPOSIT TYPES

8.1 Deposit Model

The Cañariaco Norte and Cañariaco Sur deposits are considered to be porphyry-copper systems. The following discussion of the typical nature of porphyry-copper deposits is sourced from Sillitoe (2010), Berger et al., (2008), and Sinclair (2006).

Porphyry-copper systems commonly define linear belts, some many hundreds of kilometres long, as well as occurring less commonly in apparent isolation. The systems are closely related to underlying composite plutons, at paleo-depths of 5 to 15 km, which represent the supply chambers for the magmas and fluids that formed the vertically elongate (>3 km) stocks or dyke swarms and associated mineralization.

Commonly, several discrete stocks are emplaced, resulting in either clusters or structurally controlled alignments of porphyry-copper systems. The rheology and composition of the host rocks may strongly influence the size, grade, and type of mineralization generated in porphyry-copper systems. Individual systems have life spans of circa 100,000 years to several million years, whereas deposit clusters or alignments, as well as entire belts, may remain active for 10 million years or longer.

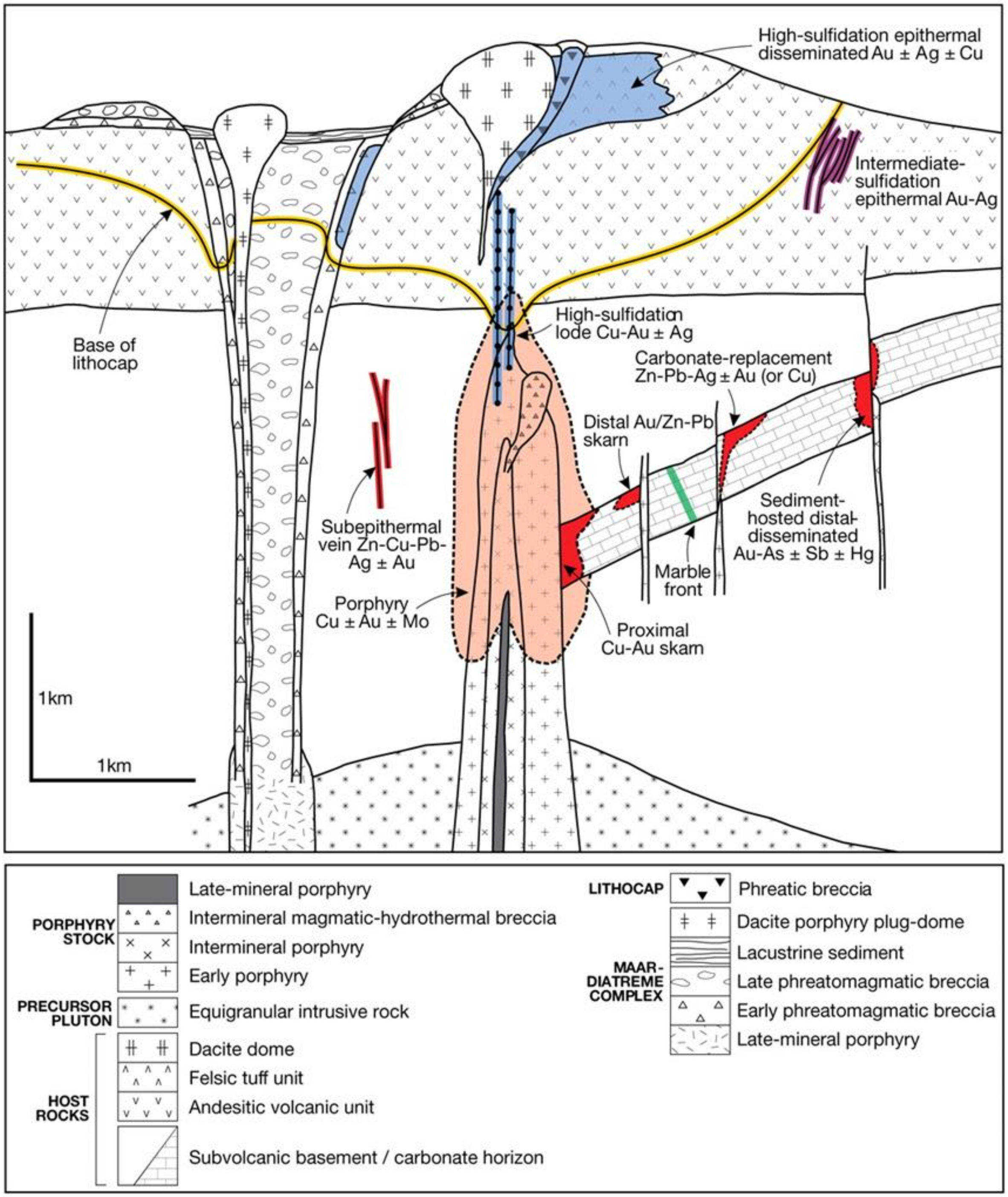
Deposits are typically semi-circular to elliptical in plan view. In cross-section, mineralized material-grade in a deposit typically has the shape of an inverted cone with the altered, but low-grade, interior of the cone referred to as the “barren” core. In some systems, the barren core may be a late-stage intrusion.

The alteration and mineralization in porphyry-copper systems are zoned outward from the stocks or dyke swarms, which typically comprise several generations of intermediate to felsic porphyry intrusions. Porphyry copper-gold-molybdenum deposits are centred on the intrusions, whereas carbonate wall rocks commonly host proximal copper-gold skarns and less commonly, distal base metal and gold skarn deposits. Beyond the skarn front, carbonate-replacement copper and/or base metal-gold deposits, and/or sediment-hosted (distal-disseminated) gold deposits can form. Peripheral mineralization is less conspicuous in non-carbonate wall rocks but may include base metal or gold-bearing veins and mantos. Data compiled by Singer et al., (2008) indicate that the median size of the longest axis of alteration surrounding a porphyry copper deposit is 4 to 5 km, while the median size area of alteration is 7 to 8 km^2.

High-sulfidation epithermal deposits may occur in lithocaps above (or over-printing within) porphyry-copper deposits, where massive sulfide lodes tend to develop in deeper feeder structures and precious metal-rich, disseminated deposits form within the uppermost 500 m.

Porphyry-copper mineralization occurs in a distinctive sequence of quartz-bearing veinlets as well as in disseminated forms in the altered rock between them. Magmatic-hydrothermal breccias may form during porphyry intrusion, with some breccias containing high-grade mineralization because of their intrinsic permeability. In contrast, most phreatomagmatic breccias, constituting maar–diatreme systems, are poorly mineralized at both the porphyry copper and lithocap levels, mainly because many such phreatomagmatic breccias formed late in the evolution of systems.

Figure 8-1: Schematic Section, Porphyry Copper Deposit



Source: Sillitoe, 2010.

Copper mineral assemblages are a function of the chemical composition of the fluid phase and the pressure and temperature conditions affecting the fluid. In primary, unoxidized or non-supergene-enriched mineralized material, the most common mineralized material –sulfide assemblage is chalcopyrite ± bornite, with pyrite and minor amounts of molybdenite. Some porphyry copper deposits also contain significant concentrations of hypogene chalcocite and covellite as in the Butte (Montana, USA) and Resolution (Nevada Arizona, USA) porphyry deposits. In supergene-enriched mineralized materials, a typical assemblage can comprise chalcocite + covellite ± bornite, whereas, in oxide mineralized materials, a typical assemblage could include malachite + azurite + cuprite + chrysocolla, with minor amounts of minerals such as carbonates, sulfates, phosphates, and silicates. Typically, the principal copper sulfides consist of millimeter-scale grains but may be as large as 1 to 2 cm in diameter and, rarely, pegmatitic (larger than 2 cm).

Alteration zones in porphyry-copper deposits are typically zoned and classified on the basis of mineral assemblages and consist of potassic, propylitic, phyllic and argillic zones. In silicate-rich rocks, the most common alteration minerals are K-feldspar, biotite, muscovite (sericite), albite, anhydrite, chlorite, calcite, epidote, and kaolinite. In silicate-rich rocks that have been altered to advanced argillic assemblages, the most common minerals are quartz, alunite, pyrophyllite, dickite, diaspore, and zunyite. In carbonate rocks, the most common minerals are garnet, pyroxene, epidote, quartz, actinolite, chlorite, biotite, calcite, dolomite, K-feldspar, and wollastonite. Other alteration minerals commonly found in porphyry-copper deposits are tourmaline, andalusite, and actinolite.

Porphyry-copper systems are initiated by injection of oxidized magma saturated with sulfur- and metal-rich, aqueous fluids from cupolas on the tops of the subjacent parental plutons. The sequence of alteration–mineralization events is principally a consequence of progressive rock and fluid cooling, from >700 to <250°C, caused by solidification of the underlying parental plutons and downward propagation of the lithostatic–hydrostatic transition. Once the plutonic magmas stagnate, the high-temperature, generally two-phase hyper-saline liquid and vapour responsible for the potassic alteration and contained mineralization at depth and early overlying propylitic and advanced argillic alteration, respectively, gives way, at <350°C, to a single-phase, low- to moderate-salinity liquid that causes the sericite–chlorite and sericitic alteration (phyllic)and associated mineralization. This same liquid also causes mineralization of the peripheral parts of systems, including the overlying lithocaps.

The progressive thermal decline of the systems combined with syn-mineral paleo-surface degradation results in the characteristic overprinting (telescoping) and partial to total reconstitution of older by younger alteration–mineralization types (argillic). Meteoric water is not required for formation of this alteration–mineralization sequence, although its late ingress is common.

8.2 Comments on Deposit Type

Cañariaco Norte and Cañariaco Sur are considered to be a porphyry systems based on the following:

- Multiple emplacements of successive intrusive phases and a variety of breccias are present;
- Copper-bearing igneous rocks are intrusive into host volcanic and sedimentary rocks;
- Mineralization is spatially, temporally, and genetically associated with the intrusive–breccia activity and hydrothermal alteration of the intrusive and breccia bodies;

- Large zones of veining and stockwork mineralization, together with minor disseminated and replacement mineralization, occur throughout large areas of intrusive–breccia and hydrothermally altered rock;
- Hydrothermal alteration is extensive and zoned, which is common to porphyry copper deposits. The alteration assemblages are consistent with the physico-chemical conditions of a porphyry environment;
- Mineralization is focused in well-developed quartz–sulfide stockworks; veins, crackle, and breccia zones are also present;
- The tenor of the copper and gold grades is typical of a porphyry deposit;
- The large tonnage is also typical.

Cañariaco Sur is considered to be an example of a porphyry system based on the following:

- Multiple emplacements of successive intrusive phases and a variety of breccias are present;
- Copper-bearing igneous rocks are intrusive into host volcanic and sedimentary rocks;
- Mineralization is spatially, temporally, and genetically associated with the intrusive–breccia activity and hydrothermal alteration of the intrusive and breccia bodies;
- Large zones of veining and stockwork mineralization, together with minor disseminated and replacement mineralization, occur throughout large areas of intrusive–breccia and hydrothermally altered rock;
- Hydrothermal alteration is extensive and zoned, which is common to porphyry copper–gold deposits. The alteration assemblages are consistent with the physicochemical conditions of a porphyry environment;
- Mineralization is focused on well-developed quartz–sulfide stockworks; veins, crackle, and breccia zones are also present;
- The tenor of the copper and gold grades is typical of a porphyry deposit;
- The large tonnage is also typical.

The QP is of the opinion that exploration programs that use a porphyry model are applicable to the Project area.

9 EXPLORATION

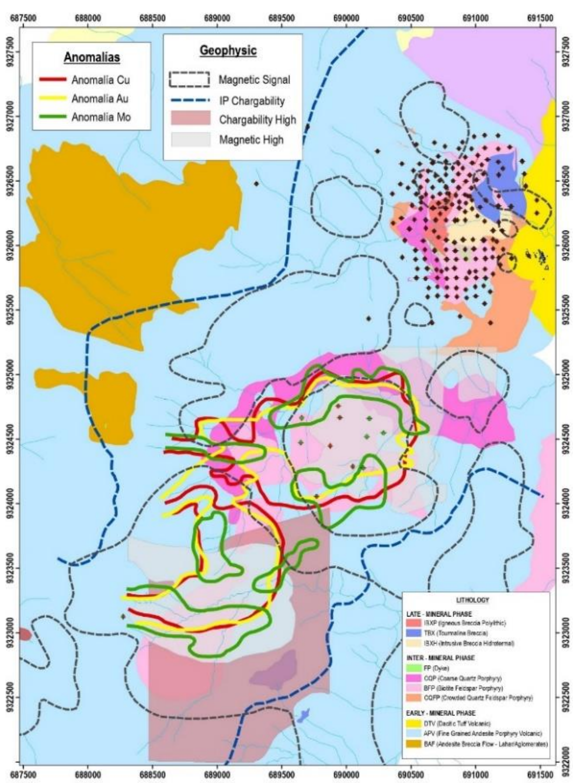
9.1 Introduction

Exploration activities such as geological mapping and geochemical sampling have been performed by Alta Copper and predecessor companies INGEMMET, Placer Dome, and Billiton. Contractors were used for activities such as geophysical surveys and core drilling.

Exploration activities on the Project have included regional and detailed mapping, stream sediment, grab, rock, and soil sampling, trenching and pitting, core drilling, ground geophysical surveys, mineralization characterization studies, and metallurgical testing of samples. Petrographic studies and density measurements on the different lithologies were completed.

A summary of the work programs completed to the Report effective date is provided in Table 6-1 and shown on Figure 9-1.

Figure 9-1: Composite Plan Geophysics, Geochemistry and Geology Cañariaco Project



Source: Alta Copper, 2022.

9.2 Grids and Surveys

GEMA was contracted in 2004 to establish the coordinates of four survey control points for site survey work. All survey data were based on these control points. The points were established by differential GPS set on the point for one hour and using the "1" order base station in Lima. The accuracy of these initial control points is not known. UTM coordinates were reported in PSAD 56.

In June 2006, GEMA was contracted to re-establish five high-precision survey control points to be used at the project as the base points for all surveying. Monuments were re-built using rebar and concrete to permanently secure their position. GEMA re-established the points with a differential GPS set on the point for three hours and using the "0" order station "Piura" as base. Accuracy was established at ±1 mm.

Reference point UTM data were reported in WGS-84, which is the main datum used for surveying in Peru. The calculated PSAD-56 UTM data was also reported.

In June 2006, the entire project was changed over from PSAD-56 (zone 17) to WGS-84 UTM (zone 17). All survey work on the project prior to the re-establishment of the control points and the datum switch was re-done. The project grid corresponds to the WGS-84 (zone 17) UTM grid.

Topographic data were based on aerial photograph coverage provided by Horizons South America S.A.C. Aerial Mapping Services.

9.3 Geological Mapping

Regional and detailed geological mapping was completed by INGEMMET, Placer Dome, and Alta Copper in a number of phases. Map scales varied from regional (1:25,000) to prospect scale (1:1,000). Map results were used to identify lithologies, areas of quartz veining, alteration, and silicification, and sulfide outcrops that warranted additional work. Air photos were interpreted to identify areas that required additional geological mapping and sampling. During the mapping program, structural measurements were collected from faults, veins, and fractures to provide additional structural detail for geological interpretations.

9.4 Geochemistry

Stream sediment, soil, and rock chip sampling was completed to evaluate the mineralization potential and generate targets for core drilling at Cañariaco Norte, Cañariaco Sur, and Quebrada Verde. Sampling was performed by INGEMMET, Placer Dome, Billiton, and Alta Copper.

Stream sediment sampling completed by INGEMMET at 100 m spacing along the drainages of Río Cañariaco during early exploration phases identified elevated copper and gold values.

Rock chip sampling outlined copper–gold–silver–molybdenum mineralization associated with quartz porphyry outcrops at Cañariaco Norte, Cañariaco Sur, and Quebrada Verde.

Stream sediment sampling provided the initial drill target areas for Cañariaco Norte. Soils with elevated levels of copper and gold, covering an area of over 0.9 x 1.4 km, were defined at Cañariaco Sur. Anomalous levels of copper and gold in BHorizon soils cover an area of approximately 0.7 x 0.9 km at Quebrada Verde (Figure 9-1). Sampling was carried out on lines usually spaced at 100 to 200 metres apart and at 25 to 50 m intervals depending on soil availability.

9.5 Geophysics

Geophysical surveys, comprising IP/resistivity and ground magnetics, were completed at Cañariaco Norte, Cañariaco Sur, and Quebrada Verde by INGEMMET, Billiton, and Alta Copper.

Results of the surveys prior to the Alta Copper work were considered inconclusive due to wide line spacings. Alta Copper completed 20-line-kms of combined IP/resistivity and ground magnetic surveys. The ground magnetic surveys at Cañariaco Norte outlined areas 0.7 km x 0.9 km of magnetite destruction and addition, thus delineating the various alteration zones and helping to confirm and/or identify fault structures. A coincident magnetic high, IP/resistivity low and chargeability high was identified at Cañariaco Sur. A large IP chargeability high centered on a resistivity anomaly and covering an area of 0.8 km x 2.0 km was identified at Quebrada Verde (refer to Figure 9-1).

9.6 Pits, Trenches, and Bulk Samples

INGEMMET excavated nine trenches and took a total of 23 rock chip samples. Placer Dome completed trenching (2,200 m) and pitting (80 test pits) at Cañariaco Norte and Cañariaco Sur.

9.7 Exploration Potential

Mineralization at Cañariaco Norte is still open at depth and in the central–western portion of the deposit.

Mineralization also remains open in all directions at the Cañariaco Sur deposit, and additional drilling is planned (see discussion in Section 26.3, and proposed drill collar locations shown in Figure 7-12 and Figure 7-13.

The Quebrada Verde prospect warrants drill testing.

9.8 Comments on Exploration

The exploration programs completed to date are appropriate to the style of the deposits and prospects within the Project. The petrographic research work supports the genetic and affinity interpretations. Exploration potential remains within the Cañariaco Sur deposit and Quebrada Verde prospect. Cañariaco Norte is still open at depth in at the central–western portion of the deposit.

10 DRILLING

10.1 Introduction

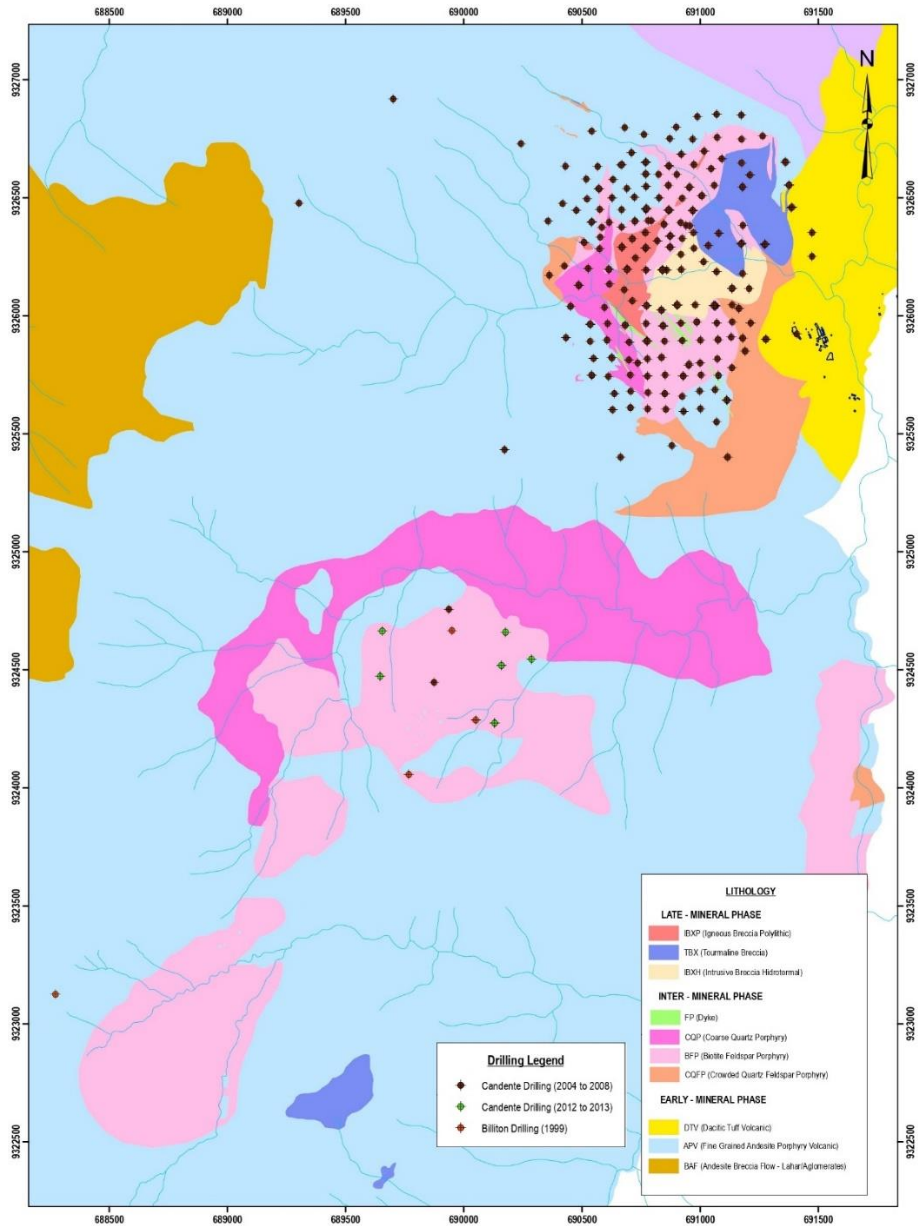
Drilling on the Project consists of 289 core holes (85,183.16 m), including geotechnical, metallurgical, and hydrogeological drilling.

Table 10-1 is a drill summary by operator and area. Drill hole locations for all drilling on the project are shown in Figure 10-1 this includes the Cañariaco Norte deposit as well as Cañariaco Sur and Quebrada Verde. In this section, all drilling completed prior to Alta Copper's involvement in the Project is termed "legacy" data.

Table 10-1: Drill Summary Table

Year	Project Operator	Deposit or Prospect	Core	
			Number of Holes	Metres
1971–1974	INGEMMET	Cañariaco Norte	5	1,500.00
1994–1997	Placer Dome	Cañariaco Norte	3	853.91
1999–2000	Billiton	Cañariaco Norte	3	555.70
1999-2000	Billiton	Cañariaco Sur	3	472.50
1999-2000	Billiton	Quebrada Verde	1	100.00
2004	Alta Copper	Cañariaco Norte	12	2,647.38
2005	Alta Copper	Cañariaco Norte	24	7,387.87
2006	Alta Copper	Cañariaco Norte	46	15,983.55
2007	Alta Copper	Cañariaco Norte	123	31,350.80
2008	Alta Copper	Cañariaco Norte Cañariaco Sur	41	13,771.75 1040.55
2013	Alta Copper	Cañariaco Norte - Met Cañariaco Norte - Geotech Cañariaco Sur	10 6 10	2,761.00 2,553.00 4,205.15
Total by Alta Copper			**289**	**85,183.16**

Figure 10-1: All Drilling to Date for the Cañariaco Project



Source: Alta Copper, 2013.

10.2 Drill Methods

The INGEMMET program cased all holes to 50 m, then used NXWL core size (60.7 mm core diameter) to 170 m, and BXWL (48.4 mm core diameter) thereafter. Both 10 ft (3 m) and 5 ft (1.5 m) core barrels were used.

No information is available on the core size for the Placer Dome drilling; however, the Billiton drill program used BQ diameter (36.5 mm).

The Alta Copper programs initially used NTW (56 mm) diameter core. During the 2005 drill program, ground conditions and depth required a reduction to a BTW core diameter (42 mm) in drill holes 05-014 at 302.35 m and 05-019 at 268 m). In 2006 and 2007, all drill holes less than 325 m in depth were drilled with NTW diameter core; holes deeper than 325 m needed to be reduced to BTW at around 250 m to 300 m depth, to allow the machine to drill to the planned depth. The 2008, 2012, 2013 drill program employed HQ (63.5 mm diameter) or NQ (47.6 mm diameter) core sizes.

For the Alta Copper programs, core was transferred to wooden core boxes. Rock quality designation (RQD) measurements were performed at the drill site. The boxes were then closed and transported by porters hired by Alta Copper to the project core processing facility, where the core was photographed and logged for geological and geotechnical information.

No information is available as to the method of core transportation used during the earlier drill programs by INGEMMET, Placer Dome, or Billiton.

10.3 Logging Procedures

During detailed core mark-up at the logging facility, Alta Copper geologists perform first-pass geological logging of the core. This initial log identified lithological boundaries, major structures, leaching depth, and broad mineralization intervals. In tandem with the logging, the geologist noted where major sample breaks should be placed at lithological boundaries.

A second phase of detailed core logging was carried out immediately after cutting on the preserved split core at Alta Copper's core storage facility in Chiclayo, where each core was logged for lithology, structure, veining, alteration, and mineralization. All logged descriptions were indexed to 2 m sample intervals, so that the mineralization, veining, structure, lithology, and alteration affecting any sample were known.

Core from all other legacy drill campaigns was re-logged by Alta Copper.

10.4 Recovery

Drill core recovery data from the legacy drill campaigns are not available. Alta Copper staff recorded core lengths and calculated core recoveries and RQD at the drill sites.

The QP reviewed the core recovery data from drill holes C07-164 through to C08-244 and found a trend of decreasing copper grade with decreasing core recovery. The copper grades decrease from 0.3 to 0.2% at a core recovery of 80%; however, only 2% of the data are affected. There is a small risk of a limited number of assays having a negatively biased copper grade because of low core recoveries, in the opinion of the QP.

10.5 Collar Survey

Alta Copper drill collar locations were picked up by a surveyor using a total station instrument. All legacy drill collars were picked up by survey in 2006 and tied into the Project grid.

10.6 Downhole Surveys

For the 2004 drill program, down-hole surveys were undertaken using a Pajari analogue survey tool with a timed locking compass and inclinometer. Readings were taken at the drill rig. Any spurious readings were discarded, and a second survey was done at the same depth. The 2005 program used the same instrumentation and procedures as in 2004. However, a number of the 2005 program drill holes do not have down-hole survey data due to an instrument malfunction.

All the drill holes from the 2006 and 2007 drill programs were down-hole surveyed with a Sperry Sun single-shot down-hole survey tool supplied by the drilling company.

The 2008, 2012 and 2013 program drill holes were down-hole surveyed for dip and azimuth at 50 m intervals with a Reflex EZ-Shot digital down-hole survey tool.

Selected drill cores from the 2006 and 2008 drill programs were oriented for geotechnical purposes using an A.C.E. tool.

The samples taken in the field for metallurgical studies in 2012 and 2013 went directly to Canada for the respective studies.

During the 2013 program six samples were taken from Cañariaco Norte by Knight Piésold Ltd (Knight Piésold) for geotechnical purposes. Core orientation was performed using a Reflex ACT II RD core orientation system.

10.7 Sample Length / True Thickness

Approximately 85% of the samples are two meters in length. A sample length of two meters is appropriate to reflect the variation in grade within the large zones (100’s of meters in scale) of veining and stockwork mineralization, found in porphyry copper-gold deposits.

The term “true thickness” is not generally applicable to porphyry-style deposits as the entire rock mass is potentially mineralized and there is often no preferred orientation to the mineralization. In areas that display porphyry-style mineralization, in general, most drill holes intersect mineralized zones at an angle, and the drill hole intercept widths reported for those drill holes are typically greater than the true widths of the mineralisation at the drill intercept point.

10.7.1 Cañariaco Norte

Drill spacing varies in the Cañariaco Norte deposit averaging less than 75 m x 75 m for most of the deposit and up to 100 m x 100 m or 100 m x 200 m spacing on the edges of the deposit. Of the 273 holes drilled to date at Cañariaco Norte, 188 were drilled vertically. Dips for the other 85 holes varied from -45 to –90 degrees but were drilled predominately at -60 to -70 degrees. The 273 holes averaged 291 m in depth and varied from 90 m to 667 m with 102 holes drilled between 200 m and 300m; 68 drilled between 300 m and 400 m and 51 holes drilled from 100 m to 200 m in depth. The selected orientation in 2004 and 2005 was based on early surface structural and geological mapping data, however in 2008 and 2012–2013 selected orientation was based on deposit modelling.

10.7.2 Cañariaco Sur

Drill spacing varies in the Cañariaco Sur deposit from 100 m x 100 m to 220 m x 220 m for the 15 holes to date. Most of the holes (13) were drilled at dips of -60 ° to -65°; 2 were drilled vertically, and 1 was at –55 °. Azimuths varied from 90–36 °; 5 were at 163 ° to 180 °; 4 at 9°; 4 at 360 °; 2 at 27 °. Depths of the holes averaged 364 m in depth and varied from 150 m to 535 m.

10.8 Significant Results and Interpretation

Please refer to the representative drill sections in Section 7 which show the relationship between mineralization and drilling.

10.9 Comments on Drilling

The quantity and quality of the lithological, geotechnical, collar, and down-hole survey data collected in the exploration and infill drill programs are sufficient to support Mineral Resource estimation, as follows:

- Core logging meets industry standards for copper, gold, and silver exploration.
- Collar surveys have been performed using industry-standard instrumentation.
- Down-hole surveys were performed using industry-standard instrumentation.
- Recovery data from core drill programs are acceptable.
- Geotechnical logging of drill core meets industry standards for planned open pit operations.
- Depending on the dip of the drill hole and the dip of the mineralization, drill intercept widths are typically greater than true widths.
- Drill orientations are generally appropriate for the mineralization style and have been drilled at orientations that are optimal for the orientation of mineralization for the bulk of the deposit area.
- Drill orientations are shown in the example cross-sections in Section 7 and can be seen to appropriately test the mineralization. The sections display typical drill hole orientations for the deposits, show summary assay values using colour ranges for assay intervals that include areas of non-mineralized and very low-grade mineralization, and outline areas where higher-grade intercepts can be identified within lower-grade sections. The sections confirm that sampling is representative of the copper, gold, and silver grades in the deposits, reflecting areas of higher and lower grades.
- No factors were identified with the data collection from the drill programs that could affect Mineral Resource estimation.

11 SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1 Sampling Methods

11.1.1 Geochemical Sampling

There is no information available on the sampling methods for the INGEMMET, Placer Dome, or Billiton geochemical sampling programs. Geochemical samples taken by Alta Copper were typically 2–5 kg.

11.1.2 Pit and Trench Sampling

There is no information available on the sampling methods for the INGEMMET and Placer Dome trench sampling programs.

11.1.3 Core Sampling

Drill core generated by INGEMMET and Placer Dome was halved; there is no information as to the typical sample intervals. The Billiton drill core was halved and sampled on 2 m intervals.

Alta Copper drill core was halved using a circular rock saw. Samples were 2 m in length unless a geological contact was present within the sample interval. In those instances, the sample interval was terminated at the contact. The subsequent sample interval terminated at the next metre depth mark that allowed a 1.50 m minimum sample length.

11.2 Density Determinations

A total of 9,424 bulk density readings were taken by Alta Copper personnel during core logging using weight in air and weight in water methods. Samples were taken every 10 to 20 m to include all rock and alteration types. Data were recorded for drill hole ID, depth, rock type, alteration, weight in air and weight submerged in water.

As a part of the update to the 2008 mineral resource estimate, 550 bulk density determinations were collected from drill core intervals. The determinations were performed using a wax-coated immersion technique (specialty assay procedure OA-GRA08a) at the ALS Chemex laboratory in Lima. AMEC assigned constant specific gravity values to each lithology (Table 11-1).

Table 11-1: Bulk Density Values used in the Mineral Resource Estimate

Rock Code	Bulk Density	
	Oxide (g/cm³)	Sulfide (g/cm³)
VC	2.13	2.51
CQFP	2.33	2.50
BFP	2.19	2.48
CQP	2.17	2.50

Rock Code	Bulk Density	
	Oxide (g/cm³)	Sulfide (g/cm³)
FP	2.03	2.56
IBXH	2.17	2.50
TBX	2.17	2.45
IBXP	2.17	2.50

11.3 Sample Security and Storage

11.3.1 Sample Security

Sample security at the Cañariaco Project during the Alta Copper drilling programs relied upon the remote nature of the site and the fact that the samples were always attended or locked at the sample dispatch facility. Sample collection and transportation were always undertaken by Alta Copper or laboratory personnel using Alta Copper vehicles.

Drill samples were picked up at site by a freight firm using a dedicated vehicle and transported to the sample preparation facility. Pulps were transported by laboratory personnel to the appropriate analytical facility.

Chain-of-custody procedures consisted of filling out sample submittal forms that were sent to the laboratory with sample shipments to ensure that all samples were received by the laboratory.

11.3.2 Sample Storage

Prior to the construction of an access road into the Alta Copper camp, all core was stored on racks within secure storage facilities. In October 2007, all existing core was moved to a secure core storage facility in Chiclayo, close to Alta Copper's regional offices, and thereafter core boxes were transported directly to the new core storage facility for detailed logging and permanent storage.

11.4 Analytical and Test Laboratories

Several primary assay laboratories were used for routine analyses over the Project history.

INGEMMET used the independent Plenge Laboratory in Lima and the non-independent INGEMMET internal laboratory, also located in Lima. No information is available as to accreditation of the laboratories at the time sampling was performed.

Placer Dome used the independent SGS-XRAL (SGS) laboratory in Lima. No information is available as to accreditation of the laboratory at the time sampling was performed.

For the Billiton sampling, sample preparation was undertaken by ALS Chemex in Trujillo, and primary analysis by ALS Chemex in Lima. The umpire laboratory was SGS. Both laboratories were independent of Billiton. No information is available as to accreditation of the laboratories at the time sampling was performed.

Activation-Skyline Laboratories (Actlabs) in Lima, Peru performed all of the sample preparation and the majority of the analyses for the Alta Copper programs. Actlabs is independent of Alta Copper and held ISO:9000 accreditation for the

Peruvian laboratory at the time the analyses were conducted. Inductively-coupled plasma (ICP) analyses were performed by the Ancaster, Canada, Actlabs laboratory, which had Standards Council of Canada (SCC) accreditation for International Standards Organization (ISO)17025.

Some analyses for the re-analysis of pre-2008 core samples for gold and ICP were undertaken by ALS Chemex in Lima. ALS Chemex also assayed 2012, and 2013 drilling campaign samples. ALS Chemex is independent of Alta Copper and held ISO:9000 accreditation for the Peruvian laboratory at the time the work was conducted.

ACME Laboratories (ACME), Lima was used as a check laboratory for pulp analyses. ACME is independent of Alta Copper and held ISO:9000 accreditation for the Peruvian laboratory at the time the work was performed.

11.5 Sample Preparation and Analysis

11.5.1 Legacy

Very limited information is available on the sample preparation and analytical methods used by INGEMMET or Placer Dome. INGEMMET samples were analyzed for copper and molybdenum, and more rarely gold and silver, using a colorimetric analytical method.

Billiton samples were 200 g splits of a 1 kg, 200 mesh homogenized sample. A split from each sample pulp was assayed for gold (fire assay with atomic absorption finish, 10 pbb detection limit) and copper, lead, zinc, molybdenum and arsenic (multi-acid, total digest), with an atomic absorption (AA) finish for each element. SGS completed check assays on a split of one in 20 pulps using the same analytical procedures as the initial analysis performed by ALS Chemex.

The samples taken during 2012 and 2013 for metallurgical studies were not analyzed.

11.5.2 Alta Copper

11.5.2.1 Sample Preparation

Sample preparation undertaken on the Alta Copper samples comprised drying, then crushing using a jaw crusher to >70% passing 10 mesh. The sample was thoroughly blended using a riffle splitter. A sub-split was taken, which was pulverized to >95% passing 150 mesh, and this pulp was submitted for analysis.

11.5.2.2 Copper

Each sample was subject to total copper and sequential copper leaching analysis which returned results for acid-soluble, cyanide-soluble and residual copper grades. Total copper analysis was performed using a three-acid digest and AA finish (laboratory method ME-3 or three-acid digestion).

The sequential leach analysis consisted of:

- An initial leach step, where samples were dissolved in sulfuric acid, and the copper grade determined by AA to give the acid-soluble copper value;

- A secondary leach step, where samples were dissolved in sodium cyanide and the copper grade determined by AA to give the cyanide-soluble copper value; and
- A third leach step, where the samples were dissolved using a three-acid digest and the copper grade determined by AA finish to give the residual copper value.
- For any given sample, copper grades were obtained by adding the three parts of the sequential copper analysis. Results were compared to the copper grade reported in the total copper analysis, and if the sum of the sequential leach grades had a >0.03% difference to the total copper grade, the analysis was repeated for both parts of the process.
- No sequential copper was performed on any sample from the 2012–2013 drilling campaign.

11.5.2.3 Gold

The sample for gold analysis was taken from the remaining pulps after copper analysis. The ±250 g pulps were homogenized, and a 30-g split was weighed out for fire assay fusion, cupelled to obtain a bead, and digested with aqua regia, followed by an AA finish, with a detection limit of 5 ppb Au (Actlabs code EF1).

Only a portion of the pre-2008 drilling originally had gold assays. These were analysed using a fire assay (FA) methodology with an atomic absorption finish (FA-AA) on a 30 g sample. The remainder of samples were not systematically analysed for gold until a large analytical in-fill campaign was conducted in March 2008. Due to the large volume of samples, analyses were split between Actlabs and ALS Chemex. The FA-AA method on a 30 g sample was used by both laboratories.

For the 2008 drill campaign, the FA-AA method on a 30 g sample was used, with all analyses performed by Actlabs. For the 2012–2013 drill campaign, the FA-AA method on a 30 g sample was used, with all analyses performed by ALS Chemex.

11.5.2.4 Multi-element

Actlabs used a 36-element inductively-coupled plasma optical emission spectrometry (ICP–OES) method following aqua regia digestion (laboratory code 1E3). ALS Chemex performed a 33-element ICP atomic emission spectroscopy (AES) method after four-acid digestion (laboratory code ME-ICP61). The samples from the 2012–2013 drilling campaign were analyzed by ALS Chemex using ICP-AES (code ME-ICP61).

11.6 Quality Assurance and Quality Control

11.6.1 Legacy

There is no information on any quality assurance/quality control (QA/QC) programs for INGEMMET and Placer Dome.

Billiton used blanks (crushed quartz every 20 samples), standard reference materials (standards) at an insertion rate of one in every 20 samples, and check assays with an insertion rate of one in every 20 samples). In Billiton's protocol, a blank sample was not to be submitted adjacent to standard but could not be any more than 10 samples away from a standard. Chain-of-custody and sample preparation protocols were also part of Billiton's QA/QC program.

11.6.2 Alta Copper

11.6.2.1 Duplicates

No field duplicates were used in the 2004 drilling program but were included from drill hole C05-013 of the 2005 drill program. Field duplicates typically comprised quarter drill core. Coarse reject duplicates were run on drill holes C04-007 and C05-023. Pulp duplicates were taken at Actlabs every 10^{th} sample.

Duplicate samples were also systematically placed during the 2012–2013 drilling program and analyzed at ALS Chemex.

Alta Copper consistently sent 5% of pulps prepared and analyzed by Actlabs to independent laboratories for check assays on total copper analyses. Five percent of the samples from drill holes C04-001 to C06-082 were sent to ACME for analysis by four-acid digestion of a 0.25 g pulp split. For drill holes C07-083 to C07-202, 5% of the samples were sent to SGS Laboratories, Lima (SGS), for analysis using the same analytical procedures as the original assay program (three-acid digestion on a 0.25 g split). All independent checks included standards, blanks, and duplicates. Pulp duplicates were also made during all independent laboratory checks at ACME and SGS.

During 2010, Alta Copper sent a suite of 530 pulp reject samples to SGS for check analyses on silver ICP analyses. The samples were analyzed by four-acid digest on a 0.3 g pulp split (method ICP40B). The samples were randomly selected from drill holes throughout the area drilled by Alta Copper. All independent checks included standards, blanks, and duplicates.

11.6.2.2 Blanks

Field blank material was obtained from a barren outcrop of volcanic host rock situated beyond the alteration halo at Cañariaco Norte. Blanks were submitted at a frequency of one in 30 samples, so that each laboratory batch of 80 always had at least two blanks. Field blank fail limits were set at 0.05% copper or five times the detection limit of 0.01%.

The 2012–2013 drilling program also had blank samples in the sample stream analyzed at ALS Chemex.

11.6.2.3 Standard Reference Materials

Alta Copper used six commercially available standards, purchased from CDN Laboratories in Vancouver, Canada. Standards were submitted at a frequency of one in 30 samples, so that each laboratory batch of 80 always had at least two standards. An additional two standards were purchased from Geostats Pty. in Australia specifically for use in the ICP check assays conducted during 2010. The selected standards had low-grade silver contents that were close to the average silver grade of the Cañariaco Norte deposit.

During 2006, SGS was retained to prepare two standards from unweathered outcrops from the Cañariaco Project area for sequential leach analysis.

Where the copper value of the standard was outside the acceptable value specified for the standard by CDN Laboratories (two standard deviations from the mean), they failed. Re-analysis of failed copper standards was not carried out at time of drilling. This was mainly due to the fact that most batches contained two, or sometimes three, standards. If one standard failed but the other did not, the batch was considered to be acceptable, particularly if the other QA/QC samples (blanks and duplicates) did not fail.

During the 2012–2013 drilling program, standards purchased from Australia with OREAS codes were systematically placed in the sample stream that was analyzed at ALS Chemex.

11.6.2.4 Re-sampling

Alta Copper conducted an extensive re-sampling program of historical pre-2004 drill hole data. Nine out of the eleven historical holes were re-sampled. Only drill holes C73-001 and C73005 were not re-sampled. Drill hole C73-001 was twinned by C04-001, and C73-005 did not have significant copper grades.

Sections of the old core were quartered and sent to Actlabs for processing. Given the state of the legacy drill core after several moves, composite samples of up to 20 m were made within mineralization types and between fixed blocks to ensure proper metreage measurements.

A comparison between the original and the Alta Copper assays shows an acceptable correlation between the datasets. Based on this, all of the historical data were added to the final database, and the Alta Copper assays from the re-sampling program were excluded because of the composite lengths.

11.6.2.5 Twin Holes

Alta Copper drilled three pairs of twinned holes to verify grade uniformity at short distances. Two of the twin sets (holes C04-007 with C07-104 and C04-023 with C07-106) show similar average grades over the same depth intervals. The third twin set (C04-005 with C07-146) shows significantly higher average grades in the original drillhole from 2004 than in its twin drilled in 2007. This may indicate that a vertically oriented mineralized vein set was intercepted in one drill hole and not the other.

11.7 Databases

All field data were recorded in field books, log books, sample sheets, logging forms or shipping forms. Various phases of record keeping were repeated in the subsequent step to confirm recorded values or numbers.

All field data were entered into Excel tables either in the Cañariaco camp or at the Alta Copper Lima office. Errors in data entry picked up during the verification stage were confirmed and corrected from filed data.

Data from third parties such as laboratories or survey contractors were generally supplied in digital and printed form. These records were printed out and kept in binders for reference during data verification.

11.8 Comment on Sample Preparation, Analyses and Security

The quality of the copper, gold, and silver analytical data is sufficiently reliable to support Mineral Resource estimation. Sample preparation, analysis, and security are performed in accordance with exploration best practices and industry standards, as follows:

- Geochemical sampling covered sufficient area, was adequately spaced to generate first-order geochemical anomalies, and thus was representative of first-pass exploration sampling.
- Sample preparation for samples that support Mineral Resource estimation has followed a similar procedure since 2004. The preparation procedure is in line with industry-standard methods for copper–gold–silver deposits.
- Core from drill programs was analysed by independent laboratories using industry standard methods for copper, gold, and silver analysis.
- Limited information is available on the QA/QC employed for the earlier drill programs completed prior to Alta Copper programs; however, twin drill holes confirm the grades and lithologies, and the core from the drill programs has been re-assayed, so that the data can be accepted for use in estimation.
- Typically, Alta Copper drill programs included insertion of blank, duplicate, and standard samples. The QA/QC program results do not indicate any problems with the analytical programs. Therefore, the copper, gold, and silver analyses from the core drilling are suitable for inclusion in Mineral Resource estimation.
- The collected data were subject to validation by built-in program triggers that automatically checked data on upload to the database.
- Verification is performed on all digitally collected data on upload to the main database, and includes checks on surveys, collar coordinates, lithology data, and assay data. The checks are appropriate and consistent with industry standards
- Sample security has relied upon the fact that the samples were always attended or locked in the on-site sample preparation facility.
- Chain-of-custody procedures consist of filling out sample submittal forms that are sent to the laboratory with sample shipments ensure that all samples are received by the laboratory.
- Current sample storage procedures and storage areas are consistent with industry standards.

12 DATA VERIFICATION

A number of data verification programs and audits have been performed over the project history, primarily in support of technical reports on the Project.

12.1 Internal Verification

The data verification programs carried out by Alta Copper were discussed in Section 11.

12.2 External Verification

12.2.1 Currie, 2004

During a site visit to support the completion of a technical report, Currie (2004) traversed the entire Cañariaco Norte zone, examining numerous outcrops. Four samples were taken from outcrop material and trenches. Analyses indicated that the assays from the outcrops were consistent with the grades seen in drill holes.

12.2.2 MineFill, 2007

MineFill performed detailed data verification for all available data from 1973 to 2006; only verified assay information was used in the estimation of Mineral Resources. Assay values were deemed verified when the original signed assay certificate or photocopy was present, and the database reflected the assay certificate values accordingly. The sample numbers and assay values on the certificates were called out by an individual, as another individual located the corresponding sample numbers within the database and verified the assay values. The data were marked as verified, corrected, or unverified, accordingly. Of the five holes drilled by INGEMMET in 1973, only three were verified. The remaining 88 core holes in the 2007 database were completely verified and corrected for use in mineral resource estimates.

12.2.3 SRK, 2008

SRK undertook the following checks:

- Detailed verification of assays using signed assay certificates;
- Assay verification from electronic laboratory files;
- Verification of down-hole survey data;
- Verification of drill hole positions in field;
- Comparison of copper grades from re-sampling of historical drill holes to original data;
- Verification of copper assay data from twinned drill holes;
- Comparison of copper assays from different analytical procedures;

- Comparison of copper assays from vertical versus inclined holes; and
- Collection and independent analysis of check assay samples.

SRK noted no errors or omissions in the data that were reviewed that could affect Mineral Resource estimation.

SRK selected a suite of 21 core samples for independent analysis at ALS Chemex, Vancouver, by four-acid digest and ICP-AES. Samples were chosen on the basis of their copper grade (determined by Alta Copper), lithology, mineralization type (presence of leachable copper), and age of drill hole, in an effort to reflect the variability in the deposit and the consistency of analytical results over time. Samples comprised bags of pieces of quartered core samples taken over a 2 m interval in an attempt to ensure that the sampled interval coincided with a Alta Copper sample interval. However, the very fractured, rubbly nature of the recovery of several intervals meant that the samples reflected a sample of gravel-sized core fragments, rather than split core.

On the whole, SRK's comparisons of results with the original Alta Copper assays yielded relatively good agreement. Graphic evaluation of the data indicated that approximately 65% and 90% of paired data fell below the 10% and 20% absolute relative difference lines. Significantly, there was no strong bias within the data, with points falling above and below the parity line. The data did not, in SRK's opinion, show an obvious correlation between the relative difference in lithology, mineralization, grade, or time of assay.

SRK also concluded that the reasonable correlation between the results from its checks using a four-acid digestion technique with ICP finish and those obtained by Alta Copper using a three-acid leach and AA finish further confirmed that the three-acid leach technique was suitable for the Cañariaco mineralization.

12.2.4 AMEC, 2010

AMEC reviewed 1,930 copper, gold, silver, and molybdenum assays, or 5.6% of the analytical data in the database, as a verification of the data quality. No errors were noted, and the analytical data were considered suitable to support Mineral Resource estimation. The QP was part of the AMEC team verifying this information.

12.3 Verification performed by the QP

The QP, David Thomas, visited the Project area on two occasions, in 2010 and 2022, as described in Section 2.4.

The QP reviewed the findings of the external data review programs to confirm that no significant issues were found with the databases or data collected at the time.

The QP is satisfied that the data is suitable to support mineral resource estimation.

12.3.1 Cañariaco Sur Assay Database

The QP reviewed 29 out of 539 assays or 5.4% of the analytical data in the 2008 Cañariaco Sur data and 115 out of 2251 assays or 5.1% of the 2012–2013 Cañariaco Sur analytical data. No errors were noted and the analytical data were considered suitable to support Mineral Resource estimation.

12.3.2 Site inspection

The QP visited the Cañariaco Sur area of the Project from February 14 to February 16, 2022. Two drill platforms were located in the field to verify the collar coordinates of drillholes CS12-005 and CS12-006. A comparison of the collar coordinates is shown in Table 12-1.

Table 12-1: Collar Coordinate Comparison

Drill Hole	Database Easting (deg)	Database Northing (deg)	Database Elevation (m)	GPS Easting (deg)	GPS Northing (deg)	GPS Elevation (m)	Difference in Easting (deg)	Difference in Northing (deg)	Difference in Elevation (m)
CS13-005	690158	9324519	3106	690166.3	9324519	3121.566	8.3	0.0	15.6
CS13-006	690177	9324660	3086	690180.1	9324660	3076.742	3.1	0.4	-9.3

The QP reviewed drill core from three drillholes (CS08-001, CS08-002 and CS 13-009). Observations of drill-core confirmed the lithological, alteration and mineralization models for Cañariaco Sur.

The QP also visited the Cañariaco Norte area from June 7 to June 11, 2010.

12.4 Comments on Data Verification

The QP is of the opinion that the data verification programs undertaken on the data collected from the Project adequately support the geological interpretations and the analytical and database quality, and therefore support the use of the data in Mineral Resource estimation:

- No major sample biases were identified from the QA/QC programs that were completed.
- Historic drill core from predecessor companies was re-analysed. The historical values were found to be sufficiently in accordance with the re-assay values that they were deemed acceptable for use.
- The collected sample data adequately reflect deposit dimensions, true widths of mineralization, and the style of the deposit.
- External reviews of the database have been undertaken in support technical reports, producing independent assessments of the database quality. No significant problems with the database, sampling protocols, flowsheets, check analysis program, or data storage were noted.
- Two generations of independent sampling support the mineralization grades reported by Alta Copper.
- Drill data are typically verified prior to Mineral Resource estimation by running a software program check.
- The QP has verified the geological models, analytical data and database for Cañariaco Sur and inspected the Cañariaco Sur area during a site visit completed in 2022. The QP also reviewed previous data verification programs and completed a site visit to Cañariaco Norte in 2010, and visually inspected the Cañariaco Norte area during the 2022 site visit.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

Four phases of test programs have been completed on the Cañariaco project by accredited metallurgical laboratories, with a heavy focus on the Cañariaco Norte deposit. Three phases of testwork were performed on the Norte deposit. With the inclusion of the Sur deposit as a blend for the Cañariaco Project for the optimized PEA, a fourth phase of testing was conducted for the Sur deposit in 2023. A summary of test programs is presented in Table 13-1.

Table 13-1: Metallurgical Testwork Summary Table

Year	Laboratory	Testwork Performed
Norte Deposit		
-	-	**Phase 1**
1999	Geomet S.A., Santiago, Chile	Sequential leach tests
2004	Kappes, Cassiday & Associates, Reno, Nevada	Column leach tests
2006	SGS Metallurgical Laboratory, Santiago, Chile	Leachability tests
2007	SGS Metallurgical Laboratory, Santiago, Chile	Scoping flotation tests
-	-	**Phase 2A**
2008	SGS Metallurgical Laboratory, Santiago, Chile	QEMSCAN examination Comminution and variability comminution tests Effects of grind sizes Effects of collectors and pH Effect of sulfidation Cleaner flotation tests Locked cycle tests
-	-	**Phase 2B**
2010	SGS Metallurgical Laboratory, Santiago, Chile	Effects of grind sizes Effects of collector, frother type and dosage, and pulp pH Effects of cleaning Flotation tests Concentrate generation Mineralogical analysis
2010	Outotec, Antofagasta, Chile	Tailings thickening tests
2010	Outotec, Sweden	Proof-of-concept roasting testwork
-	-	**Phase 3 -Variability Testwork**
2011-2012	G & T Metallurgical Services LTD	Bulk Mineral Analysis with Liberation (BMA) Effects of grind sizes Flotation variability testing
Sur Deposit		
-	-	**Phase 4**
2023	Transmin Metallurgical Consultants	Comminution testing Mineralogical analysis QEMSCAN examination Rougher Flotation tests

13.2 Cañariaco Norte Deposit Testwork

Three major phases of testwork were conducted for the Norte deposit. The initial phase examined potentially leachable copper mineralization. The leaching testwork consisted of bottle-roll leach tests and column leach tests to determine the amenability of material to heap leach extraction.

Additional drilling in 2007 indicated the mineralization consisted predominantly of sulfide copper minerals. Therefore, the testwork program was revised, and the second phase of testwork was focused on assessing the metallurgical response of the sulfide copper mineralization to flotation. This second phase was interrupted by the financial crisis of 2008. Definition work resumed in 2010 with further development of process parameters to allow primary equipment selection. The goal of the 2010 work was to support a prefeasibility study during 2010. Lastly, the mineralized material variability testing campaign conducted in 2011 and 2012 was designed to improve the geometallurgical understanding of the deposit and robustness of the proposed process flowsheet.

The testwork recommended key design parameters, as follows:

- Primary grind was established at 200 µm;
- JKSimMet breakage parameter Axb value of 52.5 and Bond ball mill work index (BWi) of 12.2 kWh/t;
- Final concentrate grade of 26% Cu;
- For a LOM feed grade of 0.40% Cu, a recovery of 89.7% is forecast at the target grind;
- Recovery will vary with feed grade (based on a grade recovery formula).

13.2.1 Phase 1 (1999 – 2007)

Initial exploration identified potentially leachable copper mineralization. Copper recovery was planned through a solvent extraction-electrowinning (SX-EW) plant to produce copper cathode. From 2005 to 2006, Kappes, Cassidy & Associates (KCA) assessed the potential of leaching samples containing high levels of cyanide-soluble copper. The leaching testwork consisted of bottle-roll leach tests and column leach tests. Initial recoveries were variable and lower than anticipated. In response, additional samples were obtained from twinned drill holes and sent to SGS Lakefield Research Chile (SGS Chile) for bottle-roll and column leaching testwork. The leaching results from SGS Chile were similar to those from KCA and varied from 52% to 70% for the higher sulfide samples.

Drilling in 2007 indicated the presence of a significantly larger copper deposit with predominantly sulfide copper mineralization. The testwork program was revised to assess the metallurgical response of the sulfide copper mineralization to flotation. In late 2007, SGS Chile re-composited untreated samples remaining from the 2006 leaching testwork program into three master composites for scoping-level comminution and flotation testwork. These samples were not considered to be representative of the entire primary sulfide deposit but were used for a process development program. The 2007 work was encouraging and confirmed that additional testwork focusing on the flotation of the sulfide mineralization was warranted.

Principal development composites were:

- CPY: primary sulfides composite;

- CN: secondary sulfides composite; and
- BS: composite from breccia lithology.

These samples were selected based on copper mineralization within the deposit and were representative of material that could potentially be mined.

13.2.2 Phase 2A (2008)

Based on the positive initial sulfide flotation testwork, comprehensive comminution and flotation testwork was initiated at SGS Chile in early 2008. Fresh samples were collected and composited, then subjected to comminution and flotation tests with the objective of defining metallurgical parameters for detailed mining studies. Although this work was halted prematurely because of the world economic situation in late 2008, it provided adequate information on the grinding and flotation responses to confirm that a smeltable concentrate could be produced. Sufficient detail was available to define metallurgical challenges, particularly arsenic in the concentrate and the potential mitigation strategy.

Principal development composites were:

- CPY: a predominantly hypogene mineralization type that is primarily chalcopyrite;
- ENT: a mixed mineralization type that has elevated arsenic values from the multi-element analysis;
- MIX: a mixed mineralization type that contains several copper sulfide minerals; and
- SEC: a mineralization type that contains predominantly enriched minerals such as chalcocite, covellite, bornite, enargite, and tennantite.

These samples were selected based on copper mineralization within selected zones of the deposit and were representative of material that could potentially be mined.

13.2.3 Phase 2B (2010)

The 2010 testwork program had the initial goal of continuing the definition of mineralization characteristics to support a prefeasibility study. During the preliminary design evaluations, it became apparent that there was an opportunity to enhance the potential deposit value by using a new approach to reduce arsenic in the bulk copper concentrate. This program consisted of testing for the optimum rougher flotation feed size, regrind feed size, and rougher and cleaner reagent additions. Products from this testwork were further analyzed for mineralogy, chemistry, and settling quality. Results indicated a coarser grind could be used and still maintain recovery through the employment of a dual collector system.

Later testwork focused on confirmation that the improvements were robust, and that the solution proposed for the penalty elements would work. This later testwork—the application of partial roasting—was demonstrated to proof-of-concept level, and development work was recommended to advance this approach.

Principal development composites were:

- ACP: selection of samples containing chalcopyrite and arsenic minerals;

- ACS: selection of samples containing secondary copper sulfide (covellite, chalcocite, bornite) and arsenic minerals; and
- ACL: selection of samples with low content of As (<0.03%As).

These samples were selected based on copper and/or arsenic mineralization within the deposit and were representative of material that could potentially be mined.

13.2.4 Phase 3: Variability Testwork (2011–2012)

The variability testwork program completed in 2011 and 2012 was designed to improve the geometallurgical understanding of the deposit and to assess the robustness in the proposed process flow sheet. The program consisted of mineralogical characterization of the composites, additional comminution tests, and roughing and cleaning flotation optimization tests. Principal development composites were:

- IBxP (Polymictic Breccia): breccia mineralization;
- TBX (Tourmaline Breccia): copper-bearing tourmaline breccia;
- IBxH (Hydrothermal breccia): a predominantly hypogene mineralization type that is primarily chalcopyrite;
- CQP (Coarse quartz porphyry): mineralization type that contains predominantly enriched minerals such as covellite, bornite, enargite, and tennantite;
- BFP (Biotite feldspar porphyry): selection of samples containing an average of Cu:As ratio equal to 15; and
- CQFP (Crowded quartz-feldspar porphyry): selection of samples with a mixed mineralization that contains several copper sulfides.

These samples were selected based on lithology, copper and/or arsenic mineralization within the deposit and were representative of material that could potentially be mined.

13.2.5 Mineralogy

Mineralogical analyses were performed during both the 2008 and 2010 testing campaigns.

In 2008, the mineralogy of three concentrate samples was studied using particle mineralogical analysis (PMA). Each sample was divided into four size fractions (#100, #200, #400, -#400) and one graphite-impregnated polished epoxy grain mount was prepared per fraction. The main gangue minerals detected in the global samples were quartz, pyrite, and sericite/muscovite. Chalcopyrite was the main copper-bearing mineral with values between 6.06 wt% and 9.30 wt%. The gangue minerals detected in lower quantities included plagioclase/albite, chlorite, clays, titanium oxides and biotite. The finest fraction (-#400) in the three samples contained a low percentage of pyrite and the main gangue mineral was phyllosilicates.

A QEMSCAN examination of metallurgical samples during the 2010 testing campaign indicated the predominant minerals and mineral associations at Cañariaco Norte samples were:

- Copper mineral species: chalcopyrite, covellite, chalcocite, bornite, enargite, tennantite, and tetrahedrite; primarily associated with pyrite and other sulfide gangue minerals.

- Non-metallic gangue minerals: quartz, micas, biotite, chlorite, and clay group minerals.

The mineral content and liberation estimates for abundant mineral species was determined by QEMSCAN using a modified Bulk Mineral Analysis (BMA) technique on eleven and twenty-eight composites in 2011 and 2012, respectively. The main findings from the analysis are:

- The sulfide content of the samples ranged from three percent to over 10 percent in the samples analyzed in 2011. In the case of the 28 composites analyzed in 2012 the sulfide content ranged from 1 percent to over 8 percent. Copper sulfides are predominantly associated with pyrite followed by chalcopyrite, tennantite, chalcocite, bornite and covellite.
- On both set of samples, silicate minerals accounted for the majority of the sample mass. None of the silicate minerals observed (quarts, muscovite, feldspar) should interfere with the flotation process.
- Copper sulfide deportment in the samples was quite variable. Some samples had chalcopyrite as the main copper sulfide and other presented significant levels of chalcocite, bornite and tennantite.
- The presence of tennantite, at various ranges, was observed in the samples analyzed in 2011. It was recommended that the distribution should be investigated further as there may be a mining solution available to control arsenic.
- The average pyrite to copper sulfide ratio in the 28 composites analyzed in 2012 was 3:1. At these levels of pyrite, maintaining chemically selective conditions will be required to produce high grade concentrates.
- The mineral fragmentation characteristics were also estimated using QEMSCAN on unisized samples of the composites generated in both 2011 and 2012. The analyses were conducted on samples at an average primary grind size of 160 µm K_{80} for the samples in 2011 and 200 µm K_{80} for the composites prepared in 2012. The following comments can be made from the results obtained in the analysis.
- Average copper sulfide liberations were relatively low averaging 35 and 38 percent in 2011 and 2012, respectively. The only exception were the samples with low pyrite levels.
- The form of copper sulfide interlocking was highly variable.
- Copper sulfide interlocking with non-sulfide gangue was also common in all the samples.
- Samples with high pyrite interlocking with copper sulfides could still achieve high copper recovery to the rougher concentrate by aggressively recovering pyrite. However, in 2012 it was noted that aggressive regrinding and selective conditions, with high pH, will likely be required to produce high grade final concentrates.
- Based on the limited data, the pyrite copper sulfide interlocking was complex and intricate. Fine regrind sizes for the rougher concentrate will be required to efficiently separate copper sulfides from pyrite.

13.2.6 Comminution Tests

The 2010-2011 engineering study work used the comminution testing, consisting of BWi and abrasion index (Ai) tests, which were performed in 2008 at SGS Chile. Initially, copper mineralization composite samples were tested to assess grindability parameters of chalcopyrite rich, arsenic-rich, and secondary copper mineral-rich materials. The results are provided in Table 13-2.

Table 13-2: Results of Comminution Testing

Composite	BWi (kWh/t)	Ai
CPY	13.3	0.12
BS	13.3	0.11
CN	12.2	0.16

An additional 48 selected spatial samples were sent to SGS Chile for comminution testing, specifically for JK drop-weight (DWT), SPI®, crusher work index (CWi) and the BWi tests at a closing size of 100 mesh (150 µm). JK drop-weight tests require coarser material and hence this testing was only possible on 10 samples (samples SJKT-01 to SJKT-10). Statistics from the comminution test results are summarized in Table 13-3.

Table 13-3: Comminution Tests Statistics

Statistics	DWT			CWi	SPI (min)	BWi (kWh/t)
	SG	A x b	t_a			
Average	2.75	68.0	0.7	9.7	56.4	11.1
75th Percentile	2.78	52.5	0.8	12.2	59	12.2
Maximum	2.87	162.2	1.1	18.1	72.2	15.8
Minimum	2.67	29.7	0.5	3.7	48.6	5.6

13.2.7 Flotation Tests

Three different testing campaigns were performed under Phase 1, Phase 2A and Phase 2B at SGS Chile using different samples to investigate the effect of particle size, reagents, pH, and modifiers. The 2006 flotation tests were performed using three lithology composites, breccia-rich, chalcopyrite-rich and copper secondary mineral-rich. In 2008, the same material was used to provide samples for variability grinding work and was used to create four composites. In 2010, SGS Chile performed additional testwork on three selected composites, a low-arsenic composite, a copper secondary mineral composite, and a composite containing high levels of chalcopyrite and arsenic. These composites were derived from the same inventory of material which supplied the material for the 2008 composites.

Results discussed in this sub-section were summarized from Bonson et al., (2008).

13.2.7.1 Particle Size

Several tests were performed at different particle sizes (K_{80} of 75, 106, 150, 212 and 250 µm) and at a pH 10. A P_{80} of 150 µm was selected for subsequent flotation testing on all 2006 composites.

In 2008, the effect of grind size (P_{80} at 75, 100, 150, 200, and 250 µm) was evaluated using rougher kinetic tests. Although finer grinding improved the copper recovery, a primary grind of P_{80} of 125 µm was selected as optimum.

In 2010, primary flotation testwork was conducted with six stages (1, 2, 4, 8, 10, 12 and 18 min) of roughing. The duration of each flotation test was 18 minutes. The effect of grind size (P_{80} at 120, 150, 175, 200, and 250 µm) was evaluated. A slight decrease in recovery was noted when going to a coarser grind with all three composites. However, the relative impact of going from 150 µm to 200 µm was fairly minor for all composite types.

After developing a comparison model that evaluated preliminary capital and operating cost requirements at various sizes versus the potential revenue, it was decided to use 200 µm as a primary grind as it allowed for a greater amount of metal to be recovered per unit cost of expenditure.

13.2.7.2 Reagent Scheme

In 2006, rougher flotation tests were carried out to determine the best reagents to achieve a high copper recovery. The collector that generated the best results was sodium thionocarbamate (AP3894) with methyl isobutyl carbinol (MIBC) frother.

Similarly, in 2008 tests were conducted with a grind size of 125 µm and pH 10 with seven collector types (AP-3894, AP-3330, PAX, AP-3477, AP-404, AP3926 and AP-3302) with frother (MIBC) addition. Higher copper selectivity was observed with collectors AP-3330, and AP-3477.

The evaluation of four frothers (MIBC, D-250, H-76, and TEB) was also carried out at grind size of 125 µm and pH 10 with the addition of collectors AP-3477 (25 g/t) and AP-3330 (25 g/t). The reagents suites that produced best results in terms of recovery and grade were AP-3300/MIBC.

In 2010, collector tests were performed as an attempt to increase the recovery of both copper and gold into the concentrate. Rougher tests indicated that the best overall recovery performance for copper and gold was achieved by a 75% A-3894 and 25% PAX combination.

13.2.7.3 pH

In 2006, the work on the effect of pH indicated that pH 10 is optimum for rougher flotation stage, giving acceptable results for all the composites tested.

Several rougher kinetic tests were conducted in 2008 at pH 7, 9, 10, 11 and 12. Tests were performed at a grind size of 125 µm with the addition of collector AP-3330 (25 g/t) and frother MIBC (20 g/t). Good performance (recovery and grade) was obtained at pH 9 and 10, with particularly higher concentrate copper grades at pH 10. A noticeable decrease in recovery was observed at natural pH for the arsenic-rich and copper secondary mineral-rich samples.

13.2.7.4 Modifiers

The initial work on modifiers was discontinued since the flotation performance (recovery and grade) was not improved by the addition of a sulfidizing reagent on the three composites samples treated in 2006.

In 2008, sulfidation tests were completed at a grind size of 125 µm, pH 10 with the addition of collector AP-3330 (25 g/t) and frother MIBC (10 g/t). The effect of NaHS addition (50 g/t and 100 g/t) was evaluated. From the samples tested, chalcopyrite-rich and copper secondary mineral-rich composites showed good copper recoveries without the addition of NaHS. The arsenic-rich sample showed slightly improved recovery with NaHS addition.

13.2.7.5 Regrind and Cleaning Stages

The results from the metallurgical campaign conducted in 2006 demonstrated that concentrate grade is sensitive to cleaning flotation pH and regrind size. Results indicated an improved performance at a P_{80} of 37 µm (80% passing 400 mesh).

Regrinds were evaluated at 4, 8, and 12 minutes instead of a specific P_{80} size to facilitate laboratory work in 2008. The chalcopyrite-rich sample required 4 minutes to achieve optimal grade-recovery relationship in batch tests. Between 8 minutes and 23 minutes were required for the other three composites. This corresponds to a P_{80} of 50 µm for chalcopyrite-rich and arsenic-rich mineralization, and a P_{80} of 35 µm for secondary copper mineral-rich and mixed sulfide type materials.

Several cleaner flotation tests were conducted at pH 11, 11.5 and 12. A pH of 11.5 achieved satisfactory results for all composites tested. 2010 tests showed that the cleaning stage requires a high pH level of 12 to produce satisfactory copper concentrate grades and the mixed reagent scheme did not perform as well as the sole use of A3894.

Flotation testwork was performed to develop the tailings flotation flowsheet and provide sample suitable for acid-base account testing.

13.2.7.6 Variability Testwork

Two additional testing campaigns were performed in 2011 and 2012 at G & T Metallurgical Services LTD to evaluate the effect of mineralized material variability on the flotation performance with previously established conditions.

In 2011, two rougher flotation tests were performed with a Master Composite to confirm initial conditions. Later a single rougher test was performed on each composite. Flotation tests were conducted with a constant grind time. Lime was added to achieve a target pH of 10. Addition of collector A3894 with PAX to collect copper sulfides.

On average, copper recovery was at 89%. Two samples presented lower recoveries due to the poor liberation levels with little interlocking with pyrite. It was suggested that finer levels of primary grind would be required to improve the metallurgical performance of these two samples.

Gold recovery from the feed to the rougher concentrate was highly variable and ranged from 33 to 90 percent.

A single cleaner test was performed on each of the composite. The rougher concentrate was reground prior to the three states of cleaning. A constant regrind time was applied, based on the initial results from the Master Composite. The pH in the cleaner circuit was maintained at pH 11.5 and collector was added in the cleaner stages.

The metallurgical response of the composites was highly variable. Two composites had the lowest copper recovery, below 65%, likely due to insufficient copper sulfide liberation at the primary grind size. Two other samples showed much better performance. These samples had the most favorable mineral fragmentations profile and the arsenic levels in the concentrate were very low for these composites. The rest of the composites had adequate copper recovery into rougher. However, the cleaner circuit performance was variable resulting in low grade concentrate (<20%) or high losses of copper to the cleaner tails. Mineralogical assessment of the cleaner tailing's streams indicated that copper sulfide-pyrite interlocking was problematic. Thus, it was suggested that the current process and regrind target was inadequate for this style of mineralization.

Gold was, on average, recovered at 45% from the feed into the final concentrate, exceeding a gold content of 1 g/tonne. Arsenic in the final concentrate was highly variable, ranging from 0.06 to 2.9 percent.

A composite of the samples with more intricate pyrite-high secondary copper sulfide mineralization was tested separately. The main objective was to optimize pH and regrind size to improve metallurgical performance. The results indicated that better performance can be achieved by reducing the grind size. At a regrind size of 16 to 17 mm K_{80}, 80 percent of the copper was recovered in a concentrate grading 30 percent copper.

Similarly, in 2012 prior to completing variability testing, primary grind size and collector dosage were evaluated over a rougher and cleaner parameter optimization.

Based on the results obtained, it was suggested that a lower PAX dosage in the rougher circuit combined with fine regrinding of the rougher concentrate and elevated pH levels in the cleaner yielded suitable metallurgical response. On average, 78 percent of the copper in the feed was recovered into a high-grade concentrate grading 33.6 percent copper.

13.2.8 Metallurgical Variability

The samples selected for metallurgical testing during the variability testing and used for study development were representative of the various styles of mineralization within the deposit. A total of 39 composites of varying geological, geographical and chemical make-up were made from drill core from all areas of the deposit. Sufficient samples were selected, and tests were performed with sufficient mass to be representative of the lithologies identified within the deposit.

13.2.9 Deleterious Elements

As part of the final conclusions of the testing campaign carried out in various testwork programs it was noted that the presence of arsenic in the concentrates could incur smelter penalties.

The work conducted in 2012 focused on establishing robust improvements and proposed solutions to deal with penalty elements. Sample of flotation concentrate were subjected to preliminary partial roasting testing at the Outotec lab in Germany. Outotec partial roasting technology has been utilized commercially for a number of years to remove arsenic from concentrates. The testwork results were positive. Based on these results a final flowsheet was developed utilizing flotation to produce a copper concentrate with a marketable copper grade followed by partial roasting of the concentrate to reduce arsenic in the final concentrate to below penalty levels.

For the 2022 PEA, the 2011-2012 flotation testwork was re-evaluated and it was determined to be more economical to build a standard copper sulphide concentrator to produce a 26% copper concentrate for sale to smelters. The concentrate will have arsenic content levels that would incur penalties but can be controlled below levels that will inhibit the sale of the concentrate.

13.2.10 Recovery Estimates

The Norte deposit metallurgical recovery estimates were derived from the 2012 variability flotation testwork program and based on the following criteria:

- Flotation test results using the optimum grind size and reagent mixture;
- Flotation test results were correlated for each of the major lithology composites (BFP, TBx, CQFP, CQP, IBxH, and IBxP); and
- The overall copper recoveries were nominalised to achieve a copper concentrate grade of 26% Cu.

Table 13-4 summarizes the recovery equations derived for the major lithology composites.

Table 13-4: Metallurgical Recoveries by Rock Type – Norte Deposit

Lithology	Recovery			
	Cu (%)	As (%)	Ag (%)	Au (%)
BFP	$91.5*(1-EXP(Cu\%_{feed}*-14))$	$14.295 \times Cu_{Recovery}^{0.4139}$	$2.138 \times Cu_{Recovery}^{0.7708}$	$0.2551 \times Cu_{Recovery}^{1.2586}$
IBxH	$91.5*(1-EXP(Cu\%_{feed}*-14))$	$6E-08 \times Cu_{Recovery}^{4.516}$	$4E-06 \times Cu_{Recovery}^{3.6335}$	$1E-05 \times Cu_{Recovery}^{3.476}$
CQFP	$89.3*(1-EXP(Cu\%_{feed}*-10))$	$3.499 \times Cu_{Recovery}^{0.7194}$	$0.0173 \times Cu_{Recovery}^{1.7542}$	$3E-05 \times Cu_{Recovery}^{3.2099}$
CQP	$89.3*(1-EXP(Cu\%_{feed}*-10))$	$3.499 \times Cu_{Recovery}^{0.7194}$	$0.0173 \times Cu_{Recovery}^{1.7542}$	$3E-05 \times Cu_{Recovery}^{3.2099}$
TBx	$80.3*(1-EXP(Cu\%_{feed}*-10))$	$2E-12 \times Cu_{Recovery}^{6.9291}$	$6E-05 \times Cu_{Recovery}^{3.1369}$	$1E-07 \times Cu_{Recovery}^{4.6209}$
IBxP	$89.3*(1-EXP(Cu\%_{feed}*-10))$	$3.499 \times Cu_{Recovery}^{0.7194}$	$0.0173 \times Cu_{Recovery}^{1.7542}$	$3E-05 \times Cu_{Recovery}^{3.2099}$

13.3 Cañariaco Sur Deposit Testwork

The first metallurgical test program for the Sur deposit was conducted by Transmin Metallurgical Consultants of Lima in November 2023. A total of 1,100 kg in 36 mineralized drill samples were collected from drill core and delivered to the Plenge Labs in Lima. It featured comprehensive chemical analysis, Bond Work Index determination, flotation tests to determine copper and gold recoveries as well as bulk densities, specific gravities, pH levels. The objective of this test program was to ensure the Sur deposit samples exhibited economic recoveries when processed through the flow sheet determined for the Norte deposit.

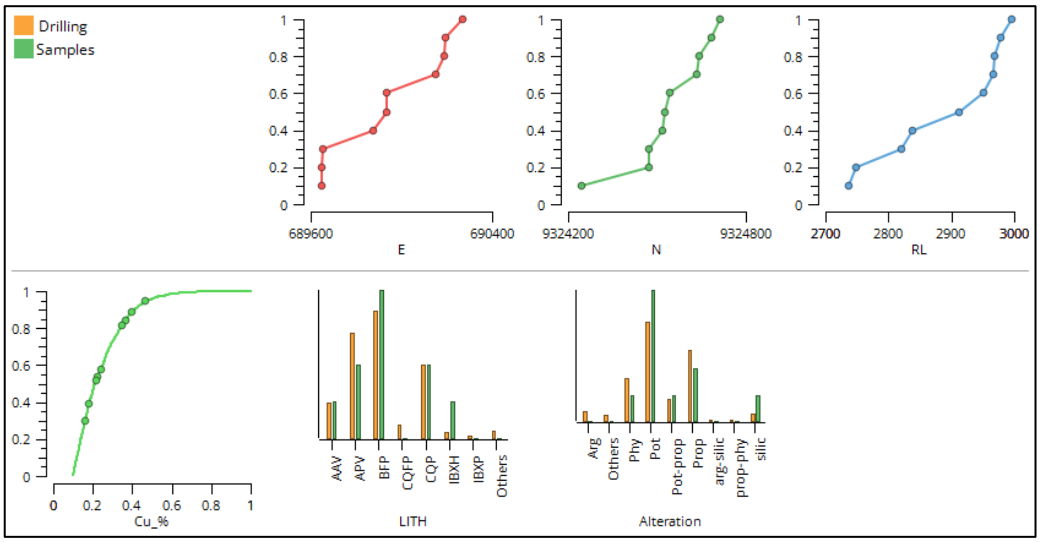
Principal development composites for this test program were:

- APV (Andesite Volcanics): mineralization mixed with chalcopyrite, bornite, and minor pyrite with copper oxide, it does not contain the later stage brecciation that introduced arsenic;
- CQP (Coarse Quartz Porphyry): mineralization type that contains predominantly enriched minerals such as covellite, bornite, enargite, and tennantite;
- BFP (Biotite Feldspar Porphyry): selection of samples with a mixed mineralization that contains several copper sulfides

- CQFP (Crowded Quartz Feldspar porphyry: selection of samples with a mixed mineralization that contains several copper sulfides.
- Two composites were obtained from the samples, CSAC-01 and CSAC-02 for flotation and comminution tests.

CSAC-01 was formed from 10 samples of the 36, with the goal to evaluate the Bond Ball Work Index (BWI), its criteria focused on diversity in lithology and alteration in the Sur deposit. The representativeness graph with respect to the Cu grade, lithology, and alteration for the composite is presented Figure 13-1.

Figure 13-1: Composites – CSA – 001 Representativity



Source: Transmin, 2024.

CSAC-002 composite was formed with two samples characterized by BFP lithology and potassic alteration which is representative of 85% of the material in the deposit. The characteristics of the two samples are summarized in Table 13-5.

Table 13-5: Composites – Chemical Analysis (Estimated)

Composite	Weight (kg)	Alteration	LITH	Chemical Characterization (Main Elements)									
				Ag (g/t)	Au (g/t)	Cu (%)	Fe (%)	C total (%)	S total (%)	As (g/t)	Cu H2SO4 (%)	Cu CN (%)	Cu Residual (%)
CSAC-001	50	Various	Various	1.14	0.18	0.28	2.72	0.15	0.31	72.0	0.050	0.069	0.15
CSAC-002	44	Pot	BFP	1.34	0.15	0.36	2.76	0.15	0.32	48.9	0.025	0.14	0.18

13.3.1 Mineralogy

The selected 36 samples in the program underwent ICP assays and were chemically characterized. The geochemical characteristics is in Table 13-6. The chemical characterization is in Table 13-7.

Table 13-6: Samples – Geochemical Characteristics of Samples

Sample Id	Drill Hole	From	To	Length	Ag, g/t	Au, g/t	Cu, %	Fe, %	As, g/t	Alteration	LITH
CSAS-001	CS08-001	224	244	20	1.25	0.1271	0.293	3.49	5	Pot	BFP
CSAS-002	CS08-001	248	268	20	1.84	0.2137	0.417	2.9	6	Pot	BFP
CSAS-003	CS08-001	300	340	40	1.56	0.1911	0.412	2.96	5	Pot	BFP
CSAS-004	CS08-001	396	418	22	1.55	0.1278	0.4	2.28	20	Pot	BFP
CSAS-005	CS08-001	460	480	20	1.29	0.0976	0.249	2.41	7	Pot	BFP
CSAS-006	CS08-002	108	140.2	32.2	1.74	0.1178	0.365	1.39	54	silic	BFP
CSAS-007	CS08-002	290	313	23	1.07	0.0631	0.175	3.05	41	Pot-prop	BFP
CSAS-008	CS08-002	360	380	20	0.79	0.0887	0.181	2.91	12	Pot-prop	BFP
CSAS-009	CS12-003	41	61	20	0.62	0.0836	0.351	2.83	4	Pot	APV
CSAS-010	CS12-003	85	105.4	20.4	1.12	0.1627	0.597	2.17	9	Phy	APV
CSAS-011	CS12-003	160	180	20	0.25	0.0703	0.223	1.06	3	Phy	CQP
CSAS-012	CS12-003	211	228	17	0.33	0.0538	0.229	0.84	306	Phy	CQP
CSAS-013	CS13-004	228	248	20	2.62	0.1121	0.311	1.34	30	Pot	CQFP
CSAS-014	CS13-004	284.3	303	18.7	2.6	0.1406	0.273	2.51	5	Pot	APV
CSAS-015	CS13-004	303	336	33	1.86	0.105	0.297	2.68	28	Pot	AAV
CSAS-016	CS13-004	360	380	20	1.63	0.1081	0.217	1.96	68	Pot	BFP
CSAS-017	CS13-005	184	194.5	10.5	1.23	0.0646	0.301	1.32	4	Pot	CQP
CSAS-018	CS13-005	194.5	202	7.5	1.81	0.1337	0.517	3.87	4	Pot	APV
CSAS-019	CS13-005	217	233	16	0.43	0.0304	0.16	1.25	6	Pot	CQP
CSAS-020	CS13-006	4	28	24	5.81	0.1147	0.18	3.68	11	Arg	APV
CSAS-021	CS13-006	52	77	25	1.31	0.1414	0.467	4.95	4	Prop	APV
CSAS-022	CS13-006	86	106	20	2.73	0.3298	0.348	4.96	22	Pot	APV
CSAS-023	CS13-006	220	240	20	2.92	0.0951	0.356	1.09	171	Prop	CQP
CSAS-024	CS13-007	200	220	20	2.04	0.0733	0.297	1.05	204	Phy	CQP
CSAS-025	CS13-007	248.1	266	17.9	3.29	0.0341	0.152	1.42	280	Phy	CQP
CSAS-026	CS13-008	65	90	25	1.24	0.0695	0.417	2.62	11	Prop	APV
CSAS-027	CS13-008	260	280	20	0.79	0.0785	0.275	2.99	12	Prop	BFP
CSAS-028	CS13-008	292	304	12	1.53	0.1397	0.363	3.07	23	Pot-prop	BFP
CSAS-029	CS13-008	304	316	12	1.07	0.1067	0.383	2.47	3	Prop	BFP
CSAS-030	CS13-008	507	525	18	1.39	0.0786	0.309	5.42	40	Pot	IBXH
CSAS-031	CS13-009	80	100	20	1.21	0.0653	0.345	2.56	10	Prop	APV
CSAS-032	CS13-009	100	120	20	1.24	0.0603	0.316	2.4	9	Prop	APV
CSAS-033	CS13-011	95.5	115	19.5	1.28	0.3679	0.24	5.4	255	Prop	IBXH
CSAS-034	CS13-012	32	54	22	0.74	0.0589	0.238	2.87	9	Prop	APV
CSAS-035	CS13-012	73	101	28	0.71	0.0913	0.265	4.45	18	Prop	IBXH
CSAS-036	CS12-003	291	311	20	0.59	0.0696	0.239	2.28	14	Pot	AAV

Source: Transmin, 2024.

Table 13-7: Chemical Characterization of Samples

Sample ID	Chemical Characterization (Main Elements) Chemical Characterization (Main Elements)									
	Ag, g/t	Au, g/t	Cu, %	Fe, %	C total, %	S total, %	As, g/t	Cu H2SO4, %	Cu CN, %	Cu Residual, %
CSAS-001	1.24	0.14	0.33	3.04	0.14	0.30	56.9	0.023	0.12	0.17
CSAS-002	1.49	0.16	0.40	2.32	0.16	0.36	36.1	0.028	0.16	0.20
CSAS-003	1.49	0.16	0.41	2.71	0.13	0.40	38.4	0.026	0.10	0.27
CSAS-004	1.48	0.13	0.41	2.05	0.16	0.38	55.1	0.037	0.14	0.22
CSAS-005	1.00	0.096	0.24	2.21	0.17	0.19	41.2	0.022	0.10	0.11
CSAS-006	1.24	0.091	0.36	1.30	0.14	0.56	76.2	0.016	0.061	0.28
CSAS-007	0.99	0.067	0.17	3.02	0.15	0.25	61.7	0.0080	0.040	0.12
CSAS-008	0.50	0.071	0.16	2.77	0.14	0.25	39.8	0.0070	0.031	0.12
CSAS-009	0.50	0.082	0.32	2.36	0.0027	0.19	42.4	0.13	0.084	0.11
CSAS-010	1.00	0.17	0.59	1.79	0.013	0.74	39.6	0.021	0.088	0.45
CSAS-011	0.50	0.074	0.22	1.07	0.22	0.29	34.7	0.0060	0.041	0.16
CSAS-012	0.50	0.060	0.22	0.77	0.081	0.46	240	0.022	0.070	0.12
CSAS-013	5.47	0.11	0.32	1.19	0.17	0.52	73.2	0.019	0.15	0.14
CSAS-014	2.97	0.13	0.28	2.49	0.17	0.41	37.0	0.029	0.13	0.11
CSAS-015	1.97	0.10	0.29	2.56	0.13	0.31	53.7	0.046	0.13	0.10
CSAS-016	1.99	0.12	0.23	1.92	0.15	0.20	90.2	0.028	0.12	0.076
CSAS-017	1.73	0.073	0.29	1.23	0.083	0.43	36.6	0.028	0.11	0.14
CSAS-018	1.97	0.13	0.51	3.92	0.12	0.50	33.4	0.052	0.18	0.26
CSAS-019	0.50	0.032	0.16	1.13	0.12	0.20	41.6	0.011	0.040	0.097
CSAS-020	5.22	0.11	0.16	3.25	0.055	0	40.5	0.068	0.0050	0.082
CSAS-021	1.24	0.14	0.45	5.07	0.023	0.0070	41.8	0.25	0.0090	0.19
CSAS-022	2.49	0.84	0.35	4.78	0.081	0.19	52.4	0.13	0.18	0.039
CSAS-023	2.98	0.085	0.36	0.96	0.19	0.66	152	0.036	0.061	0.25
CSAS-024	1.99	0.079	0.29	1.00	0.26	0.56	167	0.022	0.064	0.20
CSAS-025	3.96	0.049	0.16	1.45	0.15	1.11	257	0.0080	0.046	0.097
CSAS-026	1.48	0.078	0.41	2.38	0.12	0.90	41.3	0.014	0.049	0.32
CSAS-027	0.99	0.059	0.27	2.78	0.14	0.34	45.6	0.0080	0.048	0.21
CSAS-028	0.99	0.086	0.35	2.90	0.31	0.58	40.8	0.0090	0.084	0.25
CSAS-029	0.99	0.10	0.42	2.42	0.24	0.53	38.3	0.011	0.058	0.33
CSAS-030	0.99	0.075	0.25	4.95	0.29	1.64	63.5	0.0070	0.035	0.20
CSAS-031	1.49	0.077	0.34	2.31	0.16	0.79	40.4	0.013	0.052	0.26
CSAS-032	1.24	0.067	0.32	2.34	0.085	1.16	46.7	0.015	0.044	0.26
CSAS-033	0.99	0.21	0.23	4.94	0.26	0.78	244	0.013	0.033	0.18
CSAS-034	0.50	0.058	0.25	2.84	0.016	0.060	38.0	0.18	0.023	0.032
CSAS-035	0.99		0.27	4.13	0.20	0.55	53.8	0.015	0.042	0.20
CSAS-036	0.50	0.070	0.23	2.21	0.16	0.27	44.2	0.012	0.043	0.16

Source: Transmin, 2024.

13.3.2 Comminution Testing

The CSAC-001 composite was ground to a P_{80} of 235 and 119 and showed that the BWI does not vary significantly. The deposit material is in the moderate hardness range.

Table 13-8: Bond Work Index Results

Composite	Test Type	F_{80}, µm	P_{80}, µm	CSS_P1, µm	Gbp, g/rev	Bwi, kWh/t
CSAC-001	Original	1760	235	300	3.65	11.00
	Original	1760	119	150	2.37	11.23

Source: Transmin, 2024.

13.3.3 Flotation Testing

Exploratory flotation testing was performed on composite CSAC-002 to assess flotation response of the Sur Deposit. Kinetic rougher flotation was carried out to determine if conditions selected for the Norte deposit would suit the Sur deposit. Further variability tests were performed on 9 samples from the Sur deposit. The criteria of the flotation test were:

- Grind to a P_{80} of 200 um;
- Target flotation pulp pH of 10;
- 16 minutes of total rougher flotation test;
- A total of 38 g/t of MIBC added, 32 g/t of PAX.

In the variability testwork, the 9 samples were selected to determine the robustness of the flotation criteria for the mineralized material in the Sur Deposit. The tests were performed at the target P_{80} of 200 um with Cu recovery ranging from 80.4% to 97.4%. The tests results are displayed in Table 13-9.

Table 13-9: Variability Rougher Flotation Testing

Sample	Lithology	Alteration	Residence Time (minutes)	Mass Pull (%)	Cu Rec (%)	As Rec (%)	Au Rec (%)	Ag Rec (%)
CSAS-013	CQFP	Pot	16.0	4.6	93.8	37.4	67.7	82.7
CSAS-001	BFP	Pot	16.0	3.8	96.5	25.9	78.5	70.4
CSAS-002	BFP	Pot	16.0	4.8	96.6	17.5	65.4	77.0
CSAS-004	BFP	Pot	16.0	4.9	97.4	24.9	71.3	76.5
CSAS-006	BFP	Silic	16.0	7.4	90.0	50.4	65.2	60.5
CSAS-012	CQP	Phy	16.0	5.5	87.7	74.8	61.4	53.9
CSAS-024	CQP	Phy	16.0	7.0	80.4	59.1	59.4	76.2
CSAS-010	APV	Phy	16.0	6.7	96.3	16.2	65.9	71.0
CSAS-031	APV	Prop	16.0	8.1	94.1	15.5	60.5	71.8

Source: Transmin, 2024.

Cleaner tests were not performed.

For the nine variability tests, the average rougher flotation of copper and gold yielded 92.5% copper recovery and 66.1% gold recovery. Two tests, CSAS-012 and CSAS-024, performed on samples of CQP, contained higher arsenic and demonstrated higher arsenic recovery. The calculated head assay results are presented in Table 13-10.

Table 13-10: Calculated Head Grades of Variability Samples

Lithology	Alteration	Test	Cu (%)	As (ppm)	Au (g/t)	Ag (g/t)	Fe (%)	S (%)
CQFP	Pot	CSAS-013	0.308	65.1	0.13	2.70	1.2	0.4
BFP	Pot	CSAS-001	0.31	49.7	0.19	1.62	3.0	0.3
BFP	Pot	CSAS-002	0.40	49.0	0.17	2.07	2.6	0.4
BFP	Pot	CSAS-004	0.405	62.4	0.16	2.03	2.1	0.4
BFP	Silic	CSAS-006	0.360	72.9	0.09	2.32	1.2	0.5
CQP	Phy	CSAS-012	0.223	249.6	0.08	1.02	0.8	0.4
CQP	Phy	CSAS-024	0.290	171.2	0.08	1.96	1.0	0.5
APV	Phy	CSAS-010	0.576	41.0	0.14	1.57	1.8	0.6
APV	Prop	CSAS-031	0.374	41.7	0.08	1.63	2.4	0.8

Source: Transmin, 2024.

13.3.4 Deleterious Elements

The CSAC-002 test yielded high pay metal recoveries, with low arsenic. Variability flotation achieved good pay metal recoveries with low arsenic except for sample CSAS-012, which contained 0.3% As in concentrate. Given that the Sur deposit as defined, contains low levels of arsenic, the results achieved in test CSAS-012 represent a small portion of the Sur material and would have negligible effect on overall concentrate quality.

13.3.5 Recovery Estimates

While cleaner testing was not completed, an overall recovery estimate for the Sur deposit is established based on the BFP recovery model derived from the 2012 testwork for cleaner flotation since material from Sur deposit has very similar metallurgical flotation performance. The recovery estimate applies a 96% cleaner recovery to complement the rougher results for the composites in the Sur deposit. The recovery estimates for the major composites are presented in Table 13-11.

Table 13-11: Metallurgical Recoveries

Recovered Metal	Lithology	Recovery Of Each Element	Maximum Recovered (%)	Minimum Recovered (%)
Copper	APV	91.4%	91.4	91.4
Copper	BFP	$4.87 \times \ln[Cu\ in\ feed\ (\%)] + 96.15$	93.5	86.4
Copper	CQP	81%	81.0	81.0
Arsenic	All	$29.4 \times \ln[As\ in\ feed\ (ppm)] - 92.9$	69.4	16.3
Gold	All	$12.94 \times \ln[Au\ in\ feed\ (g/t)] + 91.13$	69.6	58.4
Silver	All	$16.38 \times \ln[Ag\ in\ feed\ (g/t)] + 91.13$	59.7	43.8

14 MINERAL RESOURCE ESTIMATES

14.1 Introduction

Mineral Resource estimates were prepared by David G. Thomas, P.Geo., Resource Geologist of DKT Geosolutions Inc.

The Cañariaco Norte block model grade estimates for copper, gold and silver are unchanged from the November 2010 Mineral Resource estimate, prepared by the QP, who at the time was working for AMEC Americas Ltd (AMEC). Arsenic has been re-estimated using lithological controls and used a probabilistic approach considering an arsenic cut-off grade of 25 ppm as the threshold for low- and high-grade populations.

The 2024 Cañariaco Norte and Sur Mineral Resource estimates are updates (effective April 25, 2024) to the January 27, 2022, resources, based on the 2024 Optimized Mine Plan by Whittle which used more current metal prices, costs and metallurgical recoveries. The update gives less than a 10% increase to the resources globally and therefore are not considered material.

14.2 Cañariaco Norte

14.2.1 Geological Models

Alta Copper provided interpretations of lithological units on north–south and east–west vertical sections that were spaced 100 m apart. As the lateral extent of the lithological interpretations in some areas of the deposit are not fully defined by drill data, the interpretation was expanded beyond the limits of the available drill hole information to cover the block model extents. Although such extrapolation is considered to be reasonable, the QP recommends that additional drilling be performed to increase the level of confidence of the lithological interpretation in these areas.

Drilling completed post-2008 was added to the existing interpretations. Bench polygons were created and extruded the bench plan polygons to the mid-point distance to the adjacent polygons to create lithological solids.

East–west oriented, 100 m spaced vertical sections for alteration types and intensities were prepared by Alta Copper. These were simplified, and the resulting vertical polygons extended beyond the limits of the available drill hole information to cover the block model extents. During the next study phase, a full review of the alteration interpretation should be completed on vertical sections reconciled to bench plans.

14.2.2 Domaining

Estimation domains were defined following evaluation of statistical distributions of lithological and alteration units. A total of seven domains were created for copper and four domains for gold and silver (Table 14-1; refer to Table 7-1 for a description of the lithological unit codes).

Table 14-1: Copper, Gold, and Silver Estimation Domains

Element	Domain Code	Lithological Unit	Alteration Unit
Copper	10	VC, CQFP	Weak argillic
	20	VC, CQFP	All but weak argillic
	30	BFP, IBXP	Weak argillic
	40	BFP, IBXP	All but weak argillic
	50	CQP, IBXH, TBX	Weak argillic
	60	CQP, IBXH, TBX	All but weak argillic
	70	FP	Weak argillic, propylitic, intermediate argillic, phyllic
Gold and Silver	10	VC, CQFP, CQP, FP	All but advanced argillic
	20	VC, CQFP, CQP, FP	Advanced argillic
	30	BFP, IBXH, TBX, IBXP	All but advanced argillic
	40	BFP, IBXH, TBX, IBXP	Advanced argillic

These domains were tagged to blocks and back-tagged to composites and were used as the basis for matching samples and blocks during the estimation process.

A supergene-enriched domain was defined that was considered only for the estimate of copper grades. The supergene layer in the Cañariaco Norte deposit is thin and laterally discontinuous.

Copper mineralization at Cañariaco Norte is related to porphyry intrusions, breccias and potassic, phyllic, argillic and advanced argillic alteration; however, the limits of mineralization are not well known in the deposit. A mineral zonation model should be built for the next phase of study. A combination of lithology, alteration and mineral zones should provide a more robust support for the mineralized envelope and the definition of grade estimation domains. The copper, gold and silver estimation domains are shown in Table 14-1.

For arsenic grade estimation the BFP, CQFP, CQP, IBXP and VC rock types were split into northern and southern subdomains reflecting the semi-circular shape of the lithologies in plan view. The arsenic estimation domains are shown in Table 14-2.

Table 14-2: Arsenic Estimation Domains

Element	Lithology Domain Code	Lithological Unit	Sub-Domain
Low-grade (< 25 ppm AS)	5, 6, 7	FP, IBXH, TBX	
	10, 20, 30, 40, 80	BFP, VC, CQFP, CQP, IBXP	North
	15, 25, 35, 45, 85	BFP, VC, CQFP, CQP, IBXP	South
High-Grade (> 25 ppm As)	5, 6, 7	FP, IBXH, TBX	
	10, 20, 30, 40	BFP, VC, CQFP, CQP	North
	15, 25, 35, 45	BFP, VC, CQFP, CQP	South
	80	IBXP	North
	85	IBXP	South

14.2.3 Assay Capping

Probability plots were evaluated to define grade outliers for copper, gold, and silver by estimation domains. Outlier values typically occur in the upper 1% of the distribution. Copper and gold values were capped at the thresholds defined. For silver, outlier values were controlled by using a restricted search ellipse with a radius of 25 m x 25 m x 15 m during grade estimation. The grade thresholds for the different element outliers are shown in Table 14-3.

Table 14-3: Outlier Thresholds for Copper, Gold, and Silver

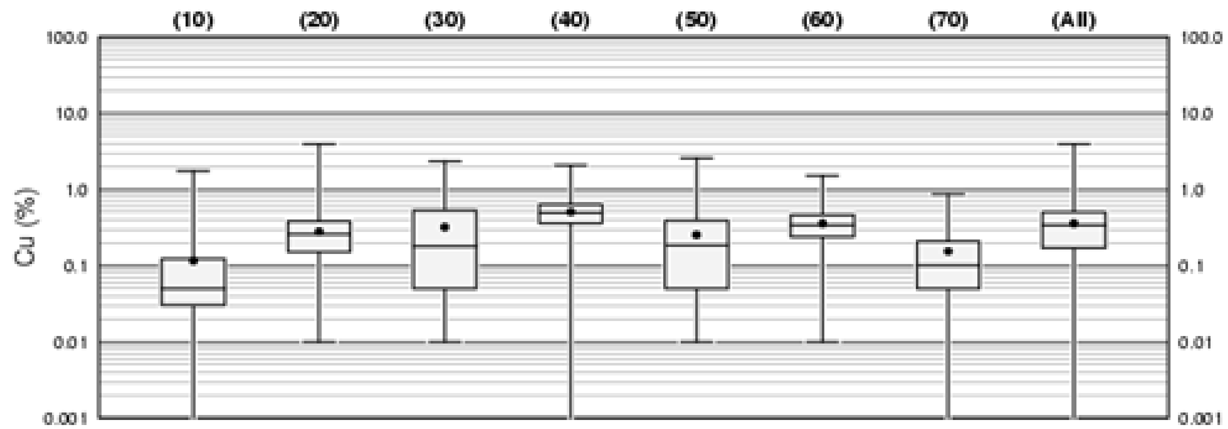
Element	Domain code	Threshold
Copper (%)	10	1.10
	20	2.00
	30	2.50
	40	2.50
	50	1.10
	60	1.20
	70	1.10
Gold (ppb)	10	-
	20	130
	30	530
	40	-
Silver (ppm)	10	12
	20	12
	30	17
	40	17

14.2.4 Exploratory Data Analysis

Exploratory data analysis comprised basic statistical evaluation of the 6 m composites for copper, gold, silver, and arsenic.

Box plots indicated that copper displayed similar grade distributions in those domains that had no weak argillic alteration. The biotite–feldspar porphyry and polylithic breccia units (BFP and IBXp) were preferentially mineralized and have higher average grades for all metals. Figure 14-1 shows the copper box plots for each domain.

Figure 14-1: Box Plot for Copper (%) by Domain



	(10)	(20)	(30)	(40)	(50)	(60)	(70)	(All)
Number of data	617	2681	985	3773	467	2192	574	11289
Mean	0.115	0.28	0.32	0.504	0.254	0.357	0.154	0.357
Maximum	1.73	3.97	2.31	2.06	2.56	1.5	0.87	3.97
Upper quartile	0.122	0.38	0.53	0.63	0.39	0.46	0.21	0.5
Median	0.05	0.26	0.18	0.49	0.186	0.34	0.1	0.34
Lower quartile	0.03	0.15	0.05	0.36	0.05	0.24	0.05	0.17
Minimum	0.0	0.01	0.01	0.0	0.01	0.01	0.0	0.0
Variance	0.0323	0.0364	0.1127	0.0503	0.0653	0.0309	0.0231	0.0615
CV	1.557	0.68	1.049	0.445	1.007	0.493	0.989	0.695
Skewness	4.271	3.677	1.522	0.782	2.33	0.825	1.611	1.274

Source: AMEC, 2010.

Coefficient of variation (CV) values for copper are low, around and below 1, in all domains but domain 10, which consists of volcanic rocks (VC) and the crowded quartz-feldspar porphyry (CQFP), where the CV is 1.5 for uncapped copper composite values.

Average gold grades are very low (below 0.1) in all domains, but box plots indicate that the combination of biotite feldspar porphyry (BFP) and the different types of breccia (IBxP, TBx, and IBxH) host most of the higher-grade gold mineralization where such mineralization is associated with any alteration type other than advanced argillic. All domains have low CV values for gold (less than 2), which confirms the low variability of the gold grades.

The same domains for silver were used for estimation as defined for gold because of their good correlation.

Silver displays similar average grades and grade ranges to gold. A higher spread of silver grade ranges was observed in domain 10.

Contact analyses were completed for copper (Table 14-4) and gold (Table 14-5) composite values, and defined soft, firm, and hard boundaries from this analysis. To represent the firm contacts, samples were shared only during the first estimation pass.

Table 14-4: Contact Matrix Defined for Copper Domains

Domains	10	20	30	40	50	60	70
10	0	1	0	1	1	1	1
20	1	0	21	0	0	0	1
30	0	21	0	1	21	1	21
40	1	0	1	0	1	0	1
50	1	0	21	1	0	1	21
60	1	0	1	0	1	0	1
70	1	1	21	1	21	1	0

Note: 0 = soft; 1 = hard; 21 = firm

Table 14-5: Contact Matrix Defined for Gold Domains

Domains	10	20	30	40
10	0	21	1	1
20	21	0	1	21
30	1	1	0	1
40	1	21	1	0

Note: 0 = soft; 1 = hard; 21 = firm

The QP capped arsenic composite values at 900 ppm As for the low-grade population and 3,000 ppm for the high-grade population. Boxplots indicate that higher arsenic values are present in lithologies BFP, VC, CQFP, CQP and IBXP and lower arsenic values are present in the FP, IBXH and TBX lithologies. Following splitting of the composites into lower and higher-grade populations using the 25-ppm probabilistic grade shell, soft contacts were defined between lithologies and a single hard contact was defined between the IBXP lithology and all other lithologies within the higher-grade population.

14.2.5 Variography

Sage2001 software was used to construct down-hole and directional correlograms for the estimation domains for copper, gold and silver. Domains that displayed soft boundaries were grouped.

For copper and gold, spherical models were used to fit the experimental correlograms. A combination of exponential and spherical models was used for silver, depending on domain.

For arsenic, the QP created correlograms for the 25 ppm as indicator threshold and subsequent grade correlograms for low- and high-grade domains sub-divided into northern and southern areas.

14.2.6 Density Assignment

Density values were assigned to blocks based upon the lithological codes. In the QP's opinion, these density values were reasonable for use in mineral resource estimation but recommended executing a continuous program of bulk density determinations from core samples, using preferably the same laboratory and determination procedures.

14.2.7 Composites

Although the nominal sample length for assays is 2 m, sample lengths in the Cañariaco Norte assay database range from 0.45 m to 100.3 m; such long intervals correspond to non-mineralized zones or to intervals of non-sampling. In order to normalize the weight of influence of each sample, the assay intervals were regularized by compositing the drill hole data into 6 m lengths using no geological or domain boundaries. The 6 m composites were backtagged using the lithological and alteration solid shapes and assigned estimation domain codes.

14.2.8 Estimation/Interpolation Methods

The block model consisted of regular blocks (15 m x 15 m x 15 m) and no rotation was used. The block size was chosen such that geological contacts were reasonably well reflected and support an open pit mining scenario.

Copper, gold and silver grades were estimated by estimation domains using ordinary kriging (OK) interpolation for the majority of domains. Inverse distance weighting to the second power (ID2) was used to interpolate gold and silver in domains 20 and 40 where variography was not considered sufficiently robust.

The process included:

- Grade estimation was completed in three passes;
- Sample sharing was based upon the matrix determined from contact profiles;
- Search orientations for all domains were based upon variogram orientations;
- A minimum of 3–11 and a maximum of 9–15 drill hole composites were required for estimation; this varied by element and estimation pass.

The QP estimated arsenic grades in three estimation passes using OK interpolation for all domains. In the first two passes, a minimum of four and maximum of 12 composites with a maximum of three composites per drill hole were required for estimation (i.e. a maximum of four holes was used). In the third pass a minimum of one and maximum of 12 composites with a maximum of two composites per hole (i.e. a maximum of six holes was used) were required for estimation. A hard boundary was used between lower-grade and higher-grade populations. Soft boundaries were used between lithologies except for the IBXP lithology in the higher-grade population where a hard boundary was used. Search orientations were based upon the directions of anisotropy shown by the variograms and were re-oriented in the northern and southern lithology subdomains.

14.2.9 Block Model Validation

The Cañariaco Norte block model was validated to ensure appropriate honouring of the input data. A nearest neighbour (NN) model was created to validate the OK model. The validation comprised:

- Detailed visual inspection of block grade versus composited data in section and plan view. The visual inspection of block grade versus composited data showed a good reproduction of the data by the model;
- A comparison between the OK and NN estimates was completed to check for global bias in the copper, gold, and silver grade estimates. Differences were within acceptable levels and no global biases were noted in the estimates;
- Swath plot validation compared average grades from OK and NN models along different directions. Except in areas where there is currently limited drilling, the swath plots indicated good agreement for all variables;
- The degree of smoothing due to kriging was assessed by considering change of support correction using Hermetian polynomials. Blocks from all copper domains were reviewed, and the results show a smoothing of 4% in copper grades but 7% more tonnes above the 0.2% Cu cut-off, resulting in a difference of only 1% in contained metal. The kriging smoothing is within acceptable ranges.

The QP validated the Cañariaco Norte arsenic block model to ensure appropriate honouring of the input data. A nearest neighbour (NN) model was created to validate the OK model. The validation comprised:

- Detailed visual inspection of block grade versus composited data in section and plan view. The visual inspection of block grade versus composited data showed a good reproduction of the data by the model;
- A comparison between the OK and NN estimates was completed to check for global bias arsenic grade estimates. Differences were within acceptable levels and no global biases were noted in the estimates;
- Swath plot validation compared average grades from OK and NN models along different directions. The swath plots indicated good agreement for all variables.

14.2.10 Classification of Mineral Resources

The following criteria were used to pre-classify blocks into categories as:

- Measured Mineral Resources: composites from a minimum of three drill holes within 75 m radius from a block centroid, or samples from two drill holes with the closest sample within 25 m of the block centroid.
- Indicated Mineral Resources: composites from a minimum of two drill holes within 110 m distance of the block centroid.
- Blocks that were not classified as Measured or Indicated but had a composite within 135 m from the block centroid were classified as Inferred. Remaining blocks were not classified. A semi-automated process was used to smooth the initial classification and avoid islands or isolated blocks of different categories.

14.3 Cañariaco Sur

14.3.1 Grade Shell

A grade shell using a 0.1% Cu threshold was created using implicit modelling. The grade shell correlates well with a surface soil geochemical anomaly for copper.

14.3.2 Assay Capping

Probability plots were evaluated to define grade outliers for copper, gold, silver and molybdenum within the grade shell estimation domain. Outlier values typically occur in the upper 1% of the distribution. Copper, gold, silver and molybdenum values were capped at the thresholds defined.

The grade thresholds for the different metals are shown in Table 14-6.

Table 14-6: Outlier Thresholds for Copper, Gold, Silver, and Molybdenum

Metal	Capping Threshold
Copper (%)	0.80
Gold (g/t)	1.0
Silver (g/t)	5.5
Molybdenum (%)	0.04

14.3.3 Exploratory Data Analysis

Exploratory data analysis comprised basic statistical evaluation of the 15 m composites for copper, gold, silver, and molybdenum.

CV values for copper, gold and silver are low, below 1. For molybdenum the CV value is 1.2 due to the very low but somewhat more erratic molybdenum values.

14.3.4 Variography

There are an insufficient number of composites to allow modelling of robust variograms.

14.3.5 Density Assignment

A density of 2.5 t/m^3 was assigned to all material below topography.

14.3.6 Composites

The nominal sample length for assays is 2 m at Cañariaco Sur. In order to normalize the weight of influence of each sample, the assay intervals were regularized by compositing the drill hole data into 15 m lengths using the grade shell boundary to split the composites. The 15 m composites were back-tagged with a code representing the grade shell wireframe shape.

14.3.7 Estimation/Interpolation Methods

The block model consisted of regular blocks (20 m x 20 m x 15 m) and no rotation was used. The block size was chosen such that geological contacts are reasonably well reflected and to support an open pit mining scenario.

Copper, gold, silver and molybdenum grades were estimated by estimation domains using ID2. The process included:

- Grade estimation was completed in two passes;
- The grade shell was considered as a hard boundary;
- The search ellipse was elongated in the vertical direction with dimensions of 250 m (vertical) x 125 m (east) x 125 m (north) in the first pass. The second pass used a search ellipse with dimensions of 500 m (vertical) x 250 m (east) x 250 m (north);
- A minimum of three and a maximum of eight drill hole composites were required for estimation in the first pass; a minimum of two and a maximum of eight drill hole composites were required for estimation in the second pass.

14.3.8 Block Model Validation

The Cañariaco Sur block model was validated to ensure appropriate honouring of the input data. A nearest neighbour (NN) model was created to validate the ID2 model. The validation comprised:

- Detailed visual inspection of block grade versus composited data in section and plan view. The visual inspection of block grade versus composited data showed a good reproduction of the data by the model;
- A comparison between the ID2 and NN estimates was completed to check for global bias in the copper, gold, silver and moly grade estimates. Differences were within acceptable levels and no global biases were noted in the estimates;
- Swath plot validation compared average grades from ID2 and NN models along different directions. Except in areas where there is currently limited drilling, the swath plots indicated good agreement for all variables.

14.3.9 Classification of Mineral Resources

All blocks falling within the grade shell were classified as Inferred Mineral Resources. The QP reviewed surface geological mapping, surface rock chip and trench samples, surface geochemistry and geophysical surveys when classifying the estimate. The surface geology and surface geochemistry allow the inference of geological and mineralization continuity between drill holes. The geological model at Cañariaco Sur is relatively simple without the multiple phases of breccias present at Cañariaco Norte.

14.4 Reasonable Prospects for Eventual Economic Extraction

The classified blocks were assessed for reasonable prospects of eventual economic extraction by applying conceptual economic parameters for potential open pit mining methods. Mining and process costs, as well as process recoveries were defined from on-going studies for the Cañariaco Norte deposit.

A pit shell, optimized using a Lerchs–Grossmann algorithm based on the parameters listed in Table 14-7, was defined. The metal prices used represent long-term estimates for Mineral Resources.

Table 14-7: Optimization Parameters for Resource Pit Shell

Mining Costs	Unit	Value
Mining Cost	US$/t mined	1.93
Process Cost, Tailings Management and G&A Cost	US$/t milled	7.25
Total Mineralization Based Costs	US$/t milled	9.18
Cu Price	US$/lb	4.25
Au Price	US$/oz	1,950.00
Ag Price	US$/oz	23.30
Selling Cost	US$/lb	0.40
Arsenic Penalty at Norte only	US/t milled	0.60
Cu Recovery	%	88
Au Recovery	%	63
Ag Recovery	%	58
Overall Pit Slope(s)	Degree	40-48 degrees at Norte, 45 degrees at Sur

Note: No dilution or mining loss adjustments were applied. The overall pit slope assumption is based on limited geotechnical data and will likely change when results from the planned geotechnical drilling become available.

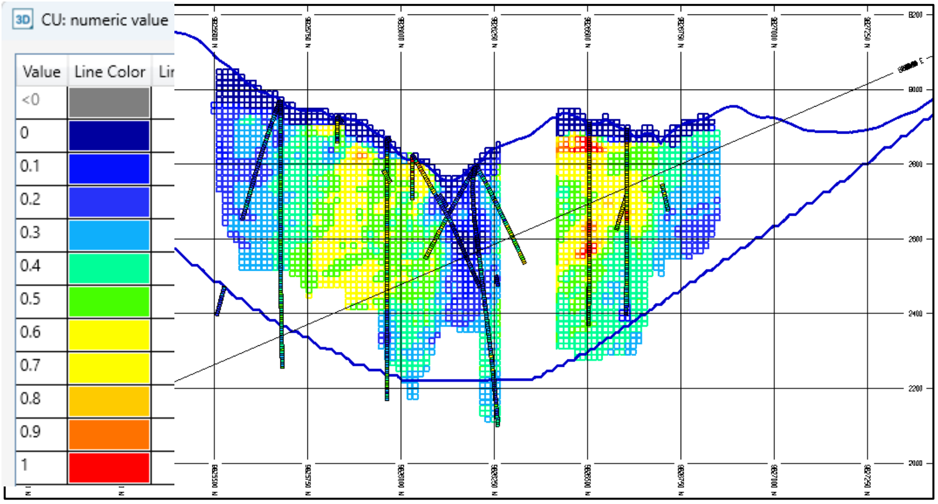
A marginal cut-off of 0.09% Cu (Table 14-8) was defined. However, Alta Copper selected a cut-off grade of 0.10% Cu for Mineral Resource reporting.

Table 14-8: Marginal Cut-Off Calculation

Parameters	Value
Processing Cost (US$/t)	7.25
Recovery (%)	87
Price (US$/lb)	4.25
Payable	96.5%
Minimum Deduction	1%
Refining Charge ($/t Cu)	165.35
Treatment Charge ($/t concentrate)	75.00
Transportation (S/t concentrate)	141.50
Cut-Off Cu (%)	0.09

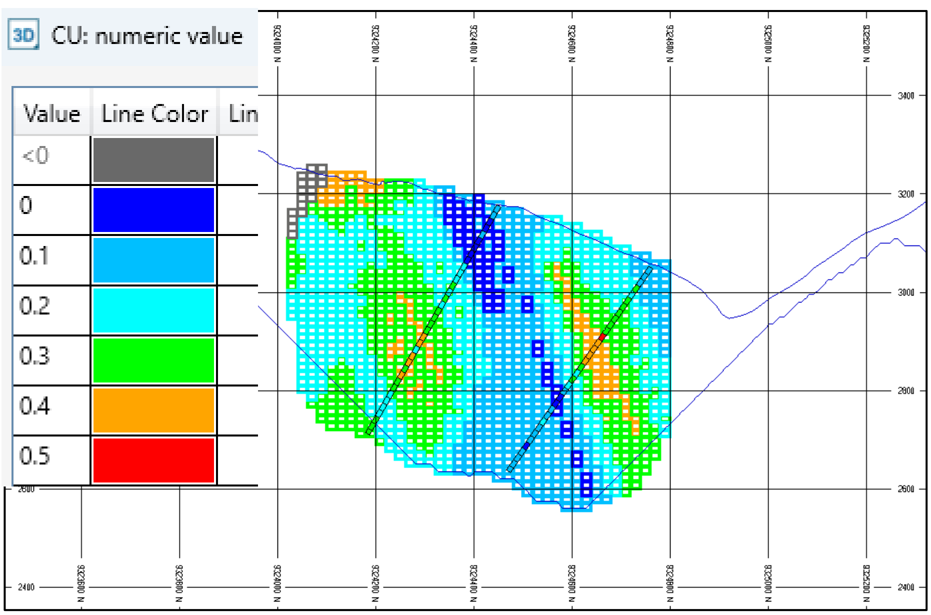
Figure 14-2 and Figure 14-3 show cross-sections displaying the resource constraining pit shells, block models and composites for Cañariaco Norte and Cañariaco Sur respectively.

Figure 14-2: Cañariaco Norte North–South Section 783,420 E Showing Resource Constraining Pit Shell, Block Model and Composites



Note: Figure prepared by DKT Geosolutions Inc., 2024.

Figure 14-3: Cañariaco Sur North-South Section 689,890 E Showing Resource Constraining Pit Shell, Block Model and Composites



Note: Figure prepared by DKT Geosolutions Inc., 2022.

14.5 Mineral Resource Statement

Mineral Resources for the Project were classified using the 2014 CIM Definition Standards.

Mineral Resources for Cañariaco Norte are tabulated in Table 14-9. The Qualified Person for the Mineral Resource estimates is David Thomas, P.Geo. Mineral Resources have an effective date of April 25, 2024 and are an update to the January 2022 resources, based on the 2024 Optimized Mine Plan by Whittle which used more current metal prices, costs and metallurgical recoveries. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Table 14-9: Mineral Resource Statement for Cañariaco Norte at a 0.10% Cu Cut-off Grade

		Grade				Contained Metal		
Category	Tonnage (Mt)	Cu Eq. (%)	Cu (%)	Au g/t)	Ag (g/t)	Copper (Blb)	Gold (Moz)	Silver (Moz)
Measured	433.3	0.47	0.43	0.07	1.9	4.07	1.00	26.15
Indicated	693.3	0.38	0.34	0.05	1.5	5.26	1.17	34.22
Measured + Indicated	1,126.6	0.42	0.38	0.06	1.7	9.33	2.16	60.37
Inferred	416.3	0.29	0.26	0.04	1.3	2.41	0.52	16.90

Notes: **1.** The Mineral Resources estimate has an effective date of April 25, 2024. The Qualified Person for the estimate is David Thomas, P.Geo., of DKT Geosolutions Inc. **2.** The Mineral Resources were reported using the definitions set out in the 2014 CIM Definition Standards. **3.** Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. **4.** The Mineral Resources are reported within a constraining Lerchs Grossmann pit shell developed using Whittle software using: **a.** A copper price of US$4.25/lb; **b.** Mining cost of US$1.93/t; **c.** A combined processing, tailings management and G&A cost of US$7.25/t; **d.** Variable pit slope angles ranging from 40 to 48 degrees; **e.** An average copper process recovery of 88%; **f.** Copper concentrate smelter terms: US$75/dry metric tonne treatment charge, US$0.075/lb refining charge and 96.5% payable; **g.** Estimated concentrate shipping costs of US$138.95/wet metric tonne of concentrate; h. An average arsenic penalty of $0.60/t processed. **5.** Copper equivalent grades including contributions from gold and silver, were estimated using metal prices of copper: US$4.25/lb, gold: US$1, 950 /oz, and silver: US$23.00/oz), metallurgical recoveries of copper: 88%, gold: 63%; silver: 58%, and smelter payables of copper: 96.5%, gold: 93%, silver: 90%). Copper grade equivalent calculation: CuEq% = (Cu % + ((Au grade x Au price x Au recovery x Au smelter payable%) + (Ag grade x Ag price x Ag recovery x Ag smelter payable)/(22.0462 x Cu price x 31.1035 g/t x Cu recovery x Cu smelter payable%). **6.** All figures are rounded to reflect the relative accuracy of the estimate. Totals may not sum due to rounding as required by reporting guidelines. **7.** The contained metal figures shown are in situ.

The sensitivity of the Cañariaco Norte mineral resource to a reduction or increase in copper cut-off grades is included as Table 14-10, with the base case in grey.

Mineral Resources estimated for Cañariaco Sur are provided in Table 14-11. The base case is highlighted. Other cases are shown as sensitivities to changes in the copper cut-off grade.

Table 14-10: Mineral Resource Statement for Cañariaco Norte Showing Sensitivity to Various Cut-offs (base case in grey)

Cut-off Cu (%)	Tonnage (Mt)	Cu Eq. (%)	Grade			Contained Metal		
			Cu (%)	Au (g/t)	Ag (g/t)	Copper (Blb)	Gold (Moz)	Silver (Moz)
Measured								
0.1	433.3	0.47	0.43	0.07	1.9	4.07	1.00	26.15
0.15	423.2	0.48	0.43	0.07	1.9	4.04	0.98	25.69
0.20	406.6	0.49	0.44	0.07	1.9	3.98	0.96	24.94
0.30	338.5	0.53	0.48	0.08	2.0	3.59	0.85	21.64
Indicated								
0.1	693.3	0.38	0.34	0.05	1.5	5.26	1.17	34.22
0.15	659.6	0.39	0.35	0.05	1.6	5.16	1.14	33.22
0.20	587.4	0.41	0.38	0.06	1.6	4.88	1.06	30.68
0.30	407.1	0.47	0.43	0.06	1.8	3.88	0.83	23.23
Measured + Indicated								
0.1	1,126.6	0.42	0.3755	0.06	1.7	9.33	2.16	60.37
0.15	1,082.8	0.43	0.3855	0.06	1.7	9.20	2.12	58.90
0.20	994.0	0.45	0.4043	0.06	1.7	8.86	2.02	55.62
0.30	745.6	0.4993	0.4543	0.07	1.9	7.47	1.68	44.87
Inferred								
0.1	416.3	0.29	0.26	0.04	1.3	2.41	0.52	16.90
0.15	355.8	0.31	0.29	0.04	1.3	2.24	0.48	15.16
0.20	254.9	0.36	0.33	0.05	1.5	1.85	0.39	12.17
0.30	130.7	0.45	0.41	0.06	1.8	1.19	0.23	7.41

Note: Footnotes to apply to this table as well.

Table 14-11: Mineral Resource Statement for Cañariaco Sur at a 0.10% Cu Cut-off Grade (base case in grey)

Cut-off Grade Cu (%)	Tonnes (Mt)	Grade					Contained Metal			
		Cu Eq. (%)	Cu (%)	Au (g/t)	Ag (g/t)	Mo (ppm)	Copper (Blb)	Gold (Moz)	Silver (Moz)	Molybdenum (Mlb)
Inferred										
0.1	474.1	0.29	0.24	0.09	1.2	23	2.52	1.34	17.61	24.04
0.15	418.0	0.31	0.26	0.09	1.20	24	2.37	1.25	16.08	22.12
0.20	311.0	0.34	0.28	0.10	1.26	23	1.95	1.02	12.57	15.77

Notes: 1. The Mineral Resource estimate has an effective date of April 25, 2024. The Qualified Person for the estimate is David Thomas, P.Geo., of DKT Geosolutions Inc. 2. The Mineral Resources were reported using the definitions set out in the 2014 CIM Definition Standards. 3. A single 0.1% Cu grade shell domain was constructed using implicit modelling. 4. Raw drill hole assays were composited to 15 m lengths broken at domain boundaries. 5. Capping of high grades was considered necessary and was completed on assays prior to compositing. Copper assays were capped to a 0.8% threshold and gold assays were capped at a threshold of 1 g/t. 6. Block grades for gold were estimated from the composites using ordinary kriging interpolation into 20 x 20 x 15 m blocks coded by the 0.1% Cu grade shell. 7. The Mineral Resource is reported above a 0.10% Cu cut-off grade. Additional cut-off grades are shown for sensitivity purposes only. 8. A dry bulk density of 2.5 g/cm3 was used for all material. 9. The Mineral Resources are reported within a constraining Lerchs Grossmann pit shell developed using Whittle software using: a. A copper price of US$4.25/lb; b. Mining cost of US$1.93/t; c. A combined processing, tailings management and G&A cost of US$7.25/t; d. 45-degree pit slope angles; e. An average copper process recovery of 87%, an average gold process recovery of 59%, an average silver recovery of 46%; f. Copper concentrate smelter terms: US$75/dry metric tonne treatment charge, US$0.075/lb refining charge and 96.5% payable; g. Estimated concentrate shipping costs of US$138.95/wet metric tonne of concentrate; arsenic grades are below the threshold that would incur a smelter penalty. 10. Copper equivalent grades including contributions from gold, silver, and molybdenum, were estimated using metal prices of copper: US$4.25/lb, gold: US$1,950/oz, silver: US$23.00/oz and molybdenum: US$11.00/lb; metallurgical recoveries of copper: 88%, gold: 63%; silver: 58% and molybdenum: 60% and smelter payables of copper: 96.5%: gold: 93%; silver :90% and molybdenum: 100%. Copper grade equivalent calculation: Cu Eq% = (Cu % + ((Au grade x Au price x Au recovery x Au smelter payable%) + (Ag grade x Ag price x Ag recovery x Ag smelter payable%) + (Mo grade x Mo price x Mo recovery x Mo smelter payable%))/(22.0462 x Cu price x 31.1035 g/t x Cu recovery x Cu smelter payable%). 11. Preliminary metallurgical test work was conducted to assess grindability and flotation recoveries on representative samples of predominant lithologies and alterations in Cañariaco Sur. Given the similarities with Cañariaco Norte, the same average recoveries were applied for conceptual pit shell generation and CuEq estimations in Cañariaco Sur. 12. All figures are rounded to reflect the relative accuracy of the estimate. Totals may not sum due to rounding as required by reporting guidelines. 13. The contained metal figures shown are in situ.

14.6 Reconciliation to Previous Mineral Resource Estimate (2022)

A comparison to the previous Mineral Resource estimate (Table 14-12) shows that the Measured and Indicated categories are very similar. The Inferred category at Norte was reduced as a result of the pit optimization parameters used. The small change in Measured and Indicated categories at Norte demonstrates that the Mineral Resource is robust with respect to changes in metal prices, costs and smelter terms.

The Inferred category Mineral Resource at Sur increased significantly due to increased metal prices. On a project-wide basis the change to the Mineral Resource estimate is not material.

Table 14-12: Resources Comparison 2024 to 2022

Area	Category	Tonnes %	CuEQ %	Cu %	Au (g/t)	Ag (g/t)	Moly (ppm)	Cu B (lbs)	Au (M Ozs)	Ag (M Ozs)	Mo (M lbs)
Norte	Measured	2.3	-1.6	-0.9	2.3	-1.1	-	0.7	1.7	1.7	-
	Indicated	3.4	-2.7	-4.4	4.8	-4.0	-	0.1	0.6	1.5	-
	Measured + Indicated	3.0	-1.1	-3.7	-0.4	-1.9	-	0.4	1.1	1.6	-
	Inferred	1.4	-9.0	-9.5	-2.5	-9.7	-	-9.4	-5.2	-6.6	-
Sur	Inferred	23.3	-9.4	-5.9	-11.9	-3.4	4.5	13.7	13.7	17.3	27.1

14.7 Factors That May Affect the Mineral Resource Estimate

Areas of uncertainty that may materially impact the Mineral Resource estimates include:

- Long-term commodity price and exchange rate assumptions;
- Changes to the assumptions used to generate the gold grade cut-off grade;
- Changes in local interpretations of mineralization geometry and continuity of mineralized zones;
- Changes to geological and mineralization shape and geological and grade continuity assumptions;
- Density and domain assignments;
- Changes to geotechnical, mining and metallurgical recovery assumptions;
- Changes to the input and design parameter assumptions that pertain to the conceptual pit constraining the estimates; and
- Assumptions as to the continued ability to access the site; ability to retain mineral titles; ability to obtain surface rights; ability to obtain authorization from the ANA to use water sufficient to support mine construction and operations; ability to obtain environment and other regulatory permits, and ability to obtain the social license to operate.

In particular, any changes to the slope angle of the pit wall as a result of more detailed geotechnical information would affect the pit shell used to constrain the Mineral Resources.

14.8 Comments on Mineral Resource Estimates

Mineral Resources are reported in accordance with the 2014 CIM Definition Standards.

There are no other environmental, legal, title, taxation, socioeconomic, marketing, political or other relevant factors known to the QP that would materially affect the estimation of Mineral Resources that are not discussed in this Report.

15 MINERAL RESERVE ESTIMATES

This section is not relevant to the report.

16 MINING METHODS

Readers are cautioned that the 2024 PEA Update is preliminary in nature. In addition to Measured and Indicated Resources it also includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the 2024 PEA Update will be realized.

Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

16.1 Overview Process Design

The Cañariaco deposits are large, near surface, bulk mineable porphyry copper deposits located in the Lambayeque region of northern Peru. This study incorporates a second pit, Cañariaco Sur, along with Norte. Open-pit mining was selected as the extraction method considering the size of the resource, grade tenor, grade distribution, and proximity to topographic surface for the deposit. The opinion of the QP is that open-pit mining offers the most viable approach for development.

Mill feed and waste will be drilled, blasted, and loaded by electric hydraulic face shovels and front-end loaders from 15 m high benches. Haul trucks will haul the above cut-off mineralized material to the mineralized material crusher, or short-term stockpile as required. Waste will be hauled during pre-production to the CPSF, and during production to the waste crusher to be located near the mineralized material crusher. During production, crushed waste will be conveyed to the plant site where it will be combined with dewatered tails and stacked at the CPSF. The mine plan considers a 120 kt/d (43.8 Mt/a) processing rate. The peak mining capacity is ~130 Mt/a.

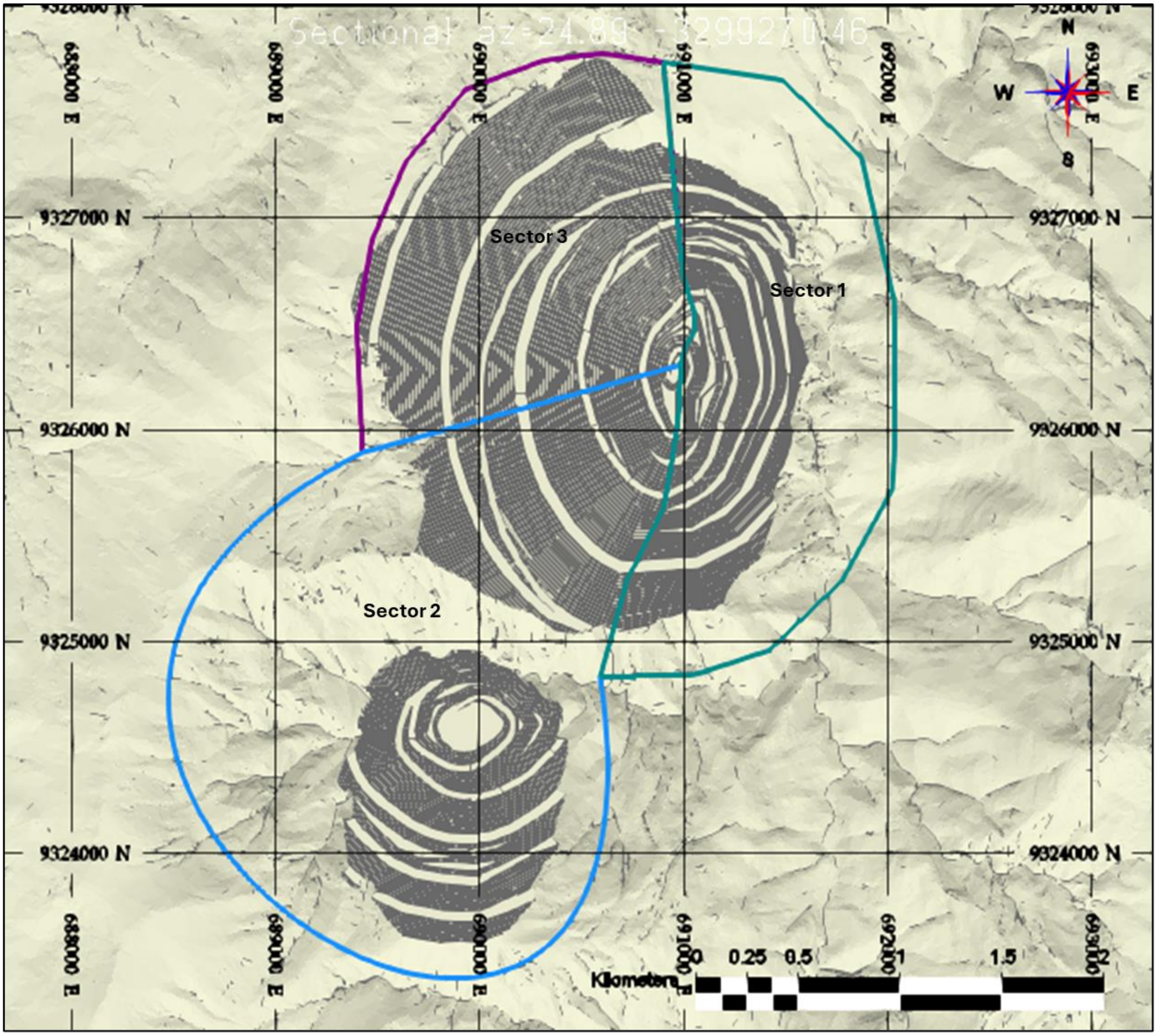
The mine planning was performed using metal prices of US$3.85/lb Cu, US$1,850/oz Au and US$23/oz Ag. Measured, Indicated, and Inferred Mineral Resources were considered for processing. The mine plan used the resource block model described in Section 14. AGP's mine planning was performed using Hexagon's MinePlan 3D software package.

16.2 Geotechnical Considerations

Knight Piésold completed a preliminary level slope stability evaluation for the proposed Cañariaco Norte open pit, which is considered suitable for PEA level mine planning. Preliminary pit slope angles were determined in accordance with a review of selected geotechnical information and corresponding data analysis. No detailed geotechnical data is currently available for Cañariaco Sur, so the parameters from the SW Geotechnical Sector (Sector 2) from Norte were used for the Sur design. This sector is closest to Sur.

The geotechnical domains are shown with the 2024 PEA ultimate pit design in Figure 16-1.

Figure 16-1: Geotechnical Domains



Source: AGP, 2024.

The recommended pit slope angles are shown in Table 16-1.

Table 16-1: Recommended Pit Slope Activities

Pit Design Sector	Pit Stage	Max Wall Height (m)	Bench Height (m)	Bench Face Angle (º)	Bench Width (m)	Inter-ramp Angle (º)	Max Inter-ramp Slope (m)	Overall Slope Angle (º)
East (Sector 1)	Starter	250	15	70	8.0	48	250	45
	Final	450			8.0	48		45
Southwest (Sector 2)	Starter	450	15	65	8.0	45	250	42
	Interim	700			9.0	43		39
	Final	1000			11.0	40		36
Northwest (Sector 3)	Starter	450	15	70	9.5	45	250	42
	Interim	700			10.5	43		39
	Final	1000			12.5	40		36

The recommended pit slope angles are considered reasonable for the early-stage pit phases within the central area of the deposit. Slope configurations for the ultimate pit are largely extrapolated from limited data and experience with other large open pit operations. A detailed review of the existing drill core is required to assess geotechnical drilling requirements. Additional geotechnical analysis will be required to support FS level pit slope designs for the Cañariaco project.

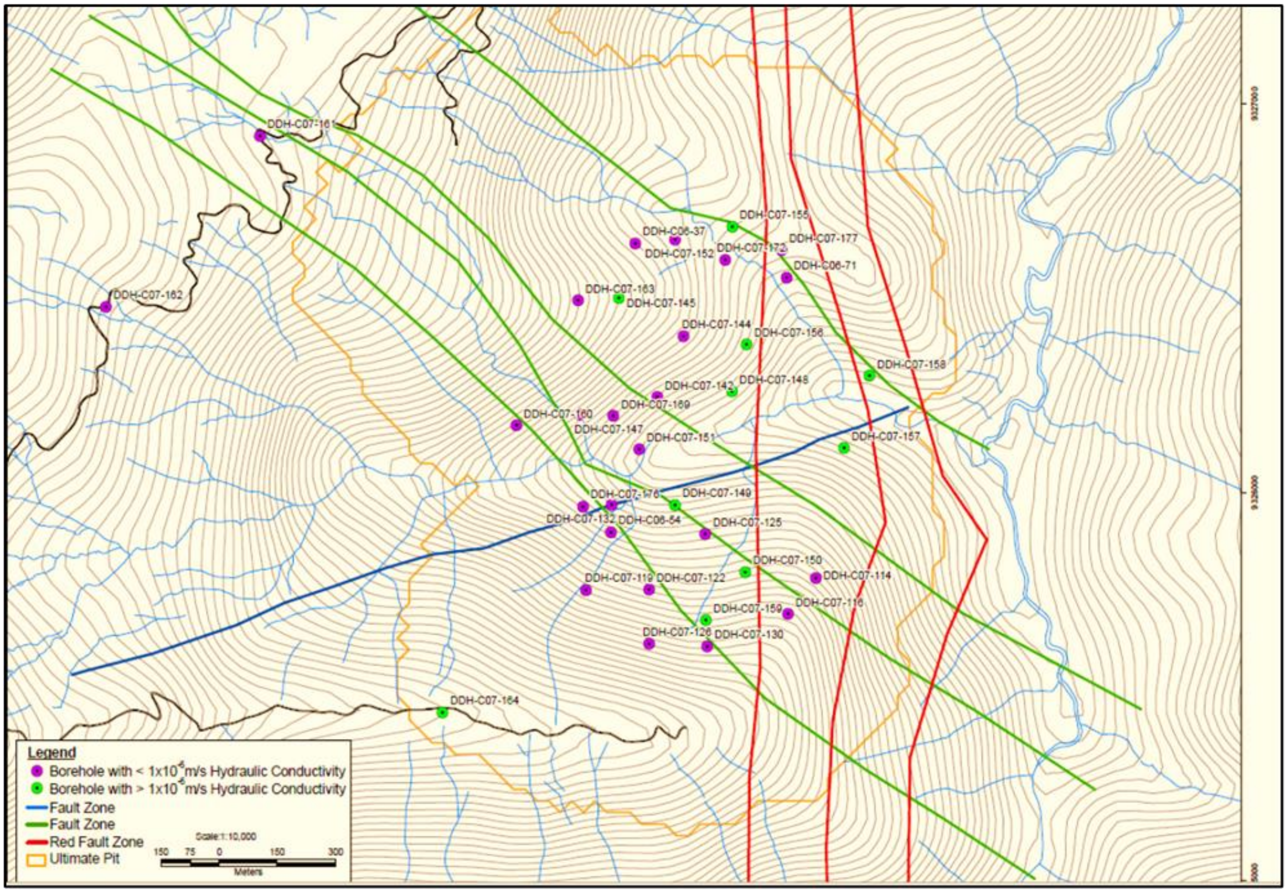
16.3 Hydrogeological Considerations

Hemmera reviewed previous work by AMEC (2010) and provided a preliminary assessment of the pit hydrological conditions, resulting in estimated quantities of pit contact water to be managed during the life of mine.

AMEC (2010) completed a 2D seepage model of the proposed pit using hydraulic conductivity (K) measurements from the mineralized zone. A uniform K of 5E-08 m/s was assumed throughout the pit to estimate flux along a section through the proposed 2-km-long pit. Maximum pit inflows were estimated at 1,700 m^3/day after 16 years of mining.

The significant pit high wall (>900 m) means that active dewatering will be required to stabilize the slope. AMEC noted that geological structures should be used to site dewatering wells since the average K measurements (1E-07 m/s) are too low for vertical wells. Packer testing (GWI, 2008) identified elevated K (> 1E-06 m/s) associated with northwest–southeast and north–south aligned faults coincident with drainages (Figure 16-2).

Figure 16-2: Packer Testing Locations and Fault Alignment



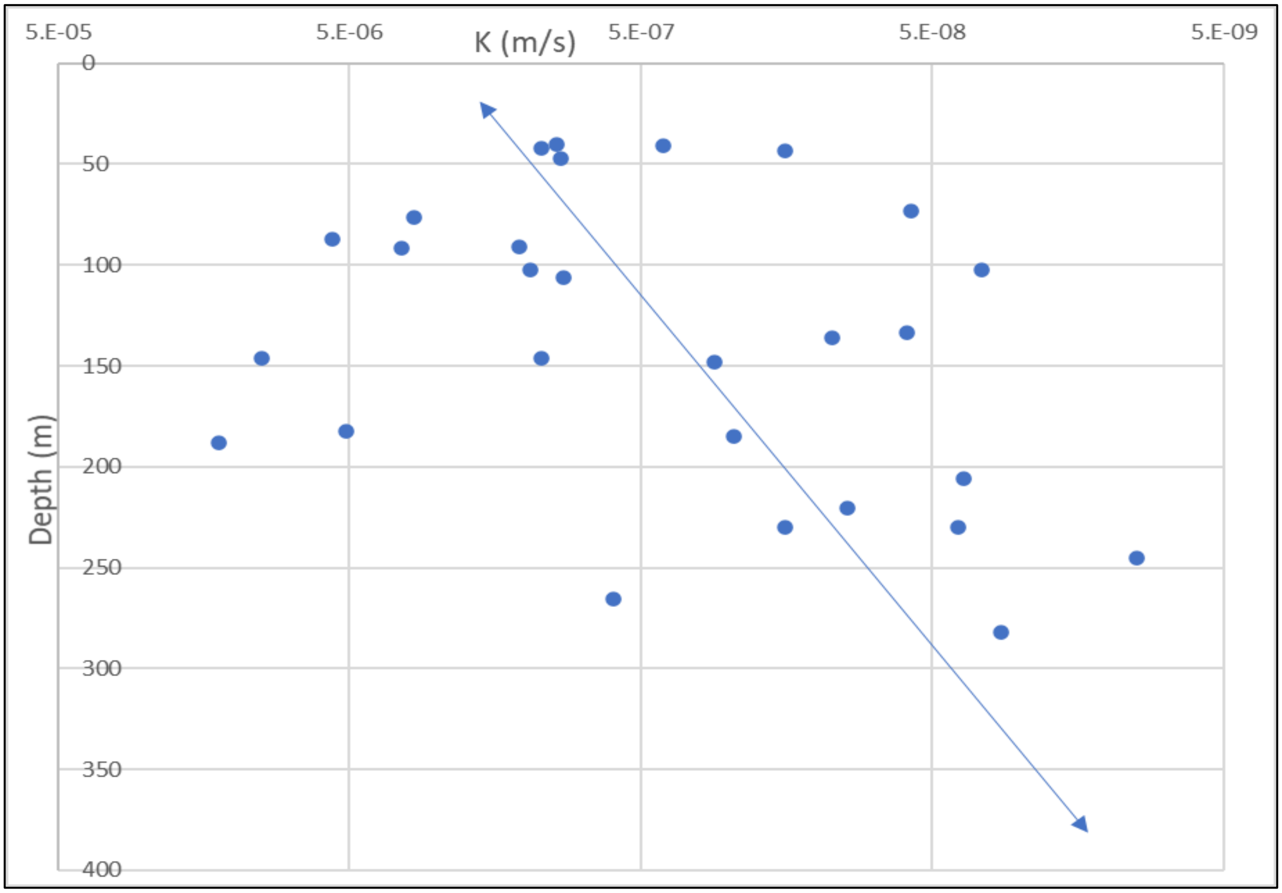
Source: GWI, 2008.

The K measurements were predominantly in the top 200 m and do not necessarily represent K at greater depth. Additional packer testing on the pit perimeter and at depth is recommended.

The proposed pit extent shown in Figure 16-2 is similar to the 2024 PEA ultimate pit design and those developed in previous studies.

Review of the K data with depth (Figure 16-3) and the average mining depth of proposed pit of 400 m, suggest that using an average K of 5E-08 m/s to represent the average pit K is on the low side; the mean for the data set (n = 43) is 3.8E-07m/s. Although most of these data are within 200 m from surface, using a higher K for inflow predictions is recommended. The data suggests an order of magnitude decrease in K every 100 m from surface.

Figure 16-3: Bedrock Hydraulic Conductivity Variation with Depth



Source: AMEC, 2010. Note: n = 43.

Figure 16-3 indicates that an average K at around 200 m depth is 1E-07 m/s or twice that used previously for pit inflow estimates.

The rock at the centre of the pit consists predominantly of breccias and granitic porphyry whereas the edges are mostly intrusive porphyry and volcanic rocks (AMEC, 2010). Of the 13 packer tests with K >1E-06 m/s, six were in intrusive rocks, five in breccias and two in volcanic rocks. Although this may lead to the conclusion that the deposit has greater K than the surrounding country rock, this may not necessarily be the case since the latter were tested less frequently due to topographical constraints (refer to Figure 16-2).

An evaluation of the (passive) inflow estimates was carried out for the proposed ultimate pit using an analytical equation that estimates lateral inflow through the side of the pit as well as through the bottom of the pit (Marinelli and Niccoli, 2000). Assumptions include:

- Average K of 1E-07 m/s for lateral inflow and 1E-08 m/s for inflow through the base of the open pit;

- Recharge equal to 100 mm of (10% of average precipitation);
- Pit radius of 750 m; and
- Groundwater level at 50 m below surface.

Pit runoff was included in this assessment as it has to be managed as contact water (either incorporated into the mill or treated and discharged). Although annual lake evaporation is 500 mm (Knight Piésold, 2012), the actual evaporation is likely less because of the high humidity in the rainy season and low or no runoff in the dry season. The combined ground water inflow and meteoric run-off water to be managed during the mine life is shown in Table 16-2.

Table 16-2: Combined Contact Water

Timeframe	Groundwater Inflow		Pit Runoff		Total Contact Water	
	(m^3/h)	L/s	(m^3/h)	L/s	(m^3/h)	L/s
Years -3 to +1	63	18	114	32	177	49
Years 2 to 10	211	59	243	67	454	126
Years 11 to 28	211	59	200	56	411	115

Note: Average conditions:1,000 mm precipitation and 10% of mean annual precipitation recharge (MAP). Reduced groundwater flows in years -3 to +1 reflect the smaller pit development area at the time.

An allowance for horizontal drains was included in the mine operating cost estimate. The possible need for vertical dewatering wells has not been considered at this time.

The Sur deposit has not had the detailed hydrogeological work completed. For the PEA study it is assumed that conditions are similar and the same pumping requirements applied. The Sur pit will also have horizontal drain holes to promote slope depressurization.

16.4 Block Model Capture

The resource block model described in Section 14 was provided in native Hexagon MinePlan 3D (formerly MineSight) format. AGP performed the mine planning activities using MinePlan 3D. Visual inspections of the model were performed, spot checking lithology codes, grades and bulk density values against provided rock type wireframes and the topographic surface.

16.5 Dilution and Mining Loss Adjustments

To address accuracy limitations with respect to grade control delineation, blast movement and digging, dilution and mining loss adjustments were made in the block model. The mineralization is generally gradational across the mineralization/waste contacts. Due to this gradational nature, a diluting method, end slice swap, was applied. As opposed to diluting methods that add a pure dilution skin, the end slice swap approach models a situation where there is a balance between dig face dilution gain and mill feed loss. In plan view, 2 m-wide slices were taken from all four sides of each 15 x 15 m block and swapped with the neighbouring slice from each adjacent block. Volumetrically, the dilution gain and the dilution loss at the dig face were equal. As the blocks had varying bulk densities, a new diluted block density was calculated on a volume-weighted basis. The diluted grades were calculated on a tonnage-weighted basis. The resulting average percentage reduction in grades from the undiluted mineral resource grades were 0.5%, 0.22% and 0.19% for copper, gold and silver respectively. The diluted bulk density and grades were then used for all NSR calculations and production reporting. An additional 2% of mining loss was applied to account for carry back and mineralization routing errors.

In light of the minor variation noted in the dilution calculation for Norte, with the inclusion of the Sur deposit no dilution was added as the gradational contact was expected to yield similar results for dilution. The material loss of 2% for mill feed was applied.

16.6 Pit Optimization

The open pit ultimate size and phasing shape guidance were determined by generating nested revenue factor (RF) Lerchs–Grossmann (LG) pit shells, utilizing various input parameters including estimates of the expected mining, processing and general and administrative (G&A) costs, as well as metallurgical recoveries by rock type, pit slopes, and reasonable long-term metal price assumptions. The mining costs were estimated based on first principles cost buildup for bulk mining in 15-m benches using vendor provided equipment pricing, consumables costs and labour costs from previous work completed by AGP, including a depth and distance hours-based model by WCL/AGP. Process and G&A costs were provided by Ausenco based on earlier internal studies and incorporated into an activity-based cost model including comminution and reagent cost by geomet type.

16.6.1 Pit Slopes

The pit slopes used for LG shell generation were based on the detailed slope design parameters provided by Knight Piésold and discussed in Section 16.1. Overall slope angles were estimated for each sector based on the depth of an assumed likely ultimate pit shell, and anticipated number of pit ramps crossing each sector. The resulting overall slopes used were 37.6º, 36.0º, and 36.4 º for the east, southwest and northwest sectors respectively.

Select shells were also generated for starter pit guidance, using the specific steeper starter pit design guidance mentioned in Section 16.2. Using a similar calculation, the resulting starter pit overall slopes used were 37.6º, 39.4º, and 38.1º for the east, southwest and northwest sectors respectively.

Pit slopes for Sur were based on the southwest geotechnical sector for Norte, which is the closest sector to the Sur pit.

16.6.1.1 Rock Type Independent Parameters

The metal prices and downstream economic parameters applied are shown in Table 16-3 through Table 16-5.

Table 16-3: Metal Prices for Pit Optimization

Metal	Price (US$/lb)
Copper	3.85
Gold	1,850
Silver	23

Table 16-4: Smelter Terms

Copper Terms and Costs	UOM	Amount
Payability	%	96.50
Minimum Deduction Unit (percentage point of con grade grade)	%	1.00
Refinery Treatment Charge Copper	$/payable lb	0.075
Refinery Treatment Charge Copper per payable copper	$/dmt	165.35
Concentrate Treatment Charge	$/dmt of con	75.00
Transportation - Land; Mine to Port (included below)	$/wmt of con	25.50
Transportation - (All concentrate logistic costs)	$/wmt of con	116.00
Moisture Content	%	9.00
Gold Terms and Costs	**Conc ppm <**	**Amount**
Refinery Payable (%)	1	90.00
Refinery Payable (%)	3	92.00
Refinery Payable (%)	5	93.00
Refinery Treatment Charge Gold	payable $/troz	5.00
Silver Terms and Costs	**Conc ppm >**	**Amount**
Refinery Payable (%) Greater Than -->	30	90.00
Refinery Treatment Charge Silver	$/payable troz	0.40
Refinery Treatment Charge Silver	$/gr	0.0129
Arsenic Penalty	**Hurdle (%)**	**Penalty ($/0.1%)**
As increment (%)	0.1	
Penalty for As greater than -->	3	9.00
Penalty for As greater than -->	0.5	5.50
Penalty for As greater than -->	0.2	2.50

Table 16-5: Concentrate Logistics Costs

Concentrate Logistics Costs	Amount
Conc. Transport to Port ($/WMT)	25.5
Port Costs ($/WMT)	30
Insurance ($/WMT)	2.5
Ocean Freight ($/WMT)	75
Transit Loss (%)	0.5

Note: WMT = wet metric tonne.

The parameters presented in Table 16-3 to Table 16-5 were based on values used in earlier internal studies. The metal prices are different from those presented in Sections 21 and 22 where higher metal prices were applied. The final metal prices in the financial model are higher. As such, the NSR block estimate has a small upside built into it.

16.6.2 Metallurgical Recoveries

Metallurgical recoveries were modeled by Ausenco as a function of head grade for each metal and rock type. The metallurgical recovery relationships provided are shown in Table 16-6.

Table 16-6: Metallurgical Recoveries by Rock Type

Lithology	Recovery			
	Cu%	As%	Ag%	Au%
BFP	$91.5*(1-EXP(Cu\%_{feed}*-14))$	$14.295 \times Cu_{Recovery}^{0.4139}$	$2.138 \times Cu_{Recovery}^{0.7708}$	$0.2551 \times Cu_{Recovery}^{1.2586}$
IBxH	$91.5*(1-EXP(Cu\%_{feed}*-14))$	$6E-08 \times Cu_{Recovery}^{4.516}$	$4E-06 \times Cu_{Recovery}^{3.6335}$	$1E-05 \times Cu_{Recovery}^{3.476}$
CQFP	$89.3*(1-EXP(Cu\%_{feed}*-10))$	$3.499 \times Cu_{Recovery}^{0.7194}$	$0.0173 \times Cu_{Recovery}^{1.7542}$	$3E-05 \times Cu_{Recovery}^{3.2099}$
CQP	$89.3*(1-EXP(Cu\%_{feed}*-10))$	$3.499 \times Cu_{Recovery}^{0.7194}$	$0.0173 \times Cu_{Recovery}^{1.7542}$	$3E-05 \times Cu_{Recovery}^{3.2099}$
TBx	$80.3*(1-EXP(Cu\%_{feed}*-10))$	$2E-12 \times Cu_{Recovery}^{6.9291}$	$6E-05 \times Cu_{Recovery}^{3.1369}$	$1E-07 \times Cu_{Recovery}^{4.6209}$
IBxP	$89.3*(1-EXP(Cu\%_{feed}*-10))$	$3.499 \times Cu_{Recovery}^{0.7194}$	$0.0173 \times Cu_{Recovery}^{1.7542}$	$3E-05 \times Cu_{Recovery}^{3.2099}$

16.6.3 Operating Costs

The process and G&A costs were provided by Ausenco based on previous internal studies at 120 kt/d milled and are shown in Table 16-7.

Table 16-7: Process and G&A Costs

Process	Quantity	Amount
Processing	(US$/t milled)	5.59
Tailings Management	(US$/t milled)	1.17
G&A	(US$/t milled)	0.50
Total Mill Feed Based Costs	**(US$/t milled)**	**7.26**

The mining operating costs were developed using a mining cost model used in earlier internal studies and considering haul destinations envisioned early in the 2024 PEA work. Mill feed and waste haul profiles were developed from top, daylight and bottom elevations of an early LG shell that was considered representative of the eventual ultimate pit. Those hauls profiles were ‘costed’ in the cost model and used to develop the base costs and incremental haul costs shown in Table 16 8.

Table 16-8: Mining Costs

Description	Range	Amount	Norte Cost	Sur Cost
Waste	Base Cost (2900)	US$/t	1.46	1.40
	above 2900	US$/t/15 m bench	0.027	0.029
	below 2900	US$/t/15 m bench	0.028	0.025
Mill feed	Base Cost (2900)	US$/t	1.46	1.40
	above 2900	US$/t/15 m bench	0.027	0.029
	below 2900	US$/t/15 m bench	0.028	0.025

The operating costs used in the LG shell are different to those presented in Section 21. When compared to the final derived operating costs, the initial estimates are considered reasonable.

After the completion of the pit optimization and pit design work, the site layout including haulage destinations changed. The original concept was a crusher feeding a conveyor through a tunnel located just to the west of the ultimate pit in the main quebrada that bisects the Norte pit. This was changed to have the crusher to the north and the conveyor on surface. The crusher was located at the same approximate elevation with a slight increase in haulage time. The costs used in mine planning are different to those presented in Section 21, which reflect the final site layout configuration.

16.6.4 Net Smelter Return (NSR) Per Tonne Calculations

Revenue will be generated from the sale of copper concentrate, which will include payable quantities of gold and silver. To assess the value of material with three payable metals, and recoveries that vary with grade and rock type, NSR estimates were performed at the block level via a script and checked with spreadsheet calculations. Using head grade–recovery relationships, recovered grades were calculated and stored in the block model. Using the copper in concentrate grade of 26%, the mass pull (dry metric tonne of concentrate per dry metric tonne milled) of each block was calculated and used to determine grades in concentrate of gold, silver, and arsenic. Gross revenues were estimated for each of the payable metals. Treatment, refining, arsenic penalties and concentrate logistics costs were subtracted from the total gross revenue, resulting in an NSR ($/t milled) estimate that was stored in the block model.

16.6.5 Nested Shell Results

LG optimized pit shells were generated using the technical and cost parameters described above. A series of revenue factor nested shells were generated by multiplying the block gross revenue by the unitless 61 revenue factors that varied from 0.22 to 1.01. The pit optimizations of Norte and Sur were run independently and limited to a max processing capacity of 43.8 Mt/y in the selection of pit shells that guide phasing. For Norte, RF shells 0.230, 0.300, 0.339, and 0.389, corresponding to Pit Shells 4, 11, 19, and 27, respectively, were selected for internal pit phase guidance. Skin analysis was used to find the potential final pit shell, and RF shell 0.5 (PitShell 39); however, the optimization of the LOM production schedule defined the final pit and included an additional RF shell 0.6 (pit Shell 46) which was tested against RF shell 0.5. The final pit shell selected for Norte corresponds to RF shell 0.6. The tonnes and grades of the set of nested shells are shown in Table 16-9 and Figure 16-4. The Sur volumetric results are shown in Table 16-10 and Figure 16-5.

Table 16-9: LG Norte Shell Volumetric Results

Revenue Factor	Ore (kt)	NSR ($/t mineralized material)	Cu (%)	Au (ppb)	Ag (ppm)	As (ppm)	Waste (kt)	Total Material (kt)	Strip Ratio
0.200	10,132	44.37	0.62	127	2.96	369.12	8,796	18,928	0.87
0.210	12,119	43.65	0.61	125	2.87	354.56	9,815	21,934	0.81
0.220	14,244	43.04	0.61	121	2.80	354.90	11,204	25,447	0.79
0.230	48,625	38.15	0.54	107	2.25	244.55	27,422	76,047	0.56
0.240	60,162	37.17	0.53	104	2.24	254.16	31,761	91,923	0.53
0.250	79,534	35.73	0.51	99	2.22	277.17	37,408	116,941	0.47
0.260	91,072	35.24	0.51	97	2.24	274.41	42,982	134,054	0.47
0.270	106,016	34.47	0.50	95	2.19	269.93	47,442	153,458	0.45
0.280	120,384	33.85	0.49	93	2.14	263.73	52,523	172,908	0.44
0.290	135,000	33.20	0.48	91	2.10	264.51	57,332	192,332	0.42
0.300	163,391	32.21	0.47	88	2.08	253.98	69,307	232,698	0.42
0.304	171,517	31.97	0.46	88	2.07	253.97	73,145	244,663	0.43
0.309	181,143	31.64	0.46	86	2.06	256.92	76,286	257,429	0.42
0.313	189,217	31.44	0.46	86	2.05	253.54	80,166	269,382	0.42
0.318	200,033	31.09	0.45	85	2.03	252.18	83,677	283,711	0.42
0.323	206,859	30.91	0.45	84	2.02	251.49	87,035	293,894	0.42
0.328	216,428	30.68	0.45	84	2.01	247.69	91,099	307,526	0.42
0.333	225,466	30.46	0.45	83	2.00	248.91	96,493	321,959	0.43
0.339	299,734	29.16	0.43	80	1.96	232.80	141,263	440,997	0.47
0.344	310,604	29.01	0.43	79	1.95	231.22	147,275	457,879	0.47
0.350	324,431	28.78	0.42	78	1.93	227.59	153,035	477,467	0.47
0.356	338,397	28.58	0.42	78	1.92	228.07	160,465	498,862	0.47
0.362	356,617	28.30	0.42	77	1.91	225.07	168,391	525,008	0.47
0.368	378,276	27.97	0.41	76	1.89	221.31	176,659	554,935	0.47
0.375	404,783	27.62	0.41	75	1.87	219.25	189,604	594,386	0.47
0.382	421,728	27.43	0.41	74	1.86	218.72	199,596	621,324	0.47
0.389	442,122	27.18	0.40	73	1.84	218.07	210,339	652,461	0.48
0.396	468,377	26.88	0.40	72	1.82	218.33	225,420	693,797	0.48
0.404	499,514	26.53	0.39	71	1.80	217.46	242,281	741,795	0.49
0.412	522,233	26.30	0.39	70	1.78	214.99	254,607	776,840	0.49
0.420	549,013	26.03	0.39	69	1.77	214.79	269,822	818,834	0.49
0.429	576,803	25.78	0.39	68	1.75	213.01	287,647	864,451	0.50
0.438	613,345	25.47	0.38	67	1.73	210.63	311,524	924,869	0.51
0.447	652,021	25.18	0.38	66	1.71	209.34	340,250	992,270	0.52
0.457	708,881	24.79	0.37	65	1.68	205.19	382,274	1,091,155	0.54
0.467	754,184	24.51	0.37	64	1.66	203.45	418,496	1,172,680	0.55
0.477	830,013	24.11	0.36	62	1.64	201.25	483,984	1,313,996	0.58
0.488	885,599	23.86	0.36	61	1.62	199.47	536,884	1,422,483	0.61

Revenue Factor	Ore (kt)	NSR ($/t mineralized material)	Cu (%)	Au (ppb)	Ag (ppm)	As (ppm)	Waste (kt)	Total Material (kt)	Strip Ratio
0.500	933,037	23.65	0.36	60	1.61	197.33	580,110	1,513,147	0.62
0.512	982,880	23.46	0.36	60	1.60	195.99	633,640	1,616,519	0.64
0.525	1,023,063	23.32	0.35	59	1.59	195.05	678,854	1,701,917	0.66
0.538	1,057,433	23.21	0.35	59	1.58	194.68	719,466	1,776,899	0.68
0.553	1,095,721	23.09	0.35	58	1.58	194.13	769,015	1,864,735	0.70
0.568	1,126,557	23.02	0.35	58	1.57	194.17	818,246	1,944,803	0.73
0.583	1,155,440	22.95	0.35	57	1.57	194.21	862,950	2,018,390	0.75
0.600	**1,184,344**	**22.90**	**0.35**	**57**	**1.57**	**194.51**	**917,442**	**2,101,787**	**0.77**
0.618	1,206,821	22.84	0.35	57	1.57	194.13	955,685	2,162,506	0.79
0.636	1,235,318	22.77	0.35	57	1.56	193.87	1,009,339	2,244,657	0.82
0.656	1,261,886	22.71	0.35	56	1.56	193.53	1,063,473	2,325,359	0.84
0.677	1,278,233	22.69	0.35	56	1.56	193.54	1,102,679	2,380,913	0.86
0.700	1,306,849	22.64	0.35	56	1.56	193.34	1,173,185	2,480,034	0.90
0.724	1,324,683	22.61	0.35	56	1.56	193.34	1,221,707	2,546,391	0.92
0.750	1,346,894	22.58	0.35	56	1.56	193.21	1,284,947	2,631,840	0.95
0.778	1,363,807	22.56	0.35	55	1.56	193.28	1,340,077	2,703,884	0.98
0.808	1,388,150	22.52	0.34	55	1.56	193.23	1,418,706	2,806,856	1.02
0.840	1,408,995	22.51	0.34	55	1.56	193.50	1,503,385	2,912,380	1.07
0.875	1,424,849	22.50	0.34	55	1.56	193.65	1,568,702	2,993,551	1.10
0.913	1,446,136	22.48	0.34	55	1.56	193.50	1,657,020	3,103,156	1.15
0.955	1,462,933	22.47	0.34	55	1.56	193.54	1,741,622	3,204,555	1.19
1.000	1,333,057	18.88	0.36	60	1.6	197.29	1,828,876	3,161,933	1.37

Note: Shell tonnes and grade reported at ((Net Revenue-Period Cost-Processing Cost-(Mining Cost Mill Feed-Mining Cost Waste))>0 as Mill Feed.

For Sur, RF shells 0.389, 0.429, 0.447, and 0.636, corresponding to pit Shells 25, 30, 35, and 46, respectively, were selected for internal pit phase guidance. Like Norte, skin analysis was used to find the possible final pit shell. The RF shell 0.636 (pitShell 46) was tested vs RF shell 0.447 (pit Shell 35) by the LOM production schedule optimization in defining the final pit. The final pit shell selected for Sur corresponds to RF 0.636. The tonnes and grades of nested shells are shown in Table 16-10 and Figure 16-5.

Table 16-10: LG Sur Shell Report Results

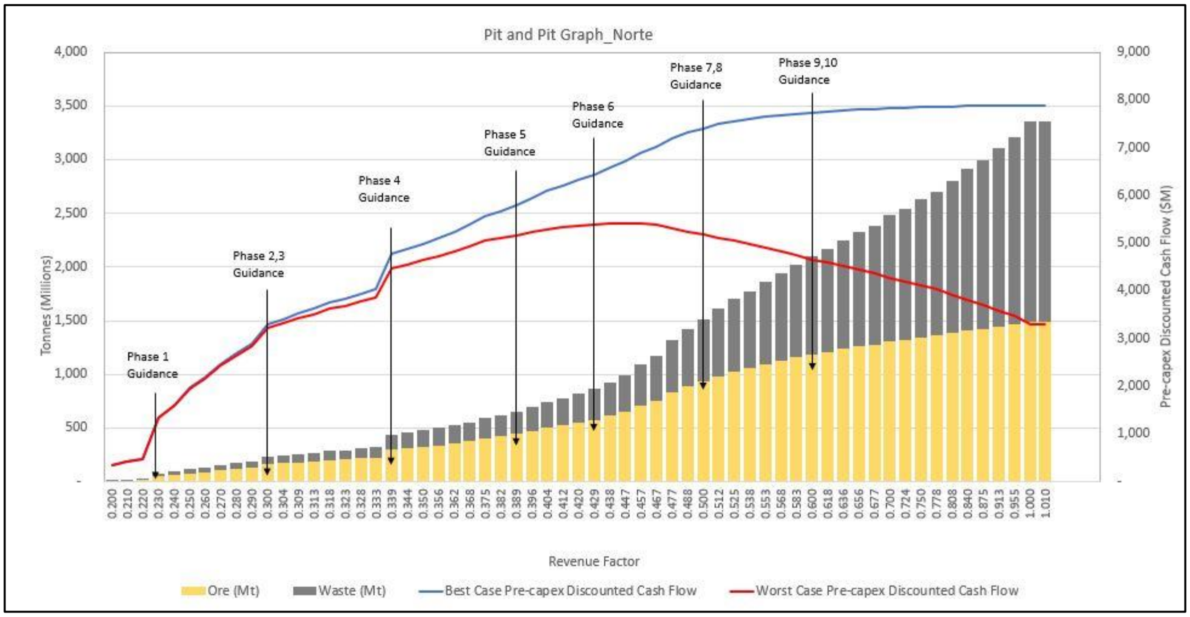
Revenue Factor	Ore (kt)	NSR ($/t mineralized material)	Cu (%)	Au (ppb)	Ag (ppm)	As (ppm)	Waste (kt)	Total Material (kt)	Strip Ratio
0.220	1	35.58	0.39	209	1.74	0.00	0	1	0.07
0.230	2	36.54	0.45	209	1.74	0.00	0	2	0.12
0.240	2	36.46	0.44	207	1.74	0.00	0	2	0.12
0.250	33	32.34	0.42	160	1.72	0.00	4	37	0.12

Revenue Factor	Ore (kt)	NSR ($/t mineralized material)	Cu (%)	Au (ppb)	Ag (ppm)	As (ppm)	Waste (kt)	Total Material (kt)	Strip Ratio
0.260	295	30.85	0.40	162	1.68	6.78	23	318	0.08
0.270	519	30.56	0.40	156	1.66	5.78	59	578	0.11
0.280	707	29.88	0.39	152	1.64	7.07	66	774	0.09
0.290	862	29.30	0.38	148	1.62	6.96	69	931	0.08
0.300	1,463	28.48	0.37	142	1.56	8.20	172	1,635	0.12
0.304	1,759	27.93	0.37	139	1.54	8.53	188	1,947	0.11
0.309	2,110	27.54	0.36	137	1.52	9.01	237	2,347	0.11
0.313	2,366	27.16	0.36	136	1.50	9.30	243	2,609	0.10
0.318	3,066	26.92	0.36	134	1.49	9.46	443	3,509	0.14
0.323	3,439	26.53	0.35	131	1.47	9.89	451	3,890	0.13
0.328	3,905	26.14	0.35	129	1.45	9.99	488	4,392	0.12
0.333	4,389	25.96	0.35	129	1.44	10.25	618	5,007	0.14
0.339	5,675	25.60	0.34	128	1.44	10.22	1,012	6,687	0.18
0.344	5,873	25.48	0.34	127	1.43	10.39	1,017	6,890	0.17
0.350	6,592	25.29	0.34	126	1.42	10.47	1,255	7,847	0.19
0.356	7,454	24.91	0.33	123	1.39	10.73	1,409	8,863	0.19
0.362	11,049	23.73	0.32	112	1.31	9.78	1,737	12,785	0.16
0.368	12,109	23.41	0.31	110	1.30	9.99	1,767	13,876	0.15
0.375	14,865	22.91	0.31	107	1.27	10.56	2,269	17,135	0.15
0.382	16,781	22.76	0.31	106	1.27	10.61	2,855	19,636	0.17
0.389	64,307	19.92	0.27	101	1.24	22.75	11,019	75,326	0.17
0.396	71,391	19.76	0.26	100	1.23	23.34	12,026	83,417	0.17
0.404	81,923	19.55	0.26	100	1.23	24.73	13,910	95,832	0.17
0.412	93,098	19.26	0.26	99	1.24	26.24	15,046	108,144	0.16
0.420	96,004	19.20	0.26	99	1.24	26.46	15,369	111,373	0.16
0.429	108,384	19.07	0.25	98	1.24	27.24	19,420	127,803	0.18
0.438	118,756	18.89	0.25	97	1.24	28.37	21,891	140,647	0.18
0.447	129,087	18.70	0.25	96	1.23	29.20	24,043	153,130	0.19
0.457	135,778	18.58	0.25	96	1.23	30.09	25,628	161,406	0.19
0.467	145,092	18.59	0.25	96	1.23	29.81	31,359	176,451	0.22
0.477	172,574	18.17	0.24	94	1.23	33.00	38,931	211,504	0.23
0.488	173,497	18.17	0.24	94	1.23	32.96	39,559	213,055	0.23
0.500	182,994	18.06	0.24	94	1.22	33.70	42,932	225,926	0.23
0.512	194,717	17.99	0.24	93	1.21	34.14	49,990	244,707	0.26
0.525	197,959	17.99	0.24	93	1.21	34.18	52,929	250,888	0.27
0.538	205,789	17.92	0.24	92	1.20	34.72	57,536	263,326	0.28
0.553	217,688	17.96	0.24	92	1.21	34.63	71,585	289,273	0.33
0.568	222,336	17.95	0.24	92	1.20	34.59	77,014	299,350	0.35
0.583	231,291	17.94	0.24	92	1.20	34.64	87,841	319,132	0.38
0.600	240,130	17.99	0.24	92	1.20	34.38	102,712	342,843	0.43

Revenue Factor	Ore (kt)	NSR ($/t mineralized material)	Cu (%)	Au (ppb)	Ag (ppm)	As (ppm)	Waste (kt)	Total Material (kt)	Strip Ratio
0.618	248,780	17.93	0.24	91	1.20	34.58	112,171	360,951	0.45
0.636	**262,994**	**17.93**	**0.24**	**91**	**1.19**	**34.51**	**135,223**	**398,217**	**0.51**
0.656	275,315	17.92	0.24	91	1.19	34.47	156,942	432,257	0.57
0.677	285,461	17.92	0.24	91	1.18	34.46	176,761	462,222	0.62
0.700	300,081	17.94	0.24	91	1.18	34.28	208,570	508,650	0.70
0.724	309,706	17.93	0.24	91	1.18	34.35	230,191	539,897	0.74
0.750	326,823	17.88	0.24	90	1.18	34.59	267,402	594,224	0.82
0.778	335,720	17.85	0.24	90	1.17	34.61	288,045	623,765	0.86
0.808	346,586	17.82	0.24	90	1.17	34.63	315,927	662,512	0.91
0.840	360,246	17.80	0.24	89	1.17	34.70	356,321	716,567	0.99
0.875	372,501	17.78	0.24	89	1.17	34.74	394,202	766,703	1.06
0.913	383,829	17.74	0.24	89	1.17	34.84	428,898	812,727	1.12
0.955	393,959	17.71	0.24	89	1.16	34.77	464,476	858,434	1.18
1.000	406,238	17.68	0.24	88	1.16	34.89	512,074	918,312	1.26

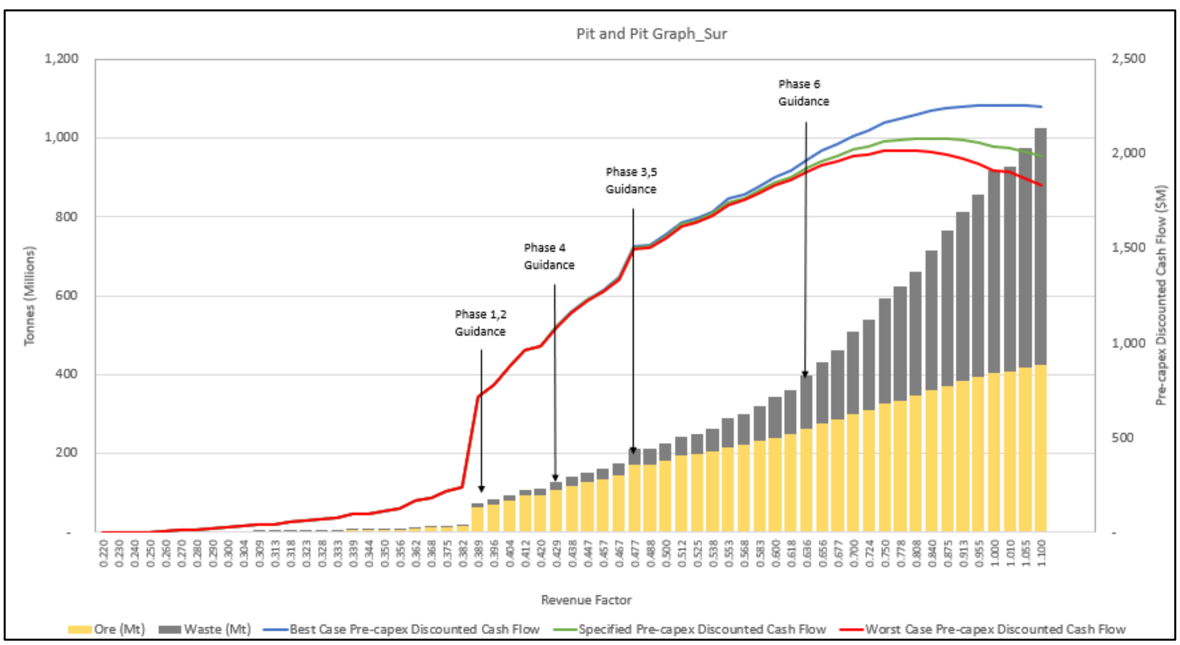
Note: Shell tonnes and grade reported at ((Net Revenue-Period Cost-Processing Cost-(Mining Cost Mill Feed-Mining Cost Waste))>0 as Mill Feed.

Figure 16-4: Norte LG Nested Shell Pit by Graph



Source: WCL, 2024.

Figure 16-5: Sur LG Nested Shell Pit by Pit Graph



Source: WCL, 2024.

The shells were selected based on the following considerations:

- A desire to maximize NPV and resource extraction with equipment appropriate minimum mining widths and practical ramp access; and
- A LOM in the range of 25 to 30 years.

16.7 Pit Designs

The Norte pit has been designed with 10 phases over the mine life. These phases were designed to release mineralized material in a timely manner and to smooth out stripping requirements on an annual basis. The nested pit optimization shells used to determine the ultimate pit were also used to outline areas of higher value for targeted early mining and phase development. The phases also take advantage of splitting the pit into north and south sides to ensure mill feed material is released consistently. The split phases help to advance to mill feed material sooner while the opposite side is still mining waste.

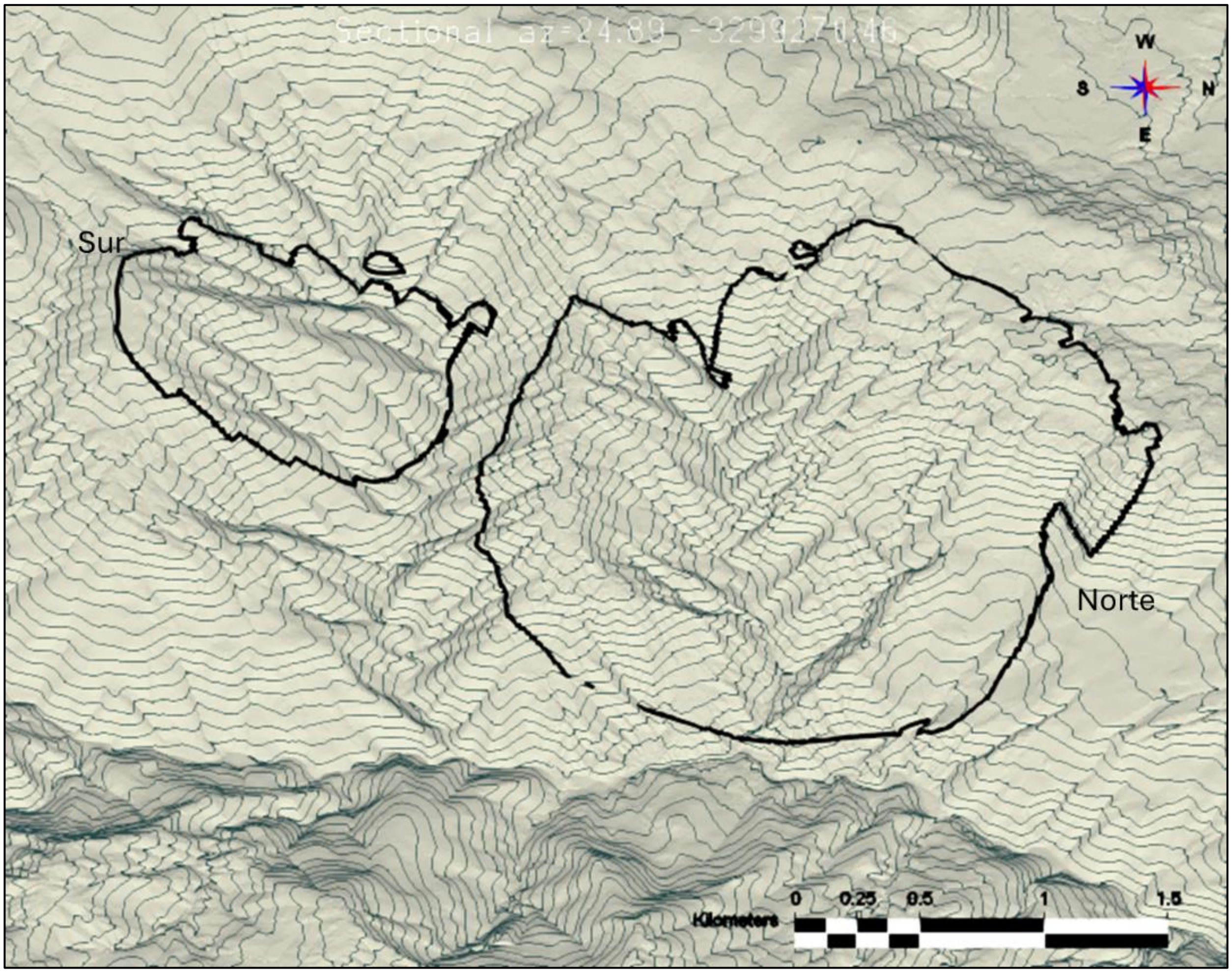
Geotechnical parameters outlined in Table 16-1 were applied to the pit phase designs. The starter pit used the steeper 'starter pit stage' slope parameters together with smaller ramps to accommodate smaller Scania 40 t trucks. The other phases used the 'final stage' parameters. The use of the steeper slopes and smaller trucks was to reduce the initial waste requirements while achieving the necessary mill feed release. The smaller equipment is also better for preparing the initial benches of the phases for the larger equipment, making it more efficient.

Ramp widths were based on the use of 290 t rigid-frame haul trucks. The operating width used for the truck is 9.0 m. Single-lane roads will be 27.1 m wide (twice the operating width plus berm and ditch) and double lane widths will be 36.1 m (three times the operating width plus berm and ditch). Ramp gradients planned to be 10% both in the pit and ex-pit for uphill gradients. Working benches were designed for a 90-m minimum mining width, considering the use of 38 m^3 electric hydraulic shovels.

The Sur pit has been divided into six phases for mining of the area. Mining geotechnical work for this pit area is lacking so the assumption was to use the same parameters as the Southwest sector (Sector 2) shown in Table 16-1.

The topography of the pit areas is generally steep with eroded drainage features. A view looking west at the Norte and Sur pit areas respectively. While the topography is steep, weathered material is present on the slopes to allow pioneering work with the smaller equipment fleet. Figure 16-6 shows the pit area outlines and topography.

Figure 16-6: West Looking View of Pit Area Topography



Source: AGP, 2024.

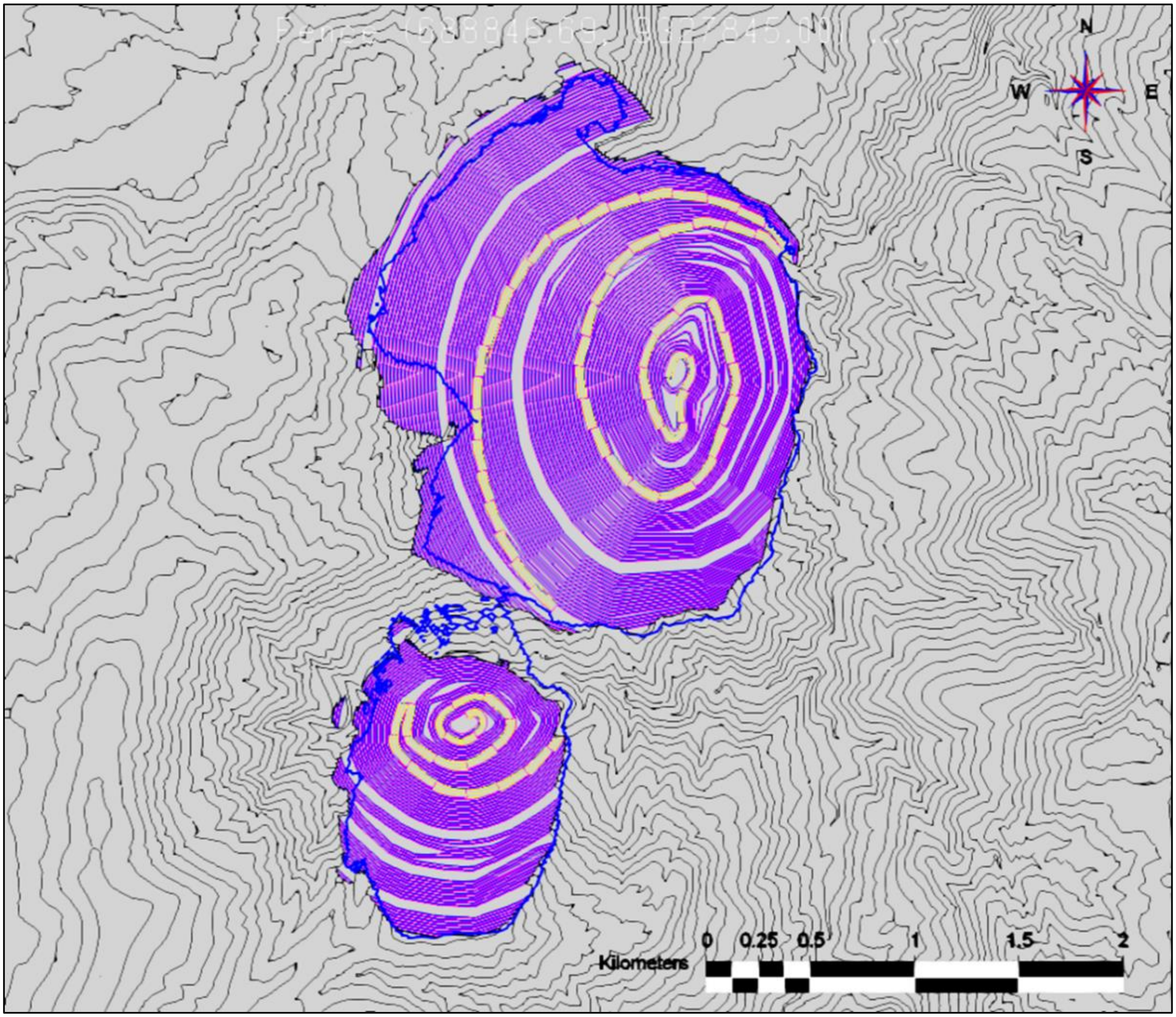
16.7.1 Ultimate Pit Design

The Norte ultimate pit design was based on the RF 0.60 LG shell. The overall dimensions of the ultimate pit are 2,700 m in the north–south direction, 2,600 m in the east–west direction and a 1,125 m maximum depth. It will have a single ramp exit point at the 2,710 m elevation, providing access to the mill feed and waste crushers, as well as the truck shop and fuel bay.

The ultimate pit design for Sur was based on the RF 0.636 LG shell. The overall dimensions of the ultimate Sur pit are 1,400 m in the north-south an 1,100 m in the east-west direction with a maximum depth of 750 metres. The final ramp connects to the southern end of Norte at the ramp elevation of 3,025 m and then down to the 2,710 m elevation for access to the crusher and conveyors.

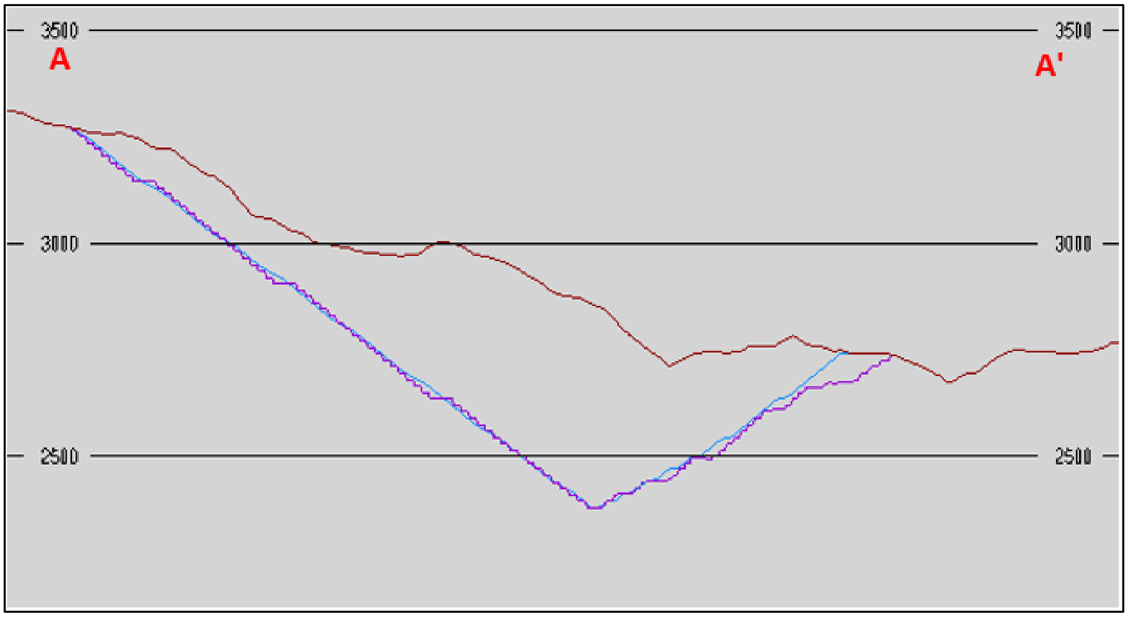
The ultimate pit designs for Norte and Sur are shown with the appropriate RF shell pit (in dark blue) in plan and vertical section view in Figure 16-7, Figure 16-8, and Figure 16-9 respectively.

Figure 16-7: Final Pit Designs – Norte and Sur



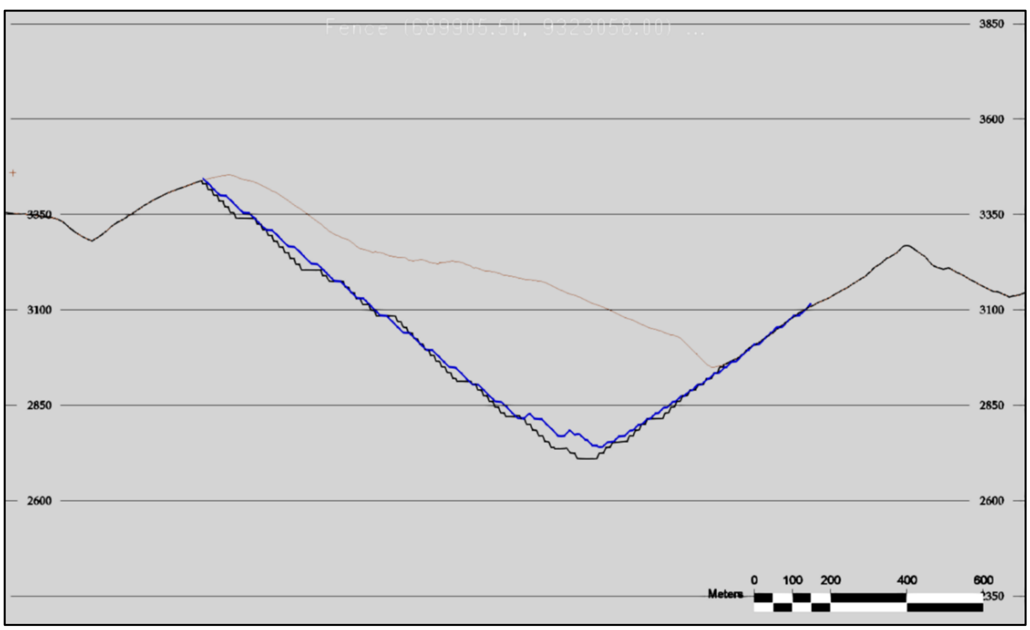
Source: AGP, 2024. Section line A–A' is location of Figure 16-7 and Section B-B' is shown in Figure 16-8.

Figure 16-8: Section View of Norte Ultimate Pit Design vs. Pit Shell (Section A-A')



Source: AGP, 2024. Section line location is provided on Figure 16-7.

Figure 16-9: Section View of Sur Ultimate Pit Design vs. Pit Shell (Section B-B')



Source: AGP, 2024. Section line location is provided on Figure 16-7.

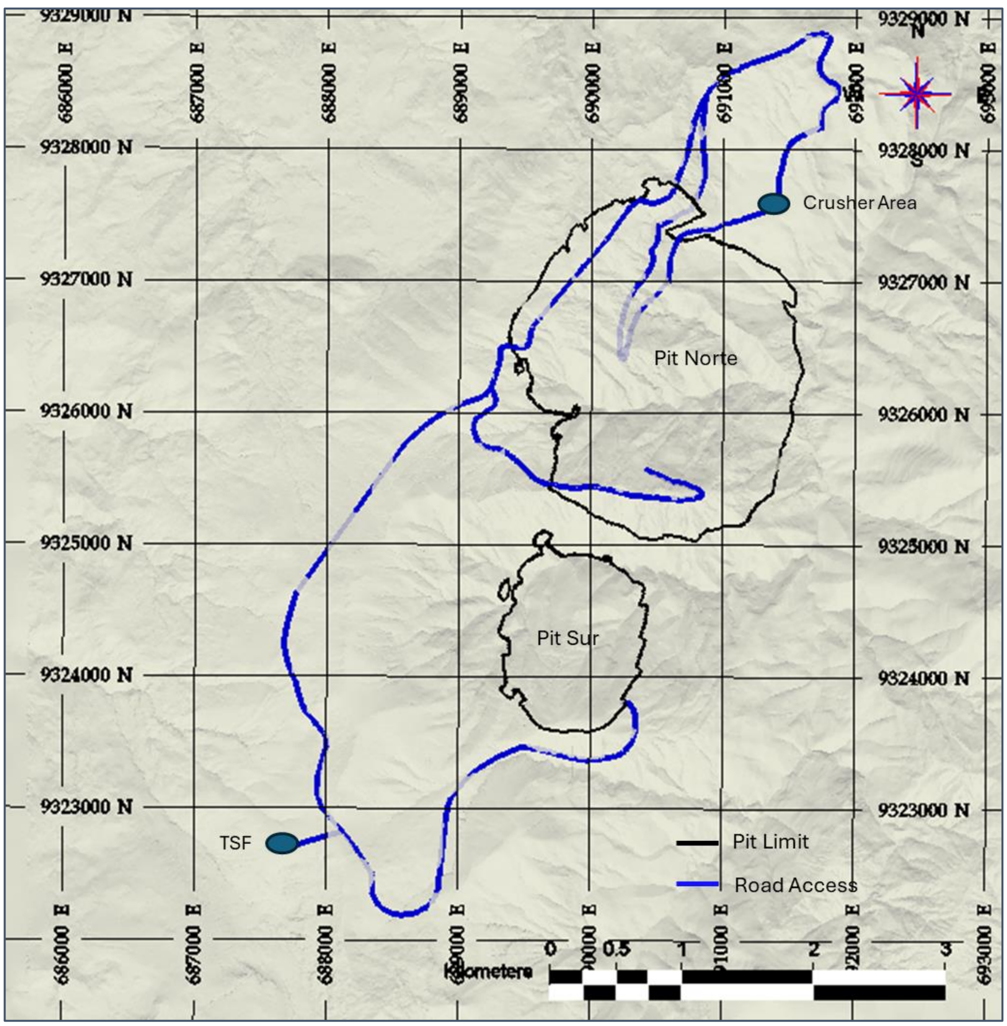
16.7.2 Internal Phase Desings

Norte Pit

The phasing strategy was to initiate mining in the central core of the deposit where mill feed is readily available. The two phases work either side of the main drainage, Quebrada Norte, that bisects the pit area. This reduced pioneering road requirements and simplified the surface water management requirements early in the pit development. The smaller equipment was used in the development of these two phases until sufficient width was available for the larger mining shovels.

The location of some of the initial pioneering roads is shown in Figure 16-10.

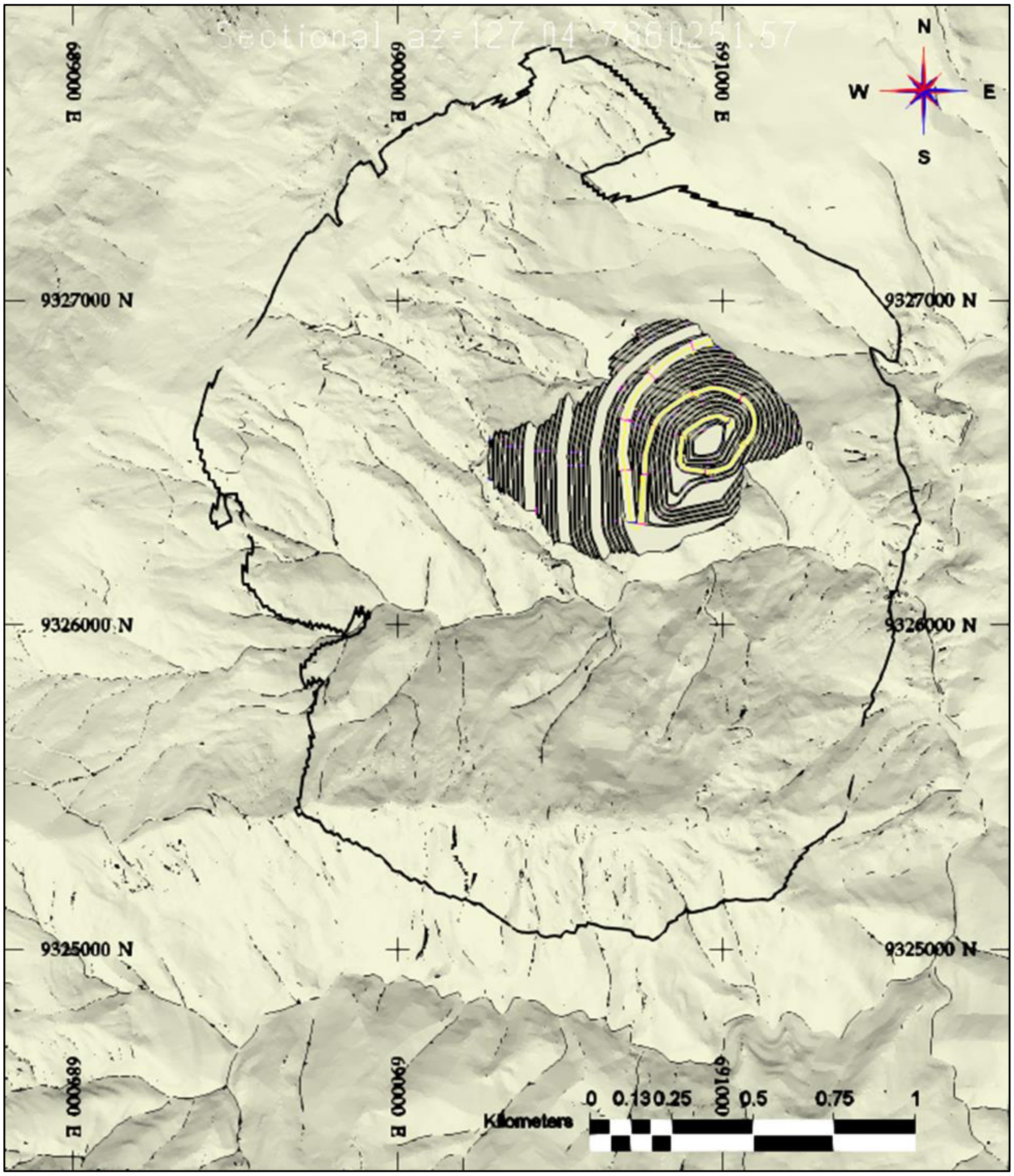
Figure 16-10: Initial Cañariaco Pioneering Road Locations



Source: AGP, 2024.

The Norte Phase 1 design is a small starter pit, in the northern portion of the ultimate footprint. It will have a crest elevation of 3,145 m, a bottom elevation of 2,680 m with the ramp exit near the primary crusher. The west highwall will contain a ramp that will be used to access the Phase 2 design. Phase 1 will be mined from Year -2 through Year 2. The Phase 1 design is shown with the ultimate pit outline in black in Figure 16-11.

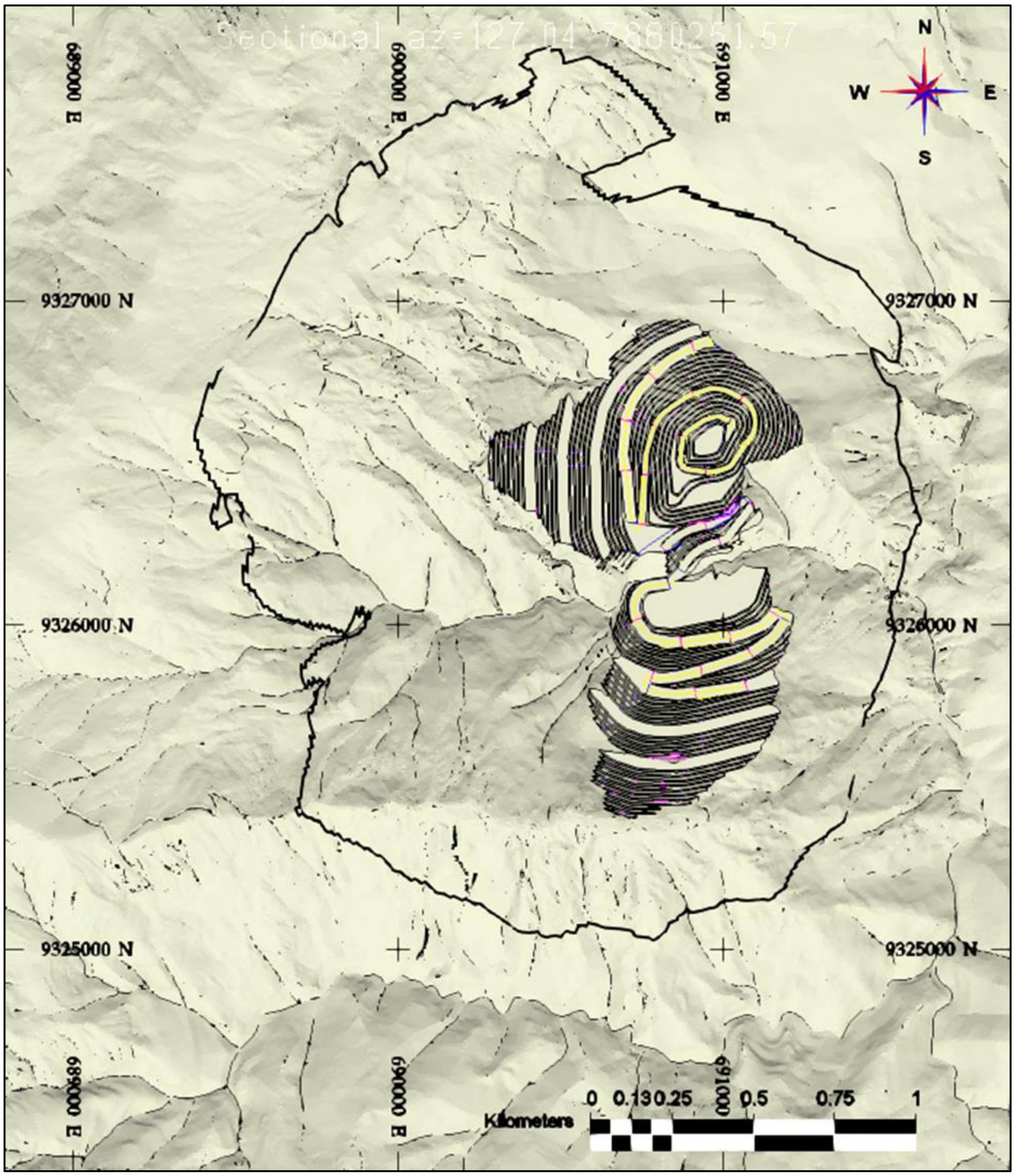
Figure 16-11: Norte Phase 1 Design



Source: AGP, 2024.

The Norte Phase 2 design will mine on the south side of the main drainage. Due to the steep topography, it was necessary for the Phase 2 design to pioneer from the southern ridge with material placed in the drainage to connect Phase 1 and 2. As with the Phase 1 design, a ramp is included in the highwall to allow access to later phases. Phase 2 will have a crest elevation of 3,190 m, a bottom elevation of 2,755. It will tie into the Phase 1 exit ramp system. Phase 2 will be mined from Year -3 through Year 4. The Phase 2 design is shown Figure 16-12.

Figure 16-12: Norte Phase 2 Design

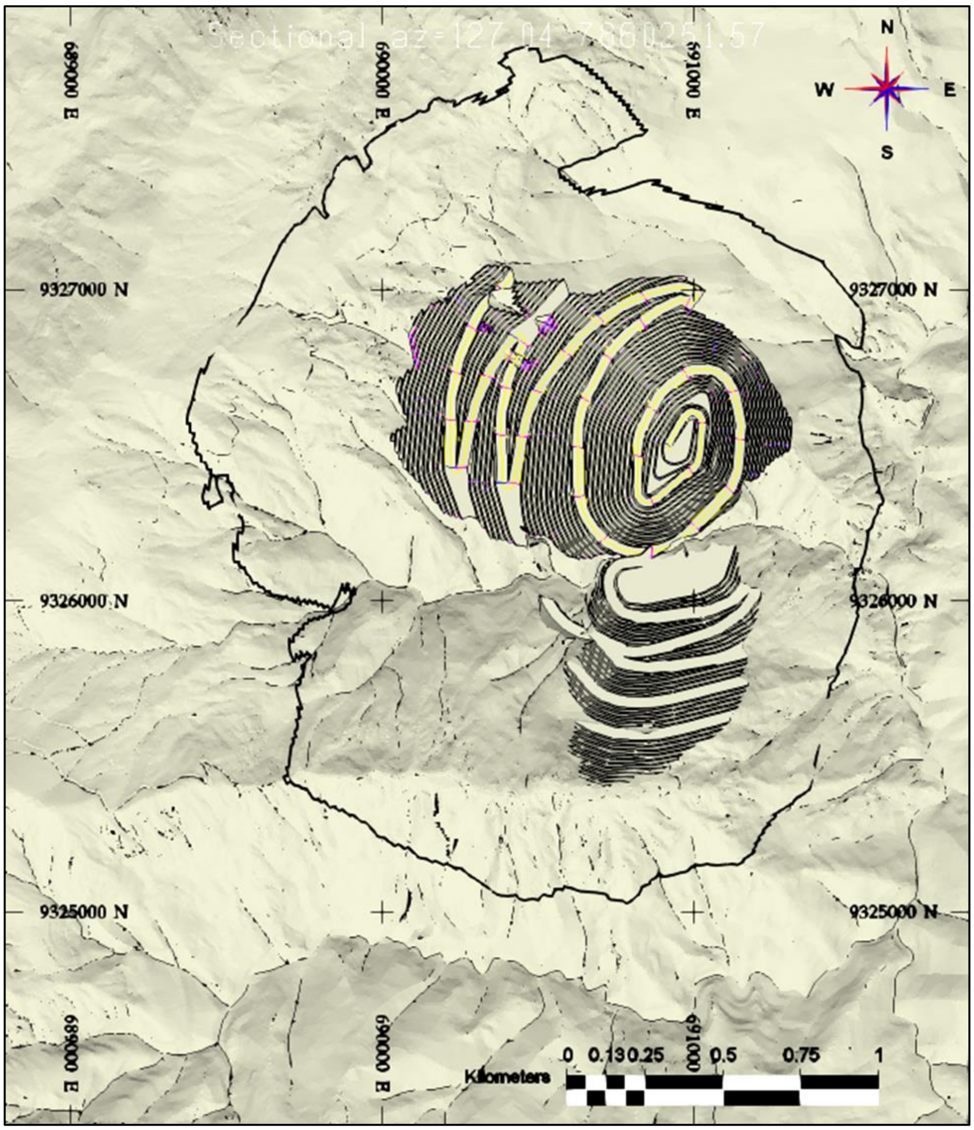


Source: AGP, 2024.

The Norte Phase 3 design will mine on the north side of Norte, but the west wall. Smaller equipment will mine the upper portions in the steeper topography until sufficient width is available for the large equipment to be efficient. The geotechnical berms left in the wall slope will be used to connect across the quebrada to the southern portion of the pit. This allows for ease of equipment movement around the pit.

A ramp will be left in the highwall to provide access to the western slope for later phases. Phase 3 will have a crest elevation of 3,235 m, a bottom elevation of 2,590 m as well as various ramp exit points to the north. Phase mining will start in production Year -3 with initial pre-stripping and end in Year 4. The Phase 3 design is shown Figure 16-13.

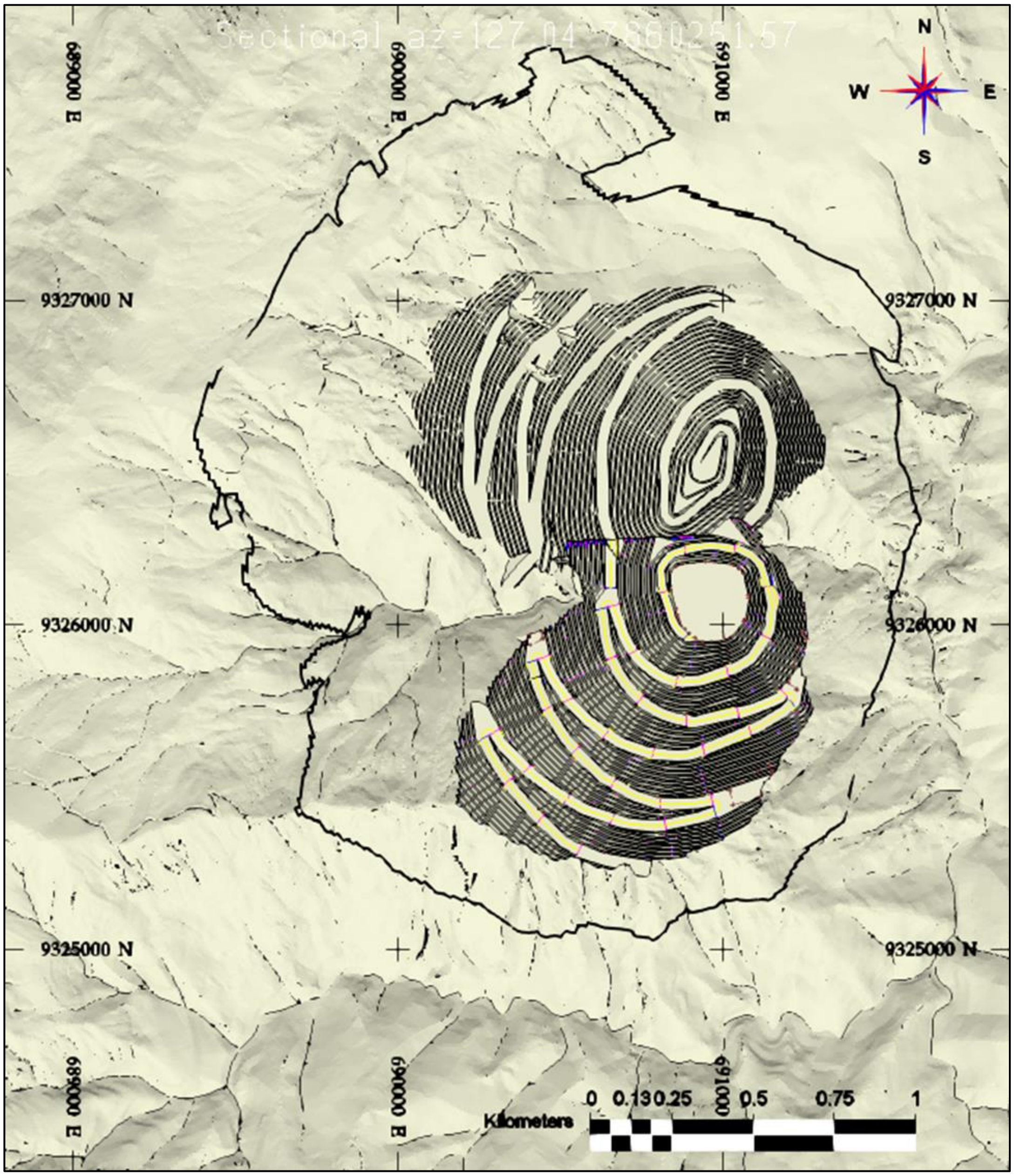
Figure 16-13:   Norte Phase 3 Design



Source: AGP, 2024.

The Norte Phase 4 design mines to the south as an expansion of Phase 2. This utilizes the earlier pioneering roads with material placement in the quebrada to connect the north and south sides. The material is later rehandled as part of normal mining. The top elevation of Phase 4 is 3,235 m elevation and the lowest level is 2,635 m. The Phase 4 design is shown in Figure 16-4.

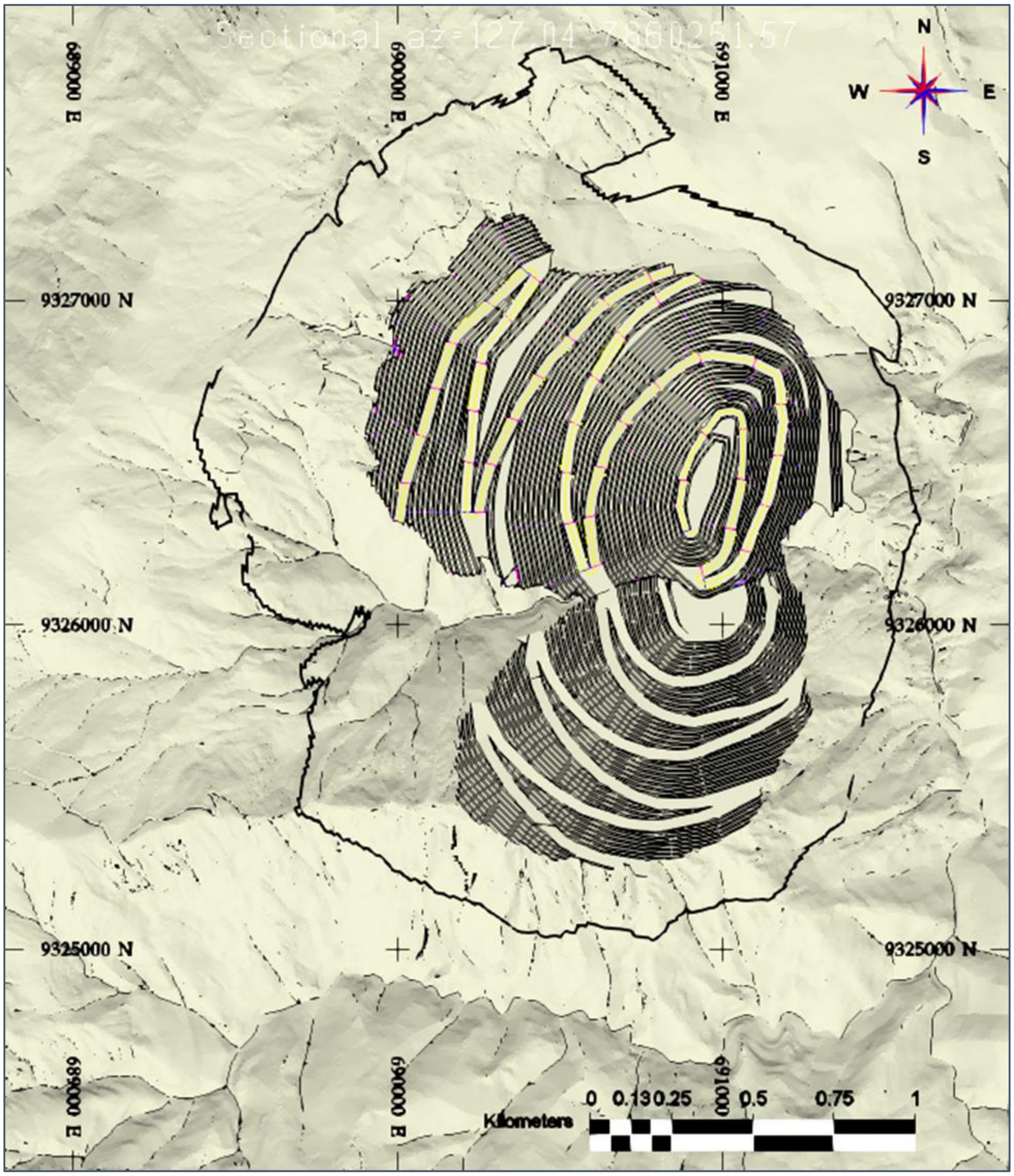
Figure 16-14: Norte Phase 4 Design



Source: AGP, 2024.

Phase 5 in Norte pushes Phase 3 further up the slope to the west utilizing the in-slope ramps and access along natural topography. Ramps are left in the wall to provide access for later phases via the upper end of the quebrada on the south side of the phase. The top elevation is 3,265 m and the bottom is 2,515 m. The phase design is shown in Figure 16-15.

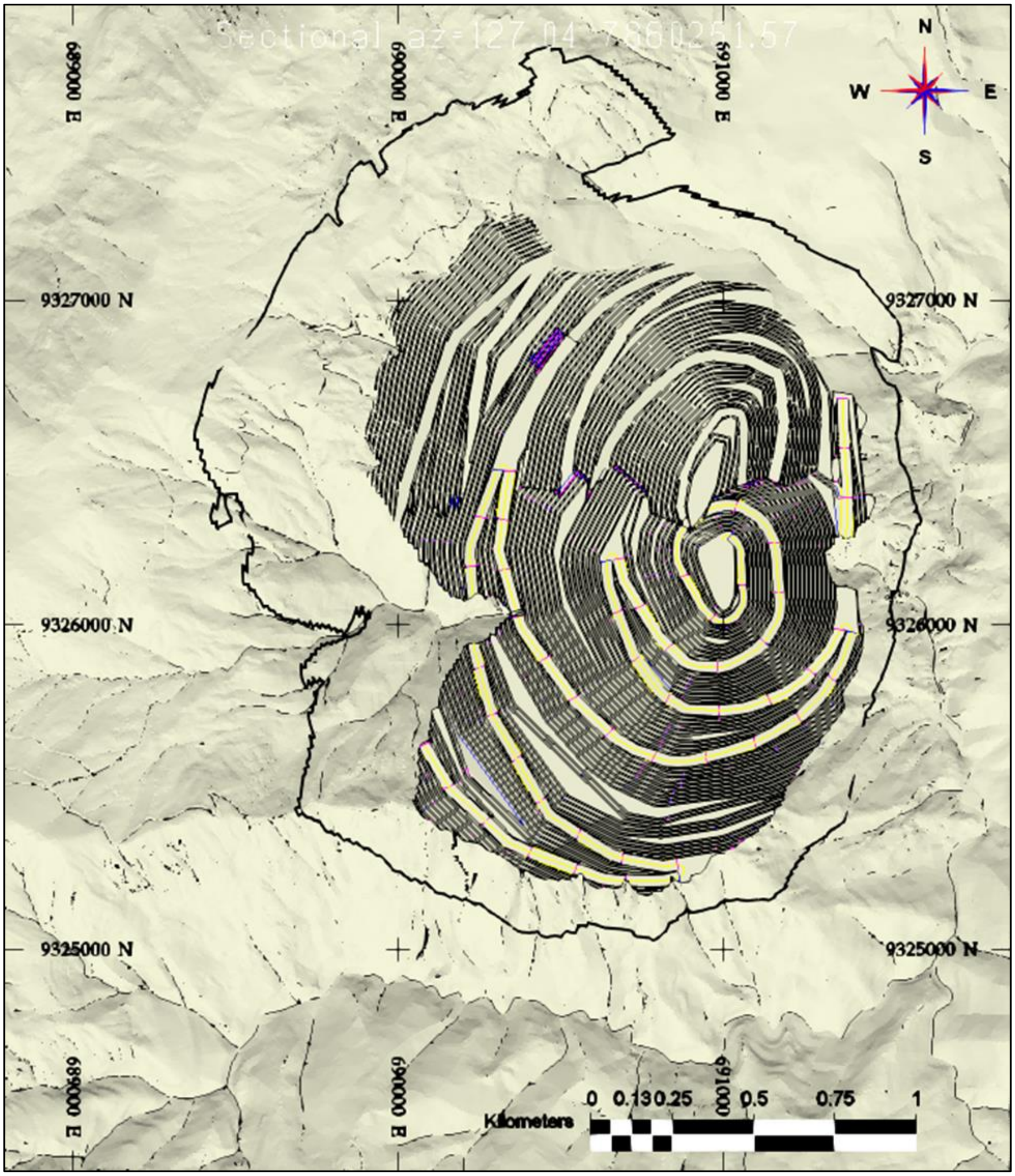
Figure 16-15: Norte Phase 5 Design



Source: AGP, 2024.

Norte Phase 6 mines on the southwest side of the Norte pit expanding Phase 4 in that direction. Access is maintained from Phase 5 across the quebrada. The top elevation of Phase 6 is 3,250 m with a bottom elevation of 2,455 m. The Phase 6 design is shown in Figure 16-16.

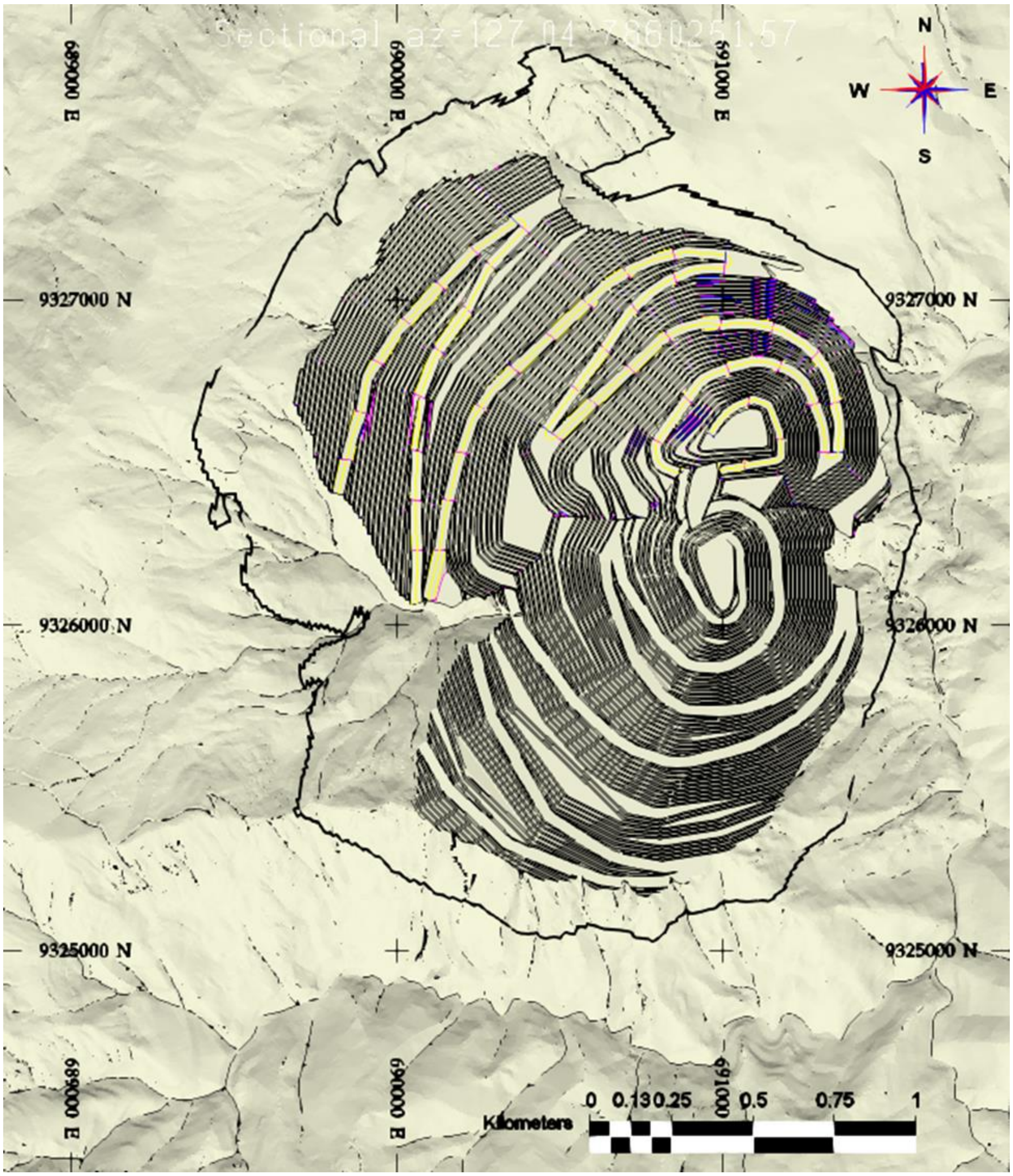
Figure 16-16: Norte Phase 6 Design



Source: AGP, 2024.

Phase 7 in the Norte pit toggles again to the northwest pushing the Phase 5 slope further to the west. To access this the natural topography and the quebrada are used as well as the ramps left in the slope from Phase 5. Ramps are left in place in Phase 7 also for future access. The top elevation of Norte Phase 7 is 3,325 m with a bottom elevation of 2,470 m. The design is shown in Figure 16-17.

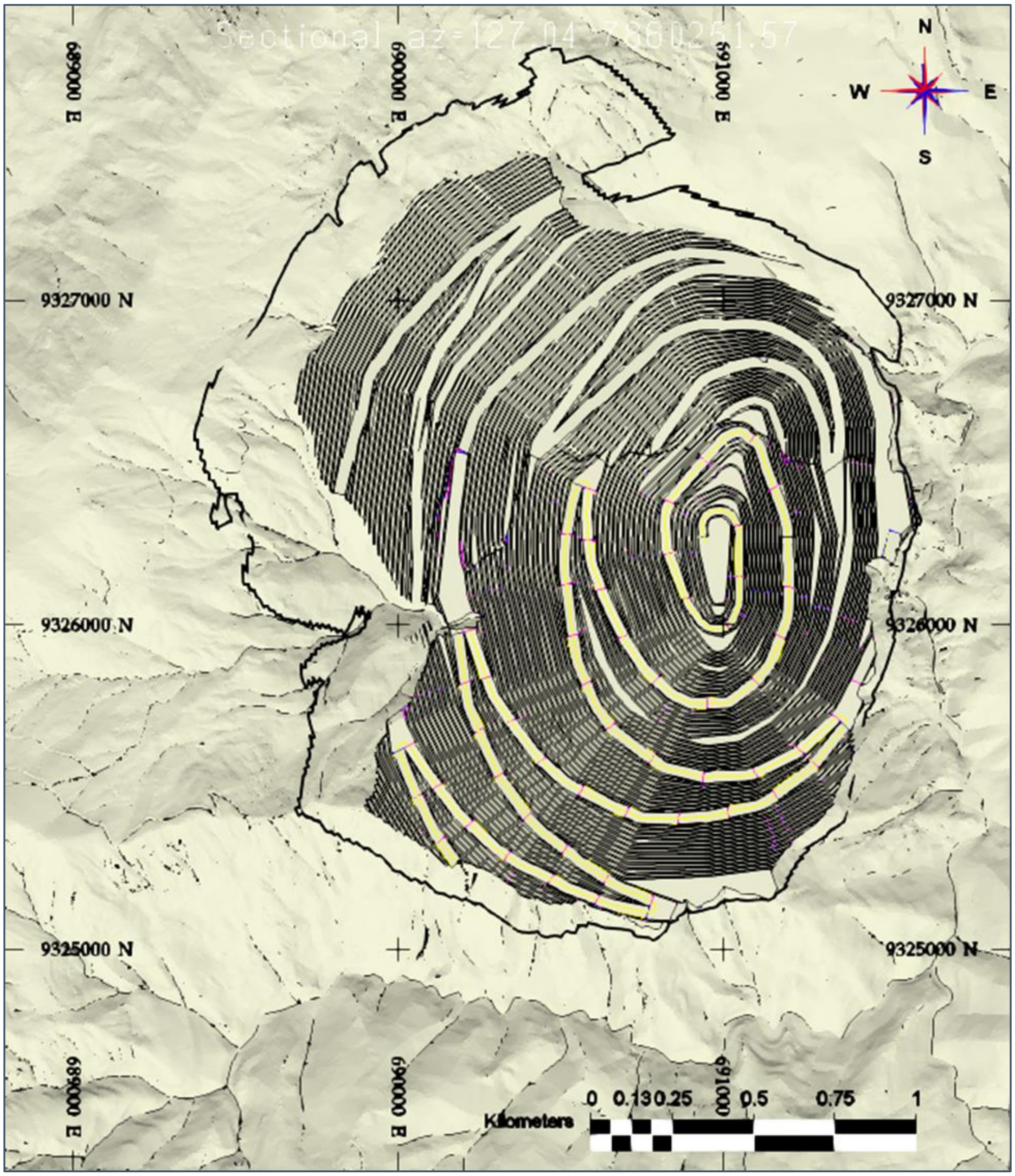
Figure 16-17: Norte Phase 7 Design



Source: AGP, 2024.

Norte's Phase 8 design expands to the southwest and east in the pit. The eastern side is pushed closer to the ultimate limit and the southwest is expanded further. Access to the top is from the early pioneering roads from the plan as well as via the main quebrada using the ramps left from Phase 6. The top elevation is 3,265 m and the bottom is 2,335 m. The phase design is shown in Figure 16-18.

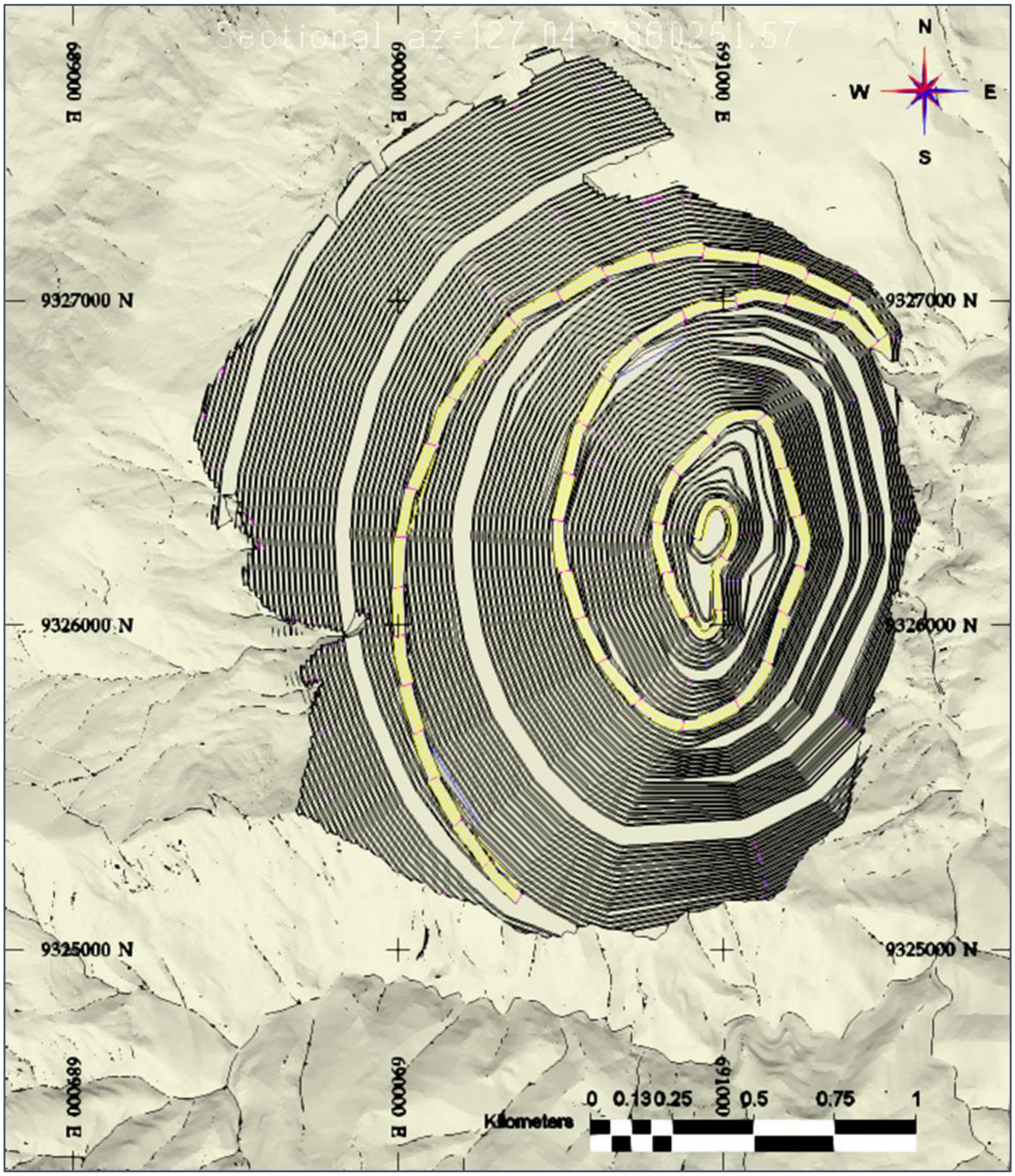
Figure 16-18: Norte Phase 8 Design



Source: AGP, 2024.

Phases 9 and 10 of the Norte pit push back to the ultimate limits to the west and north. A ramp is left in the slope to provide truck access to the Sur pit at the 2,710 m level but otherwise is mined to the steepest slope under the guidelines with the appropriate geotechnical berms left in place. This access is for ease of moving personnel and equipment between the pit areas. The highest level in the phase is 3,385 m with the lowest level at 2,260 m. The phase design is shown in Figure 16-19.

Figure 16-19: Norte Phase 9 and 10 Design



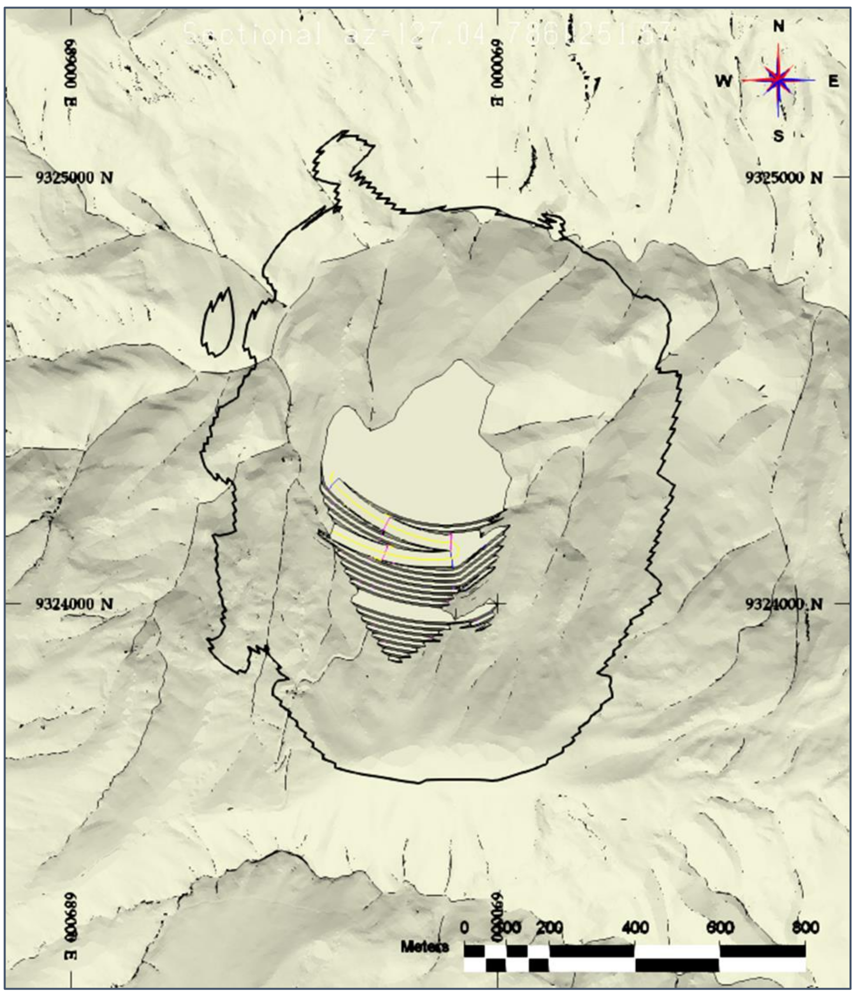
Source: AGP, 2024.

Sur Pit

In the same manner as Norte, the Sur pit was developed in the core of the deposit to start expanding to the south then into the eastern and western flanks. The concept was to develop a tunnel connecting Sur and Norte with a crusher/sizer in Sur. In the design process it became apparent with the timing that road access across the drainage could be maintained and access to Norte for easier equipment movement possible. The connecting elevation in Norte is on the south end ramp at the 3,025 m elevation. The design reflects the use of a crusher and conveyor through the ridge to shorten the waste and mill feed haulage.

Phase 1 in Sur is accessed from a pioneering road from the plant as well as with a road along the north side of the Sur quebrada to allow quick movement of equipment between the two pit areas. The top level of Sur is at 3,385 m with the bottom level at 3,145 m. The design for Phase 1 is shown in Figure 16-20.

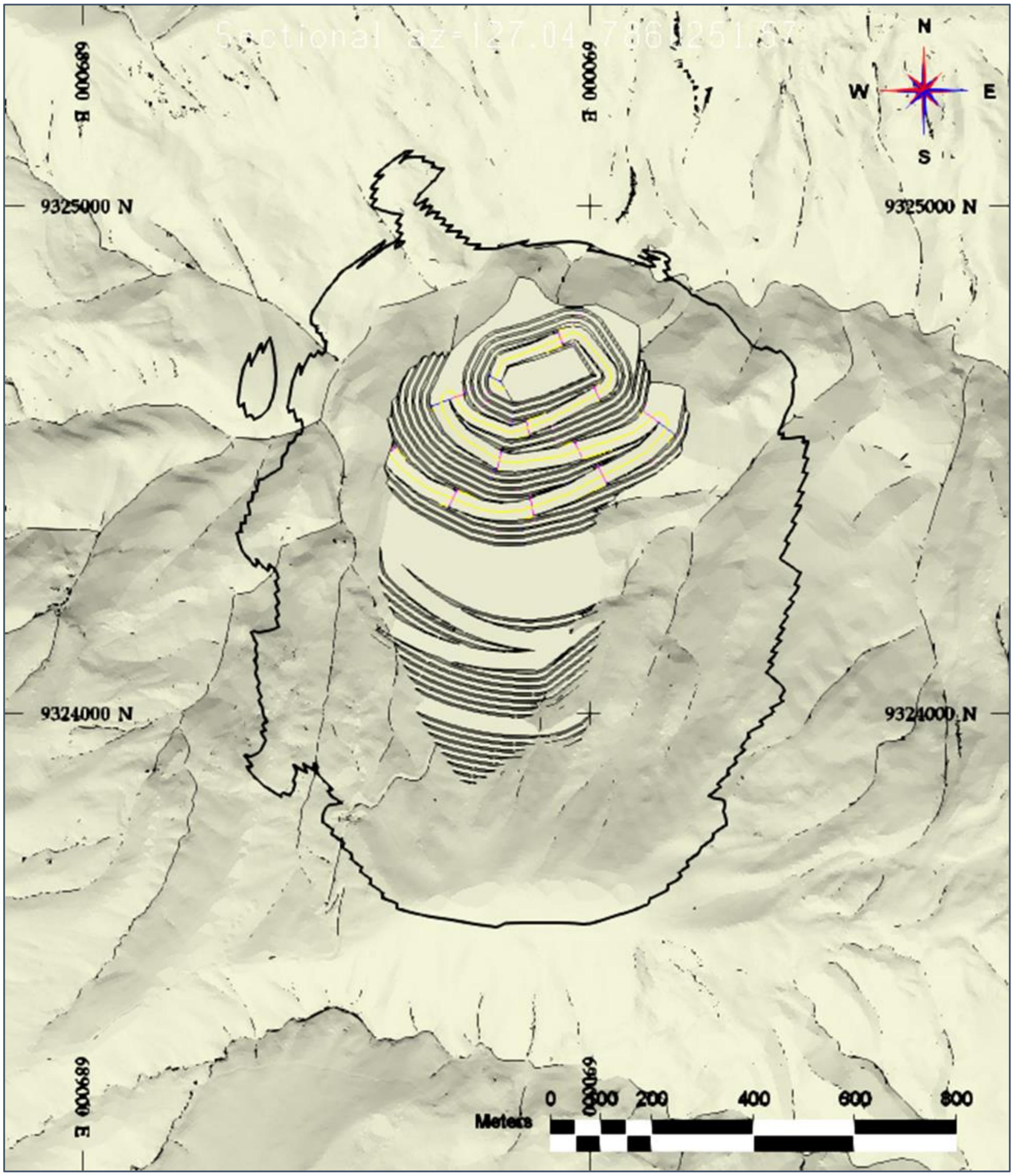
Figure 16-20: Sur Phase 1 Design



Source: AGP, 2024.

Sur Phase 2 expands deepens to the north while keeping the same southern slope as Phase 1. The top elevation is now 3,130 m but the depth is 2,950 m. The design for Sur Phase 2 is shown in Figure 16-21.

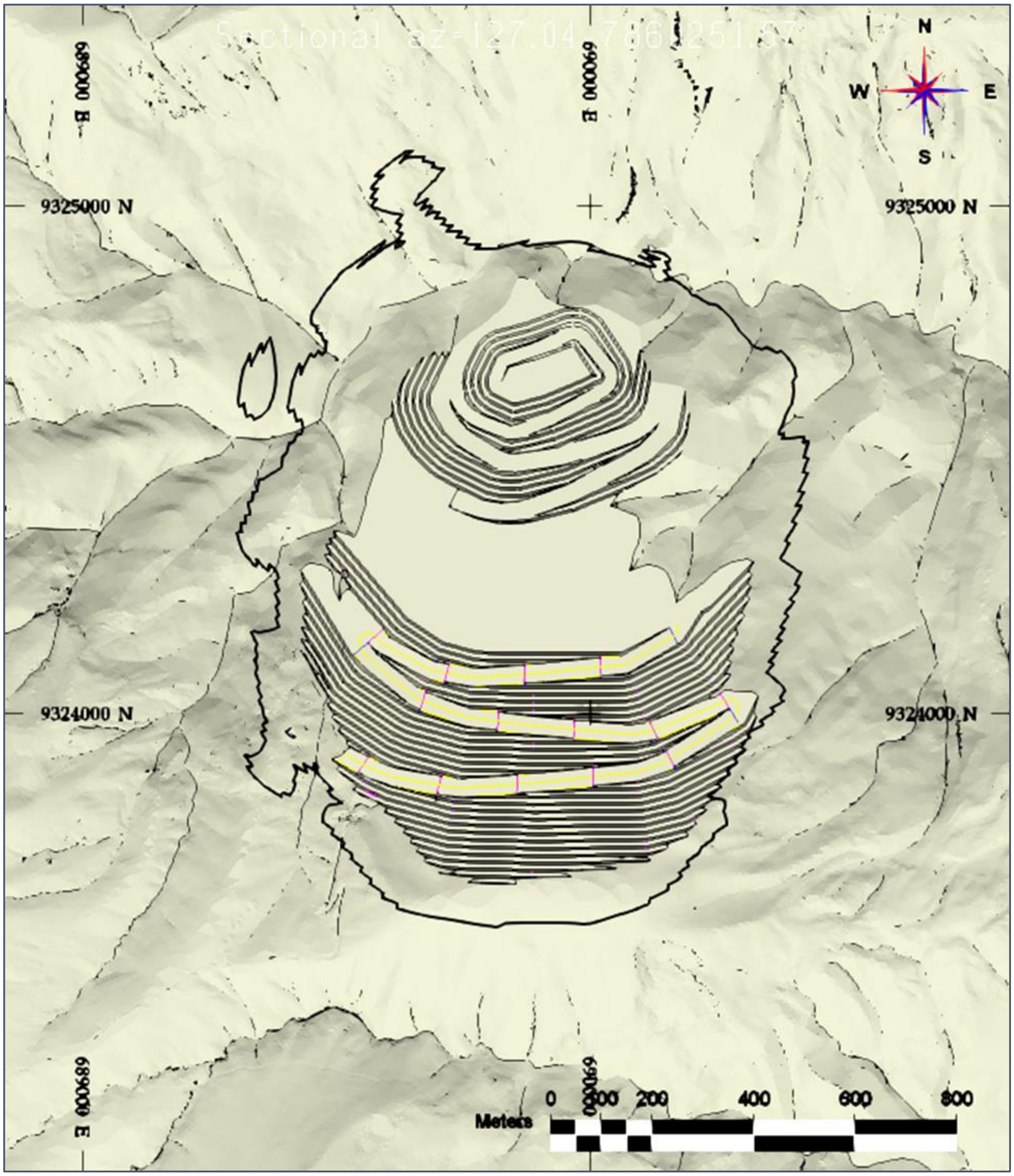
Figure 16-21: Sur Phase 2 Design



Source: AGP, 2024.

Phase 3 in Sur expands to the south with a top elevation of 3,460 m and stops at the 3,100 m level. Ramp access is maintained in the south wall to allow later phases to be developed. The Phase 4 design is shown in Figure 16-22.

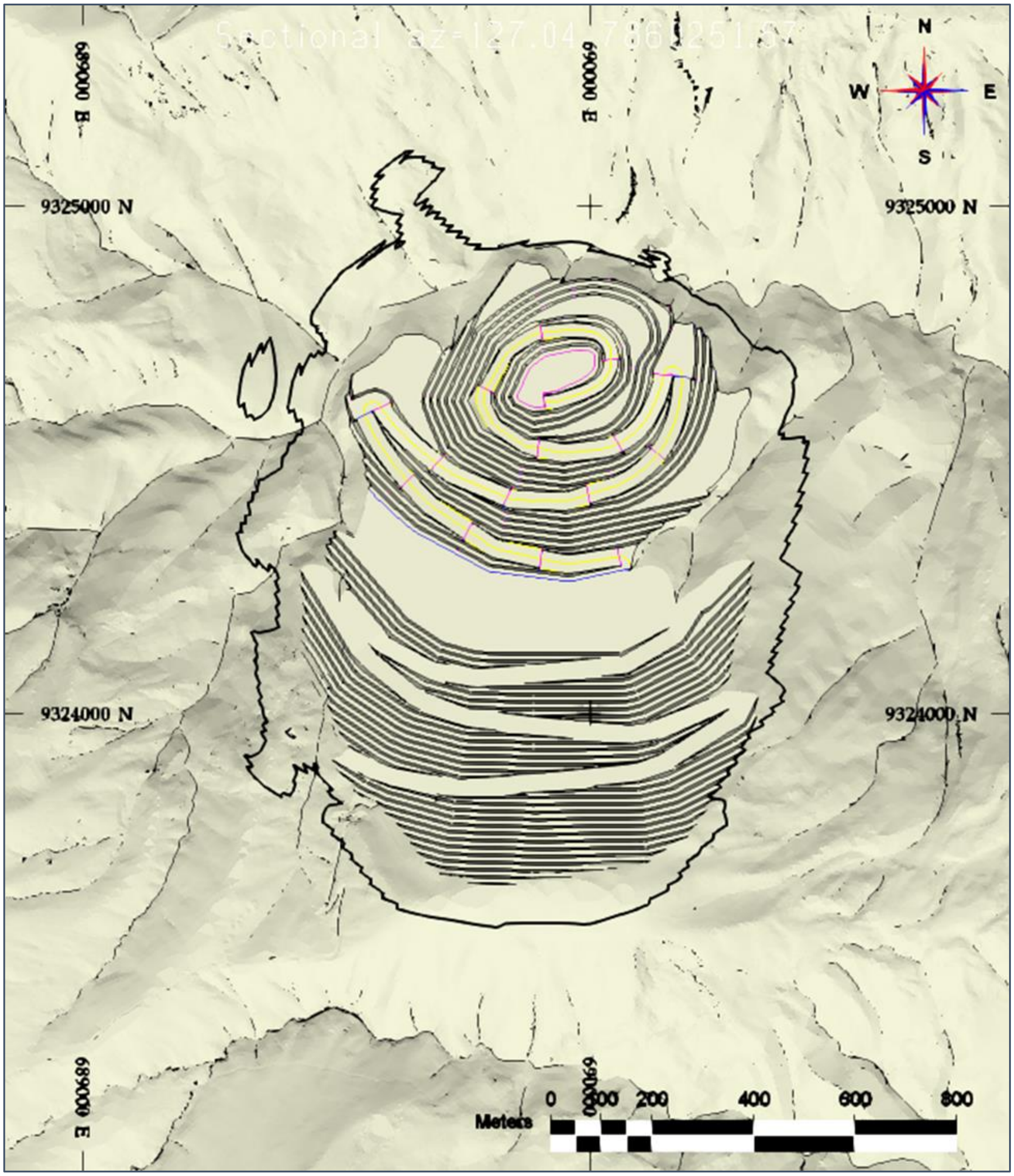
Figure 16-22: Sur Phase 3 Design



Source: AGP, 2024.

In the same manner as Phase 1 and 2 in Sur, Phase 4 deepens the pit from Phase 3. This phase starts at the 3,085 m level and goes to a depth of 2,875 m. Ramp access is maintained for a tunnel or truck haulage to Norte over the ridge. The design for Phase 4 is shown in Figure 16-23.

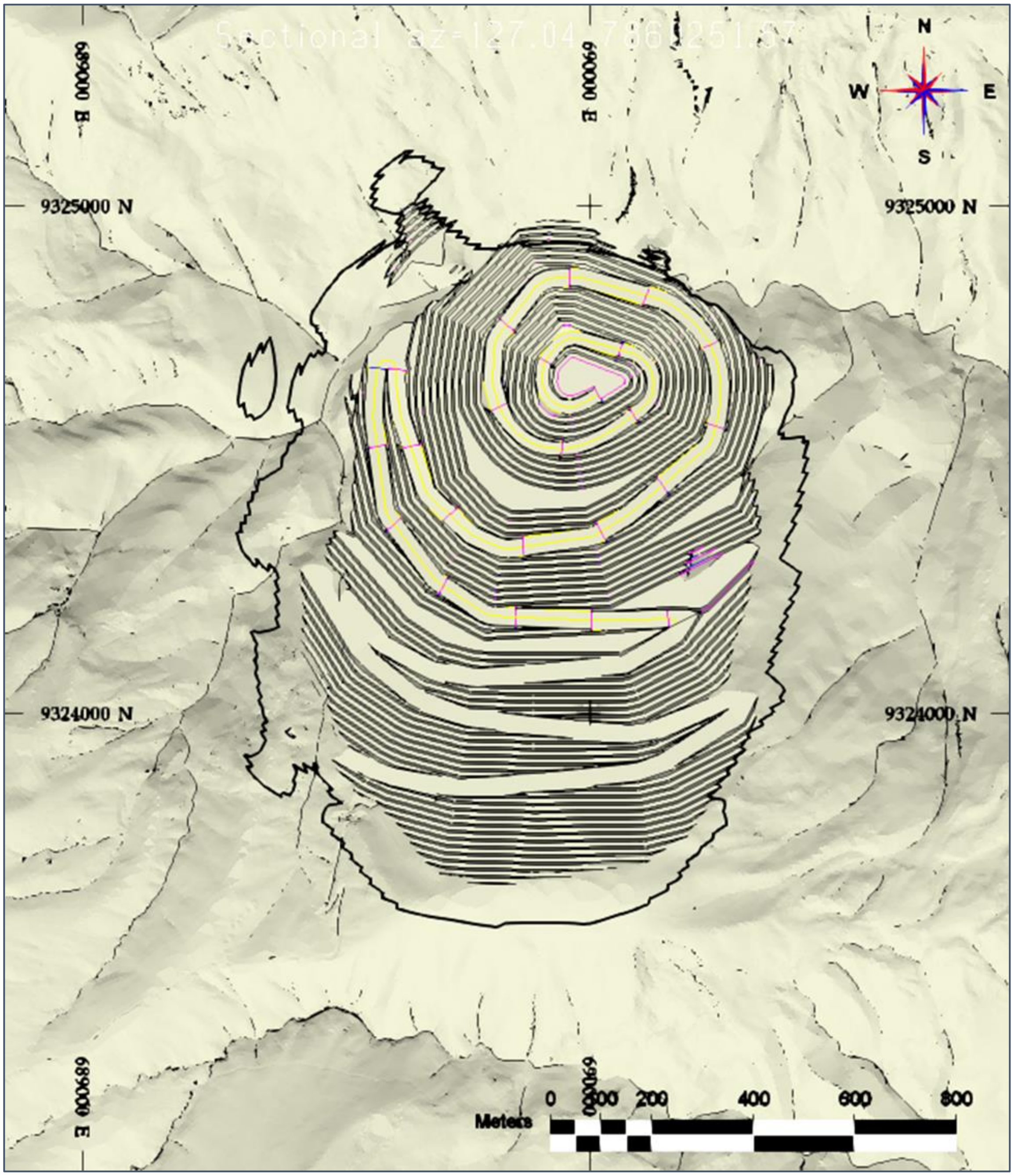
Figure 16-23: Sur Phase 4 Design



Source: AGP, 2024.

Sur Phase 5 deepens Phase 4 and starts at 3,085 m but goes to 2,770 m as its bottom. Ramp access is maintained up the slope. Sur Phase 5 is shown in Figure 16-24.

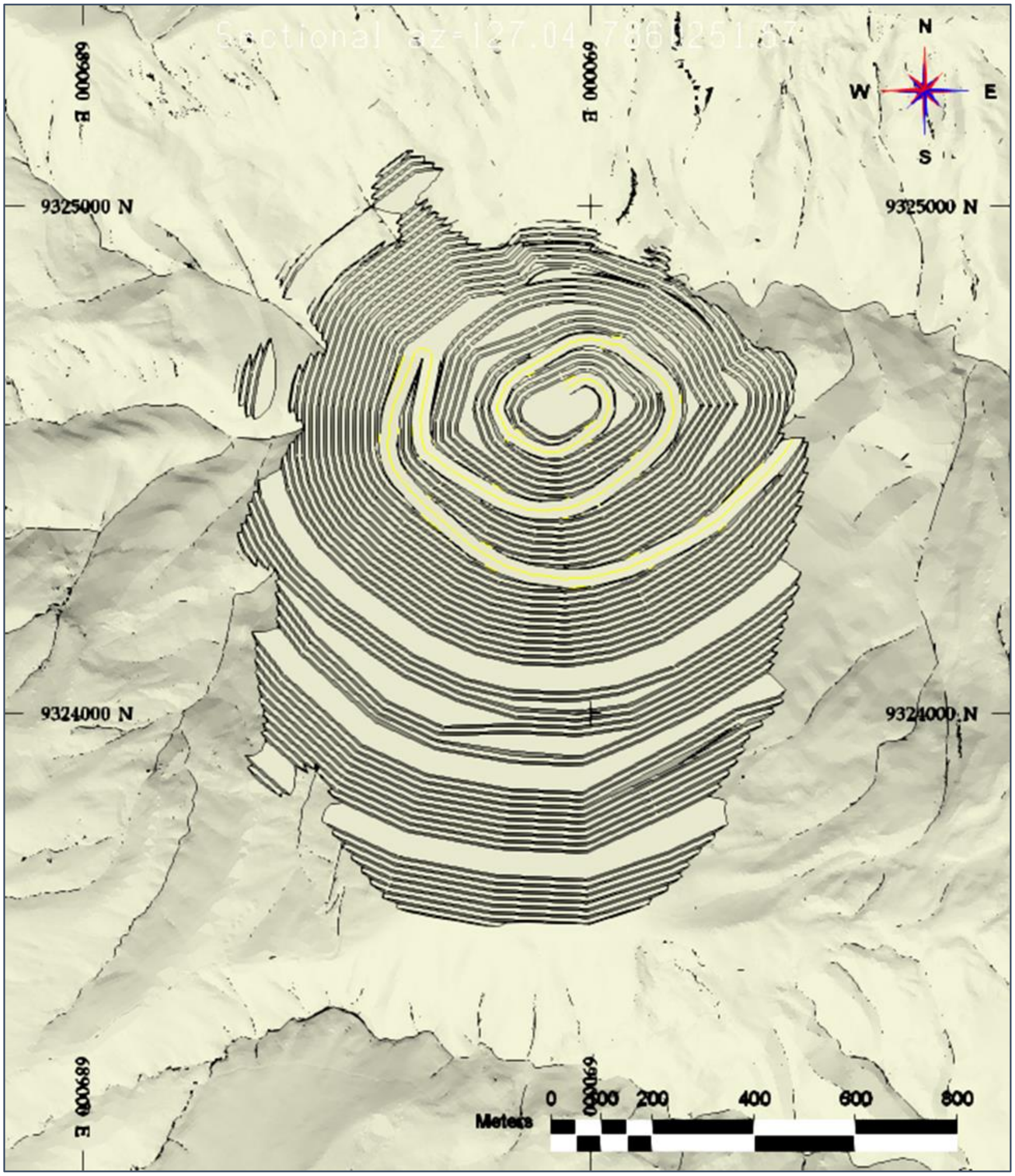
Figure 16-24: Sur Phase 5 Design



Source: AGP, 2024.

The final phase of Sur is Phase 6. It has a top elevation of 3,460 m and a bottom elevation of 2,710 m. No additional ramp access is left in the walls except for the one connecting to the portal to Norte for mill feed and waste. The Sur Phase 6 design is shown in Figure 16-25.

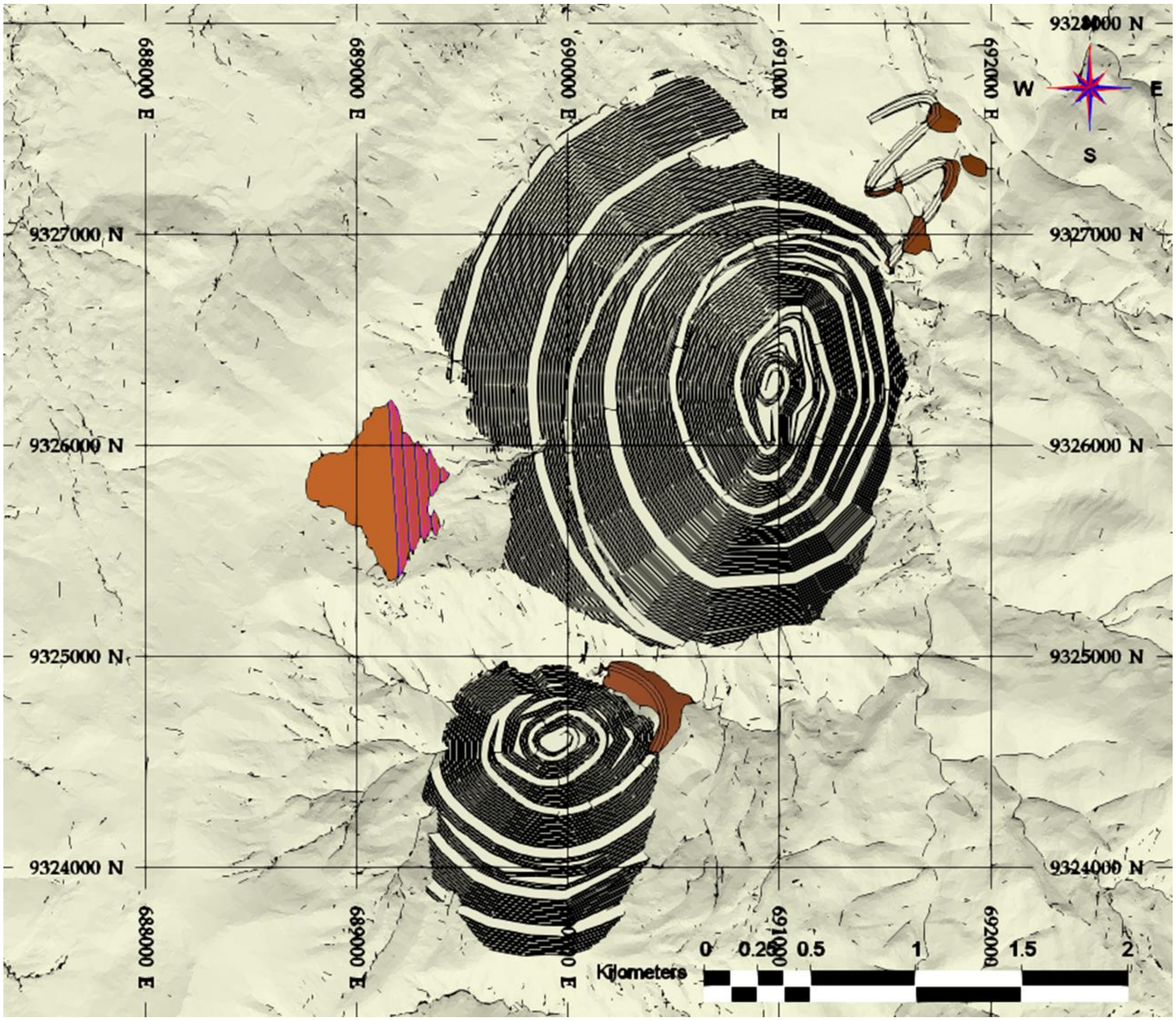
Figure 16-25: Sur Phase 6 Design



Source: AGP, 2024.

To illustrate further the Sur access to the portal and converyor to Norte, Figure 16-26 was prepared.

Figure 16-26: Sur Access Ramp



Source: AGP, 2024.

16.7.3 Phase Volumetrics

The pit phase volumetrics by pit area are shown in Table 16-11 and Table 16-12.

The Norte pit phases are shown with NSR grade blocks in vertical section view in Figure 16-27. The Sur pit phases are shown in Figure 16-28.

Table 16-11: Norte Pit Phase Volumetrics

Phase	Mill Feed						Waste (kt)	Total Material (kt)	Strip Ratio
	Feed (kt)	NSR (US$/t)	Cu Dil (%)	Au Dil (ppb)	Ag Dil (ppm)	As Dil (ppm)			
PH1	44,500	36.00	0.48	108	2.27	238.25	31,800	76,200	0.71
PH2	15,500	25.14	0.34	82	2.64	233.09	15,400	30,900	0.99
PH3	84,400	32.83	0.45	86	1.81	230.24	51,900	136,300	0.61
PH4	70,600	28.23	0.39	71	1.99	195.02	85,100	155,800	1.21
PH5	88,200	28.55	0.40	75	1.60	211.83	82,600	170,800	0.94
PH6	139,500	26.37	0.37	59	1.74	202.88	103,400	243,000	0.74
PH7	77,600	23.68	0.34	61	1.49	208.54	202,500	280,100	2.61
PH8	200,100	26.43	0.38	55	1.59	191.28	181,100	381,200	0.91
PH9	68,000	23.06	0.34	50	1.61	227.38	368,900	436,800	5.43
P10	142,700	25.00	0.36	52	1.54	201.85	252,300	395,000	1.77
Total	931,100	27.08	0.38	64	1.70	207.41	1,375,000	2,306,100	1.48

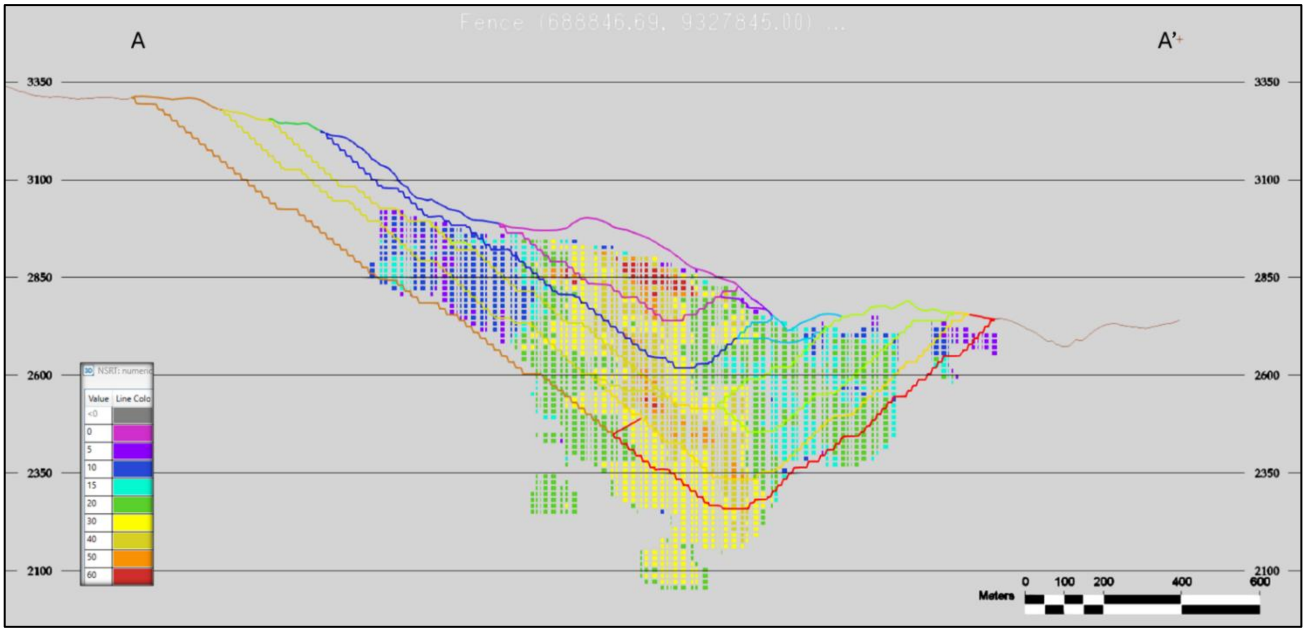
Note: All figures are rounded to reflect the relative accuracy of the estimate. Totals may not sum due to rounding as required by reporting guidelines.

Table 16-12: Sur Pit Phase Volumetrics

Phase	Mill Feed						Waste (kt)	Total Material (kt)	Strip Ratio
	Feed (kt)	NSR (US$/t)	Cu Dil (%)	Au Dil (ppb)	Ag Dil (ppm)	As Dil (ppm)			
PH1	11,700	21.47	0.26	107	1.31	13.39	9,700	21,400	0.83
PH2	32,100	18.22	0.22	94	1.29	36.87	2,800	34,900	0.09
PH3	15,300	22.18	0.28	95	1.20	12.65	46,500	61,800	3.03
PH4	39,100	18.05	0.22	92	1.18	39.83	6,500	45,600	0.17
PH5	73,100	19.23	0.24	93	1.18	39.51	14,100	87,200	0.19
PH6	73,800	18.34	0.23	85	1.07	39.25	93,300	167,000	1.27
Total	245,100	18.93	0.24	91	1.17	36.21	172,900	417,900	0.71

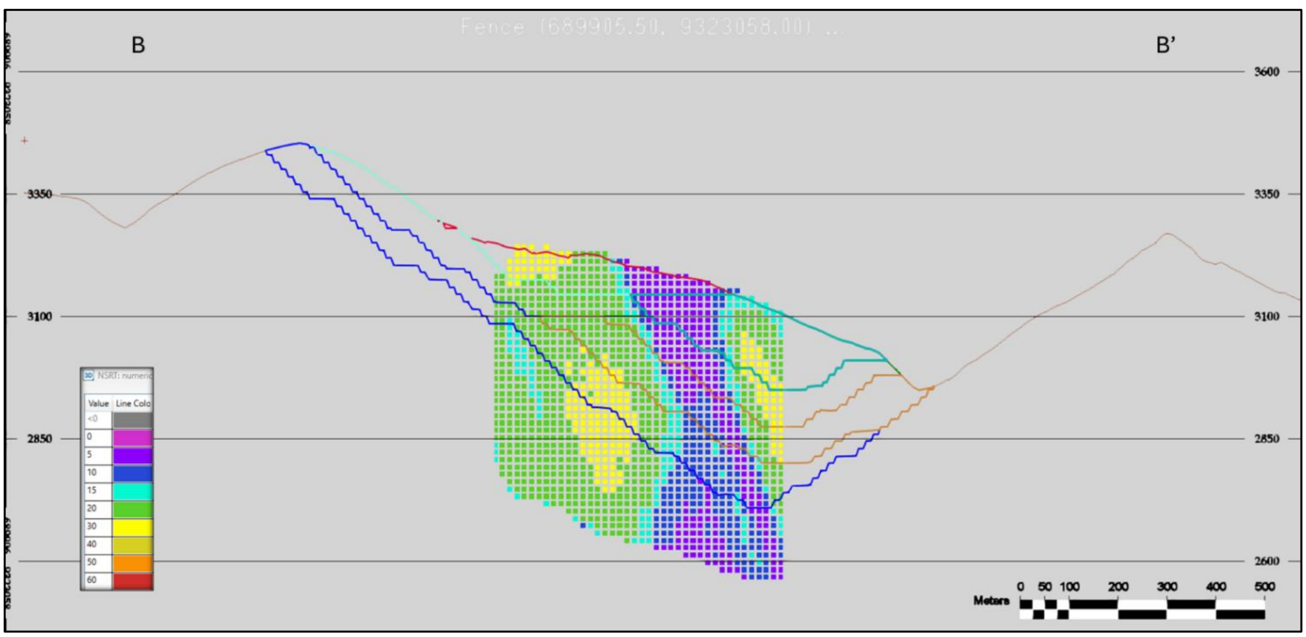
Note: All figures are rounded to reflect the relative accuracy of the estimate. Totals may not sum due to rounding as required by reporting guidelines.

Figure 16-27: Norte Pit Phases with NSR Block Grades



Source: AGP, 2024.

Figure 16-28: Sur Pit Phases with NSR Block Grades



Source: AGP, 2024.

16.7.4 Cut-Off Grades

The mine planning has been performed based on marginal cut-offs applied to the NSR grade item. Feed and stockpile material was routed by period, and the cut-off for each period is different due to location of mill feed in the pit, head grades, geometallurgical parameters, and overall mill feed availability. The marginal cut-off is the net value (NV), which is the NSR minus the sum of the mill feed based operating costs, which are processing, G&A, and tailings management. The global average for mill feed operating costs is US$7.26/t. These costs vary by material type, so the cut-off in terms of NV can vary by location and period.

Within the ultimate pit, a positive Net Value was required for material to be processed as mill feed.

16.8 Production Schedule

The mine plan presented in this Report was developed using Hexagon MinePlan 3D's Schedule Optimizer and Whittle Consultings Prober software. Prober established the annual surfaces and adjustments were made in Schedule Optimzer to balance tonnages. Descent rates were generally around 5-7 benches a year with a maximum of 14 benches in pioneering where the benches contained very little material. The mine is scheduled to work 365 (d/a), with five days worth of delay time due to weather disruptions. The plant is scheduled to operate 365 d/a.

16.8.1 Pre-Production

Three years of pre-production mining activities are required to carry out the following tasks:

- Develop 8 km of cut-and-fill haul roads to connect the upper elevations of the Norte Phases 1, 2 and 3 to the truck shop area, and crusher/conveyor system and to the TSF. Year -3 activities consist of road building/pioneering primarily.
- Strip 65.1 Mt of material from the pit, exposing initial mill feed material, and hauling the waste directly to the TSF until the crusher/conveyor system is available in Year -2.
- Stockpile 1.85 Mt of mill feed at the crusher for use in Year 1 onwards.

16.8.2 Production

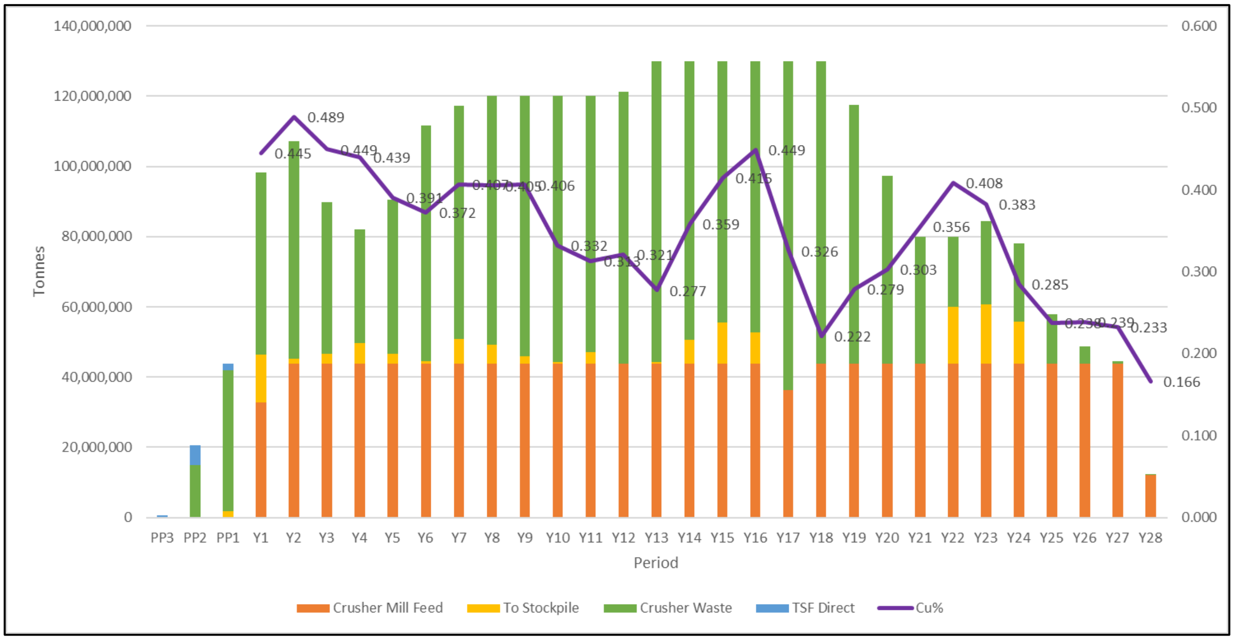
Mill feed delivery to the crusher in the first production year is forecast at 32.85 Mt. In production Year 2 onwards the full 43.8 Mt (120,000 t/d) will be delivered to the crusher area. In year 17 only 36.4 Mt will be delivered as the sequence did not allow full production to be achieved even with stockpiling at the crusher. This is also the year that Sur comes online as part of the production profile.

The last year of production, Year 28, will be a partial year with 12.0 Mt processed. The mine material movement is shown in Table 16-13 and Figure 16-29. Mill feed tonnes and grades are shown in Table 16-14. Select end-of-period surfaces are shown in Figure 16-30 to Figure 16-38.

Table 16-13: Mine Material Movement Summary

Year	Mill Feed (kt)	To Stockpile (kt)	From Stockpile (kt)	Crusher Waste (kt)	TSF Waste Direct (kt)	Total Material Movement (kt)	Stockpile Balance (kt)
-3	0	0	0	0	600	600	0
-2	0	0	0	15,000	5,500	20,500	0
-1	0	1,900	0	40,100	2,000	44,000	1,900
1	32,900	13,600	0	51,900	0	98,400	15,500
2	43,800	1,400	0	62,000	0	107,200	16,800
3	43,800	2,800	0	43,300	0	89,900	19,600
4	43,800	5,800	0	32,500	0	82,100	25,400
5	43,800	2,800	300	44,000	0	90,600	27,900
6	43,800	600	5,900	67,200	0	111,600	22,600
7	43,800	7,000	0	66,500	0	117,300	29,600
8	43,800	5,400	300	70,800	0	120,000	34,600
9	43,800	2,000	100	74,200	0	120,000	36,600
10	43,800	600	10,100	75,600	0	120,000	27,100
11	43,800	3,300	10,300	72,900	0	120,000	20,100
12	43,800	0	5,100	77,500	0	121,300	15,000
13	43,800	500	5,400	85,700	0	130,000	10,100
14	43,800	6,900	4,400	79,300	0	130,000	12,600
15	43,800	11,700	1,900	74,500	0	130,000	22,400
16	43,800	8,900	10,600	77,300	0	130,000	20,800
17	36,400	0	20,800	93,600	0	130,000	0
18	43,800	0	0	86,200	0	130,000	0
19	43,800	0	0	73,800	0	117,600	0
20	43,800	0	0	53,500	0	97,300	0
21	43,800	0	0	36,200	0	80,000	0
22	43,800	16,200	0	20,000	0	80,000	16,300
23	43,800	17,000	0	23,600	0	84,400	33,300
24	43,800	12,000	0	22,300	0	78,100	45,300
25	43,800	0	3,800	14,000	0	57,800	41,500
26	43,800	0	5,800	4,800	0	48,600	35,700
27	43,800	0	26,500	700	0	44,500	9,200
28	12,000	0	9,100	200	0	12,200	0
Total	**1,176,200**	**120,300**	**120,200**	**1,539,200**	**8,100**	**2,844,000**	**0**

Figure 16-29: Mine Material Movement



Source: AGP, 2024.

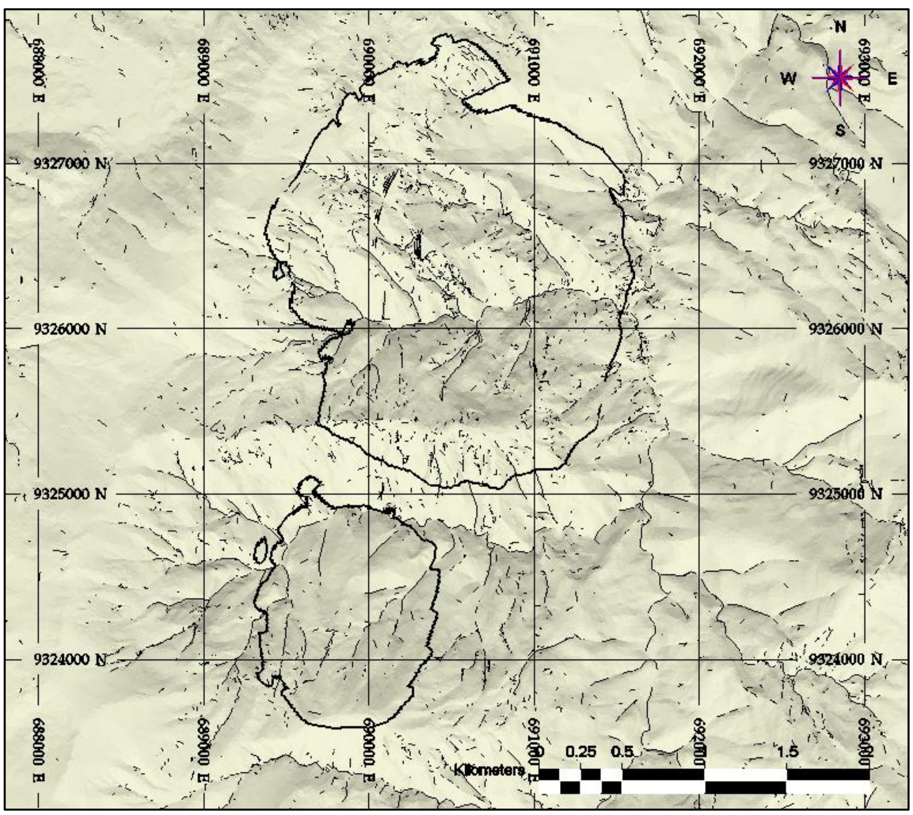
Table 16-14: Mill Feed Summary with Diluted Grades

Year	Mill Feed (kt)	NSR ($/t)	Cu (%)	Au (g/t)	Ag (g/t)	As (g/t)
1	32,900	33.20	0.45	0.10	2.61	304.23
2	43,800	36.24	0.49	0.09	2.00	237.76
3	43,800	33.43	0.45	0.09	1.70	185.87
4	43,800	32.15	0.44	0.08	2.03	233.00
5	43,800	28.38	0.39	0.07	1.98	204.21
6	43,800	26.58	0.37	0.08	1.61	188.96
7	43,800	29.43	0.41	0.07	1.54	200.37
8	43,800	28.16	0.41	0.06	1.89	293.68
9	43,800	29.74	0.41	0.07	1.90	134.85
10	43,800	23.32	0.33	0.06	1.54	176.35
11	43,800	21.62	0.31	0.05	1.38	237.78
12	43,800	22.46	0.32	0.06	1.52	186.13
13	43,800	18.23	0.28	0.04	1.38	240.94
14	43,800	24.77	0.36	0.05	1.59	196.99
15	43,800	29.90	0.41	0.07	1.72	166.37
16	43,800	32.60	0.45	0.07	1.77	164.68
17	36,400	23.37	0.33	0.06	1.50	152.66

Year	Mill Feed (kt)	NSR ($/t)	Cu (%)	Au (g/t)	Ag (g/t)	As (g/t)
18	43,800	16.79	0.22	0.07	1.19	59.51
19	43,800	19.07	0.28	0.05	1.40	209.36
20	43,800	20.61	0.30	0.04	1.41	185.01
21	43,800	24.59	0.36	0.05	1.57	212.88
22	43,800	29.19	0.41	0.07	1.70	200.77
23	43,800	28.40	0.38	0.08	1.50	113.07
24	43,800	21.76	0.28	0.08	1.30	71.44
25	43,800	18.90	0.24	0.09	1.22	52.20
26	43,800	19.06	0.24	0.09	1.27	49.29
27	43,800	18.23	0.23	0.08	1.09	45.20
28	43,800	13.52	0.17	0.07	1.00	37.49
Total	**1,176,200**	**25.38**	**0.35**	**0.07**	**1.59**	**171.74**

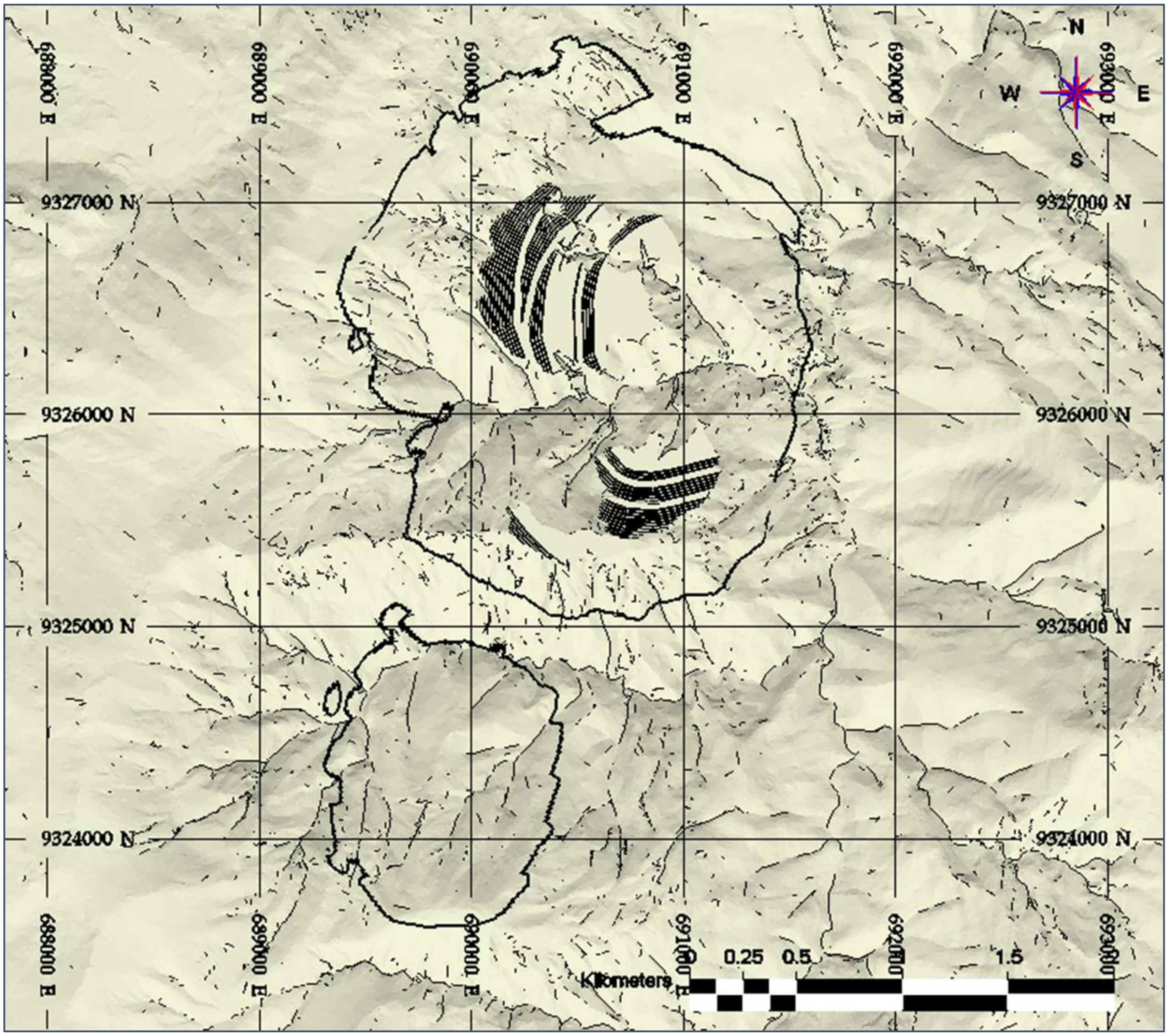
Mill feed tonnages by classification are as follows: **1.** Measured = 33.5% (393.5 Mt grading 0.42% copper). **2.** Indicated = 38.8% (456.6 Mt grading 0.36% copper). **3.** Inferred = 27.7% (326.1 Mt grading 0.25% copper).

Figure 16-30: Year 3 End of Period Surface



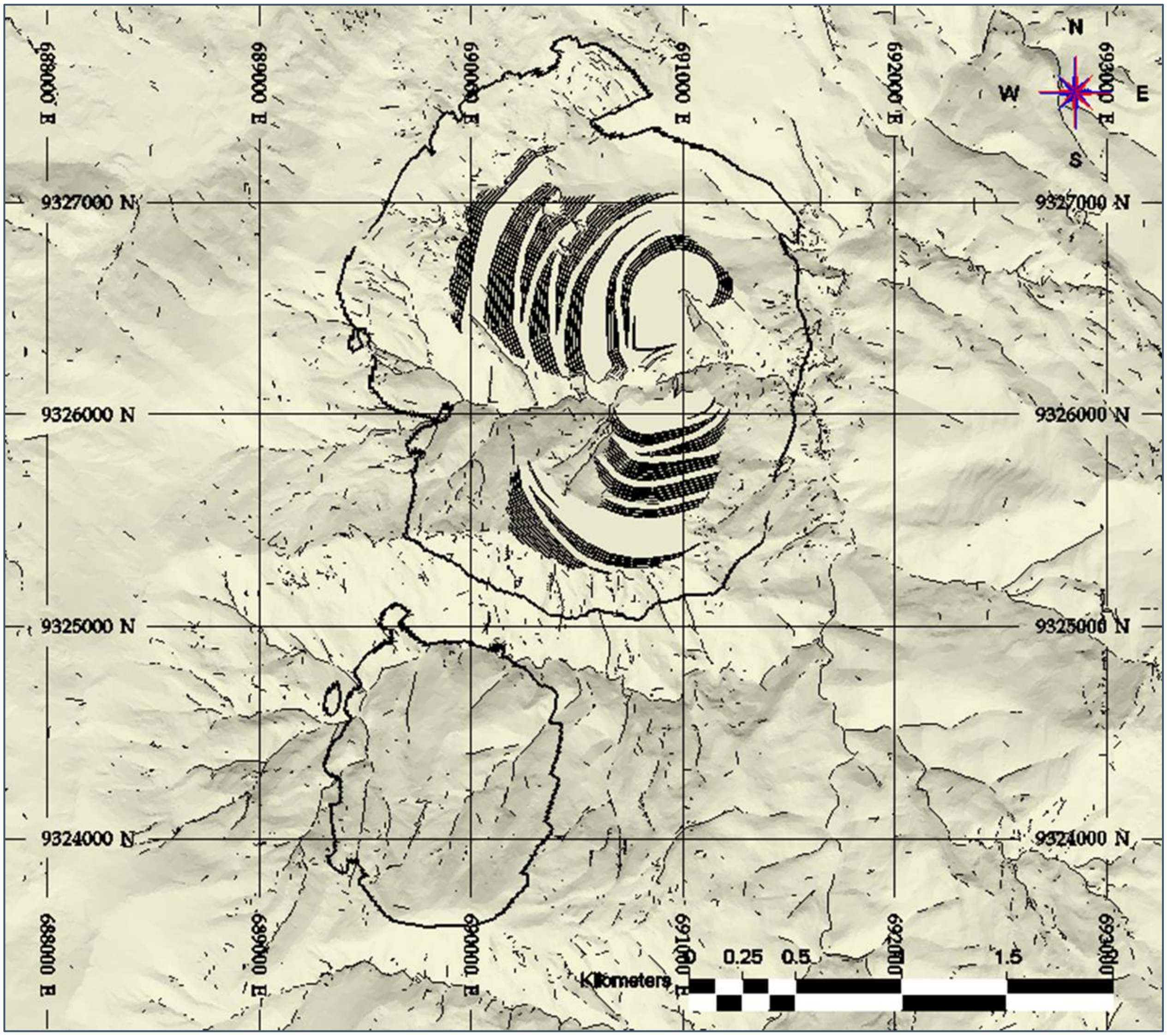
Source: AGP, 2024.

Figure 16-31: Year -1 End of Period Surface



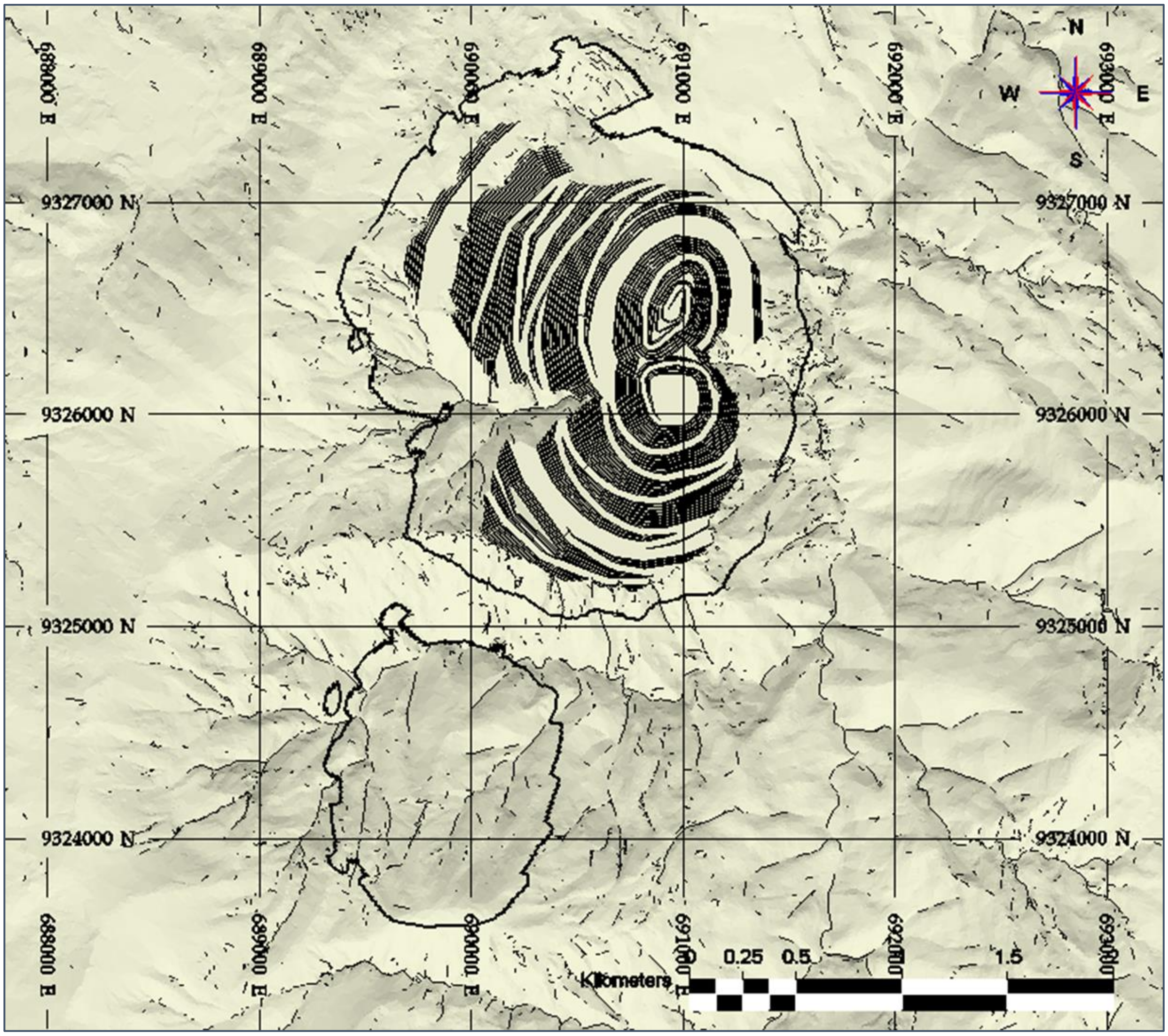
Source: AGP, 2024.

Figure 16-32: Year 1 End of Period Surface



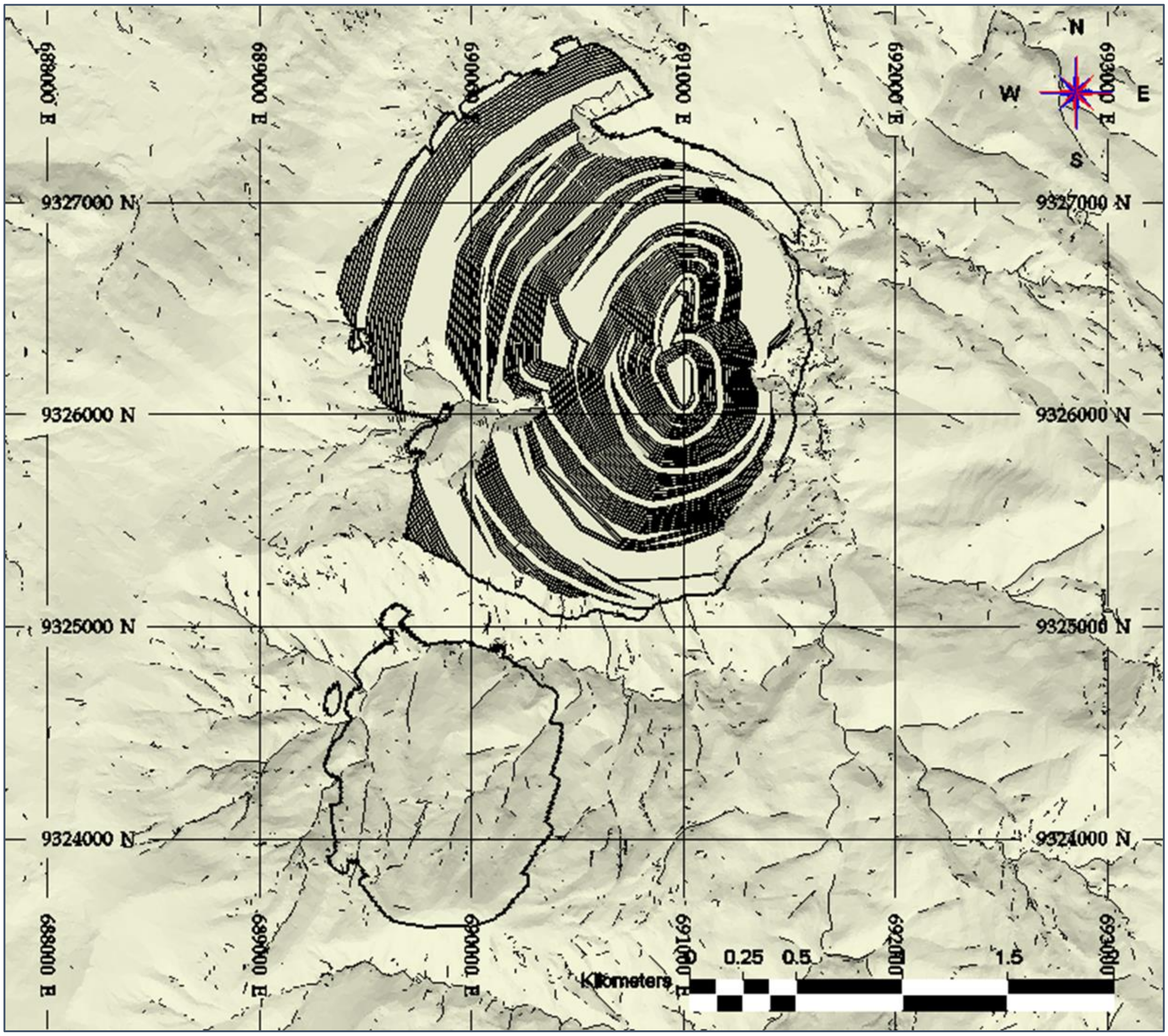
Source: AGP, 2024.

Figure 16-33: Year 5 End of Period Surface



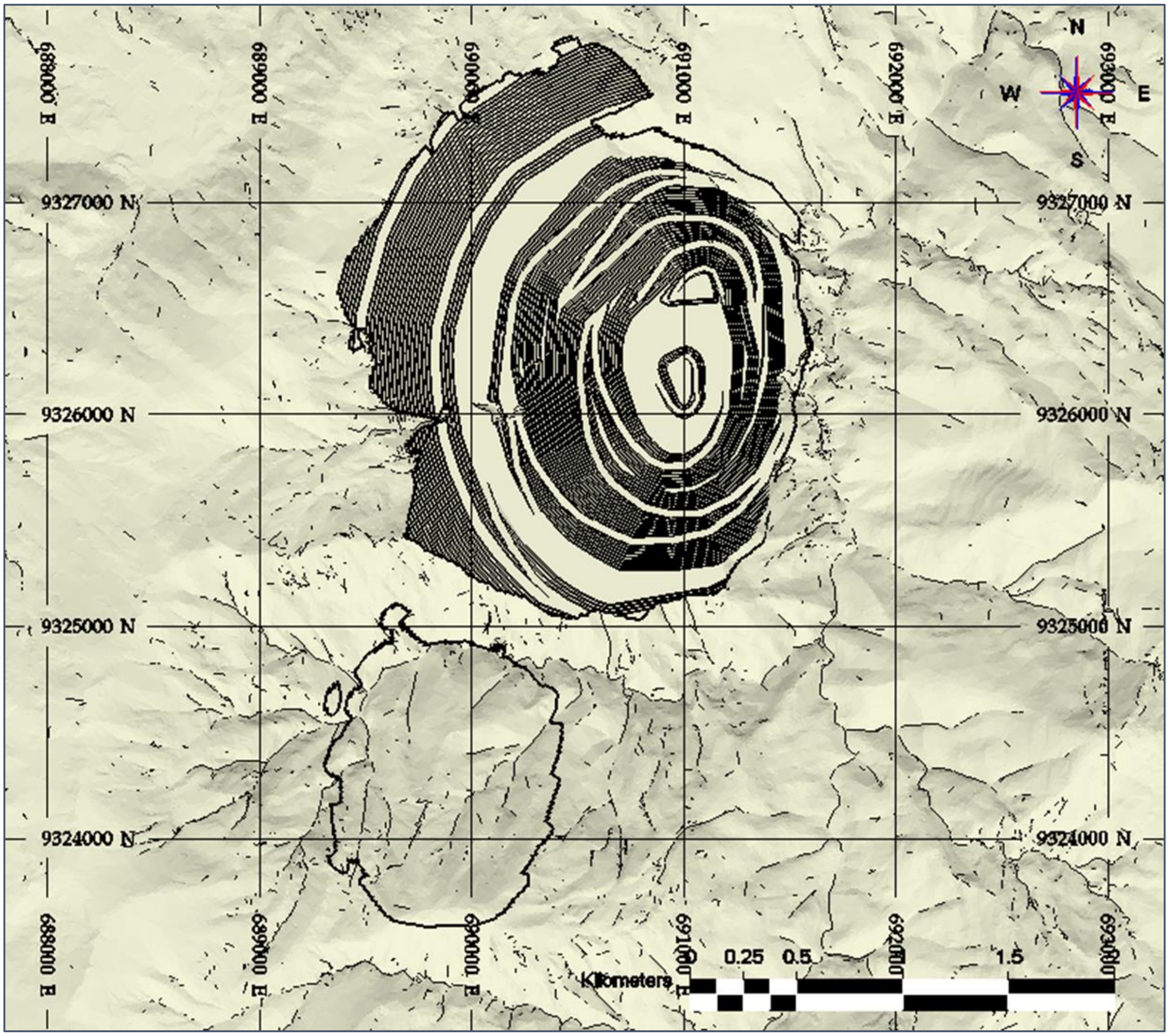
Source: AGP, 2024.

Figure 16-34: Year 10 End of Period Surface



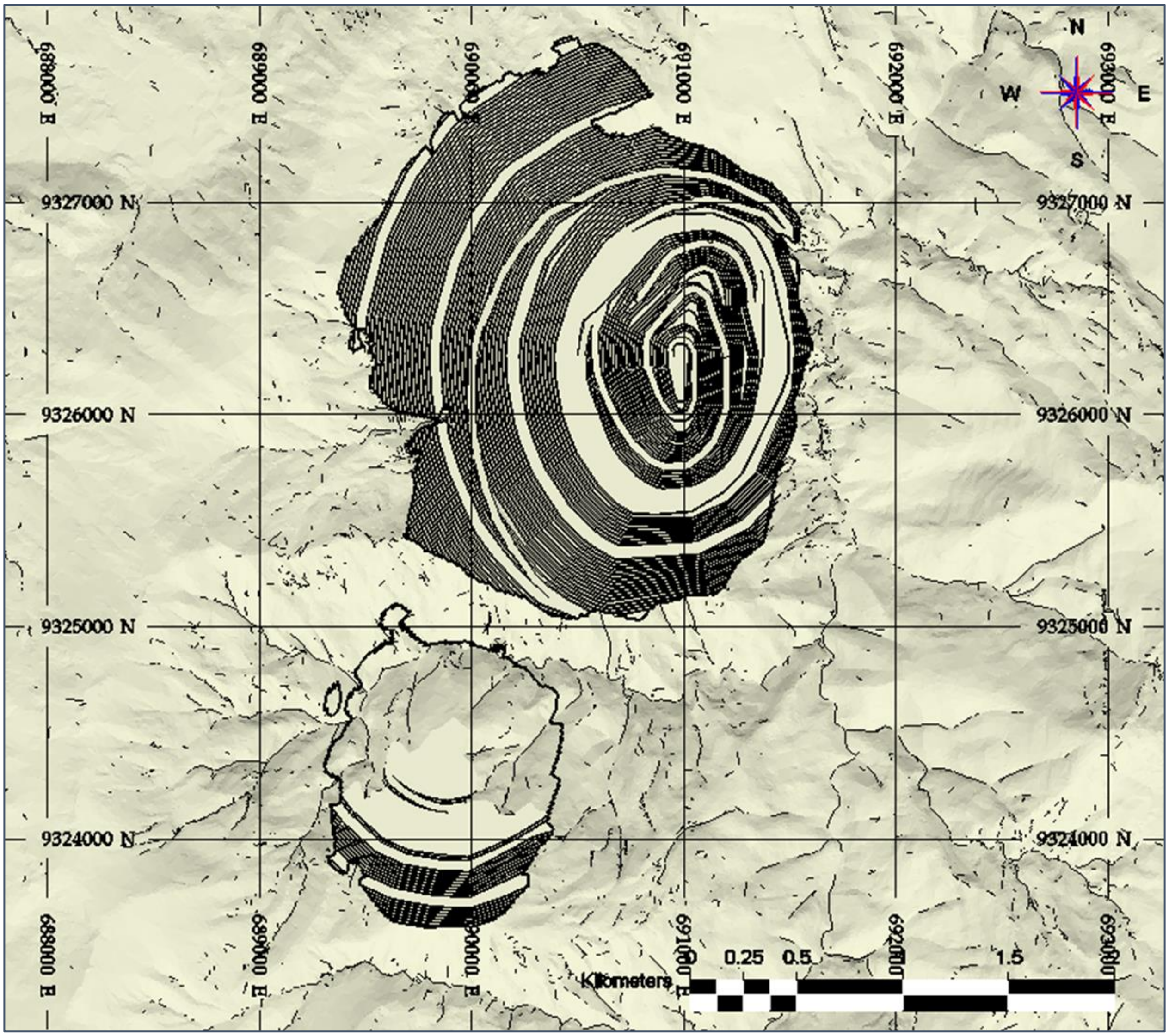
Source: AGP, 2024.

Figure 16-35: Year 15 End of Period Surface



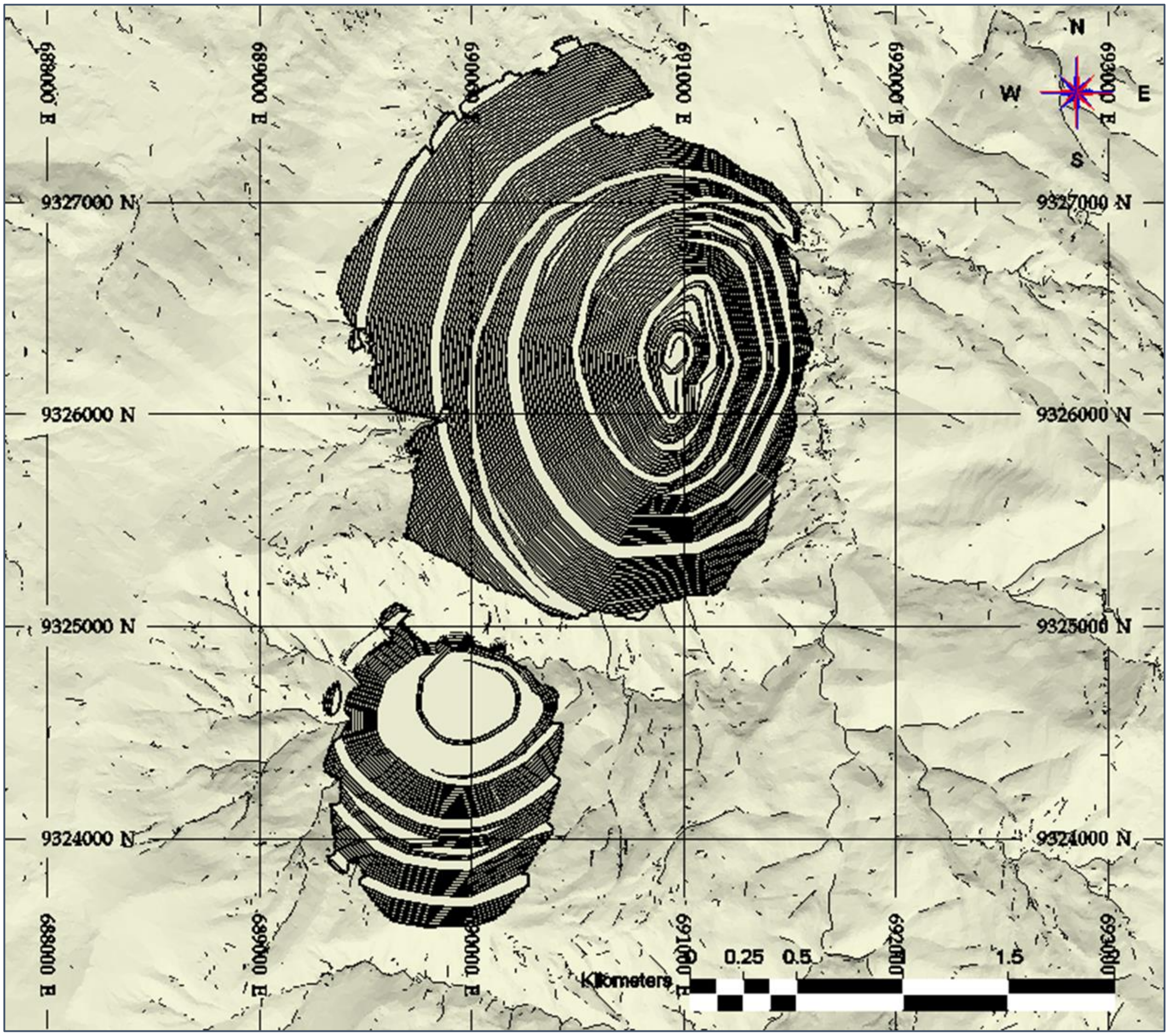
Source: AGP, 2024.

Figure 16-36: Year 20 End of Period Surface



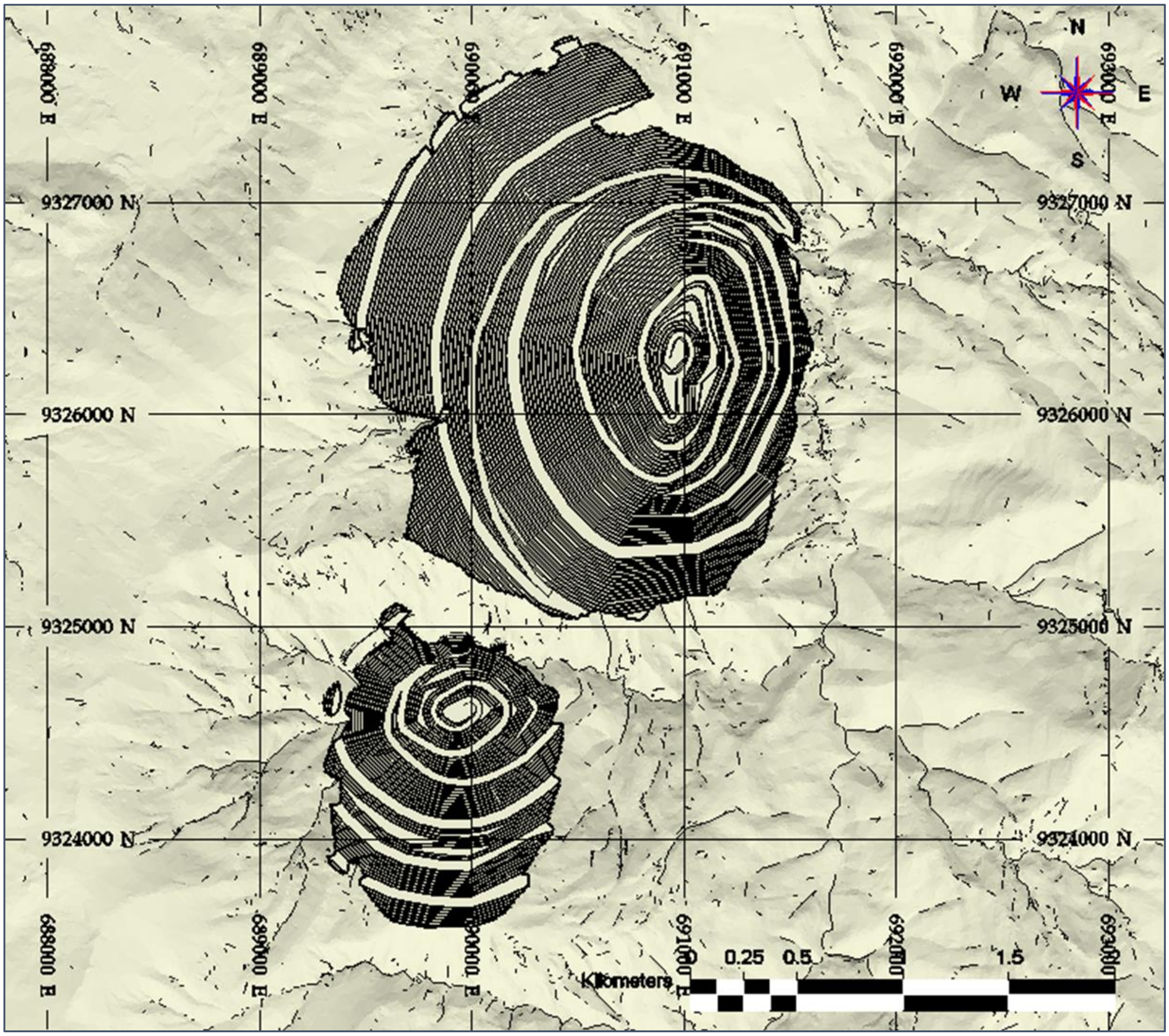
Source: AGP, 2024.

Figure 16-37: Year 25 End of Period Surface



Source: AGP, 2024.

Figure 16-38: Year 28 End of Period Surface



Source: AGP, 2024.

16.9 Mining Operations

The Cañariaco deposits are large and relatively continuous in grade, allowing a bulk mining scenario. The pit operations will work two 12-hour shifts per day with four crews on a standard rotation. Engineering, geology and some operations supervisory / support positions will be on day only 12-hour shifts which will also rotate weekly.

The following sub-sections discuss the selection of equipment and peak requirements. A summary table of the primary production equipment is provided in Table 16-15.

Table 16-15: Primary Production Equipment

Equipment Type	Equipment Class	Maximum Fleet Size
Haul truck	290t	38 (Year 18+)
Haul truck	227t	34 (Year 2+)
Haul truck	40t	43 (Year -1)
Hydraulic shovel	38 m^3	6 (Year 18+)
Front-end loader	33 m^3	2 (Year -1+)
Track dozer	4.7-m blade	7 (Year -1+)
Grader	4.3-m blade	3 (Year -3+)
Blast hole drill	270 mm bit	6 (Year 6+)
Secondary drill	160 mm bit	3 (Year 1+)

16.9.1 Loading

Production loading duties will be performed by a variety of loading tools over the life of the mine. The largest will be the 38 m^3 electric hydraulic face shovels, with a 33 m^3 front-end loader assisting with pit loading as well as mill feed rehandle from the short-term stockpile. The smaller truck fleet will be loaded by 6.8 m^3 excavators in the narrower working areas. The larger equipment is well matched to the 15-m bench height and the smaller equipment would mine this in various flitch heights. The peak loader requirements are six shovels and two front end loaders but the fleet in the first 5 years is only 3 shovels and 2 loaders.

16.9.2 Hauling

The geometric shapes of the pit phases, mountainous terrain and relative location of the mill feed and waste crushers will result the need for a mixed fleet to properly mine the deposit and keep operating costs lower as the mine advances. The initial mine fleet will be the 40t rigid trucks (Scania) which are common in Peru. These will be used to pioneer roads and initial mining. At various points in the mine life there is a need for additional capacity and these trucks are employed in a “peak shaving” mode in those instances. They can be mobilized quickly and more capital efficient then adding larger trucks.

The initial large truck fleet is the 227t class. This will be replaced with larger 290t class trucks as the mine stripping requirements increase and the initially purchased trucks are due for replacement. The 290t trucks will then be used for the remainder of the mine life.

16.9.3 Drilling and Blasting

Production drilling will be performed on 15-m benches. The drill bit size selected for main production holes was 270 mm diameter. A production drill rig with 38,000 kg pulldown was selected that could drill holes in a single pass, without the need to add or remove steel. A secondary drill rig capable of drilling 160-mm diameter holes was selected for pioneering work, assisting with production drilling, installing horizontal drains and secondary blasting as required. Pre-splitting has not been considered at this time. Peak requirements are six primary drills and three secondary drills.

The pattern sizes for mill feed material and waste are the same and were determined by targeting reasonable powder factor for the rock strengths and densities. The production drill patterns are shown in Table 16-16.

Table 16-16: Drill Pattern Specifications

Specification	Unit	Primary Drill	Secondary Drill
Bench height	m	15	15
Sub-drill	m	1.6	1.0
Blasthole diameter	mm	270	160
Pattern burden - staggered	m	7.9	5.0
Pattern spacing- staggered	m	9.1	5.8
Powder factor	kg/t	0.28	0.28

An opportunity exists to investigate semi-autonomous drilling during the next stage of project planning.

A bulk loaded blended emulsion product will be used for blasting and is expected to give better performance and have better water resistance compared to ANFO. The product selected is composed of 40% emulsion and 60% AN by weight and will have a loaded density of 1.18 g/cm^3.

The blasting cost is estimated based on a quotation from a local vendor. Unit costs for bulk, packaged and initiating explosives, delivered to site, were provided. The vendor also quoted a monthly service fee to cover the cost of capital and personnel to provide a full blasting service (priming, loading, stemming, sequencing, firing and magazine management). The blasting supplier will provide two mobile manufacturing units (MMUs), magazine storage, offices, storage tanks and pumps.

The mine will be responsible for providing the following at no cost to the blasting vendor: meals, accommodation, electricity, water, diesel and stemming aggregate, and any other special accessories.

16.9.4 Support and Ancillary Equipment

Roads, pit floors and dumps will be maintained by a fleet of track dozers, wheel dozers, and graders. Appropriate support equipment was selected for this size of mining operation.

16.9.5 Grade Control

Grade control will be performed by a group of geologists and geologic technicians within the mine operations department. Samples will be collected from the blastholes during the drilling process and delivered to the process facility for sample preparation and assay determinations. Assay results for total copper, gold, silver and arsenic, plus sequential copper determinations (acid soluble, cyanide soluble and residual copper) will be used to estimate recovered metals and NSR in a similar manner to the long-range planning process. An estimate of annual sample quantities was developed assuming all mill feed material plus 25% of waste blastholes would require assay determinations. No waste characterization determinations were considered. Assaying will be performed by the plant laboratory. A grade control block model will be developed and used to create 'diggable' homogeneous grade control 'packets' that will be uploaded to the shovels and loaders for 'stakeless' mineralized material and waste digging.

The grade control group will also be responsible for performing regular reconciliations between the resource model, the grade control model and process production reporting.

16.9.6 Pit Slope Monitoring

Deformation monitoring of the pit slopes during mining will be undertaken to:

- Maintain safe operational practices for personnel, equipment, and near-pit facilities;
- Provide warning of slope instability;
- Confirm design assumptions; and
- Provide geotechnical information for slope designs to assist in making subsequent modifications, should they be required, to achieve the desired slope performance.

A ground control management plan will be developed and implemented for the pit slopes of the proposed mine during operations. This plan will include:

- Daily visual inspections of pit crest and slopes by mine staff with results recorded in a slope hazard logbook to be reviewed on a regular basis by the site geotechnical engineer;
- Monitoring of slope movements using total stations to survey a network of reflector prisms;
- A trigger action response plan (TARP) associated with the slope monitoring; and
- A monitoring database to store the prism survey records with the ability to plot the time-series graphs.

The need for more complex monitoring systems, such as slope stability radar, LiDAR monitoring, or subsurface instrumentation, should be assessed throughout the mine's operation. If slope instabilities develop, the monitoring system should be upgraded to allow for continued safe operation of the mine.

16.10 Workforce

The peak mine operations workforce will consist of 526 hourly operators and maintenance workers and 61 staff. Additionally, there will be nine blasting contractors on site at all times. The peak total mine operations workforce in camp is 294 people.

16.11 Comments on Mining Methods

The steep topography in the upper elevations of the pit designs has resulted in designs and a mining schedule that utilizes smaller equipment to properly prepare the larger fleet for efficient operation. Significant pre-stripping was included in the mine schedule which helped advance the higher grade in the process schedule. This had positive impacts in the overall project economics. Refinements from future iterations of road access and phase designs may be able to improve upon the current designs and reduce capitalized stripping costs while still maintaining or improving the grade profile.

17 RECOVERY METHODS

17.1 Overview

The process flowsheet for the Cañariaco project is designed for 120,000 t/d at a 91.3% availability and was developed based on extensive metallurgical testwork completed on samples of Norte mineralized material and on the preliminary metallurgical testwork results for the Sur deposit. The unit operations selected are standard technologies typically used in copper processing plants for treating copper sulfide mineralized material for copper concentrate production.

The proposed flowsheet features conventional processes for the following circuits:

- One mill feed primary gyratory crusher near the Norte pit with a dedicated overland mill feed conveyor to the Mill feed stockpile at the process plant.
- One waste rock mineral sizer near the Norte pit with a dedicated overland waste rock conveyor to the CPSF.
- One gyratory crusher near the Sur pit for both mill feed and waste rock with a single mill feed and waste conveyor to the Norte crushing plant and a diverter chute to direct mill feed and waste rock to their respective conveyors at the Norte crushing plant, deferred until Sur mining commences in year 17 of operation.
- Single GMD SAG mill with pebble crushing and screen classification.
- Two lines of ball mills with cyclone classification.
- Rougher flotation followed by secondary regrind with cyclone classification.
- 3 stages of cleaner flotation, with cleaner scavenger flotation cells, for copper concentrate production.
- Copper concentrate thickener, filtration, and loadout facilities.
- Tailing thickener and filtration for dry stack tailings disposal.

17.2 Process Design Criteria

The project's key process design criteria were established after a review of available metallurgical testwork and comparable industry benchmarks. They are summarized in Table 17-1.

Table 17-1: Process Design Criteria

Description	Units	Value
Plant Throughput	kt/d	120
Life of Mine	Y	28
Plant Feed Grade, Year 1-10		
Copper, Year 1-10	%	0.41
Copper Equivalent (Cu, Au, Ag), Year 1-10	%	0.48
Plant Feed Grade, LOM		
Copper, LOM average	%	0.35
Copper Equivalent (Cu, Au, Ag), Average	%	0.41
Operating Hours per Year, Primary Crushing	h/a	6,570
Operating Hours per Year, Grinding and Flotation	h/a	8,000
Operating Hours per Year, Concentrate Filtration	h/a	7,358
Operating Hours per Year, Tailings Filtration	h/a	7,888
Bond Crusher Work Index (CWi), design	kWh/t	5.4
JK SMC test parameters Axb	-	52.5
Bond Ball Mill Work Index (BWi), design	kWh/t	12.2
Bond Abrasion Index (Ai), design	-	0.11
ROM Specific gravity (SG)	-	2.6
Comminution Circuit		
Crushing plant capacity, nominal	t/h	6,667
Crushing circuit product size, P_{80}	mm	85
Grinding circuit capacity, nominal	t/h	5,475
Grinding circuit configuration	-	SACB
Grinding circuit product size, P_{80}	µm	200
Classification cyclones O/F pulp density	%w/w	25
Copper Flotation Circuit		
Rougher Flotation Capacity, nominal	t/h	5,475
Rougher Flotation Stage Mass pull	%	15.5
Regrind Circuit Capacity, nominal	t/h	835
Copper Cleaner 1 Stage Mass Pull	%	19
Cleaner 1 Scavenger Stage Mass Pull	%	5
Copper Cleaner 2 Stage Mass Pull	%	58
Copper Cleaner 3 Stage Mass Pull	%	83
Copper Recovery, at Design Feed grade	%	90.3
Copper Concentrate Filtration		
Type of Filtration	-	Pressure filtration
Feed Rate	t/h	89
Filter Cake Moisture	% solids (w/w)	9
Tailings Filtration		
Type of Filtration	-	Pressure filtration
Feed Rate	t/h	5,386
Filter Cake Moisture	% solids (w/w)	15

17.3 Process Plant Description

The flowsheet selected is typical for treating porphyry copper sulfide mineralized material. It is aimed to process plant feed from the open pit mine to produce copper concentrate containing 26% Cu.

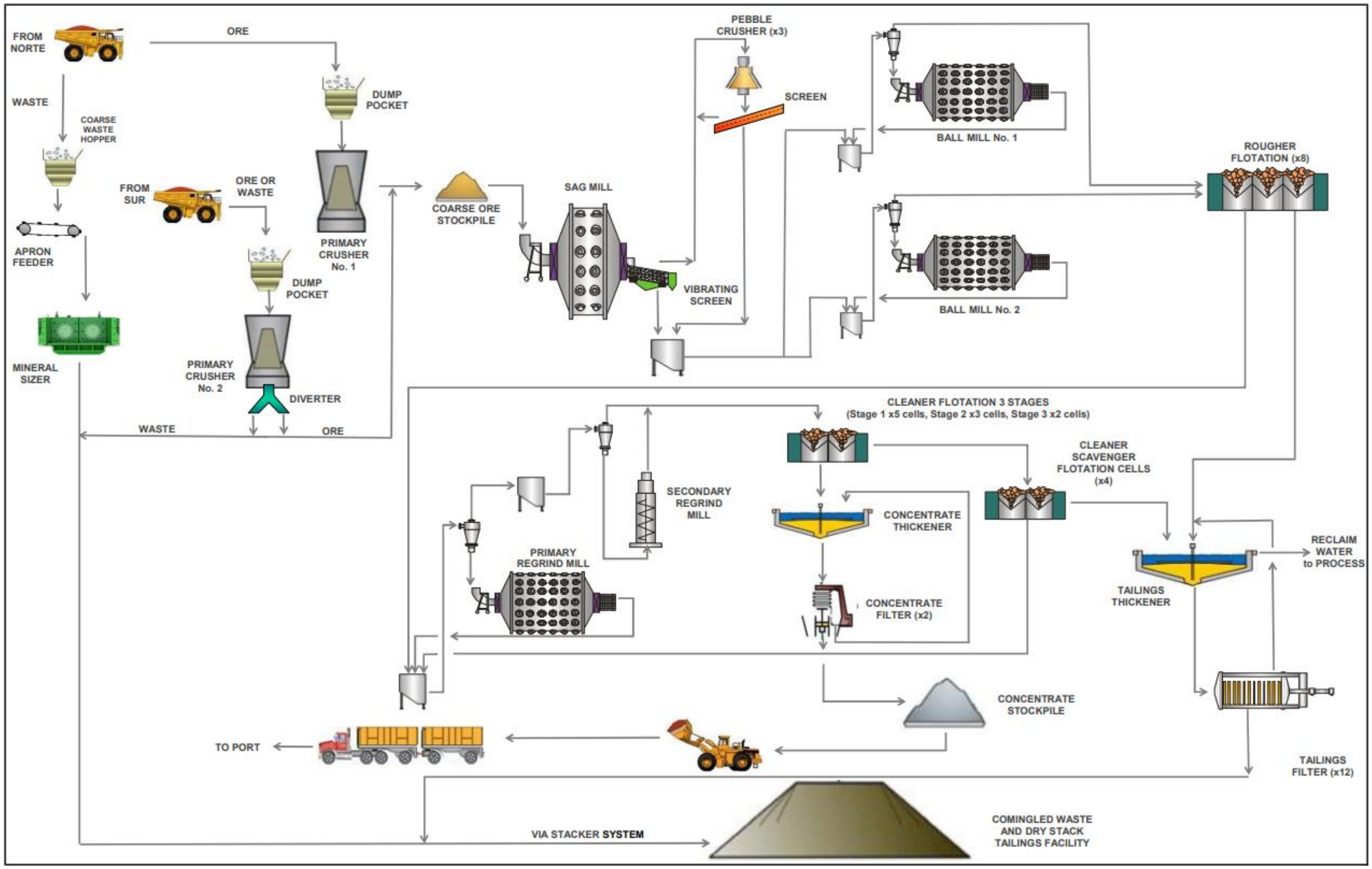
The equipment quantities will remain the same throughout the life of mine. The process design is comprised of the following units, process, and facilities:

- Two primary gyratory crushers for ROM mill feed, one for the Norte deposit and one for the Sur deposit in year 16;
- One primary mineral sizer for waste rock fed with waste rock from haul trucks;
- Feed overland conveyor system to transport mill feed to the coarse material stockpile at the process plant;
- Waste overland conveyor system to transport mine waste to the CPSF fed from the sizer from the Norte deposit and hauls trucks from the Sur deposit;
- Coarse mineralized material stockpile and reclaim system;
- Gearless Mill Drive Semi-autogenous grinding mill (GMD SAG) followed by ball mills with cyclone classification in a closed circuit;
- Rougher flotation followed by two-stage regrind of rougher concentrate;
- Three-stage cleaner flotation of regrind concentrate to achieve final concentrate grade;
- Cleaner–scavenger flotation to maximize copper recovery;
- Flotation concentrate thickening, filtering, and transport;
- Tailings thickening, filtering, and tailings stacking system at the CPSF;
- Reagent storage and distribution; and
- Water services and management (process water, treated water, raw water).

17.3.1 Process Flowsheet

An overall proposed flowsheet is shown in Figure 17-1.

Figure 17-1: Process Flowsheet



Source: Ausenco, 2024.

17.3.2 Crushing/Conveying

The Norte Pit mineralized material is hauled from the Norte pit and directly tipped into the mill feed gyratory crusher dump pocket and fed into the primary gyratory at a feed size (F_{80}) of 312 mm and crushed to product size (P_{80}) of 85 mm.

The waste rock of the Norte Pit is directly tipped into a waste rock dump pocket above an apron feeder which feeds the waste rock mineral sizer and sized to a P_{100} of 500 mm and conveyed to the CPSF.

The crushing circuit will have an availability of 75% and the total throughput of the mineralized material will have an average capacity of 6,667 t/h and the waste at 11,598 t/h. Mineralized material and waste rock is transferred via overland conveyors to their respective destinations. The mineralized material and waste rock conveyors were sized for a maximum instantaneous throughput 7200 t/h and 16,700 t/h respectively.

Once mining starts in the Sur pit, run-of-mine mineralized material and waste rock is hauled from the Sur pit and direct tipped into the Sur gyratory crusher dump pocket. The Sur crusher will alternate between crushing waste rock and mineralized material in batches to a P_{80} of 85 mm and be conveyed in discrete lots to the Norte crushing facility to be diverted either onto the mill feed or waste rock overland conveyors.

Major equipment in this area includes:

- Norte Pit mineralized material primary Gyratory crusher (1,650 kW), with a two tip position dump pocket (1.5 truckloads each);
- Norte Pit waste rock apron feeder and mineral sizer (800 kW) , with a two tip position dump pocket (1.5 truckloads each);
- Mill feed overland conveyor (18 MW);
- Waste rock overland conveyor (45 MW);
- Sur Pit mineralized material/waste rock primary gyratory crusher (1,650 kW) , with a two tip position dump pocket (1.5 truckloads each), to be installed in Year 16;
- Dual purpose (feed/ waste) overland conveyor to Norte crusher station (0.4 MW), to be installed in Year 16.

17.3.3 Stockpile and Reclaim

Once the mineralized material from the Norte and Sur pits has been conveyed to the mill on the mill feed overland conveyor, the mineralized material will be stored in a stockpile. The stockpile will have a live capacity of 14 hours and is reclaimed by two apron feeders to load the SAG mill feed conveyor for grinding.

Major equipment in this area includes:

- Two Reclaim Feeders
- One Sag Mill Feed Conveyor

17.3.4 Grinding

The SAG mill feed conveyor will deliver the crushed material to the SAG mill feed chute. The grinding circuit will consist of a GMD SAG mill operating in open circuit with discharge screen oversize reporting to a pebble crusher. The pebble crusher is closed circuit with screening. SAG screen and pebble crusher screen undersize reports to two parallel ball mills in closed circuit configuration with hydrocyclones. It will treat on average 5,475 t/h of mineralized material at an availability of 91.3%. The circuit is sized based on a grinding circuit feed size of F_{80} of 85 mm and a circuit product size of P_{80} of 200 µm.

Major Equipment in this area includes:

- One GMD SAG mill (30 MW) with discharge screen
- Three pebble crushers (1 MW each) with discharge screens
- Two ball mills (18.5 MW each) with cyclone clusters

17.3.5 Rougher Flotation and Regrind

Ball mill cyclone overflow gravitates to the rougher flotation circuit via a trash screen. The trash screen will remove any debris or trash from the slurry before flotation. The undersize slurry will be conditioned with lime to achieve pulp pH of 10 and a mixture of A3894 and potassium amyl xanthate (PAX) collector and MIBC frother. Conditioned slurry flows into a single line of flotation cells with a residence time of 27 minutes. The rougher concentrate is then pumped into the regrind feed hopper. A similar reagent scheme will be used in the cleaning stage.

The rougher tailings stream will be combined with the cleaner scavenger tailings, thickened and filtered.

The rougher concentrate is mixed with the primary regrind mill product and cleaner scavenger concentrate in the primary regrind cyclone feed pumpbox and pumped to the primary regrind cyclones for classification. The primary regrind cyclone underflow will report to the primary regrind mill which will overflow back into the primary regrind cyclone feed pumpbox. The primary regrind cyclone overflow reports to the secondary regrind cyclone feed pumpbox. Secondary regrind cyclone overflow will report to the first stage of cleaner flotation while the underflow will report to the secondary regrind mill prior to reporting to the first stage of cleaner flotation.

The reground slurry will be cleaned in 3 stages of cleaner flotation cells. Concentrate recovered from the first copper cleaner will be pumped to the second cleaner for further upgrading. The first copper cleaner tailings will be directed to the cleaner scavenger flotation cells to maximize copper recovery.

The second cleaner concentrate will advance to the third cleaner flotation and the tailings will return to the first cleaner flotation to maximize copper recovery.

The third cleaner concentrate, which will be the final concentrate product, will be pumped to the concentrate thickener. The third cleaner tailings will return to the second cleaner flotation to maximize copper recovery.

Major equipment in this area includes:

- Eight rougher flotation cell 600m^3 (450 kw);
- Primary Regrind Mill (11 MW ball mill);
- Primary regrind cyclone pack;
- Secondary regrind mill (9 MW high intensity mill);
- Secondary regrind cyclone pack;
- Five cleaner 1 cells 200 m^3 (200 kw);
- Four cleaner scavenger cells 200 m^3 (200 kw);
- Three cleaner 2 cells 200 m^3 (200 kw);
- Two cleaner 3 cells 200 m^3 (200 kw).

17.3.6 Copper Concentrate Thickening and Filtration

The concentrate dewatering circuit consists of thickening and filtration equipment to allow for loadout and shipment of dry concentrate.

The copper concentrate thickening and filtration circuit will consist of a single high-rate thickener, filter feed tank and a pressure filter. Copper concentrate will be pumped from the copper flotation Cleaner 3 circuit to the copper concentrate thickener via a thickener feed box.

Flocculant will be added to the thickener feed stream to enhance settling. The concentrate thickener overflow will report to the process water tank. Copper concentrate solids will settle for collection at the underflow cone at a density of 60% w/w solids. The thickener underflow stream will be pumped to an agitated filter feed tank prior to filtration.

A dedicated air compressor will supply high pressure air for the copper concentrate filter. Copper concentrate filter cake will be discharged from the filter, directed to the concentrate stockpile, and loaded onto trucks via a front-end loader.

Major equipment in this area includes:

- One high-rate thickener (25 m);
- Two horizontal plate pressure filters.

17.3.7 Tailings Thickening and Filtration

Copper rougher tailings and copper cleaner scavenger tailings are combined and pumped to the tailings thickener. Flocculant is added to the thickener feed to enhance settling resulting in an underflow density of 60% w/w solids. The overflow is directed to the process water tank. The underflow is then pumped to tailings filter feed tank. The slurry is then fed into plate & frame pressure filters to achieve a moisture level below 15 %w/w and conveyed to a transfer station to be mixed with waste rock from the waste rock stockpile. This mixed material is conveyed by the waste rock overland conveyor to the dry stack tailings facility.

Major equipment in this area includes:

- One high-rate thickener (87 m);
- Twelve vertical plate pressure filters and discharge belt feeders;
- Filtered cake conveyor.

17.4 Energy, Water, and Process Materials Requirements

17.4.1 Process Materials Requirements & Storage

Reagent and consumables consumptions are based on test work results and standard industry practices. A summary of the necessary handling and storage of required reagents is presented in Table 17-2. Consumption of reagents and consumables are listed in Table 17-3 and Table 17-4.

Table 17-2: Reagents Handling and Storage

Reagents	Preparation Method	Usage
Quicklime	Received as powder in bulk bags; slaked with raw water and transferred to a storage tank and dosed to circuits	pH control
PAX	Received as pellets in bags; mixed with raw water for storage tank and then dosed to flotation circuits.	Collector
MIBC	Received in IBC totes in solution; used as frothing aid for flotation. Dosed to flotation and cleaner cells.	Frothing aid
Magnafloc 5250	Received as powder in bulk bags; mixed with raw water to storage tanks. Dosed the concentrate and tailings thickeners	Flocculant Agent
Antiscalant (Belclene 110)	Received in IBC totes in solution; used as antiscalant for cooling water and other industrial process waters.	Antiscalant
Biocide	Received in drums in solution; used to maintain slurries and protect materials from corrosion and decay.	Site Maintenance

Table 17-3: Nominal Annual Reagents Consumption

Reagents	Unit	Quantity
Quicklime	t/a	70,080
PAX	t/a	830
MIBC	t/a	2,845
Magnafloc 5250	t/a	1,535
Antiscalant (Belclene 110)	t/a	220
Biocide	t/a	90

Table 17-4: Nominal Annual Media Consumption

Reagents	Unit	Quantity
SAG Mill Media	t/a	8,365
Ball Mill Media	t/a	7,800
Regrind Mill Media	t/a	1,160
Lime Slaker Media	t/a	20

17.4.2 Water requirements

17.4.2.1 Raw water

Raw water will be provided to a storage tank, where it is further pumped to various processes. It will supply water to the process plant for distribution to process water make-up, gland water services, cooling water services, reagent services, and potable water plant. Total raw water to the process plant will be 902 m^3/h.

17.4.2.2 Fire Water

The bottom portion of the raw water tank will be reserved for fire water.

17.4.2.3 Potable Water

An on-site potable water plant will treat water from the raw water tank and store it in the potable water tank for distribution around the processing plant, camp and offices.

17.4.2.4 Process Water

The process water distribution to the plant will be 11,000 m^3/h and will require a raw water make-up of 596 m^3/h at the design throughput. The majority of the process water will be recycled water from the thickening and filtration process of the concentrate and tailings. Contact water from the site and CPSF as well as Pit dewatering will be the primary source of make-up water.

17.4.3 Air requirements

The air services capacity will be designed to accommodate the flotation and filtration equipment as well as providing shop and instrument air in all areas of the processing plant.

17.4.4 Power requirements

The total power requirement for the process plant is 1,447,606 MWh/a. The average power demand for the process plant and estimated power consumption for each area is presented in Table 17-5.

Table 17-5: Power Requirements

Area	Average Demand (kW)	Power Consumption (MWh/a)
Primary Crushing and Sizing	3,290	21,615
Crushed Material Conveying	63,400	416,538
Grinding and Pebble Crushing	59,954	479,632
Flotation and Regrind	27,383	219,062
Concentrate Thickening, Filtration and Loadout	894	6,953
Tailings Thickening and Filtration	14,875	115,407
Reagents	119	950
Plant Services	15,980	127,838
Tailings Management	7,733	59,608
Total	**193,627**	**1,447,606**

18 PROJECT INFRASTRUCTURE

18.1 Overview

Infrastructure at the Cañariaco Project includes on-site infrastructure including earthworks development, site facilities and buildings, on-site roads, water management systems, and site electrical power facilities. Off-site infrastructure includes site access roads, fresh water supply, power supply, piping, camp, tailings storage facility, and the Norte and Sur deposits along with their respective crushing stations. The plant and camp site were selected based on suitable topography and access corridors and its proximity to other project facilities.

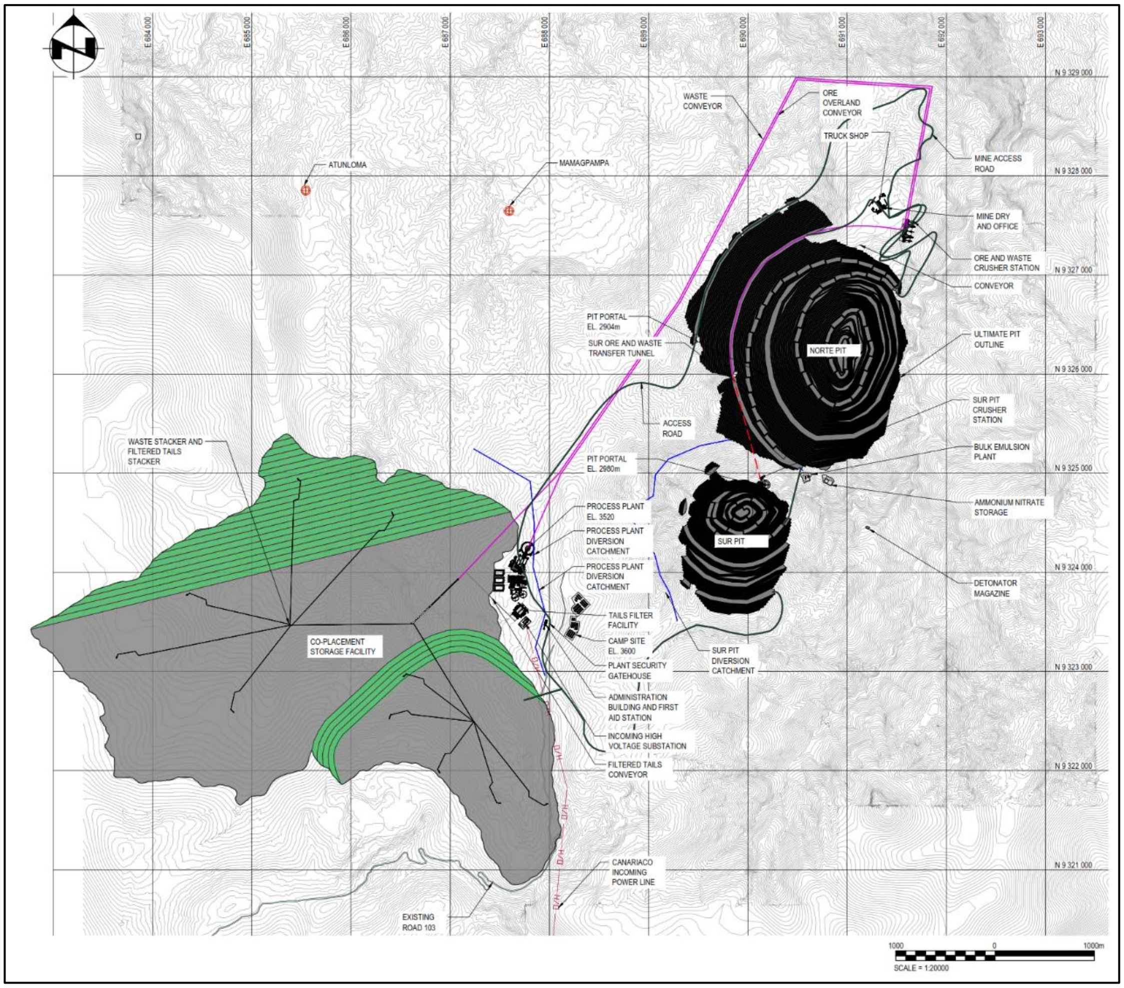
The site infrastructure will include:

- Mine facilities include administration offices, truck shop and wash bay, and mine workshop.
- Common facilities, including an entrance/exit gatehouse, a security/medical office. Overall site administration building, potable water and fire water distribution systems, compressed air, power distribution facilities, diesel reception, and communications area.
- Process facilities housed in the process plant, including grinding and classification; flotation; concentrate thickening, filtering and handling; assay laboratory; process plant workshop; and warehouse. The tailings filter plant is adjacent to the process plant building.
- Other infrastructure includes on-site camp, waste management facilities, and co-placement storage facility.

The project infrastructure selection was determined based on the following considerations:

- The facilities described above must be located on the Cañariaco patent land.
- Locating each of primary crushing plants close to the Norte and Sur deposits to reduce haul distance.
- Utilize the natural high ground for the run-of-mine (ROM) pad as much as possible.
- Separate heavy mine vehicle traffic from non-mining, light vehicle traffic.
- Locate the process plant near an existing primary access road.
- Place administration, and process plant staff offices close together to limit walking distances between them.

Figure 18-1: General Site Layout



Source: Ausenco, 2024.

18.2 Site Access

18.2.1 Plant Access Road

Road access to the site will be via private road to be built to connect site with paved highway 103, which runs along the Huancabamba Valley. The private road is estimated to be 24 km in length, but further studies are required to establish exact routing. Internal project site roads will be constructed to the process plant, and the Norte and Sur pits. The project access road will be designed to meet Peruvian guidelines, as presented in the Peruvian road design manual "*Manual de diseño geometrico para carreteras*", and will be suitable for the transport of concentrate, freight, and equipment, including all oversized equipment needed for construction and operation of the mine/plant facilities. The entrance to the road will be controlled through the plant security gatehouse, thereby restricting the road to authorized vehicles only. All users will need two-way radios, as the road is planned to be radio-controlled to increase safety and usability.

18.2.2 On-Site Roads

General vehicle access roads will be provided around facilities with service roads to remote structures and haul roads around the primary crushing plant, truck shop and pit. During the life of the mine, roads and lay down areas will be modified and added as needed. General access roads will be two-way and 8 m wide, the service roads will be one-way and 5 m wide with pullouts. The haul roads will contain a running surface of 36 m wide, while the access road to the truck shop area will be 15 m wide.

The roads will have the following safety features:

- Rock safety berms for haul roads;
- Mandatory and advisory traffic signs;
- Speed limit signs;
- Radio communications systems between trucks; and
- Pullouts every kilometre on one-way roads.

18.3 Built Infrastructure

A major feature of the Cañariaco Project is the overland conveyors that will convey the mineralized material to the process plant and waste rock from the pits to the CPSF. An overland conveyor will be constructed from the Norte pit's crusher station to carry mill feed to a crushed material stockpile feed prior to the process plant. A larger overland conveyor for mine waste will be built along the same route of the mill feed overland conveyor but diverge prior to the milling plant to the CPSF. The conveyors will follow the topography to reduce earthwork and supports required, keeping away from creeks and valleys to maintain reasonable conveyor-slope profiles. The route of the two lines of overland conveyors is shown in Figure 18-1.

18.4 Accommodation

The camp and administration building areas are shown in Figure 18-1. The main facilities will be the sleeping dormitories, administration offices, and medical clinic. Supporting infrastructure will include an incinerator, water storage tanks, potable water treatment plant (PWTP), and sewage treatment plant (STP). The site development plans will also incorporate a helipad and control centre for operational requirements and emergency medical evacuations.

18.4.1 Camp

A permanent camp will be constructed to house the 420-person workforce during production.

The camp will consist of the following:

- Dormitories with ablution facilities, toilets, and self-serve laundry;

- Food preparation and serving facilities; and
- Recreation facilities and small commissary.
- The construction camp will also have a check-in and administration office module.

18.4.2 Administrative Building

The administration building and medical clinic will be constructed south of the process plant and adjacent to the security gatehouse and plant site access road. The building will be a modular pre-assembled building. The facility will be built early in the construction phase and used for construction administration.

18.5 Stockpiles

The mine plan and processing schedule includes long term stockpiling of mineralized material. A coarse mineralized material stockpile between the crushers and SAG mill will be utilized to buffer material flow between the crushers and mills. Crushed waste rock will be conveyed directly into the CPSF.

18.6 Co-placement Storage Facility

The waste material generated from mining operations at Cañariaco project consist of waste rock and tailings. Based on a trade-off study it was decided to place both material in a single facility to minimize environmental impacts and improve overall stability of the co-placement storage facility (CPSF). The materials will be stored in a co-placement storage facility, i.e. the tailings and waste rock material will be transported independently, but not mixed to form a single discharge stream. The waste rock will be transported to the facility using overland conveyor and end dumped using mobile stacking system to create internal and external embankments, benches, rock drains, internal and external berms (erosion protection with the filtered tailings placed in the interior). The filtered tailings will be conveyed to the CPSF in a similar manner. A portion of the filtered tailings will spread and compacted near exterior slopes to improved overall stability of the facility.

18.6.1 Hazard Classification

The design standards for the CPSF are based on the relevant Peruvian and International guidelines for construction of mine waste storage facilities. The following regulations and guidelines were used to determine the dam hazard classification and suggested minimum target levels for some design criteria, such as the inflow design flood (IDF) and earthquake design ground motion (EDGM):

- Technical Bulletin – Application of Dam Safety Guidelines to Mining Dams (CDA, 2019); and
- “Guidelines for Mine Waste Dumps and Stockpile Design” (2017).

The CPSF was classified as high or moderate hazard classification under both CDA and GMWDSD guidelines, respectively. The recommended IDF during operations is defined as 1/100-year return period moderate waste storage facility in accordance with guidelines for mine waste dumps. However, the minimum for closure is 1/200-year return period, which was utilized for both operations and closure. The design earthquake is characterized as the 1/2,475-year return period for high dam classification in accordance with CDA guidelines.

18.6.2 Tailings and Waste Rock Characteristics

Tailings and waste rock characterization are based on historical testing programs and are shown in Table 18-1.

Table 18-1: Tailings and Waste Rock Characteristics

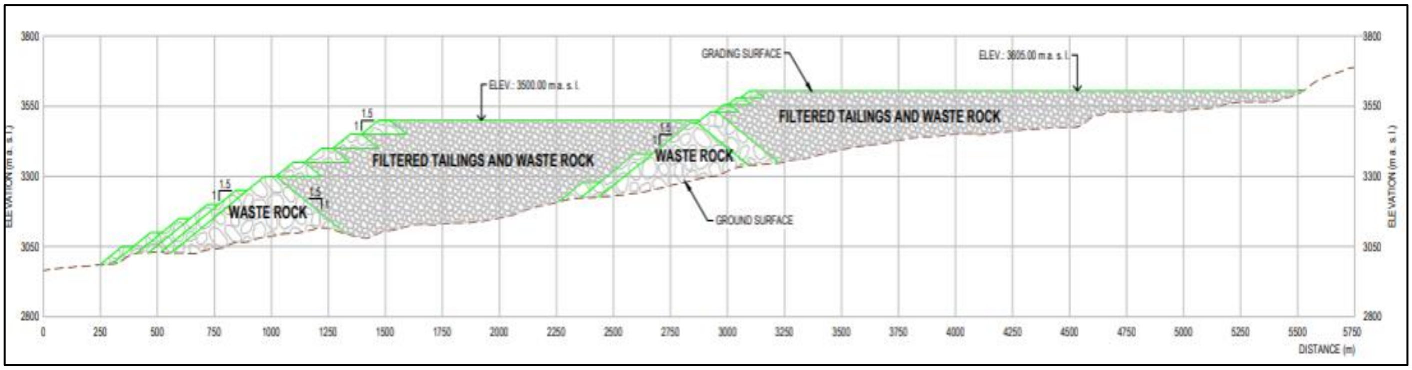
	Units	Value
Tailings		
Specific Gravity	Unitless	2.60
Deposition Void Ratio	Unitless	0.5
Deposition Dry Density	t/m^3	1.65
Moisture Content of Filtered Tailings	%	15
Tailings Gradation (P_{80})	µm	200
Classification	USCS	ML (inorganic silt)
Friction Angle	°	31
Waste Rock		
Specific Gravity	Unitless	2.20
Deposition Void Ratio	Unitless	0.23
Deposition Dry Density	t/m^3	2.00
Moisture Content of Filtered Tailings	%	Varies
Tailings Gradation (P_{80})	Mm	155
Classification	USCS[2]	GP (poorly grade gavel)
Friction Angle	°(varies with depth)	Leps[1]

Notes: **1.** Leps, T. M. (1970). Review of shearing strength of rockfill. Journal of the Soil Mechanics and Foundations Division, 96(4), 1159-1170. **2.** Unified Soil Classification System (USCS).

18.6.3 Co-Placement Storage Facility Design

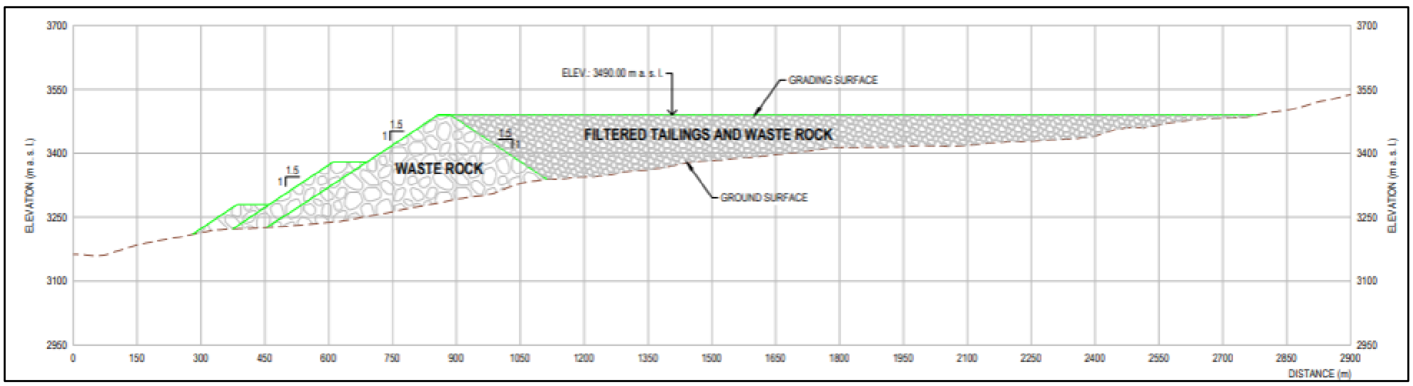
During a portion of pre-production, mine waste rock will be trucked to the waste storage facility and used to build the starter embankment. In pre-production once the crusher, overland conveyor and mobile conveyor systems have been installed, the mine waste will be crushed and conveyed to the CPSF. It is assumed that the pre-production waste rock will be non-acid generating (NAG), and any potential acid generating (PAG) waste rock will be mixed and encapsulated within the filtered tailings to mitigate metal leaching and acid rock drainage (ML/ARD). During operations the waste rock delivered to the CPSF to build embankments, benches, rock drainages, and interior and exterior berms or conveyed and placed with the filtered tailings using two overland and mobile conveyor systems (refer to Figure 18 1). Based on the potential acid generation of some materials, if required, calcium carbonate will be added as part of the ARD mitigation measures along with encapsulation within the filtered tailings. It is assumed that PAG materials will become more prevalent with depth. There will be insufficient waste rock to "co-mingle" with the tailings waste stream to take advantage of the waste rock strength. Therefore, the waste rock will be used to develop embankments, interior and exterior berms and the balance of waste rock will be placed with tailings near the front of the facility to improve the facility physical stability.

Figure 18-2: Ultimate CPSF - Section 1



Source: Ausenco, 2024

Figure 18-3: Starter CPSF - Section 1



Source: Ausenco, 2024.

The CPSF will be constructed in several phases over the 28-year operating mine life. The CPSF includes a contact water management pond (CWMP) downstream of the facility, surface water management structures, rock drains, access road, waste conveyor systems, mechanical equipment for spreading (dozers) and compacting (compactors) portions of the materials, and a water reclaim pipeline for the CWMP. The CPSF and CWMP footprint areas that will be cleared of organic material and topsoil for foundation preparation. Basin preparation will include removal of unsuitable materials (soft soils and saturated soils) and compaction of the subgrade in the CWMP and embankment footprints. It is assumed that less than an average 1 m of overburden removal will be required over the footprint of the CWMP and under any embankment footprints.

The starter embankment, bench, and rock drain for the CPSF will be constructed during pre-production using 39.2 Mm^3 of NAG waste rock to support the deposition of waste rock and tailings during operations. A starter embankment will be constructed to an elevation of 3,480 masl along with a rock drain up the thalweg of the valley that will promote drainage of the tailings and waste rock to the front of the starter CPSF, maintaining a low phreatic head in the downstream portion of this facility. Over the life of mine additional embankments, benches and berm will be developed downstream of the starter facility to contain filtered tailings, PAG waste rock, and excess NAG waste rock.

A network of rock drains will be developed within the base of the facility using selective placement of NAG waste rock (Refer to Section 18.6.6). In addition, a non-contact water diversion channel will be constructed to intercept surface runoff and convey it around the CPSF to natural drainages. The CWMP will also be constructed at the foot of the ultimate facility to provide sediment management.

The ultimate non-contact water diversion channel will be constructed in Year 2 and convey surface runoff above the ultimate facility around the ultimate CPSF footprint and CWMP.

18.6.4 Waste Placement

During a portion of pre-production and operations, crushed waste will be conveyed by an overland conveyor to the CPSF and then to mobile conveyor stacking system to construct embankments, benches, rock drains, and berms or placed with filtered tailings within the CPSF. Filtered tailings will be conveyed to the CPSF on an overland conveyor and then transition to mobile stacking system within the facility. The waste rock and tailings production rates are shown in Table 16-13.

The waste rock to create embankments will be stacked across the valley at angle of repose, then addition wrap around embankments will be placed downstream of the initial embankment in benches to create and overall slope of 2.25:1 (H:V) for the starter embankment to a final height 3,605 masl by the end of year 2. Internal embankments will be constructed in a similar fashion and the starter facility to store filtered tailings and waste rock. The ultimate embankment will be constructed with a downstream slope of 2.5:1 (H:V) to and elevation of 3,500 masl by the end of the mine life. The filtered tailings will be stacked in 5-meter lifts and the surface spread and compacted to reduce infiltrations and promote runoff.

18.6.5 Instrumentation and Monitoring

Instrumentation and monitoring will be required to assess embankment and facility performance. Vibrating wire piezometers will be installed to monitor pore pressure within the embankment and interior of the facility along with slope inclinometers and survey monuments within embankments to monitor for any potential slope movement and deformation.

18.6.6 CPSF Water Management

The surface of the filtered tailings and waste rock within the CPSF will be graded and compacted to encourage flow to defined surface water collection points within the facility. The collected surface water will be directed to the CWMP through channels designed to convey up to the 1:100-year storm event. The water stored in the CWMP will be used as a source of make-up water for the process plant.

Water that infiltrates the surface of the facility will fill any void space remaining within the waste materials and may end up contributing to the phreatic surface at the base of the facility. The phreatic surface may also be elevated because of loading from subsequent waste placement if the material is placed at or near saturation. A drainage system will be installed that includes a rock drain to facilitate collection of water and transfer to the CWMP. The HDPE pipe and rock drain will be extended as the waste surface rises.

Non-contact water diversion channels will be constructed in phases around the perimeter of the CPSF to capture surface water above the facility and convey it below the CWMP to discharge into the watershed below the facility. The channels are designed to convey up to the 1:200-year storm event. The channel has a light vehicle service road next to the channel.

18.7 Power and Electrical

Electrical power will be supplied to the Project site by a 220 kV overhead transmission line from the local utility substation at Carhuaquero, a distance of 57 km from the mine site. The incoming transmission line will terminate at a new main site substation where it will be stepped down from 220 kV to the site distribution/utilization level of 25 kV. The anticipated connected average load will be 207 MW year-round, at a rate of 0.072 US$/kWh.

Power will be supplied to the various project facilities through radial feeders originating at the main substation and routed on site through either installed on overhead powerlines, direct buried, or in duct banks. Step-down transformers will provide equipment utilization voltages from the site distribution voltage. All process electrical and control rooms will be modular units constructed off-site, with all electrical controls and instrumentation equipment installed, wired, and completely tested before shipment to site. They will be located as close as practical to the electrical loads to minimize voltage drop concerns and reduce cabling costs, and be equipped with heating and ventilation, lighting, small power transformers, distribution boards and uninterrupted power supply systems.

18.8 Fuel

Fuel will be delivered to the mine site via tanker trucks. The fuel storage and dispensing station will be located at the truck shop, and contain several above ground tanks, including diesel tanks, gasoline tanks and various propane tanks. The tanks will be contained in a lined containment berm to ensure no fuel can leak into the environment.

18.9 Water Supply and Management

18.9.1 Raw Water Supply

Raw fresh water will be sourced from the seepage collection pond and mine pits and will have a live capacity of 185,000 m^3. It will be used to supply the main process use in the form of process makeup water, gland seal water, cooling water and general facility use of fire water and potable water.

18.9.2 Process Water Supply

Process water will be made up from three sources, the reclaimed water from the thickening and filtration process of the concentrate and tailings, the contact water from the tailings disposal facility, and raw water supplied from the mine pits. Process water will be stored in a process water tank and pumped to various circuits in the process plant.

18.9.3 Potable Water Supply

An on-site potable water treatment plant will treat the raw water to supply potable water for site, as well as for the building facilities and process plant. Potable water is stored in a tank for distribution to the processing plant.

18.10 Site Water Balance

This section covers the site-wide water management, including the design of water management structures, hydrological considerations, and the water balance analysis.

18.10.1 Climate Data

Monthly precipitation and evaporation data were obtained from the El Limon weather station. Meteorological and hydrological parameters used for this study are shown in Table 18-2 and Table 18-3, based on available site and regional data.

Table 18-2: Climate Data for Alta Copper Site

Parameter	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Annual Total
Average Monthly (°C)	25.3	24.9	25	25.2	25.3	25.1	25	25.7	26.3	26.4	26.2	25.7	306.1
Precipitation (mm)	103	131	187	113	51	35	24	33	59	96	75	92	999
Evaporation (mm)	41	25	26	27	34	38	54	66	53	48	47	41	500

Table 18-3: Rainfall Depth-Duration-Frequency Statistics

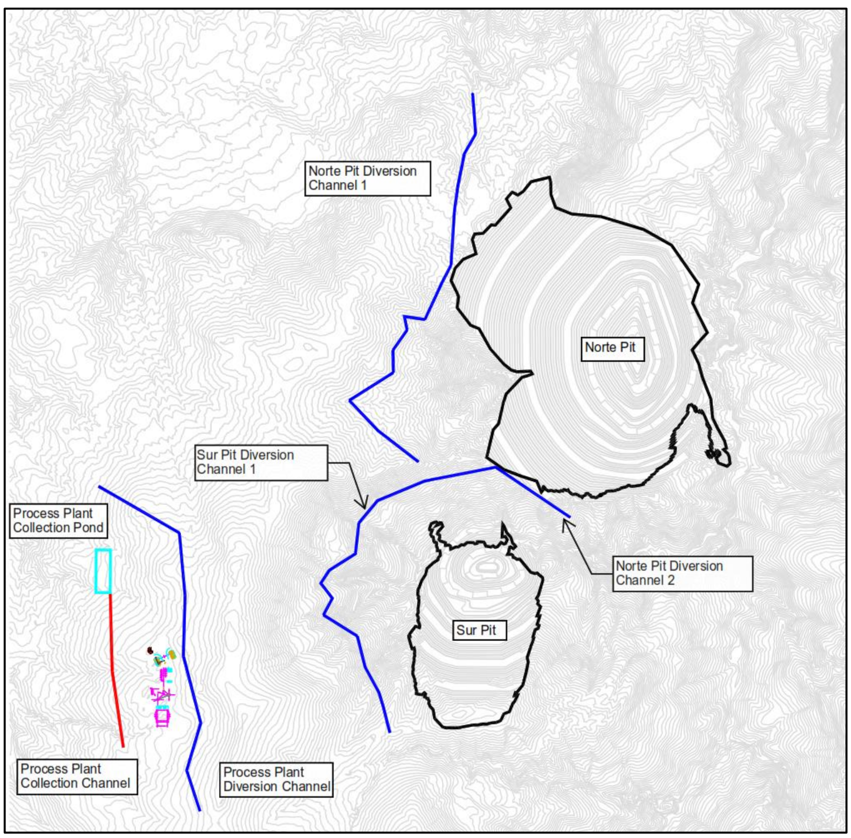
Return Period Duration	0.25	0.5	1	6	12	24
2	31	37	45	74	90	109
5	40	49	60	100	121	148
10	47	57	70	117	142	173
20	53	65	79	133	162	198
50	61	75	91	154	188	230
100	67	82	101	170	207	254
200	73	90	110	185	227	278
500	81	99	122	206	252	309
1,000	87	107	131	222	272	333

18.10.2 Water Management Structures

Water management infrastructure at the site has been proposed based on the current pit locations, process plant area, and tailing management facility. Runoff from the process plant will be directed to a settling pond via a contact water channel. Diversion channels have also been incorporated into the proposed plan to divert non-contact water from the active mine areas and reduce quantity of mine-affected water requiring management. The overall site-wide surface water management strategy is shown in Figure 18-4 and comprises:

- Collection Channel: The contact water collection channel will capture the runoff from the process plant and convey runoff to a collection pond located northwest of the process plant. The design criterion for collection channel was the conveyance of 1/100-year, 24-hour peak flow without overflow. Geometry of the channels is shown in Table 18-4.
- Diversion Channels: Four channels will divert non-contact runoff away from mine infrastructure to minimize the amount of contact runoff to be managed at the site. The design criterion for the diversion channels was to ensure the conveyance of 1/100-year, 24-hour event without overflow. Geometry of the channels is shown in Table 18-5.
- Collection Pond: Contact water captured in the collection channel will be conveyed to a collection pond. Pond has been sized to store contact runoff from a 1/100-year 24-hour event. Stored contact water is either treated and released into the environment or reused for process purposes. Geometry of the collection pond is shown in Table 18-6.
- A preliminary assessment of excavation, riprap volumes and geotextile liner area was conducted based on the proposed geometries of the structures and elevation profiles along the alignment of channels and ponds. Table 18-7 shows the estimated quantities.

Figure 18-4: Water Management Structure



Source: Ausenco, 2024.

Table 18-4: Collection Channel Parameters

Segment name	Channel Shape	Side-Slope (H:V)	Length (m)	Design Channel Depth (m)	Bottom Width (m)	Design Slope (m/m)
Process Plant Collection Channel	Trapezoidal	2.5:1	1094	3.85	2	0.3

Table 18-5: Diversion Channel Parameters

Segment name	Channel Shape	Side-Slope (H:V)	Length (m)	Design Channel Depth (m)	Bottom Width (m)	Design Slope (m/m)
Norte Pit Diversion Channel 1	Trapezoidal	2.5:1	3,255	4.50	5	0.2
Norte Pit Diversion Channel 2	Trapezoidal	2.5:1	636	2.02	2	0.3
Sur Pit Diversion Channel	Trapezoidal	2.5:1	2,586	6.64	3	0.2
Process plant Diversion Channel	Trapezoidal	2.5:1	2660	3.43	3	0.5

Table 18-6: Dimensions of Collection Ponds

Item	Dimensions (m)	
	Length	Width
Process Plant Pond Collection	300	100

Table 18-7: Material Take Off (MTO) for Water Management

Item	Volume (m³)			Geotextile Liner Area (m²)
	Excavation	Fill	Riprap	
Diversion channels	127,585	-	42,824	112,326
Collection channels	8792	-	2865	9,549
Collection ponds	46,500	-	-	-
Total	**182,877**	**-**	**45,689**	**121,875**

18.10.3 Site-wide Water Balance

A preliminary site-wide water balance analysis was conducted for the mine site, and the results are summarized in this section.

In this analysis, a comparison between water requirements, and available water from the collection system was made to identify the site-wide water balance. Key factors considered in the water balance included pit dewatering (groundwater and precipitation on the pits) and surface water on contact areas. Assessments for each facility were performed using input data, including average monthly precipitation, evaporation rates, and appropriate runoff coefficients. This analysis has been conducted based on average climate conditions at the site.

The following water components were considered in this calculation:

- Surface runoff from precipitation on Norte and Sur pits, process plant and Tailing Storage Facility;
- Evaporation from the process plant collection pond;

- Process water requirement;
- CPSF reclaim capacity.

As shown in Table 18-8, there is a net annual water excess of 518 m^3/hr during average meteorological conditions. It should be noted that groundwater modelling was not conducted at the time of this report, and pit dewatering values are calculated based solely on precipitation.

Table 18-8: Site-Wide Water Balance for Average Year

Water component	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Mean Annual
Contact Water from Pits (m^3/hr)													
Precipitation on Norte Pit it (Norte Pit dewatering)	380	535	690	431	188	133	89	122	225	354	286	339	314
Precipitation on Sur Pit (Sur Pit dewatering)	131	185	239	149	65	46	31	42	78	122	99	117	109
Contact Water from Net Precipitation and Evaporation (m^3/hr)													
Process Plant Area	79	111	143	89	39	28	18	25	47	73	59	70	65
Ponds Direct Precipitation	72	102	131	82	36	25	17	23	43	67	54	65	60
Ponds Evaporation	29	19	18	20	24	28	38	46	38	34	34	29	30
Water Deficits/Excess (-/+) in Average Conditions	**633**	**913**	**1,185**	**732**	**304**	**205**	**117**	**166**	**354**	**583**	**465**	**562**	**518**

19 MARKET STUDIES AND CONTRACTS

19.1 Market Studies

Alta Copper retained an external consultant, Open Mineral, for a review of metal payables (including penalty scales). The market terms for this study are based on the terms determined by Open Mineral as well as recently published terms from other similar studies. The QP is of the opinion that the marketing and commodity price information is suitable to be used in cashflow analyses to support this report. No contracts are in place for development of the project, which is appropriate for a PEA study.

Alta Copper's management were provided with indicative smelter terms. The net payabilities and penalties for metals in concentrate are summarized as:

- Copper in concentrate is payable at 96.5% subject to a minimum deduction of 1 tonne of copper per 100 tonnes of concentrate.

Gold in concentrate is payable at:

  - 0% for Au grade below 1 g/t.
  - 90% for Au grade above or equal to 1 g/t and below 3 g/t.
  - 92% for Au grade above or equal to 3 g/t and below 5 g/t.
  - 93% for Au grade above or equal to 5 g/t.

Silver in concentrate is payable at:

  - 0% for Ag grade below or equal to 30 g/t.
  - 90% for Au grade above 30 g/t.

Arsenic in concentrate results in a penalty calculated on the following progressive scale:

  - US$2.50/t per 0.1% for As grades above 0.2% and less than 0.5%.
  - Then, US$5.50/t per 0.1% for As grades above 0.5% and less than 3.0%.
  - Then, US$9.00/t per 0.1% for As grades above 3.0%.

Copper concentrates are widely traded and can be marketed domestically or internationally with significant optionality regarding the ultimate customer base. An independent assessment indicated that the concentrate produced is expected to be of sufficient quality to be marketable to smelters globally.

19.2 Commodity Price Projections

Project economics were estimated based on long-term flat metal prices of US$4.00/lb Cu, US$1,850/oz Au, and US$23.00/oz Ag. These prices are in accordance with consensus market forecasts from various financial institutions and are consistent with historic prices, shown in Table 19-1, sourced from Capital IQ on April 19, 2024. The QP also considers the prices used in this study to be consistent with the range of prices being used for other project studies.

Table 19-1: Summary of Historic Commodity Pricing

Metal	3-Year Trailing Average
Copper (US$/lb)	4.03
Gold (US$/oz)	1,882
Silver (US$/oz)	23.24

Source: Capital IQ, April, 2024.

19.3 Contracts

No contracts for transportation or off-take of the concentrate are currently in place, but if they are negotiated, they are expected to be within the industry norms. Similarly, there are no contracts currently in place for the supply of reagents, utilities, or other bulk commodities required to construct and operate the Project.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Introduction

This section provides an overview of the setting of the Cañariaco Norte and Sur deposits within the Cañariaco Project. It outlines existing biological and physical baseline conditions, proposed new and ongoing baseline studies to support existing and future permitting and regulatory requirements, including water and waste management requirements. In addition, this section also discusses socio-economic baseline conditions, the status of community engagement, and mine closure and reclamation planning for the Project. This information was summarized in a previously published NI 43-101 report completed in 2022 (Ausenco). This section has been revised from 2022 to account for additional information made available since 2022, and changes to the scope of the Project including inclusion of the nearby Sur deposit to complement the Norte deposit, and changes to infrastructure including processing facilities, roads, production rates, mine life and footprint of the CPSF.

20.2 Environmental Considerations – Existing Environmental Setting

The information contained in this section is derived mainly from baseline studies, investigations and field work carried out in 2021 through 2023 by Yaku Consultants for the development of the semi-detailed Environmental Impact Assessment (EIAsd 2021) as well as previous baseline studies conducted by AMEC from 2007 to 2014 to support the development of a partially completed Environmental Impact Statement (EIA).

The following environmental components were studied:

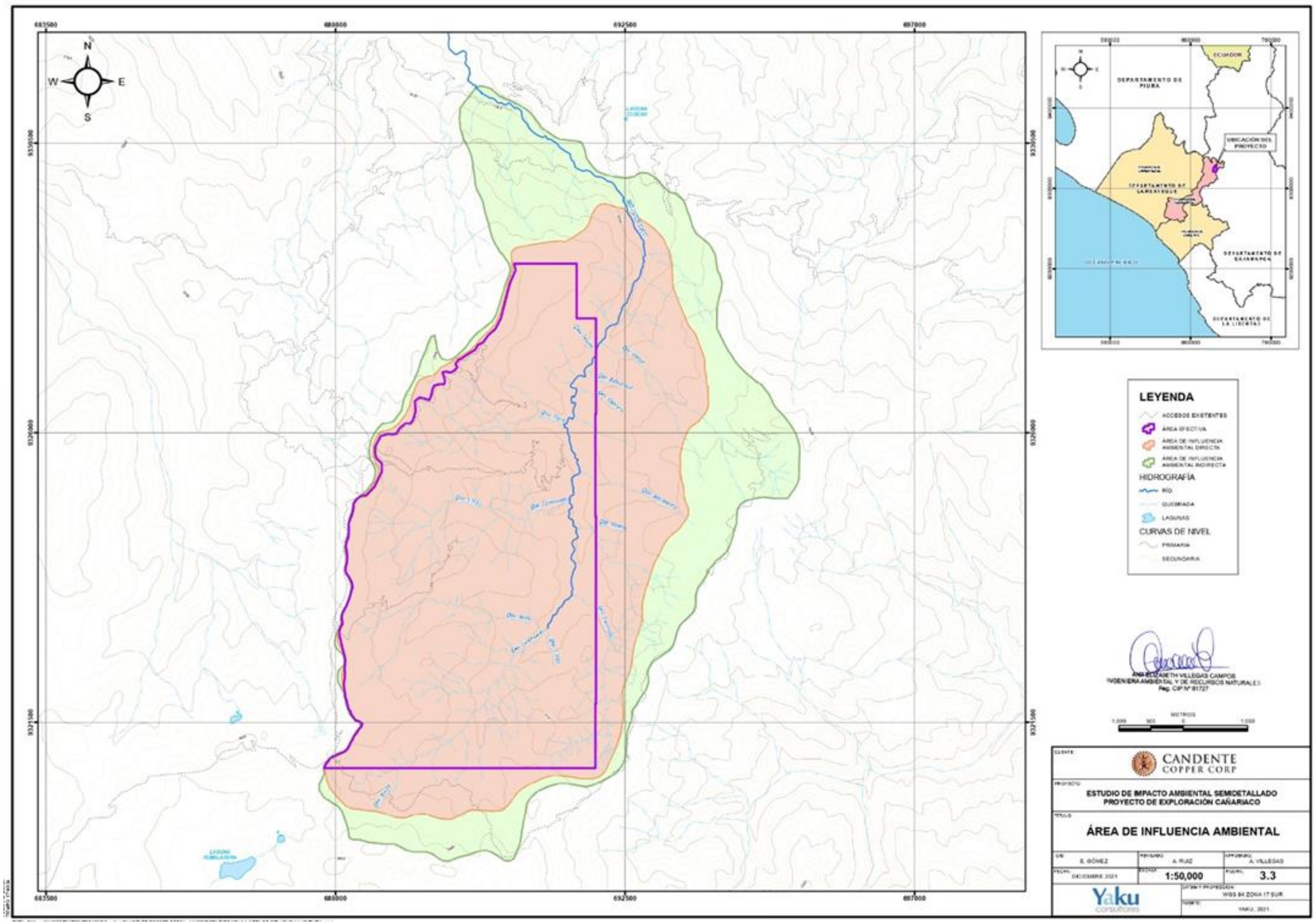
- Physical components: climate and meteorology, air quality and environmental noise, vibrations, soils, geology, geomorphology, hydrology and hydrogeology, and water quality;
- Biological components: ecosystems, flora, fauna and hydrobiology; and
- Socioeconomic and archaeological components.

In addition, some sub-sections include a summary of the information obtained between 2008 and 2011 for the development of the 2012 EIAsd prepared by AMEC and approved by Directorial Resolution No. 177-2012-MEM/AAM in May 2012.

20.3 Study Area

The Project is located on the surface land of the Community of San Juan de Cañaris, in the district of Cañaris, province of Ferreñafe, department of Lambayeque. The environmental study area was defined as shown in Figure 20-1, and limited to the upper watershed Cañariaco River (tributary of the Huancabamba River), or Jatun Yacu river watershed, as the river is called in that section. The lower limit is located before the confluence with the Achicamonte Creek, where the river morphology changes significantly and is remote from the Project's activities. Small, pre-existing disturbed areas are observed in the study area.

Figure 20-1: Environmental Study Area



Source: Yaku Consultants, 2021.

20.4 Air Quality and Ambient Noise

For the analysis of air quality and ambient noise, the data obtained in the May 2011 campaign (AMEC, 2012) and in the campaigns of September 2020, dry season, and January 2021, wet season (Yaku Consultants, 2021) were used.

In the 2012 EIAsd, there were five air sampling stations, whose locations are detailed in the Table 20-1 for air quality and in the Table 20-2 for ambient noise quality.

Table 20-1: Air Quality Sampling Stations

Station Code	Geographical Location UTM		Altitude (masl)	Description
	East	North		
K-AQ-01	691,535	9,325,922	2,749	Station located near the dining room of the exploration camp.
K-AQ-02	691,197	9,327,490	2,916	Station located in the area called Palmapampa, north-east of the exploration camp.
K-AQ-03	691,042	9,325,552	3,037	Station located in the area called Delta Point
K-AQ-04	689,231	9,326,090	3,384	Station located in the zone called Point 27.
K-AQ-05	690,324	9,327,324	3,239	Station located in the zone called Punto Venado.

Note: Coordinates in Datum WGS84, Zone 17S.

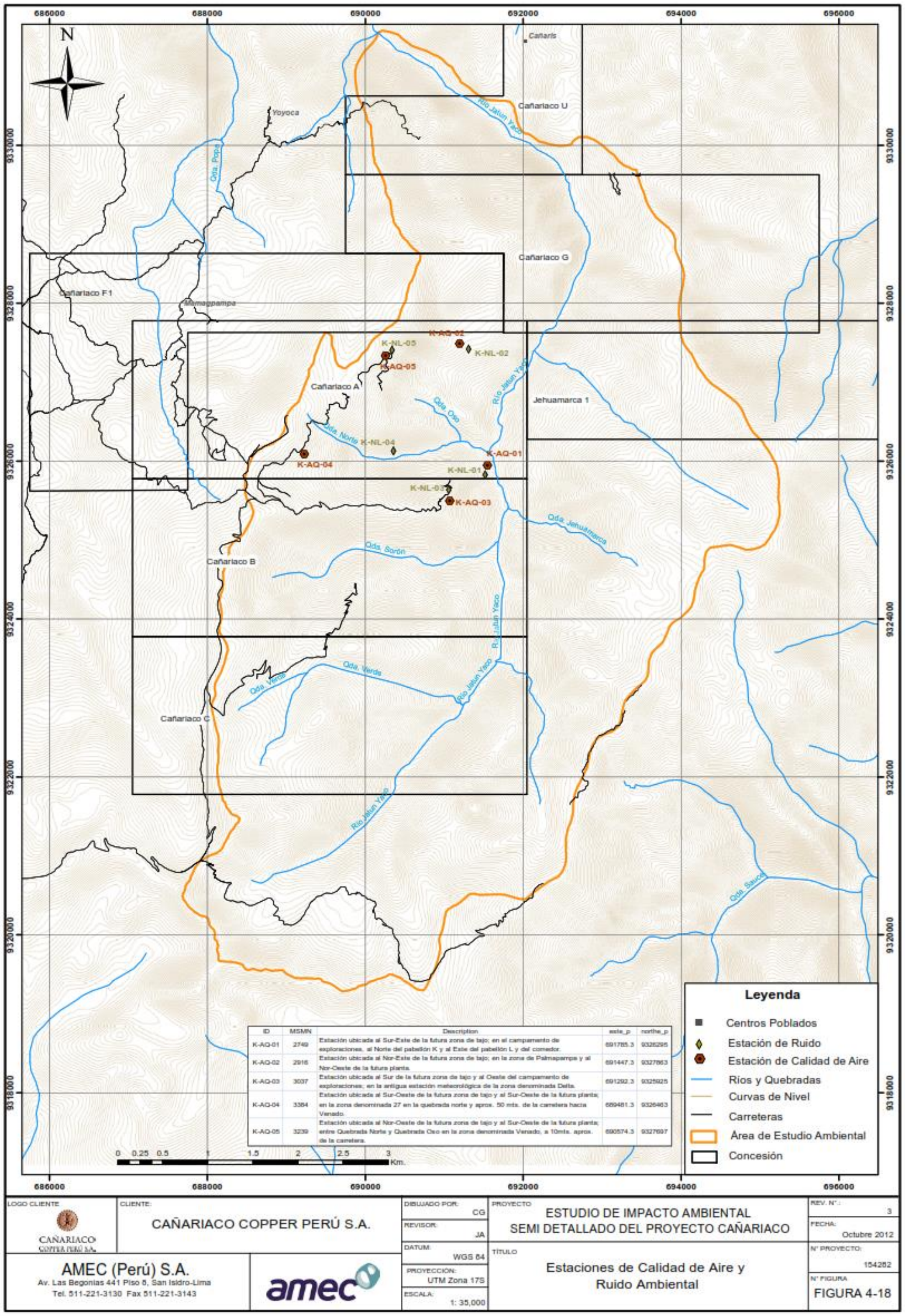
Table 20-2: Ambient Noise Sampling Stations

Station Code	UTM Coordinates		Altitude (masl)	Description
	East	North		
K-NL-01	691,511	9,325,908	2,752	Station located near the dining room of the exploration camp.
K-NL-02	691,310	9,327,418	2,906	Station located in the area called Palmapampa, north-east of the exploration camp.
K-NL-03	691,041	9,325,566	3,072	Station located in the area called Delta Point
K-NL-04	690,357	9,326,128	3,380	Station located in the zone called Point 27.
K-NL-05	690,357	9,327,350	3,231	Station located in the zone called Punto Venado.

Note: Coordinates in Datum WGS84, Zone 17S.

Figure 20-2 shows the location map of the air quality and ambient noise monitoring stations for the 2012 survey.

Figure 20-2: Location of Air Quality and Ambient Noise Monitoring Stations for the 2012 Survey



Source: Amec (Peru) S.A., 2012.

Three monitoring stations were used in the 2021 EIAsd 2021. Their locations are detailed in Table 20-3 for air quality and for ambient noise quality.

Table 20-3: Air Quality and Noise Monitoring Stations

Station		UTM Coordinates WGS 84-Zone 17S		Altitude (masl)	Description
Air	Noise	East	North		
AR-01	RU-01	688,101	932,2878	3,598	Located north of the project area
AR-02	RU-02	691,506	932,5846	2,703	Located to the northeast of the project area
AR-03	RU-03	690,241	932,7491	3,256	Located southwest of the project area

20.4.1 Air Quality

The data obtained for the parameters analyzed (particulate matter with a particle size of ≤10 µm (PM10), particulate matter with a particle size of ≤2.5 µm (PM2.5), Pb, SO2, H2S, CO and NO2) in 2012 and 2021 were compared with the National Environmental Quality Standards (ECA, for its acronym in Spanish) for Air Quality (Supreme Decree (D.S., for its acronym in Spanish) N° 074-2001-PCM-CONAM, D.S. N° 003-2008-MINAM and D.S. N° 003-2017-MINAM), following the guidelines and methodology recommended in the "Air Quality and Emissions Monitoring Protocol for the Mining Sub-Sector" of the General Directorate of Environmental Affairs (DGAA, for its acronym in Spanish) - Ministry of Energy and Mines (MINEM).

The concentrations obtained for the parameters analyzed show that they do not exceed the ECAs established for each parameter.

20.4.2 Environmental Noise

The results of noise level monitoring obtained during daytime and nighttime hours during the 2012 and 2021 surveys were compared with the ECAs for Noise, approved by D.S. N° 085-2003-PCM. The norm establishes the primary environmental quality standards for noise. It considers the Equivalent Continuous Sound Pressure Level with A-weighting for a given time interval (NPSAeqT), for different application zones: special protection, residential, industrial and commercial.

The stations complied with the ECA for daytime and nighttime hours during the entire analysis period in 2012.

In 2021 the NPSAeqT values recorded at stations RU-01, RU-02 and RU-03 in the daytime and nighttime period, taken in dry and wet season were as follows:

- NPSAeqT Daytime: both the values recorded at station RU-01 and station RU-02 comply with the ECA Noise for industrial zone (80 Db(A)); however, stations RU-01 and RU-03 exceeded the ECA Noise for residential zones (60 dB(A)) in the current standard.

- NPSAeqT Nocturnal: both the values recorded at stations RU-01, RU-02 RU-03 complied with the ECA Noise for industrial zone (70 Db(A)); however, all stations exceed the ECA Noise for residential zone (50 dB(A)) in the current standard.

The exceedances at stations RU-01 and RU-03 are due to the circulation of vehicles on the road near these stations and the presence of strong winds. The exceedances at station RU 02 are attributable to the location of this point at the Cañariaco Norte camp.

20.5 Hydrology

Almost the entire study area is located within the sub-watershed of the Jatun Yacu River, which is part of the upper watershed of the Cañariaco River. The sub-watershed is divided into a number of micro-watersheds, including the Gaspar, Jehuamarca, Norte, Uchuimarai, Vicente, Sunawaka Yacu, Zonahuaca, Tembladera, Popa and Inter watersheds. Meteorological information for the Project is provided in Section 18.10.1 and was obtained from the El Limon weather station.

The hydrological characteristics of the hydrographic units evaluated are directly related to precipitation.

Three defined seasonal periods can be distinguished in the study area: a wet period from January to April, a dry period from June to September, and two transition periods from October to December and May to June. Surface runoff is present in all months of the year.

Since there are no hydrometric stations within the study area, monthly flow series were generated using the flow transposition method based on measurements from the Cañariaco hydrometric station.

Estimated water flows included:

- Wet season occurs between the months of January and April, with flows ranging from 67.5 L/s for the Jehuamarca micro-watershed to 452.7 L/s for the Zonahuaca micro-watershed; and
- Dry season occurs between June and September, with flows ranging from 29.5 L/s for the Jehuamarca micro-watershed to 198.1 L/s for the Zonahuaca micro-watershed.

20.6 Hydrogeology

The hydrogeological system is characterized as a volcanic to subvolcanic aquitard, fractured intrusive with low permeability. Water movement is gravitational, from the higher elevations, where the greatest recharge occurs, to the lower elevations. At the present time there has been no site-based hydrogeological monitoring or testing conducted for the Project.

20.7 Surface Water Quality

Water quality analysis information was generated for the 2012 EIAsd based on a single sampling campaign carried out during May 2011.

Results were compared to standards sent out in the ECAs (D.S. N° 002-2008-MINAM), Category 3 (irrigation of vegetables and animal drinking) and Category 4 (Conservation of aquatic environments - Highland rivers).

The parameters analyzed in 28 samples were as follows:

- Physical-chemical: pH, conductivity, total dissolved solids (TDS), total suspended solids (TSS), carbonate and bicarbonate, total alkalinity.
- Inorganic parameters: sulfate, phosphate, chloride, fluoride, ammoniacal nitrogen, total nitrogen, total organic nitrogen, nitrates, nitrites.
- Organic parameters: oils and fats, phenolic compounds, geochemical oxygen demand (BOD5+, total and dissolved metals: B, Ag, Al, As, Ba, Ba, Be, Ca, Cd, Co, Cu, Cu, Fe, Hg, Li, Mg, Mn, Na, Ni, Pb, Se, (standard series by ICP).
- The following is a summary of the results for the most relevant parameters of environmental interest:
- Hydrogen potential (pH): the results are within the range established by the ECA in Category 3 and Category 4, except for the waters at sampling station K-WQ-04 located in the Oso Creek, which recorded a value below the established standards. The acidic behavior in these bodies of water is due to the mineralogical characteristics of the area, mainly due to the presence of sulfides, which tend to lower the pH of the water.
- Dissolved oxygen, conductivity, STS, carbonates and bicarbonates, sulfates, nitrates and BOD: the values were below the ECAs considered in each case. Lead exceeded the ECA Category 4 criteria.
- Nitrites and organic parameters (oils and fats): values below the laboratory detection limit were recorded at all stations. The results are below the ECAs considered in each case.
- Total nitrogen and ammonia and copper: limits exceeded in some of the stations the ECAs considered.
- Surface water monitoring programs completed during 2021 were carried out in the dry (September 2020) and wet seasons (January 2021).
- A total of eight surface water quality monitoring stations were sampled. Results were compared to the ECA for water, Category 3 D1: Vegetable Irrigation and D2: Animal Drinking of the D.S. N° 004-2017-MINAM.

Results included:

- Hydrogen potential (pH): The pH results at the surface water quality monitoring stations, for the most part, registered neutral characteristics and in some specific cases acidic characteristics.
- Dissolved oxygen: The results obtained at all stations in September 2020 were above the minimum value established in the ECA Water Category 3 (D1 and D2).
- Conductivity, fluoride, nitrate and zinc: the results obtained at all stations were below the ECA Category 3 Water value (D1 and D2), as applicable.
- Sulfate, chloride, chemical oxygen demand, biochemical oxygen demand, cyanide WAD, oils and fats, nitrite, bicarbonates, arsenic, copper, cadmium, mercury, lead, selenium, fecal coliforms and escherichia coli: the results obtained in most of stations were below the laboratory detection limit, in that sense the values are below the ECA value Water Category 3 (D1 and D2), depending on the case.

20.8 Soils

Two soil types were defined:

- Land suitable for pasture, of low agricultural quality, soil limitations and erosional risk: covers 445 ha (22%) of the Project area; and
- Land with a higher use capacity, protection lands, soil limitations and erosional risk: covers 1,600 ha (78%) of the Project area.

Soil quality sampling was conducted on 10 samples, with samples analyzed for organic and inorganic parameters:

- Inorganic parameters arsenic, barium, cadmium, free cyanide, hexavalent chromium, total chromium, mercury and lead; and
- Organic parameters: hydrocarbon fraction F1 (C6-C10), hydrocarbon fraction F2 (C10-C28) and hydrocarbon fraction F3 (C28-C40).
- Analytical results were compared to the ECAs for soils approved by D.S. N° 011-2017-MINAM.

None of the 10 samples analyzed exceeded the ECA limit value for both standards for both agricultural and industrial use.

20.9 Ecosystems

Four terrestrial ecosystems were identified (Yaku Consultants, 2021):

- Western Andean montane forest: characterized by medium-sized trees (25-30 m);
- Jalca: located in low slope areas at an altitude above 3,000 masl where the temperature and rainfall are moderate;
- Shrub thicket: characterized by the presence of shrubs and bushes; and
- Coastal and Andean Agriculture: associated with agricultural activity, either currently active or not currently in use.

20.9.1 Flora

A flora monitoring program was completed over seven transects in February 2021 (wet season) and September 2021 (dry season) as part of the 2021 EIAsd, using the variable transect method (Foster et al., 1995).

A total of 473 species of flora were recorded, distributed in 41 orders, 84 families and 238 genera:

- During the wet season, a total of 305 species of flora distributed in 39 orders, 79 families and 193 genera were recorded;
- During the dry season (September 2021), a total of 318 species of flora distributed in 35 orders, 70 families and 182 genera were recorded.

Seven species were classified as vulnerable (Vu), one as endangered (EN), six as critical (CR) and one as near threatened (NT) under the criteria set out in national legislation, S. D. N° 043-2006-AG.

A total of 49 species were in the category of least concern (LC), seven were in the vulnerable category (VU), two species in the data deficient category (DD) and one was categorized as near threatened (NT) under the International Union for Conservation of Nature (IUCN) listing (2021–2022).

There were 42 species that were listed in the Convention on International Trade in Endangered Species (CITES) Appendix II (2021).

20.9.2 Fauna

A monitoring campaign was conducted during January 2021 (wet season) and September 2021 (dry season), to characterize mammal, bird, reptile, amphibian and arthropod species in the study area. The evaluation was conducted in accordance with the requirements of the Wildlife Inventory Guide approved by Ministerial Resolution No. 057-2015-MINAM. Results are as follows:

- During the 2021 wet season, a total of 15 species of wild mammals were recorded. These species are represented by 15 genera, 12 families and five orders.
- In the dry season of 2021, a total of 17 species of wild mammals were recorded. These species are represented by 17 genera, and 15 families.
- Nine species of mammals were recorded as threatened, with Antavaca or mountain tapir (Tapir pinchaque) being the most endangered species: Critically Endangered (CR) according to Peruvian legislation, EN according to the IUCN, and within CITES Appendix I. The second most endangered species was the Andean bear (Tremarctos ornatus), which is listed as VU by the IUCN and Peruvian legislation and is also listed in CITES Appendix I.
- During the wet season, 52 species of birds were identified, which are taxonomically distributed in 12 orders and 24 families.
- During the dry season, 52 bird species were identified, distributed in 10 orders and 22 families.
    - In the study area, one bird species found on the list of Threatened Species of Wildlife in Peru (D.S. N° 004-2014-AG) was identified. Two species, according to the IUCN classification (2021–2022) are NT.
    - A number of bird species were on the CITES Appendix II listing (2021).
    - In the baseline study, the only endemic bird species reported is Thaumasius taczanowskii "Taczanowski's Hummingbird"; all other species have a wide geographic distribution.
    - The species composition of amphibians and reptiles in all sampling stations for the wet and dry seasons was 12 species, four families and two orders.
    - Of the amphibian and reptile species identified, one is considered EN, one as DD, one as LC and two as VU according to the IUCN Red List of Threatened Species (2021–2022).
    - According to national legislation (D.S. N°043-2006-AG), one species is considered VU, while none are within the CITES lists.

- Eight hydrobiological stations were evaluated in the wet and dry seasons during 2021. No fish were present in either season, and the absence of fish was confirmed by interviews with local residents.

20.10 Water and Waste Management

Section 18.6 provides details of the Co-placement Storage Facility (CPSF) design, instrumentation and monitoring, and CPSF water management plan. Tailings and waste rock from the process will be placed in the CPSF. Section 18.10 provides details and descriptions of water management structures including collection channels, diversion channels, and collection ponds, as well as a preliminary site water balance.

The primary objective for water management at the Cañariaco Norte Project is to minimize the volume of runoff contributing to the SCF while maintaining as much fresh water downstream as possible. This objective will be achieved by following the philosophy outlined below:

- Divert – as much non-contact water as possible will be diverted around the Project site and discharged downstream of the SCF into the Río Cañariaco.
- Collect – all surface runoff (contact water) and seepage from within the project footprint will be collected at the seepage collection facility. Runoff and seepage sources include the open pit, CPSF, Mill Site and surrounding haul roads.
- Recycle – as much water as possible will be recycled from the CPSF facility as make-up water to minimize the amount of freshwater make-up required from the SCF.
- Treat – all water collected at the SCF that is surplus to the project requirements will be treated to meet or exceed local water treatment standards and the treated water will be discharged downstream into the Río Cañariaco.
- Contain – all pipelines will be constructed in dedicated channels such that any spillage will be routed to separate emergency spill ponds and contained for recycle.

20.11 Closure and Reclamation Planning

20.11.1 Closure and Reclamation Plans

Site reclamation will comply with Peruvian environmental regulations and the International Finance Corporation (IFC) Environmental, Health and Safety Guidelines for Mining (IFC, 2007). The IFC guidelines state that closure and post-closure activities should be considered as early as possible in the planning and design stages.

The reclamation and closure plan will evolve hand in hand with the design as the Project progresses through feasibility and necessary permitting. Site-specific knowledge will be acquired during mine development and operations, and the closure plan will be updated to incorporate this knowledge in addition to the environmental and social conditions and circumstances at the time of closure.

The primary reclamation and closure objectives will be to:

- Minimize adverse socioeconomic impacts and maximize socioeconomic benefits to local communities;
- Ensure that future public health and safety will not be compromised;

- Ensure that subsequent use of the site is beneficial and sustainable for communities in the long term;
- Return areas disturbed by mining operations to their pre-mining land use, where possible; and
- Ensure the physical, chemical and ecological integrity of site features.

Despite these objectives, it will not be possible to fully return some areas to their pre-development conditions due to the physical changes that will result from mining operations.

Closure and post-closure plans will include appropriate aftercare and ongoing monitoring of the site, emissions and related potential impacts. The recommended minimum duration of post-closure monitoring is five years after chemical and physical stability has been achieved. Reductions in monitoring frequency will be a function of environmental performance and regulatory requirements. Closure costs are estimated at US$216 M.

20.11.2 Site Facilities Subject to Closure

The open pits, plant site, camp site and ancillary infrastructure, CPSF, and associated infrastructure comprise the reclamation sites for the Project and will be the focus of the following closure concepts. Responsibility and control for reclamation and closure of the latter two areas will be transferred to the government or other corporations at the time of closure.

20.11.3 Decommissioning and Reclamation Measures

Alta Copper plans to undertake progressive reclamation where feasible to enhance the overall reclamation of the site. Procedures at the Cañariaco Project will be site specific to achieve the required results. The plan will be updated as the Project progresses.

20.12 Permitting Considerations

In accordance with the requirements of the Peruvian Legal Framework and the Ministry of Energy and Mines, Alta Copper must initiate a process to obtain all environmental, construction and mining permits pertaining to the Project for all the various levels of exploration and development. The main permitting requirements identified to date for the commencement of construction and mining operations are summarized as follows:

- Environmental studies and permits;
- Water authorization;
- Archaeological assessment;
- Closure plan;
- Mining operation certificate (MOC);
- Easement by agreement;
- Construction permit;
- Municipal permits to build infrastructure within their limits (if any);
- Labour permits;

- Surface water use license;
- Groundwater use license;
- Sanitary authorization for wastewater treatment;
- Permits to build roads;
- Deed of transportation of controlled substances and products;
- Beneficiation Concession (necessary to process the mineral); and
- Authorization to start operation.

Alta Copper is committed to complying with the applicable Peruvian environmental and mining regulations. Legislation will need to be continually reviewed to identify all required social and environmental studies, permits and authorizations, as well as the permitting process and the schedule itself so that the Project schedule can be matched to the licensing schedule.

20.13 Pre-Construction Permits

This sub-section describes some of the high priority permits that Alta Copper must acquire prior to commencing the construction phase. The process for obtaining construction and operational permits will depend on whether all required pre-construction permits have been obtained.

20.13.1 Surface Land Use

The area proposed for Project development is located within the lands of the Community of San Juan de Cañaris. The community is recognized by the Peruvian Government as the owner and beneficiary of the surface land in its name. To ensure that it can access all sites of the Project facilities, Alta Copper must enter into an agreement with Community of San Juan de Cañaris representatives to obtain permission to use the surface land. Alta Copper currently has an agreement with the Community of San Juan de Cañaris to proceed with environmental, social and engineering studies. An agreement for surface land use for drilling, construction and mining operations will be applied for as a key component of the project development process, prior to project construction. A second level access negotiation will also be required, primarily for the open pit area; this area is parceled between individual landowners.

The time required for Alta Copper to acquire the necessary surface rights for the Project will depend on the willingness of the other parties to negotiate and the formalities that must be completed to ensure that all parties acknowledge the validity of all land disposition agreements.

20.13.2 Archaeological Evaluation

Mining Projects in Peru must ensure that the development property is evaluated for the potential presence of historical or archaeological remains (Proyecto de Evaluación Arqueológica). Depending on the results of the evaluation, the area in question could be subject to an Archaeological Excavation Evaluation by the General Directorate of Archaeological Heritage of the National Institute of Culture (INC) or by an archaeologist registered in the National Registry of Archaeologists, which is administered by the INC. This evaluation, in turn, could lead to the need for an Archaeological Rescue Project, as recommended by the National Archaeological Technical Committee. Once the evaluation or rescue

Project is completed and approved, as appropriate, a Certificate of Non-existence of Archaeological Remains (CIRA) will be granted.

To date, Alta Copper has carried out the Archaeological Evaluation Project for the Cañariaco Norte and Sur open pit areas and the Quebrada Verde exploration area. The results showed that these areas and the surrounding wooded areas have no archaeological sites or remains. As a result, the government has issued CIRA Certificates for these three areas confirming the absence of archeological remains. It is suspected that other forested areas within the Project footprint will have similar results. Other Project components including the process plant, camp and comingled waste rock and dry stack tailings facilities will require archeological assessments prior to construction. Environmental Impact Assessment and Closure Plan.

20.13.3 Environmental Impact Assessment

In accordance with the Environmental Protection Standard on Mining and Metallurgical Activities, Alta Copper must define all Project components as part of an Environmental Impact Assessment (EIA). Many regulatory procedures and permits are directly dependent on the approval of the EIA.

A semi-detailed Environmental Impact Assessment (EIAsd) for the exploration stage of the Cañariaco Project was prepared by AMEC and approved by Directorial Resolution No. 177-2012-MEM/AAM in May 2012. Further work was conducted in 2014 by GEMA to support drilling permits and AMEC worked on the EIA detailed (for exploitation) from 2007 to 2014. Yaku Consultants conducted additional environmental studies for the project from 2021 through 2023.which was used to support the Declaracion de Impacto Ambiental (“DIA”) drilling permit application submitted to the Ministry of Energy and Mines (“MINEM”) in December 2023. According to DS 033-2005-EM, a closure plan must be submitted within a maximum period of one year from the approval of the EIA. However, for Project activities to proceed in a timely manner, it is recommended that the Closure Plan be submitted within three months of EIA approval.

20.13.4 Others

Once the EIAd (Detailed EIA) is submitted to SENACE, applications can be prepared and submitted for other permits as listed below; however, approvals cannot be granted until the EIAd has been approved:

- Benefit Concession;
- Mining Transport Concession;
- Definitive Electricity Concession;
- Establishment of Electric Easement;
- Water use license for domestic or mining use;
- Favourable technical report for direct consumers of liquid fuel; and
- Deforestation permit for holders of operations and activities other than forestry.

The receipt of the above authorizations is subject to various conditions and procedures.

20.13.4.1 Powerline Easement

Alta Copper will need a powerline easement if its electrical supply line crosses an area owned by residents or third parties. The easement cannot be granted until the final grant is approved.

20.13.4.2 Water Use

The National Water Resources System was established by the Water Resources Law 29338 and its regulation (DS 001-2010-AG), which stipulates that the National Water Authority (ANA) is responsible for granting water use rights through decentralized offices, known as Local Water Administrators (ALA), in each of the main regional basins. The ALA of the Jaen Basin will be responsible for granting the water use license for the Cañariaco Project. The license must consider all the requirements and procedures of current legislation.

According to ANA, water license studies include hydrological information, a site description, downstream water use estimates, and a water balance study. In addition, a feasibility-level hydraulic engineering description is required for all facilities related to water abstraction. These elements will be included in the future EIA and/or engineering studies.

20.13.4.3 Road Use

Authorization to cross existing access roads or railroad tracks is framed in the Single Text of Administrative Procedures (TUPA, for its acronym in Spanish) of the Ministry of Transportation and Communications (MTC, for its acronym in Spanish), but the General Directorate of Roads and Railroads has not established specific procedures for obtaining permits for such crossings. In practice, however, it is appropriate to request permission to cross existing roads and railroads because they are public property. S. D. N° 034-2007-MTC and its Annex detail road classifications as national, regional or municipal.

If any of the Project's facilities or access roads are in an area considered by a Provincial Municipality as Urban Housing Development or Urban Expansion Area, then Alta Copper must obtain all required permits from the Municipality.

S. D. N° 021-2008-MTC approved the National Regulation for Land Transportation of Hazardous Materials and Waste, which seeks to regulate activities, processes and operations for land transportation of hazardous materials and waste. These procedures have not yet been incorporated into the MTC's TUPA, but this is expected to change. The MTC will be the competent authority for licensing vehicles for the transportation of hazardous materials and wastes, including explosives. This responsibility previously fell to the Directorate for the Control of Security Services, Control of Firearms, Ammunition and Explosives for Civilian Use.

20.14 Construction and Operations Permits

20.14.1 Construction Phase

Once all required permits for the pre-construction phase have been obtained, the following licenses and permits are considered most critical:

- Authorization for the private provision of telecommunications services.

- License for the operation of an explosives depot.
- Authorization for the eventual use of explosives and related materials.
- Explosives handler's license.
- Issuance of the explosives transit guide.

20.14.2 Operations Phase

Prior to the commencement of the operations phase, the following authorizations and regulatory requirements may be necessary:

- Authorization to start exploitation activities in metallic mining concessions.
- Mining Operation Certificate.
- Multidisciplinary Civil Defense Technical Safety Inspection.
- Permits and licenses from municipalities as required.
- Biannual authorization for use of explosives, supplies and related materials (global authorization).
- Registry or updated registry or information for the control of chemicals and controlled products.
- Monthly reports on special registers of chemicals and controlled products, maintained manually or electronically, even when there have been no changes.

20.15 Social Considerations

20.15.1 Socioeconomic Background Information

The Cañariaco Project is located within the Lambayeque Region, in the province of Ferreñafe, in the district of Cañaris in northern Peru. In addition to this political-administrative division, the Community of San Juan de Cañaris has surface rights in the development area of the proposed Cañariaco Project.

In Peru, a campesino (peasant) community is defined as a group of families with collective rights to land ownership. These communities are registered with the Public Registry Office and are considered a legal entity, such as an association or company. The community must renew its registration every two years by submitting a complete list of its active members, mainly adult men. A village is defined as a group of houses in a rural area. A social group consists of several people who interact and share a goal and norms. For example, the farming community itself is a social group comprising several primary social groups.

The Community of San Juan de Cañaris was formally recognized by the State on October 10, 1956, and now comprises more than 40 villages and 3,000 families (primary social groups). The village of Cañaris is adjacent to an unnamed tributary of the Cañariaco River on the east side of the Cañariaco valley, 6 km from the Project site. According to the Instituto Nacional de Estadística (INEI), the village of Cañaris has 323 inhabitants grouped into 81 families. The community's origins are pre-Hispanic and despite transformations over the centuries, the community retains the main characteristics that have allowed it to organize and govern the underdeveloped territory in which it is located. These include:

- Partial collective management of the surficial natural resources within its territory (land, water, forest, natural pastures).
- An own social organization in which authorities and representatives are democratically elected. The community board, which is elected every two years, is the body that represents the community consisting of all community members.

Road infrastructure within the Community of San Juan de Cañaris and its various farms is limited, such that roads connecting to the two main paved road leading to the nearest intermediate cities, such as Chiclayo (Lambayeque Region) and Jaén (Cajamarca Region) make those cities up to 100 km away from many farms and hamlets.

Except for the access road and power line, the Project components will be constructed along the Cañariaco River and Quebrada Yerma watershed, covering altitudes ranging from 2,600 to 3,600 masl. The territory includes villages, communal agricultural areas or individual plots, collective use pastures and the Cañaris forest zone. In the latter case, the forest is partially parceled among landowners, but it is also recognized that the entire forest area in the Community of San Juan de Cañaris is a common community resource, especially for water harvesting.

The residents of the Community of San Juan de Cañaris are defined by the United Nations Statistical Commission as "absolute and relatively poor". In 2007, AMEC conducted a household survey and found that living standards were well below the national average. For example, the study showed that 86% of families have monthly incomes of less than US$100. Another important indicator of vulnerability is the high illiteracy rate of 46%.

Adequate sanitation is scarce; the AMEC survey found that more than 50% of households do not have running water. Water supply is derived directly from streams that are also used for livestock, ablution and defecation and as waste receptacles for plastics, cans and household waste. Ninety-five percent of the families use wood obtained from the cloud forests for cooking inside their homes, without adequate exhaust, which generates a high rate of respiratory diseases, especially among women and children who spend most of their time indoors.

20.15.2 Archaeological Studies

The Cañariaco Project area has been granted CIRA certificates for the Cañariaco Norte, and Sur mine areas and the Quebrada Verde exploration area.

In 2007, 2010, May 2011 and June 2014, archaeological evaluations were carried out, the results of which were presented as part of the EIAsd and the Modification of the EIAsd, approved by Directorial Resolution No. 177-2012-MEM/AAM and Directorial Resolution No. 462-2014-MEM/DGAAM, respectively. In October 2021, Yaku Consultants conducted an archaeological assessment to complement the evaluation of the proposed study area. Study evaluations and results include:

- Archaeological Evaluation Project (2007): this study analyzed the area of the potential open pit (area of 121 ha and a perimeter of 7,000 m), in the Cañariaco A, Cañariaco B, and Jehuamarca concessions. A total of 55 boreholes were drilled and no archaeological evidence was recorded (AMEC, 2014).
- Archaeological Reconnaissance Report (2010): the area of the Cañariaco B and Cañariaco C concessions was evaluated, which included the surface lands of the Atunloma and Mamagpampa settlements, part of the

Community of San Juan de Cañaris. No record of archaeological evidence or settlements were noted on the surface of the Quebrada Verde area (Cañariaco C and Cañariaco B Concession) (AMEC, 2014).

- Archaeological Reconnaissance Report (May 2011): the area corresponding to the Achicamonte and Jatun Yacu streams, which are located in the towns of Atunloma, Mamagpampa, San Juan de Yoyoca, Mitubamba and Cañaris, all belonging to the Community of San Juan de Cañaris, was evaluated. During the archaeological reconnaissance, it was found that there is no movable or immovable archaeological evidence in the towns and their surroundings in the area of direct influence of the Project. Ceramic evidence was observed in the village of San Juan de Yoyoca; however, the pottery was highly deteriorated and very disturbed. Ceramics were observed in the middle of farms and on roads (AMEC, 2014).
- Archaeological Reconnaissance Report (June 2014): an area of 1,300 ac was evaluated, covering the mining concessions Cañariaco B, Cañariaco C, Cañariaco D, Jehuamarca 2, Jehuamarca 3 and Jehuamarca 4. No archaeological evidence was identified in the area. There was no indication or trace of pre-Hispanic human occupation (AMEC, 2014).
- Archaeological Reconnaissance Report (October 2021): five areas were evaluated that together cover 313.34 ac, located between 2,300 to 3,066 masl. This archaeological reconnaissance did not include any type of physical intervention (excavations) or collection of material and consisted only of a photographic record and georeferencing of the findings.
- Archaeological Reconnaissance Report (September 2023): sixteen areas in Cañariaco Sur were evaluated to obtain the respective CIRA. The total area was 4,366,865.517 m2 with a perimeter of 36,085.796 m located between 3,000 to 3,620 masl. This archaeological survey was carried out with the accompaniment of the Representatives of the Ministry of Culture who validated and approved the respective CIRAs. This recognition did not include any type of physical intervention (excavations) or collection of material, it only consisted of a photographic record and visualization in the field.

20.15.3 Public Consultation and Engagement

The public consultation and engagement process for the preparation of the 2012 EIAsd and the 2023 DIA were carried out in accordance with the provisions of D.S. N° 028-2008-EM (Regulations for Citizen Participation in the Mining Sub-Sector) and according to the rules approved by Ministerial Resolution (M.R.). N° 304-2008-MEM/DM, which details the participation mechanisms to be implemented at different stages of the development of a mining project.

There are also ongoing and recent meetings with various levels of Peruvian authorities to discuss Company activities, potential benefits, legal regulations and required permissions to support exploration and mining (Alta Copper press release dated March 13, 2024).

Based on recent news releases from Alta Copper (February 29, 2024), the Company continues to advance its involvement with the local community near to the Project by means of:

- Financial and resource commitments to substantially contribute towards the development of sustainable projects in the area.
- Considerable investments in bringing local agricultural producers technical assistance and access to markets at fair prices.

- Articulation of social infrastructure improvements to be financed by other public and private sources.
- Alta Copper's community team including local community members and consultants conduct camp and road maintenance, environmental tasks and archaeological studies to support drilling programs and . Involving local community members in this work provides opportunity for community engagement.
- Establishment of four offices allow for ongoing opportunities to engage with the local community and to understand community concerns and needs.

20.16 Comments on Environmental Studies, Permitting and Social or Community Impact

Environmental baseline studies are well advanced for the Project and will help to form the basis for Environmental Impact Assessment and future permitting applications. There remain a few areas where baseline studies need to be initiated and advanced including groundwater studies and modelling, geochemistry studies, and further refinement of the mine water balance. These studies will help to inform future Project design, mine water effluent predictions and effluent treatment requirements. Continued socio-economic studies and community engagement efforts will help to identify community needs and provide a basis for targeted community investment in local development projects, training, education, and employment. These activities will help to gain community support that will be required to progress the Project on a timely basis through the regulatory process. Implementation of the recommendations provided in Section 26.8 will help to address and mitigate permitting and community risks.

21 CAPITAL AND OPERATING COSTS

21.1 Introduction

The capital and operating cost estimates presented in this PEA provide substantiated costs that can be used to assess the preliminary economics of the Cañariaco Project. The calculations are based on an open pit mining operation, processing plant development, infrastructure, co-placement waste rock and tailings storage facility, and the Owner's expenses and provisions. The project anticipates a life of mine life (LOM) for 28 years with an annual average mill throughput of 43.8 Mt.

21.2 Capital Costs Estimate

21.2.1 Capital Cost Summary

The capital cost estimate conforms to Class 5 guidelines for a PEA-level estimate with a -30% to +50% accuracy according to the Association for the Advancement of Cost Engineering International (AACE International). The capital cost estimate was developed in Q2 2024 US dollars based on Ausenco's in-house database of projects and studies, budgetary level equipment quotes as well as experience from similar operations.

The estimate includes the open pit mine, process plant, site services and utilities, The capital cost summary is presented in Table 21-1. The total initial capital cost for the Cañariaco Project is US$2,160.2 M; and life-of-mine sustaining costs are US$518.3 M. Closure costs are estimated at US$216 M.

Table 21-1: Summary of Capital Costs

WBS	WBS Description	Initial Capital Cost (US$M)	Sustaining Capital Cost (US$M)	Total Capital Cost (US$M)
2000	Mine	429.0	186.0	615.0
3000	Process Plant	788.7	25.0	813.7
4000	Site Services and Utilities	105.9	-	105.9
5000	On-site Infrastructure	150.8	216.5	367.3
6000	Off-site Infrastructure	41.9	-	41.9
-	**Total Directs**	**1,516.3**	**427.5**	**1,943.8**
7000	Common Construction Facilities and Services	245.1	13.7	258.8
8000	Owner's Cost	30.3	-	30.3
-	**Total Indirects**	**275.4**	**13.7**	**289.1**
9000	Provisions	368.5	77.1	445.6
-	**Project Total**	**2,160.2**	**518.3**	**2,678.5**

Note: Totals may not sum due to rounding.

21.2.2 Basis of Estimate

All capital and operational costs were developed based on Ausenco and AGP's in-house database of costs and labour rates. The estimate is prepared in United States dollars (currency: USD; symbol: US$). Where applicable, pricing has been converted to United States dollars using the exchange rates in Table 21-2.

Data for the estimates have been obtained from numerous sources, including the following:

- Mine schedules;
- Conceptual engineering design by Ausenco and AGP mining;
- Mining costs were estimated by AGP; remaining scope by Ausenco;
- Major mechanical equipment costs are based on vendor quotations, first principles, and Ausenco's database of historical projects;
- Cost for concrete, steel, instrumentation, in-plant piping, and platework were factored by benchmarking against similar projects with equivalent technologies and unit operations;
- Earthworks material take-offs are developed based on the overall site layout.
- Engineering design at a PEA level;
- Topographical information from existing topographical maps, LIDAR and air-photo imagery;
- Data from similar recently completed studies and projects; and
- Site layout (which provided preliminary footprint areas of buildings).

Table 21-2: Currency

Currency Abbreviation	Symbol	Currency	Exchange Rate
CAD	C$	Canadian Dollar	0.74
USD	US$	United States Dollar	1.00
SOL	S/	Peruvian Sol	0.27
EUR	€	Euro	1.08

The following parameters and qualifications were considered:

- No allowance has been made for exchange rate fluctuations;
- There is no escalation added to the estimate, and
- A contingency allowance was included.

  Major cost categories (permanent equipment, material purchase, installation, subcontracts, indirect costs, and Owner's costs) were identified and examined.

21.2.3 Mine Capital Costs (WBS 2000)

Direct capital costs were those costs that pertain to the permanent equipment, freight, materials and labour associated with the physical construction of the facilities including refurbishment costs. Contractor's indirect costs, which included contractor's distributable costs, are contained within the direct costs. Ausenco and AGP provided the direct costs associated with the works in their respective discipline areas.

The mining capital cost estimate was grouped into initial and sustaining costs in six main categories as shown in Table 21-3.

Table 21-3: Mining Capital Cost Breakdown

WBS	Mining Capital Description	Initial Cost (US$ M)	Sustaining Cost (US$ M)	Total Capital Cost (US$ M)
2200	Haul Roads and Heavy Civil Works	5.0	-	5.0
2300	Open Pit Mine Equipment	35.2	133.5	186.7
2400	Mine Services	4.8	18.0	4.8
2500	Mine Infrastructure	64.9	9.0	83.7
2600	Mine Prestrip	238.3	-	238.3
2800	Mine Waste Rock Conveyor	80.8	25.4	106.2
-	**Total**	**429.0**	**186.0**	**615.0**

Note: Totals may not sum due to rounding.

21.2.3.1 Haul Roads and Heavy Civil Works

The costs accumulated in this category consist of various haul road and access road construction requirements. The road from the pit to the CPSF will be 4 km long with a 36 m wide running surface. Within the mining area, an initial 4 km of haul road is also included.

21.2.3.2 Open Pit mine equipment

Mine capital costs have been derived from historic data by AGP mining.

The mining equipment capital costs reflected the use of financing of the major equipment and some support equipment. Equipment prices used current quotations from local vendors. A 20% down payment was included in the capital cost for those units financed. The remaining cost was included in operating costs (refer to Section 21.3.3). The capital cost, the cost of financing, and down payment, are shown in Table 21-4.

Table 21-4: Major Mine Equipment – Capital Cost, Full Finance Cost and Down Payment

Equipment	Unit	Capacity	Capital Cost (USM$)	Full Finance Cost (USM$)	Down Payment (USM$)
Production drill	mm	160	1.3	1.5	0.3
Production drill	mm	270	3.4	3.9	0.7
Production excavator	m^3	6.7	1.6	1.8	0.3
Production loader	m^3	33	10.2	11.8	2.0
Electric hydraulic shovel	m^3	38	12.8	14.8	2.6
Haulage truck	t	40	0.3	0.3	0.0
Haulage truck	t	227	5.1	5.9	1.0
Haulage truck	t	290	6.6	7.7	1.3
Crusher loader	m^3	20	5.4	6.3	1.1
Track dozer	kW	474	1.2	1.4	0.3
Grader	kW	163	0.3	0.4	0.1

The cost of spare truck boxes, shovel and loader buckets were included in the capital cost for the major equipment cost estimate.

The distribution of capital costs was completed using the number of units required within a period. If new or replacement units were needed, that number of units, by the unit cost (20% of that for major equipment) was applied to the capital cost in that period. There was no allowance for escalation in any of these costs.

The balancing of equipment units based on operating hours was completed for each major piece of mine equipment. The smaller equipment was based on number of units required, based on operational experience. This included such items as pickup trucks (dependent on the field crews), lighting plants, mechanics trucks, etc.

The most significant piece of major mine equipment is the haulage trucks. There are two fleets of large haulage trucks at various moments in the mine life. The 227 t trucks are initially purchased but as the stripping requirements increase to maintain the production rate the replacement trucks will be 290 t class trucks. At their peak there will be 34 units of 221 t capacity will be necessary to maintain mine production. The 290 t trucks will peak at 38 units over the mine life. The production fleet will consist of 34 m^3 hydraulic shovels with a 33 m^3 loader as backup. Drilling will be completed by 270 mm drills with a smaller 160 mm drill for pre-shear holes, horizontal drain holes and backup.

The support equipment was scheduled for replacement on a number of years of usage basis. For example, pickup trucks will be replaced every three years, with the older units possibly being passed down to other departments on the mine site. However, for the purpose of the capital cost estimate, new units were considered for mine operations, engineering, and geology.

The number of pieces of major equipment required by year are shown in Table 21-5.

Table 21-5: Mine Equipment On-Site

Equipment	Yr-3	Yr-2	Yr -1	Yr 1	Yr 5	Yr 10	Yr 15	Yr 17	Yr 20	Yr 25	Yr 28
Production drill (160 mm)	1	2	2	3	3	3	3	-	2	2	2
Production drill (270 mm)	-	1	2	4	5	6	7	-	6	6	6
Production excavator (6.7 m^3)	2	3	4	4	4	4	4	-	4	4	4
Production loader (33 m^3)		1	2	2	2	2	2	-	1	1	1
Electric hydraulic shovel (34 m^3)	-	-	-	1	3	5	6	-	6	5	5
Haulage truck (40 t)	8	40	43	32	12	-	-	22	-	-	-
Haulage truck (221 t)	-	7	13	27	34	34	34	-	27	-	-
Haulage truck (290 t)	-	-	-	-	-	9	36	-	38	29	29
Crusher loader (20 m^3)	-	-	-	1	1	1	1	-	1	1	1
Track dozer (455 kw)	4	7	7	7	7	6	6	-	6	6	6
Grader (146 kW)	3	3	3	3	3	3	3	-	3	3	3

There will be one full-time loader at the primary crusher when the plant commences operation. Its role will be to tram material from the short-term stockpile as required.

The expected equipment life is:

- Production drill (small): 25,000 hrs;
- Production drill (large): 60,000 hrs;
- Production excavator: 35,000 hrs;
- Production loader: 60,000 hrs;
- Electric hydraulic shovel: 84,000 hrs;
- Haul trucks (227 t, 290 t): 84,000 hrs;
- Haul truck (40 t) : 12,000 hrs ;
- Crusher loader: 40,000 hrs;
- Track dozer: 35,000 hrs; and
- Grader: 25,000 hrs.

Other support equipment replacement is determined in number of years and varies by the support equipment duty in the mine. Lighting plants for example will be replaced every four years. The integrated tool carrier for site support will be purchased at the Project start and will be replaced every 10 years over the mine life.

21.2.3.2.1 Mine Services

The mine services estimate included the mine dewatering system, dispatch and engineering office equipment including mine design software.

21.2.3.2.2 Mine Infrastructure

Mine infrastructure all costs associated for the includes the radio communication network, pit ring powerline and the truck shop sufficient for the mining fleet proposed.

21.2.3.2.3 Mine Pre-production Stripping

Mining activity will commence in advance of the process plant achieving commercial production. This includes the movement of 78.3 Mt of waste and 1.8 Mt of mill feed prior to the primary crusher being ready for mill feed material. The mine operating costs in Year -3, -2 and Year -1 were included in the capital cost estimate with the total expected cost to be US$238.3 M. This cost covered all associated management, dewatering, drilling, blasting, loading, hauling, support, engineering and geology departments labour, grade control costs and mine equipment financing costs.

21.2.3.2.4 Mine Waste Rock Conveyor

This section accounts for all costs for the conveyors constructed to transport the mineralized material and waste rock from the North pit to the process plant and CPSF.

21.2.4 Process Capital Costs (WBS 3000)

Process plant costs are summarized in Table 21-6. The definition of process equipment direct costs includes all contractors' direct and indirect labour, permanent equipment, materials, freight, and mobile equipment associated with the physical construction of the areas.

Process equipment requirements are based on conceptual process flowsheets and process design criteria as defined in Section 17. Major mechanical equipment was sized based on the process design criteria to derive equipment lists. The estimate of major equipment was developed based on a compiled and priced mechanical equipment list containing a combination of historical costs from Ausenco's database and direct vendor quotes.

In support of the major mechanical and electrical equipment packages, the process plant and infrastructure engineering designs were completed to a PEA study level of definition. Bulk material quantities were derived for earthworks and priced from other benchmark projects. All other quantities for electrical and instrumentation, concrete, steel, piping, cable, and platework were factored and priced.

The sustaining capital costs are derived from the crushing station installed in the Sur deposit in Year 16.

Table 21-6: Summary of Process Plant Capital Costs

WBS	WBS Description	Initial Capital Cost (US$M)	Sustaining Capital Costs (US$M)
3100	Primary Crushing	29.6	25.0
3200	Process Plant	517.1	-
3300	Overland Conveyors	80.2	-
3400	Reagents Storage	3.4	-

WBS	WBS Description	Initial Capital Cost (US$M)	Sustaining Capital Costs (US$M)
3500	Plant Services	11.6	-
3600	Tailings Filter Plant	146.8	-
3000	**Process Plant Total**	**788.7**	**25.0**

Note: Totals may not sum due to rounding.

21.2.5 Site Services and Utilities (WBS 4000)

Site services and utilities are summarized in Table 21-7. The costs are developed based on the Ausenco's in house database of costs and include the following:

- Site Water Management;
- Plant fuel storage and distribution;
- Sewerage and waste management;
- Communications, IT, and computing;
- Onsite power supply;
- Mobile equipment fleet; and
- Potable water treatment plant and distribution

Table 21-7: Summary of Site Services and Utilities

WBS	WBS Description	Initial Capital Cost (US$M)	Sustaining Capital Costs (US$M)
4100	Site Stormwater Management	6.4	-
4200	Plant Fuel Storage and Distribution	0.7	-
4300	Sewerage and Waste Management	2.1	-
4400	Communications, IT, and Computing	0.5	-
4500	Onsite Power Supply (Substation)	94.4	-
4600	Mobile Equipment Fleet	1.3	-
4700	Onsite Potable Water Treatment	0.4	-
4800	Onsite Fresh and Potable Water	0.1	-
4000	**Site Services and Utilities Total**	**105.9**	**105.9**

21.2.6 On-site Infrastructure Capital Costs (WBS 5000)

On-site infrastructure costs are summarized in Table 21-8. The costs were developed based on Ausenco's in-house database of costs and include the following:

- Site preparation and bulk earthworks;
- Drainage for site;

- Warehousing, office, and workshops;
- Administration Building and Medical Clinics;
- Site water management services;
- Assay/met lab;
- Permanent site camp with 420 beds; and
- Comingled waste and dry tailings storage facility equipment and required civil works.

Sustaining costs reflect the earthworks required for the construction of the Sur deposit crushing station in year 16 and the ongoing civil works for the comingled waste and dry tailings storage facility.

Table 21-8: Summary of On-Site Infrastructure Capital Costs

WBS	WBS Description	Initial Capital Cost (US$M)	Sustaining Capital Cost (US$M)
5100	Buildings and Facilities	9.9	-
5200	Roads	1.4	-
5300	Permanent Camp	2.4	-
5400	Plant Site Earthworks and Drainage	34.1	6.3
5500	CPSF Stacking System	76.8	-
5600	CPSF Heavy Civil Works	26.1	210.2
5000	**On-site Infrastructure Total**	**150.8**	**367.3**

Note: Totals may not sum due to rounding.

21.2.7 Off-site Infrastructure Capital Costs (WBS 6000)

Off-site infrastructure costs are summarized in Table 21-9. The costs were developed based on Ausenco's in-house database of costs and labour rates. The off-site infrastructure includes:

- Upgrades to the main access roads and other external roads up to 24 km;
- Allowance for the port of Salaverry;
- Allowance for wells and piping to site; and
- External power supply through a 220 kV 55km power line.

Table 21-9: Summary of Off-Site Infrastructure Capital Costs

WBS	WBS Description	Initial Capital Cost (US$M)	Sustaining Capital Cost (US$M)
6100	External Roads	14.4	-
6200	Port Allowance	9.5	-
6300	External Water Supply	3.2	-
6400	External Power Supply	14.7	-
6000	**Off-Site Infrastructure Subtotal**	**41.9**	0

Note: Totals may not sum due to rounding.

21.2.8 Indirect Capital Costs (WBS 7000, 8000, 9000)

Indirect capital costs are calculated as a percentage of the direct non-mining costs. WBS 7000 covers common construction facilities and services provided in the initial years of construction. WBS 8000 covers the owner's costs and WBS 9000 covers the provisions to mitigate the risks of uncertainties that can occur. The costs are summarized in Table 21-10 and are described in the following subsections.

Table 21-10: Summary of Indirect Costs

WBS	WBS Description	Initial Capital Cost (US$M)	Sustaining Capital Cost (US$M)
7100	Field Indirects	61.5	8.1
7200	Project Delivery	110.8	-
7300	Vendor Representatives	4.7	0.8
7400	Spares	14.0	1.6
7500	First Fills	9.4	-
7600	Commissioning Support	2.3	3.2
7700	Construction Camp	15.0	-
7800	Construction Camp Catering, Cleaning and Maintenance	27.4	-
8100	Owner's Cost	30.3	-
9100	Contingency – Mining	14.3	-
9200	Contingency – Plant and Infrastructure	354.2	69.5
7000,8000,9000	**Total Indirects and Contingency**	**643.9**	**83.2**

Note: Totals may not sum due to rounding.

21.2.8.1 Field Indirects (WBS 7100)

Field indirects are required during the project delivery period to enable and support construction activities. Indirect costs include the following:

- Temporary construction facilities and services;
- Commissioning representatives and assistance;
- On-site materials transportation and storage;
- Spares (commissioning, initial, and insurance);
- Freight and logistics; and
- Engineering, procurement, and construction management services.

Field indirect costs have been based on Ausenco's historical project costs of similar nature and have been included at a rate of 5% of the total direct non-mining cost or US$61.5M.

21.2.8.2 Project Delivery (WBS 7200)

The project delivery cost has been calculated at 9% of processing plant & infrastructure based on Ausenco's historical project costs of similar nature. This includes the following:

- Engineering, procurement, and construction management services (EPCM); and
- Project delivery.

Costs are estimated at US$110.8M.

21.2.8.3 Vendor Representatives (WBS 7300)

Vendor representative costs during commissioning and construction are estimated at 1.0% of total equipment supply based on Ausenco's experience.

Costs are estimated at US$4.7M.

21.2.8.4 Spares and First Fills (WBS 7400 & 7500)

The spares and first fills have been calculated at 3% and 2% respectively of total equipment cost in the process plant based on Ausenco's historical project costs of similar nature.

Costs are estimated at US$23.4M.

21.2.8.5 Commissioning Operations Readiness (WBS 7600)

The commissioning operations and project readiness cost has been calculated at 0.5% of total equipment supply in the process plant based on historical projects in the same region. This includes the following:

- Commissioning of the plant to assist in achieving plant nameplate capacity (EPCM); and

Costs are estimated at US$2.3M.

21.2.8.6 Accommodations (WBS 7700, 7800)

WBS 7700 and 7800 covers the construction costs and the catering, cleaning and maintenance of the construction camp during preproduction. The cost is benchmarked from projects in the similar region. Camp catering, cleaning and maintenance have been calculated for a total of $25/person-day for the preproduction period based on historical project costs of similar size to the Cañariaco project.

21.2.8.7 Owner's Capital Costs (WBS 8000)

The Owner's costs are estimated at USD$30.3 M. Owner's cost include the following:

- Project staffing and miscellaneous expenses;
- Pre-production labour;

- Home office project management; and
- Home office finance, legal, and insurance.

21.2.8.8 Contingency (WBS 9000)

Contingency is a provision of funds for unforeseen or inestimable costs within the defined project scope relating to the level of engineering effort undertaken and estimate/engineering accuracy and applied to provide an overall level of confidence in costs and schedule outcomes. The contingency is meant to cover events or incidents that occur during the project that cannot be quantified during the estimate preparation and does not include any allowance for project risk.

The estimate contingency does not accommodate the following:

- Abnormal weather conditions;
- Changes to market conditions affecting the cost of labour or materials;
- Changes of scope within the general production and operating parameters;
- Effects of industrial disputations;
- Financial modelling;
- Technical engineering refinement; and
- Estimate inaccuracy.

The total estimated contingency for the project is US$368.5 M during the construction and US$ 69.5 M for sustaining.

21.2.8.9 Closure Costs

The estimated total reclamation and closure costs, exclusive of taxes and contingency, for the Cañariaco project is US$216M.

21.2.8.10 Salvage Value

Salvage value for the Cañariaco project is estimated at US$76.4 M. Salvage value was calculated as 15% of the total process and mining equipment direct costs.

21.2.9 Life Of Mine Sustaining Capital

The life of mine sustaining cost for the project is estimated at US$518.3M. Mining sustaining costs include US$186.0M direct costs and US$7.5M indirect costs. The sustaining costs of additional facilities include USD$241.5M direct costs and US$83.2M indirect costs.

21.2.9.1 Additional Facilities

The sustaining cost under additional process facilities includes the CPSF expansion and process plant costs of the inclusion of the Sur crushing station and overland conveyors. The crushing circuit and the overland conveyor connecting the Sur deposit to the plant will occur during year 16 The total direct LOM sustaining costs of the co-placement waste rock and dry stack tailings storage facility and the Sur deposit crushing station are US$216.5M and US$25.0M, respectively.

21.2.10 Exclusions

The following costs and scope are excluded from the capital cost estimate:

- Land acquisitions;
- Taxes not listed in the financial analysis;
- Sales taxes;
- Scope changes and project schedule changes and the associated costs;
- Any facilities/structures not mentioned in the project summary description;
- Geotechnical unknowns/risks;
- Residual value of temporary equipment and facilities;
- Environmental approvals;
- This study or any future project studies, including environmental impact studies;
- Senior finance charges;
- Special incentives (schedule, safety or others);
- No allowance has been made for loss of productivity and/or disruption due to religious union, social and/or cultural activities;
- Escalation costs; and
- Owner's foreign exchange exposure.

21.3 Operating Costs

21.3.1 Introduction

Operating costs for the project consist of those related to mining, processing of mineralized material, comingled waste rock and dry stack tailings storage, maintenance, power and general administration activities. The operating cost estimate is presented in Q2 2024 US dollars. Table 21-12 provides a summary of the project operating costs.

Table 21-11: Operating Cost Summary

Cost Area	Life-of-Mine Cost (US$M)	Unit Cost (US$/t milled)
Mining	6,317.5	5.37
Process	5,847.2	4.97
G&A	531.9	0.45
Co-Mingle Facility	115.8	0.10
Total	**12,812.2**	**10.89**

Note: Totals may not sum due to rounding.

Table 21-12: Operating Cost Summary based on Produced Copper

Area	Unit	US$	Unit	US$ /lb Cu
On-site Costs	-	-	-	-
Mining	$/t milled	5.37	lb Cu	0.79
Processing	$/t milled	4.97	lb Cu	0.73
Co-mingle Tailings	$/t milled	0.10	lb Cu	0.01
General & Administration	$/t milled	0.45	lb Cu	0.07
Sub-total Site Costs	$/t milled	**10.89**	lb Cu	**1.60**
Off-site Costs	-	-	-	-
Concentrate Transport	$/t dry concentrate	172.5	lb Cu	0.30
Smelting & Refining	$/t dry concentrate	155.1	lb Cu	0.27
Sub-total Off-site Costs	$/t dry concentrate	327.6	lb Cu	**0.57**
Total Cost On/Off Site	-	-	lb Cu	**2.17**
Credits (Gold, Silver)	-	-	lb Cu	**(0.35)**
Total Cost			**lb Cu**	**1.82**

Note: Table shown as in press release

21.3.2 Basis of Estimate

Common to all operating cost estimates are the following assumptions:

- Cost estimates are based on Q1 2024 pricing without allowances for inflation;
- Majority of the labour requirement is assumed to come from neighbouring municipalities;
- Equipment and materials will be purchased as new;
- Grinding media consumption rates have been estimated based on the material characteristics as described in Section 13;
- Reagent consumption rates are based on metallurgical testwork results and in-house benchmark;
- The annual power costs were calculated using a unit price of US$0.072/kWh; and
- The fuel price provided was USD$1.35/L delivered to the site.

21.3.3 Mine Operating Costs

The operating costs were estimated from first principles with vendor quotations for repair and maintenance costs and other suppliers for consumables. Key inputs to the mine cost were fuel and labour. The mine fleet will be primarily diesel powered except for the loading shovels, which will be electric powered. The dewatering pumps will also be diesel powered.

21.3.3.1 Mine Labour

Labour costs for the various job classifications were obtained from recent project work in Peru and other operations. A burden rate of 40% was applied to expatriate positions and 150% applied to the local rates. Labour was estimated for both staff and hourly on a 12-hour shift basis using a rotation of two weeks on/two weeks off. Mine positions and salaries are shown in Table 21-13.

Table 21-13: Mine Staffing Requirements and Annul Employee salaries (year 5)

Position	Employees	Annual Salary (US$/a)
Mine Maintenance		
Maintenance Superintendent	1	239,500
Maintenance General Foreman	1	164,500
Maintenance Shift Foremen	6	118,400
Maintenance Planner/Contract Administration	4	65,800
Clerk	2	23,000
Subtotal	**14**	
Mine Operations		
Mine Operations/Technical Superintendent	1	257,900
Mine General Foreman	1	164,500
Senior Shift Foreman	4	118,400
Junior Shift Foreman	8	98,700
Trainers	1	118,400
Road Crew/Services Foreman	1	118,400
Clerk	2	23,000
Subtotal	**18**	
Mine Engineering		
Chief Engineer	1	164,500
Senior Engineer	1	131,600
Open Pit Planning Engineer	2	118,400
Geotechnical Engineer	1	118,400
Blasting Engineer	1	118,400
Blasting/Geotechnical Technician	2	65,800
Dispatch Technician	6	65,800
Surveyor/Mining Technician	2	65,800
Surveyor/Mining Technician Helper	2	52,600

Position	Employees	Annual Salary (US$/a)
Clerk	1	23,000
Subtotal	**19**	
Geology		
Chief Geologist	1	164,500
Senior Geologist	1	131,600
Grade Control Geologist/Modeller	3	118,400
Sampling/Geology Technician	4	52,600
Clerk	1	23,000
Subtotal	**10**	-
Total	**61**	-

The mine staff labour remains constant from Year 1 until the end of the mine life. During the pre-production period there are two trainers, and in Year 3 that drops to one trainer position in mine operations. Hourly employee labour force levels in mine operations and maintenance fluctuate with production requirements. The Year 5 hourly labour requirements are shown in Table 21-14.

Table 21-14: Hourly Manpower Requirements and Annual Salaries (Year 5)

Position	Employees	Annual Salary (US$/a)
Mine General		
General Equipment Operator	12	60,000
Road/Pump Crew	4	48,700
General Mine Labourer	8	45,400
Light Duty Mechanic	6	73,000
Tire Technician	8	73,000
Lube Truck Driver	8	50,300
Subtotal	**46**	-
Mine Operations		
Driller	20	64,900
Blaster	-	79,500
Blast Helper	-	48,700
Loader Operator	4	79,500
Hydraulic Shovel Operator	20	79,500
Haul Truck Driver	156	56,800
Dozer Operator	17	60,000
Grader Operator	6	60,000
Crusher Loader Operator	4	73,000
Water Truck	11	48,700
Subtotal	**238**	-
Mine Maintenance		
Heavy Duty Mechanics	59	73,000

Position	Employees	Annual Salary (US$/a)
Welder	30	73,000
Electrician	5	73,000
Apprentice	9	56,800
Subtotal	**103**	-
Total Hourly	**387**	-

Labour costs are based on an Owner-operated scenario, with Alta Copper responsible for the maintenance of the equipment with its own employees.

A Mine Operations Superintendent will oversee all the mine operations, maintenance, engineering, and geology functions. This person would have the Mine General Foreman and Maintenance Superintendent reporting to them, as well as the Chief Engineer and Chief Geologist.

The Mine General Foreman would have the Shift Foremen report directly to them.

The mine will have four mine operations crews, each with a Senior Shift Foremen who will have two Junior Shift Foreman reporting to them. Over the mine life, there will also be a Road Crew/Services Foreman responsible for roads, drainage, and pumping around the mine. This person would also be a backup Senior Mine Shift Foreman. The Mine Operations department will have two Clerks.

The Chief Engineer will have one Senior Engineer and two Open Pit Engineers reporting to them. The Blasting Engineer would be included in the Short-Range Planning Group and would double as Drill-And-Blast Foreman as required. The Geotechnical Engineer would cover all aspects of the wall slopes and WRSFs, together with shared technicians in blasting.

The Short-Range Planning Group in Engineering will have two Surveyor/Mine Technicians and two Surveyors/Mine Helpers. These employees will assist in the field with staking, surveying, and sample collection with the geology group; they will have a Clerk/Secretary to assist the team.

In the Geology Department, there will be one Senior Geologist reporting to the Chief Geologist. There will also be three Grade Control Geologists/Modellers; one will be in short range, one in grade control drilling, and the other will be in long range/reserves. There will also be four Grade Control/Sampling Technicians and one Clerk/Secretary.

Six Mine Maintenance Shift Foremen will report to the Maintenance General Foreman who in turn will report to the Maintenance Superintendent. There will be four Maintenance Planners/Contract Administrators and two Clerks.

The hourly labour force includes positions for the Light Duty Mechanic, Tire Men, and Lube Truck Drivers. These positions will all report to the Maintenance Department. There will generally be one of each position per crew. Other general labour includes General Mine Labourers (two per crew) and Trainees (one per crew) plus two Road/Pump Crew personnel per crew for water management.

The drilling labour force is based on one operator per drill, per crew. This peaks at 32 Drillers in Year 8 and maintains that level until Year 10 and then drops down over time as the drilling hours are diminished.

Shovel and Loader Operators raise to 32 in Year 7 and peak at 44 in Year 18 when the number starts to taper off. Haulage Truck Drivers peak at 228 in Year 18 and then tapers off to the end of the mine life.

Maintenance factors are used to determine the number of Heavy-Duty Mechanics, Welders and Electricians are required and are based on the number of equipment operators. Heavy Duty Mechanic requirements work out to 0.25 mechanics required for each Drill Operator for example. Welders are 0.25 per operator and Electricians are 0.05 per operator.

The number of Loader, Truck and Support Equipment Operators is estimated using the projected equipment operating hours. The maximum number of operators is four per unit, to match the mine crews.

21.3.3.2 Equipment Operating Costs

Vendors provided repair and maintenance costs for each piece of equipment selected for the project. Fuel consumption rates were estimated from the supplied information and knowledge of the working conditions. The costs for the repair and maintenance area are expressed in $/h form.

Tire costs were also collected from various vendors for the sizes expected to be used. Estimates of tire life are based on AGP's experience. The operating cost of the tires is expressed in a $/hr form. The life of the haulage truck tires is estimated at 5,000 hours per tire for the 221 t trucks with proper rotation from front to back. Each truck tire for the 221-t truck costs US$57,600, resulting in a cost per hour for tires of US$69.14/hr.

Ground engaging tools costing is estimated from other projects and is an area that would be fine-tuned once the project was operational.

Drill consumables are estimated as a complete drill string using the parts list and component lives provided by the vendor. Drill productivity is estimated at 26.4 m/h for mill feed and waste. The equipment costs used in the estimate are shown in Table 21-15.

Table 21-15: Major Equipment Operating Costs – No Labour (US$/hr)

Equipment	Fuel/ Power	Lube/ Oil	Tires/Undercarriage	Repair & Maintenance	Ground Engaging Tools / Consumables	Total
Production drill (160 mm)	67.50	6.75	-	68.56	73.04	215.85
Production drill (270 mm)	148.50	14.85	6.00	100.88	108.40	378.63
Production excavator (6.7 m^3)	90.45	9.05	20.00	59.61	20.00	199.11
Production loader (33 m^3)	195.75	29.36	60.55	255.90	68.60	610.16
Electric hydraulic shovel (38 m^3)	136.51	-	144.80	349.34	88.10	718.75
Haulage truck (40 t)	25.25	2.52	3.00	12.00	1.00	43.77
Haulage truck (221 t)	243.00	24.30	69.14	108.56	6.00	451.00
Haulage trucks (290 t)	264.60	26.46	50.40	209.20	5.00	555.66

Equipment	Fuel/ Power	Lube/ Oil	Tires/Undercarriage	Repair & Maintenance	Ground Engaging Tools / Consumables	Total
Crusher loader (20 m^3)	255.15	38.27	60.55	166.27	71.00	591.24
Track dozer (455 kw)	75.60	7.56	26.20	63.12	12.10	184.58
Grader (146 kW)	19.85	1.98	3.75	22.99	6.00	54.57

21.3.3.3 Drilling

Drilling in the open pit will use down the hole hammers for the 160 mm drill and rotary for the 270 mm drill rig. The pre-production drilling will be with the smaller drill in small working areas until sufficient bench width is available for the larger drill. The pattern size is constant for both mill feed and waste. The material will be smaller and finer to improve productivity and reduce maintenance costs as well as improve plant performance. The drilling pattern parameters are shown in Table 21-16.

Table 21-16: Drill Pattern Specifications

Specification	Unit	160-mm Drill		270-mm Drill	
		Mill Feed	Waste	Mill Feed	Waste
Bench height	m	15	15	15	15
Sub-drill	m	1.0	1.0	1.6	1.6
Blasthole diameter	mm	160	160	270	270
Pattern spacing - staggered	m	5.8	5.8	9.1	9.1
Pattern burden – staggered	m	5.0	5.0	7.9	7.9
Hole depth	m	16.0	16.0	16.6	16.6

The parameters used to estimate drill productivity are shown in Table 21-17.

Table 21-17: Drill Productivity Criteria

		160-mm Drill		270-mm Drill	
Drill Activity	Unit	Mill Feed	Waste	Mill Feed	Waste
Pure penetration rate	m/min	0.55	0.55	0.50	0.50
Hole depth	m	16	16	16.6	16.6
Drill time	min	29.09	29.09	33.20	33.20
Move, spot and collar hole	min	3.00	3.00	3.00	3.00
Level drill	min	0.50	0.50	0.50	0.50
Add steel	min	0.50	0.50	0.00	0.00
Pull drill rods	min	1.50	1.50	1.00	1.00
Total setup/breakdown time	min	5.50	5.50	4.50	4.50

		160-mm Drill		270-mm Drill	
Total drill time per hole	min	34.6	34.6	37.7	37.7
Drill productivity	m/hr	27.8	27.8	26.4	26.4

21.3.3.4 Blasting

A heavy ANFO product (60% ANFO, 40% emulsion) will be used. The powder factors used are shown in Table 21-18.

Table 21-18: Design Powder Factors

	Unit	160-mm Drill		270-mm Drill	
		Mill Feed	Waste	Mill Feed	Waste
Powder Factor	kg/m^3	0.68	0.68	0.70	0.70
Powder Factor	kg/t	0.27	0.27	0.28	0.28

The blasting cost was estimated using quotations from a local explosive vendor. The explosives price was US$120.00/100 kg. The explosives vendor will provide an all-in load and shot service for the mine. The total monthly cost was US$250,000 per month.

21.3.3.5 Loading

Loading costs for both mill feed, and waste are based on the use of hydraulic shovels and front-end loaders. The shovels will be the primary diggers with the front-end loader as backup/support units. The average percentage of each material type that the various loading units are responsible for is shown in Table 21-19 at Year 5. This highlights the focus of the shovels over the loader.

Table 21-19: Loading Parameters – Year 5

Loading Parameters	Unit	Hydraulic Shovel		Front End Loader	
Bucket capacity	m^3	38	38	33	33
Truck capacity loaded	t	227	290	227	290
Waste tonnage loaded	%	75	75	25	25
Mill feed tonnage loaded	%	75	75	25	25
Bucket fill factor	%	86	86	97	97
Cycle time	sec	33	33	40	40
Trucks present at loading unit	%	80	80	80	80
Loading time	min	2.35	2.90	2.70	2.70

The trucks present at the loading unit refers to the percentage of time a truck is available to be loaded. To maximize truck productivity and reduce operating costs, it is more efficient to slightly under-truck the loading unit. One of the largest operating cost items is haulage and minimizing this cost by maximizing the truck productivity is crucial to lower operating costs. The value of 80% comes from the standby time shovels typically encounter due to a lack of trucks and the loader has a slightly lower availability of trucks due to its backup role.

21.3.3.6 Hauling

Haulage profiles were determined for each pit phase to the primary crushers, and to the co-mingle facility during pre-production. Cycle times were generated by period by destination and phase to estimate truck requirements and resulting haulage costs. Maximum speed on the trucks is limited to 50 km/hr for tire life and safety considerations. Travel speeds for various segments are shown in Table 21-20.

Table 21-20: Haulage Cycle Speeds

Haulage	Flat (0%) On Surface	Flat (0%) In pit, Crusher, Dump	Slope Up (8%)	Slope Up (10%)	Slope Down (8%)	Slope Down (10%)
Loaded (km/hr)	40	20	16	15	15	15
Empty (km/hr)	40	20	35	20	35	35

21.3.3.7 Support Equipment

Support equipment hours and costs are determined on factors applied to various major pieces of equipment. For the PEA, some of the factors used are shown in Table 21-21.

Table 21-21: Support Equipment Operating Factors

Mine Equipment	Factor	Factor Units
Track dozer	25%	Of haulage hours to maximum of 5 dozers
Grader	15%	Of haulage hours to maximum of 3 graders
Crusher loader	40%	Of loading hours to maximum of 1 loader
Water truck	10%	Of haulage hours to maximum of 2 trucks
Pit support backhoe	10%	Of loading hours to maximum of 1 backhoe
Road crew backhoe	4	hours/day/unit
Road crew dump truck	4	hours/day/unit
Road crew loader	4	hours/day/unit
Lube/fuel truck	6	hours/day/unit
Mechanics truck	14	hours/day/unit
Integrated tool carrier	4	hours/day/unit
Light plants	12	hours/day/unit
Pickup trucks	10	hours/day/unit

These factors resulted in the need for seven track dozers, three graders, and one crusher loader. Their tasks will include clean-up of the loader faces, roads, CPSF, and blast patterns. The graders will maintain the haul roads. In addition, water trucks will have the responsibility for patrolling the haul roads and controlling fugitive dust for safety and environmental reasons. The small backhoe and road crew dump trucks will be responsible for cleaning out sedimentation ponds and ditch maintenance.

The hours estimated in this manner were applied to the individual operating costs for each piece of equipment. Many of these units will be support equipment, so no direct labour is allocated to them due to their variable function. The operators will come from the General Equipment operator pool.

21.3.3.8 Grade Control

Grade control will be performed using blasthole assays. The nature of the deposit and gradational contacts noted did not warrant a separate reverse circulation program for grade control.

The assay cost for this has been included in the laboratory cost of the plant and is not included as a mine operating cost.

21.3.3.9 Dewatering

Pit dewatering will be an important part of the mining process. Water will be diverted around the pit where possible but with the size of the pit, significant volumes will need to be pumped. Initial dewatering requirements are estimated at 4,200 m^3/d until Year 2, when they will increase to 10,900 m^3/d. In Year 11 the rate reduces to 9,800 m^3/d. This level is maintained for the remainder of the mine life.

The dewatering is planned to be completed with a set of four pumps in the pit and two pumps on the surface. These pumps will be diesel powered.

Additional dewatering in the form of horizontal drill holes is included as part of the dewatering costs; 1,200 m of horizontal drain drilling per years has been costed.

Dewatering is expected to cost US$26.3 M over the proposed mine life.

21.3.3.10 Financing

Financing of the mine fleet is considered a viable option to reduce initial capital. Various vendors offer this as an option to help select their equipment. Both Caterpillar and Komatsu have the ability, and desire, to allow financing of their product lines.

Indicative terms for financing provided by the vendors are:

- Down payment = 20% of equipment cost;
- Term length = 3-5 years (depending on equipment);
- Interest rate = 7.60%; and
- Residual = $0.

The initial capital, down payments, and annual leasing costs were included in Section 21.2.2.

The support equipment fleet is calculated in the same manner as the major mining equipment.

All of the major mine equipment, and the majority of the support equipment, where it was considered reasonable, was assumed to be financed. If the equipment had a life greater than the finance term length, then the following years onward of the term did not have a payment applied. In the case of the mine trucks, with an approximate 10-year working life, the finance period would be complete, and the trucks would simply incur operating costs after that time. For this reason, the operating cost would vary annually depending on the equipment replacement schedule and timing of the financing packages for each piece of equipment.

Using the financing option adds US$0.25/t to the mine operating cost over the LOM.

21.3.3.11 Total Mine Costs

The total LOM operating costs per tonne of material moved and per tonne of mill feed processed are shown in Table 21-22 and Table 21-23.

Table 21-22: Open Pit Mine Operating Costs – with Financing (US$/t Mined)

Open Pit Category	Unit	Year 1	Year 5	Year 10	LOM Average
General Mine and Engineering	US$/t mined	0.09	0.11	0.09	0.10
Drilling	US$/t mined	0.14	0.13	0.13	0.13
Blasting	US$/t mined	0.41	0.41	0.41	0.42
Loading	US$/t mined	0.23	0.22	0.24	0.23
Hauling	US$/t mined	0.88	0.99	1.01	1.09
Support	US$/t mined	0.16	0.17	0.14	0.16
Financing costs	US$/t mined	0.51	0.34	0.09	0.25
Dewatering	US$/t mined	0.01	0.03	0.03	0.03
Total	**US$/t mined**	**2.43**	**2.41**	**2.14**	**2.38**

Table 21-23: Open Pit Mine Operating Costs – with Financing (US$/t Mill Feed)

Open Pit Category	Unit	Year 1	Year 5	Year 10	LOM Average
General Mine and Engineering	US$/t mill feed	0.27	0.22	0.22	0.22
Drilling	US$/t mill feed	0.42	0.27	0.32	0.29
Blasting	US$/t mill feed	1.23	0.86	1.03	0.93
Loading	US$/t mill feed	0.68	0.46	0.61	0.52
Hauling	US$/t mill feed	2.63	2.04	2.53	2.43
Support	US$/t mill feed	0.48	0.36	0.36	0.35
Financing costs	US$/t mill feed	1.52	0.69	0.22	0.56
Dewatering	US$/t mill feed	0.04	0.08	0.08	0.06
Total	**US$/t mill feed**	**7.29**	**4.97**	**5.37**	**5.37**

21.3.4 Process Operating Costs

The operating costs are estimated from benchmarks on available operational data for equivalent copper operations. The overall life of mine processing operating cost is US$5,847.2 M. The average annual operating costs for pre-expansion and post-expansion phases are shown in Table 21-24.

Table 21-24: Process Operating Cost Summary

Cost Area	LOM Cost (US$M)	LOM Annual Cost (US$M/a)	LOM Unit Cost (US$/t milled)
Power	2,656.1	97.4	2.26
Labour	279.3	10.0	0.23
Reagents	1,252.1	46.0	1.06
Grinding Media and Consumables	1,054.1	38.7	0.89
Maintenance, including Vehicle & Mobile Equipment	605.5	22.2	0.51
Processing Subtotal	**5,847.2**	**214.2**	**4.97**
General & Administration	532.0	19.0	0.45
Process and G&A Total	**6,374.2**	**233.2**	**5.42**

Note: Totals may not sum due to rounding.

21.3.4.1 Reagents

Reagent usage was estimated based on an interpretation of the metallurgical testwork well as benchmarked usage from comparable operations. Reagent costs were based on internal data, which is developed from vendor quotations for other projects. The major reagent costs and their addition rates are presented in during a production year of 43.8 Mt in Table 21-25.

Table 21-25: Reagents Costs

Reagent	Reagent Unit Cost US$/t	Annual Consumption (t/a)	LOM Average Annual Cost (USM$/a)	LOM Unit Cost (US$/t Milled)
Quicklime	120	70,080	8.3	0.19
Collector 1	3,968	832	3.3	0.08
Frother	3,616	2,847	10.2	0.23
Flocculant (Concentrate)	3,265	657	2.1	0.05
Flocculant (Tailings)	3,265	876	2.8	0.07
Antiscalant	2,509	219	0.5	0.01
Biocide	1,759	88	0.2	0.00
Calcium Carbonate	120	187,332	18.5	0.42
Reagents Total	-	-	**46.0**	**1.06**

Note: Totals may not sum due to rounding.

21.3.4.2 Consumables

The consumables considered in this cost summary are crusher and mill liners and mill grinding media. Costs of liners and grinding media were estimated based on vendor information in Ausenco's database. The consumption of liners was estimated based on vendor information and benchmarking of similar plants. Grinding media consumption rates were estimated from grinding consumption rates from the abrasion index (Ai) and the expected average mill power draws. The details during a 43.8 Mt production year are shown in Table 21-26 and Table 21-27.

Table 21-26: Grinding Media Costs

Area	Annual Consumption t/a	Annual Cost (US$M/a)	Unit Cost (US$/t Milled)
SAG Mill Media	8,366	11.7	0.27
Ball Mill Media	7,796	8.6	0.20
Regrind Media	1,162	4.7	0.11
Lime Slaker Media	21	0.04	0.00
Subtotal	**-**	**25.1**	**0.58**

Note: Totals may not sum due to rounding.

Table 21-27: Consumables Costs

Area	Annual Cost (US$M/a)	Unit Cost (US$/t Milled)
Crushing and conveying	1.1	0.03
Concentrator	12.4	0.28
Reagents	0.02	0.00
Plant Services	0.01	0.00
Tailings Management	0.05	0.00
Crushed Waste Transport System	0.05	0.00
Subtotal	**13.7**	**0.31**

Note: Totals may not sum due to rounding.

21.3.4.3 Labour

Labour costs include all the processing and maintenance staff in Table 21-28.

Table 21-28: Labour Costs

Cost Centre	Number	Annual Cost (US$M/a)	Unit Cost (US$/t Milled)
Management	8	0.7	0.02
Technical Services	25	1.0	0.02
Operations	104	2.8	0.06

Cost Centre	Number	Annual Cost (US$M/a)	Unit Cost (US$/t Milled)
Maintenance	98	3.3	0.07
Laboratory	25	0.7	0.02
Contract Maintenance Labour	-	1.5	0.04
Subtotal	**260**	**10.0**	**0.23**

Note: Totals may not sum due to rounding.

Processing production labour was developed using benchmarks from similar projects and are grouped into 6 categories:

- General Management, including the plant manager, alongside the superintendents for maintenance, mill operations, mechanical, CPSF and training supervisors and clerks;
- Technical services, metallurgists, engineers, and the assay and met team;
- Operations, operators through the whole plant, including crushing, grinding, flotation, regrind, thickening, CPSF, and concentrate loading along with their associated shift supervisors;
- Maintenance, maintenance crew including electricians, boilermakers, mechanics, technicians and their associated supervisor and foreman;
- Laboratory, featuring chemists, met and lab technicians and their associated shift supervisors; and
- Contract Maintenance Labour, brought in for major maintenance activities such as mill relines and crusher mantle/concave changeouts.

Each position was defined and classified as salary and wages. Costs included taxes and benefits. The total estimated labour force for plant operations and plant maintenance was estimated at 260 people. The estimate was based on providing a labour force to support continuous operations at 24 h/d, 365 d/a. The total process plant labour operating cost including management is US$10.0M/a.

21.3.4.4 Power

The power cost is calculated from the estimated power draw determined from the preliminary mechanical equipment list plus an allowance for other auxiliary services. The total installed power is estimated at 262 MW with an estimated average power draw of 207 MW and an annual consumption of 1,447,606 MWh. Electricity will be provided to site at a unit cost of US$0.072/kWh. The LOM power cost is US$2,656.1M.

21.3.4.5 Maintenance Consumables

Annual maintenance spares and consumables costs were developed through a percentage factor from the total direct cost of each WBS area. The factor was applied to mechanical equipment, platework and piping plus an allowance for freight charges. The percentage factors ranged between 3.0% to 5.0% depending on the area, for a total weighted average of 4.6%. The maintenance consumables are presented in Table 21-29.

Table 21-29: Maintenance Consumables Cost Summary Breakdown

WBS	Area	% Installed Direct Cost	Annual Cost (US$/a)	Unit Cost (US$/t Milled)
2110	Primary Crushing (total)	5.0	1,222,760	0.03
2210	Stockpile Feed Conveyor	4.0	2,643,947	0.06
2200	Stockpile and Reclaim	4.0	243,929	0.01
2300	Grinding	5.0	4,811,737	0.11
2130	Pebble Crushing	4.0	742,620	0.02
2400	Flotation	4.0	989,418	0.02
2400	Regrind	4.0	873,811	0.02
2510	Concentrate Thickening	4.0	60,825	0.00
2530	Concentrate Filtration and Load Out	4.0	311,909	0.01
2610	Tailings Thickener	4.0	208,936	0.00
2620	Tailings Filtration	5.0	2,885,522	0.07
2700	Reagents	4.5	49,073	0.00
2800	Plant Services	3.0	124,124	0.00
3110	Waste Rock and Tailings Management	5.0	4,200,550	0.10
3120	Crushed Waste Transport System	4.0	2,020,320	0.05
-	**Total**	**4.6**	**21,389,480**	**0.49**

Note: Totals may not sum due to rounding.

21.3.4.6 Vehicles and Mobile Equipment Estimate

Operating costs of the surface vehicle fleet for the plant were estimated through predicted fuel and spares usage throughout production. The total costs accumulated to US$0.8M/a or US$0.02/t milled.

21.3.5 General and Administrative Operating Costs

General and administrative costs are expenses not directly related to the operation of the process plant but required to support safe and effective operation of the facility and satisfy legislative requirements in some cases. These costs were developed using Ausenco's in-house data on existing operations, and include costs such as the following:

- Human resources, including the consultancy, training, community relation, communication, and community development;
- Site Administration, including the office supplies and maintenance for the administrative offices;
- Health and safety, including the necessary first aid and hospital services and onsite PPE supply;
- Environmental, including the monitoring and analysis of soil, water, noise, air, and biodiversity around the project area. It also includes CPSF monitoring and acid rock drainage (ARD) monitoring and sampling costs;
- IT and telecommunications, including hardware, software, and support services;
- Contract services, insurance, and legal, including site wide insurance, license and legal fees and sitewide waste management, camp and travel costs; and
- Administrative costs, general office and auditing expenses throughout the year.

The annual G&A cost is estimated at US$19.0M. Table 21-30 shows the breakdown of the G&A costs.

Table 21-30: G&A Costs Breakdown

G&A Area	Annual Cost US$M/a	Cost per Tonne US$/t Milled
Human Resources	1.9	0.04
Site Admin, Maintenance	0.1	0.00
Health & Safety	0.5	0.01
Environmental	2.3	0.05
IT & Telecommunications	1.2	0.03
Contract Services, Insurance, Legal	9.9	0.23
Administrative Costs	0.2	0.00
G&A Labor	2.8	0.06
Total Costs	**19.0**	**0.45**

Note: Totals may not sum due to rounding.

21.3.6 Co-Placement Storage Facility Operating Costs

The CPSF operational costs are derived from the operation of the necessary compactors, graders, dozers, and water trucks for the CPSF. The mobile equipment costs, fuel costs and operator wages were calculated. The annualized cost of the CPSF accumulate to US$4.1M/y for a total of USD$115.8 M in the LOM.

22 ECONOMIC ANALYSIS

22.1 Forward-Looking Information Cautionary Statements

The results of the economic analyses discussed in this Section represent forward-looking information as defined under Canadian securities law. The results depend on inputs that are subject to known and unknown risks, uncertainties, and other factors that may cause actual results to differ materially from those presented here. Information that is forward looking includes the following:

- Mineral resource estimate.
- Assumed commodity prices and exchange rates.
- The proposed mine production plan.
- Projected mining and process recovery rates.
- Assumptions as to mining dilution and ability to mine in areas previously exploited using mining methods as envisaged, and the timing and amount of estimated future production.
- Sustaining costs and proposed operating costs.
- Assumptions as to closure costs and closure requirements.
- Assumptions as to environmental, permitting, and social risks.

Additional risks to the forward-looking information include:

- Changes to costs of production from what is assumed.
- Unrecognized environmental risks.
- Unanticipated reclamation expenses.
- Unexpected variations in quantity of mineralized material, grade, or recovery rates.
- Accidents, labour disputes and other risks of the mining industry.
- Geotechnical or hydrogeological considerations during mining being different from what was assumed.
- Failure of mining methods to operate as anticipated.
- Failure of plant, equipment, or processes to operate as anticipated.
- Changes to assumptions as to the availability of electrical power, and the power rates used in the operating cost estimates and financial analysis.
- Ability to maintain the social license to operate.
- Changes to interest rates.
- Changes to tax rates.

22.2 Methodologies Used

The project has been evaluated using a discounted cash flow (DCF) analysis based on an 8% discount rate. Cash inflows consist of annual revenue projections. Cash outflows consist of capital expenditures, including pre-production costs, operating costs, taxes, and royalties. These are subtracted from the inflows to arrive at the annual cash flow projections.

Cash flows are taken to occur at the mid-point of each period. It must be noted that tax calculations involve complex variables that can only be accurately determined during operations and, as such, the actual post-tax results may differ from those estimated. A sensitivity analysis was performed to assess the impact of variations in metal prices, discount rate, head grade, recovery, total operating cost, and total capital costs.

The capital and operating cost estimates developed specifically for this project are presented in Section 21. The economic analysis has been run on a constant dollar basis with no inflation.

22.3 Financial Model Parameters

The economic analysis was performed assuming the copper price of US$4.00/lb, gold price of US$1,850/oz, and silver price of US$23.00/oz; these prices are based on historic metal pricing, consensus analyst estimates and recently published economic studies as discussed in Section 19. The forecasts used are meant to reflect the average metals price expectation over the life of the Project. No price inflation or escalation factors were taken into account. Commodity prices can be volatile, and there is the potential for deviation from the forecast.

The economic analysis also used the following assumptions:

- Total operating mine life of 27.3 years following construction.
- Cost estimates in constant Q2 2024 US$ with no inflation or escalation factors considered.
- Results based on 100% ownership with a 0.5% NSR royalty applicable to the entire resource.
- Capital cost funded with 100% equity (no financing cost assumed).
- All cash flows discounted to start of construction period using mid-period discounting convention.
- All metal products are sold in the same year they are produced.
- Project revenue is derived from the sale of copper concentrate with gold and silver credits payable.
- No contractual arrangements currently exist for offtake of copper concentrate.

22.3.1 Taxes

The project has been evaluated on a post-tax basis to provide an approximate value of the potential economics. The tax model calculations are based on the tax regime as of the date of the PEA technical report. At the effective date of this report, the Project is assumed to be subject to the Peruvian Corporate Income Tax rate of 29.5% plus 2% during the term of the assumed Stability Agreement, a mandated Employee Profit Sharing rate of 8% up to a maximum of 150% of salaries paid, a Mining Retirement Fund, a variable Special Mining Tax rate between 2% and 8.4% based on operating profit and a government Mining Royalty rate between 1% and 12% also based on operating profit.

Depreciation on capital equipment, development, and exploration cost as permitted by Peru tax regulations has been applied. The tax schedule calculated over the life of the Project result in a total tax and government royalty payable of US$5,292 M.

22.4 Economic Analysis

The economic analysis was performed assuming an 8% discount rate. On a pre-tax basis, the NPV discounted at 8% is US$4,169.20M; the internal rate of return (IRR) is 32.4%, and the payback period is 2.6 years. On a post-tax basis, the NPV discounted at 8% is US$2,345.9M; the IRR is 24.1%, and the payback period is 3.1 years. A summary of project economics is tabulated in Table 22-1. The analysis was done on an annual cashflow basis; the cashflow output is shown in Table 22-2 and cashflow is represented graphically in Figure 22-1 on a post-tax basis.

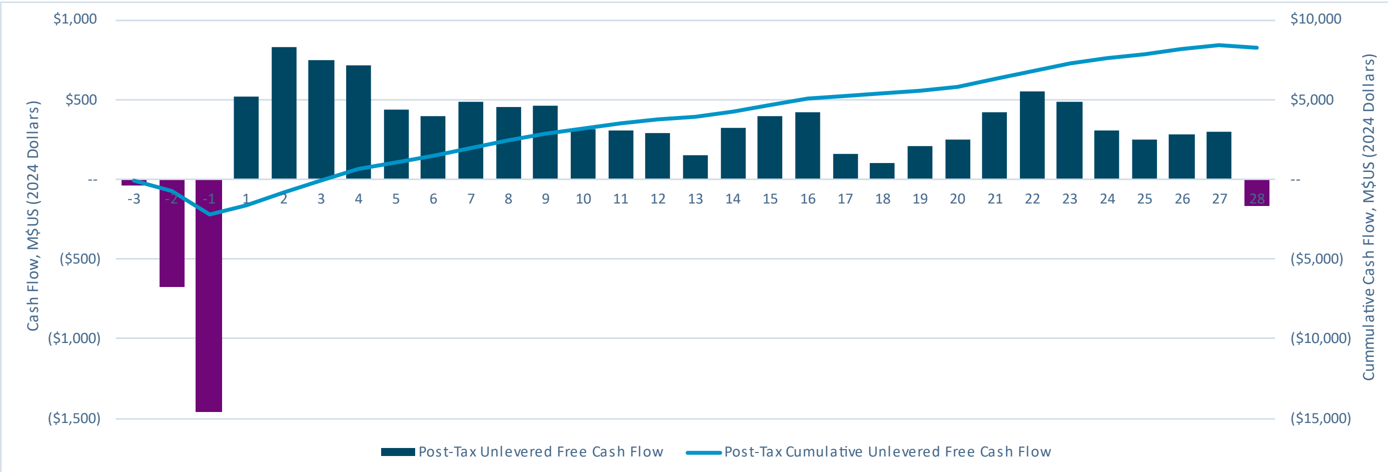
Readers are cautioned that the PEA is preliminary in nature. It includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the PEA will be realized.

Table 22-1: Economic Analysis Summary Table

General	Units	LOM Total / Avg.	
Copper Price	US$/lb	4.00	
Gold Price	US$/oz	1,850	
Silver Price	US$/oz	23.00	
Mine Life	Years	27.3	
Total Mineralized Material Processed	Mt	1,176	
Total Waste	Mt	1,563	
Avg. CuEq Head Grade	%	0.41	
Production	**Units**	**LOM Total / Avg.**	
Avg. Head Grade – Cu	%	0.35	
Avg. Head Grade – Au	g/t	0.07	
Avg. Head Grade – Ag	g/t	1.59	
Avg. Recovery Rate – Cu	%	88.2	
Avg. Recovery Rate – Au	%	63.3	
Avg. Recovery Rate – Ag	%	55.3	
Total Payable Metal – Cu	M lbs	7,717	
Annual Payable Metal – Cu	M lbs/a	283	
Total Payable Metal – Au	k oz	1,538	
Annual Payable Metal – Au	k oz/a	56.3	
Total Payable Metal – Ag	k oz	29,867	
Annual Payable Metal – Ag	k oz/a	1,094	
Operating Costs	**Units**	**LOM Total / Avg.**	
Mining Cost	US$/t mined	2.38	
Mining Cost	US$/t milled	5.37	
Processing Cost	US$/t milled	4.97	
Co-mingle Tailings Cost	US$/t milled	0.11	
G&A Cost	US$/t milled	0.45	
Operating Cash Costs*, by-product basis	US$/lb Cu	1.20	
Total Cash Costs**, by-product basis	US$/lb Cu	1.82	
All-in Sustaining Costs (AISC)***, by-product basis	US$/lb Cu	1.91	
Capital Costs	**Units**	**LOM Total / Avg.**	
Initial Capital (Incl. Capitalized Opex)	US$M	2,160	
Sustaining Capital	US$M	518	
Closure Costs	US$M	216	
Financials	**Units**	**Pre-Tax**	**Post-Tax**
NPV (Discounted at 8%)	US$M	4,169.2	2,345.9
IRR	%	32.4	24.1
Payback	Years	2.6	3.1

*Operating cash costs consist of mining costs, processing costs, co-mingle tailings operating costs, and G&A. **Total cash costs consist of operating cash costs plus transportation cost, off-site treatment & refining, transport costs, and non-governmental royalties. ***AISC consist of total cash costs plus sustaining capital, and closure cost.

Figure 22-1: Free Cash Flow – Post Tax



Source: Ausenco, 2024.

Table 22-2: Cashflow Statement on an Annualized Basis

Dollar figures in Real 2024 US$M unless otherwise noted	Unit	Year	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27	28
Copper Price	US$/lb	4.00	4.00	4.00	4.00	4.00	4.00	4.00	4.00	4.00	4.00	4.00	4.00	4.00	4.00	4.00	4.00	4.00	4.00	4.00	4.00	4.00	4.00	4.00	4.00	4.00	4.00	4.00	4.00	4.00	4.00	4.00	4.00
Gold Price	US$/oz	1,850	1,850	1,850	1,850	1,850	1,850	1,850	1,850	1,850	1,850	1,850	1,850	1,850	1,850	1,850	1,850	1,850	1,850	1,850	1,850	1,850	1,850	1,850	1,850	1,850	1,850	1,850	1,850	1,850	1,850	1,850	1,850
Silver Price	US$/oz	23.00	23.00	23.00	23.00	23.00	23.00	23.00	23.00	23.00	23.00	23.00	23.00	23.00	23.00	23.00	23.00	23.00	23.00	23.00	23.00	23.00	23.00	23.00	23.00	23.00	23.00	23.00	23.00	23.00	23.00	23.00	23.00
Revenue	US$M	34,403	--	--	--	1,295	1,836	1,681	1,634	1,446	1,339	1,487	1,462	1,443	1,211	1,121	1,125	972	1,298	1,531	1,582	865	836	996	1,085	1,279	1,500	1,407	1,042	898	937	911	183
Operating Cost	US$M	(12,812)	--	--	--	(463)	(515)	(465)	(445)	(460)	(504)	(499)	(511)	(509)	(477)	(459)	(505)	(547)	(576)	(606)	(573)	(489)	(530)	(501)	(466)	(420)	(401)	(421)	(403)	(337)	(325)	(303)	(104)
Off-Site Costs	US$M	(4,587)	--	--	--	(185)	(247)	(217)	(225)	(197)	(182)	(202)	(222)	(184)	(162)	(168)	(154)	(158)	(184)	(199)	(200)	(113)	(94)	(149)	(157)	(183)	(203)	(167)	(119)	(98)	(101)	(99)	(19)
Royalties	US$M	(149)	--	--	--	(6)	(8)	(7)	(7)	(6)	(6)	(6)	(6)	(6)	(5)	(5)	(5)	(4)	(6)	(7)	(7)	(4)	(4)	(4)	(5)	(5)	(6)	(6)	(5)	(4)	(4)	(4)	(1)
EBITDA	US$M	16,854	--	--	--	642	1,065	991	958	783	647	781	723	744	567	489	461	263	533	720	802	260	208	342	458	671	889	812	515	459	508	505	59
Initial Capex	US$M	(2,160)	(32)	(674)	(1,454)	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Sustaining Capex	US$M	(518)	--	--	--	(42)	(37)	(64)	(1)	(65)	(20)	(1)	(3)	(1)	(40)	(0)	(5)	(14)	(16)	(58)	(98)	(1)	(9)	(1)	(33)	(0)	(3)	(1)	(1)	(1)	(3)	(0)	(0)
Closure Capex	US$M	(216)	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	(216)
Changes in Working Capital	US$M	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Pre-Tax Unlevered Free Cash Flow	**US$M**	**13,960**	**(32)**	**(674)**	**(1,454)**	**600**	**1,029**	**927**	**956**	**719**	**627**	**780**	**720**	**743**	**527**	**489**	**455**	**249**	**518**	**661**	**705**	**259**	**199**	**341**	**425**	**671**	**886**	**811**	**514**	**458**	**505**	**504**	**(157)**
Income Tax, Mining Tax & Mining Royalties	US$M	(5,434)	--	--	--	(79)	(194)	(177)	(242)	(283)	(232)	(288)	(265)	(277)	(210)	(180)	(167)	(94)	(193)	(260)	(279)	(88)	(73)	(118)	(157)	(236)	(318)	(291)	(182)	(167)	(191)	(191)	(4)
Post-Tax Unlevered Free Cash Flow	**US$M**	**8,525**	**(32)**	**(674)**	**(1,454)**	**520**	**835**	**749**	**715**	**435**	**395**	**492**	**455**	**466**	**317**	**309**	**288**	**155**	**325**	**401**	**426**	**171**	**127**	**223**	**267**	**435**	**568**	**520**	**333**	**291**	**314**	**313**	**(160)**
Production																																	
Total Material Mined	**kt**	**2,739,161**	**686**	**35,577**	**43,925**	**98,347**	**107,164**	**89,925**	**82,125**	**90,339**	**105,728**	**117,276**	**119,688**	**119,924**	**109,884**	**109,717**	**116,273**	**124,650**	**125,629**	**128,110**	**119,417**	**109,249**	**130,000**	**117,559**	**97,349**	**80,000**	**80,000**	**84,455**	**78,130**	**54,022**	**42,868**	**18,061**	**3,085**
Strip Ratio	w:o	1.33	--	--	--	1.12	1.37	0.93	0.66	0.95	1.75	1.31	1.45	1.62	2.21	1.98	2.00	2.20	1.71	1.39	1.83	6.00	1.97	1.68	1.22	0.83	0.33	0.39	0.40	0.35	0.13	0.04	0.09
Project Life	yrs	27.3	--	--	--	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	0.3
Mill Feed	kt	1,176,207	--	--	--	32,850	43,800	43,800	43,800	43,800	43,800	43,800	43,800	43,800	43,800	43,800	43,800	43,800	43,800	43,800	43,800	36,355	43,800	43,800	43,800	43,800	43,800	43,800	43,800	43,800	43,800	43,800	12,002
Mill Head Grade (Cu)	%	0.35%	--	--	--	0.45%	0.49%	0.45%	0.44%	0.39%	0.37%	0.41%	0.41%	0.41%	0.33%	0.31%	0.32%	0.28%	0.36%	0.41%	0.45%	0.33%	0.22%	0.28%	0.30%	0.36%	0.41%	0.38%	0.28%	0.24%	0.24%	0.23%	0.17%
Mill Recovery (Cu)	%	88.24%	--	--	--	90.9%	90.4%	89.6%	90.0%	89.9%	86.6%	88.9%	89.3%	87.2%	89.4%	87.6%	84.4%	87.1%	89.7%	90.4%	86.9%	78.2%	87.4%	86.7%	88.3%	88.9%	89.9%	88.2%	86.0%	85.9%	88.9%	89.4%	88.7%
Recovered Copper	mlbs	8,026	--	--	--	293.10	427.28	388.89	382.15	339.39	311.44	348.93	349.52	341.90	286.80	264.58	261.81	233.21	310.69	361.94	376.60	204.03	187.28	233.05	258.32	305.32	354.48	326.17	236.55	197.52	204.92	201.14	39.01
Mill Head Grade (Au)	g/t	0.07	--	--	--	0.10	0.09	0.09	0.08	0.07	0.08	0.07	0.06	0.07	0.06	0.05	0.06	0.04	0.05	0.07	0.07	0.06	0.07	0.05	0.04	0.05	0.07	0.08	0.08	0.09	0.09	0.08	0.07
Mill Recovery (Au)	%	63.33%	--	--	--	72.4%	70.2%	70.0%	68.5%	65.5%	62.6%	68.3%	62.2%	60.2%	63.7%	65.7%	61.1%	59.0%	64.6%	69.5%	62.4%	50.2%	58.3%	60.6%	65.2%	64.5%	64.9%	63.7%	58.2%	56.1%	58.9%	60.2%	60.0%
Recovered Gold	koz	1,674	--	--	--	76.62	91.50	90.35	75.90	63.71	67.21	69.77	52.93	57.72	49.35	46.13	54.82	32.67	46.50	63.85	63.08	37.68	57.84	45.39	40.38	48.07	63.74	75.36	66.83	71.52	77.03	71.29	17.23
Mill Head Grade (Ag)	g/t	1.59	--	--	--	2.61	2.00	1.70	2.03	1.98	1.61	1.54	1.89	1.90	1.54	1.38	1.52	1.38	1.59	1.72	1.77	1.50	1.19	1.40	1.41	1.57	1.70	1.50	1.30	1.22	1.27	1.09	1.00
Mill Recovery (Ag)	%	55.30%	--	--	--	65.5%	62.9%	63.8%	59.9%	56.0%	56.6%	59.4%	53.6%	54.0%	53.2%	60.9%	55.2%	51.5%	56.2%	61.0%	54.8%	44.8%	46.0%	55.0%	58.5%	54.9%	56.2%	54.4%	46.3%	42.1%	45.5%	46.5%	46.6%
Recovered Silver	koz	33,219	--	--	--	1,808.11	1,770.00	1,526.35	1,716.07	1,560.09	1,281.83	1,290.36	1,428.25	1,445.13	1,151.96	1,181.21	1,180.42	997.01	1,260.08	1,477.06	1,369.78	784.42	767.99	1,082.85	1,165.98	1,214.56	1,340.30	1,147.54	844.58	720.93	810.34	714.62	180.70
Wet Concentrate Produced	kt	15,387	--	--	--	562	819	746	733	651	597	669	670	655	550	507	502	447	596	694	722	391	359	447	495	585	680	625	453	379	393	386	75
Payable Copper	mlbs	7,717	--	--	--	282	411	374	367	326	299	336	336	329	276	254	252	224	299	348	362	196	180	224	248	294	341	314	227	190	197	193	38
Payable Gold	koz	1,538	--	--	--	70.49	84.18	83.13	69.83	58.62	61.84	64.19	47.64	53.10	45.40	42.44	50.44	29.40	41.85	58.74	56.77	34.67	53.80	41.76	36.34	43.26	58.64	69.33	62.15	66.51	71.63	66.30	16.02
Payable Silver	koz	29,897	--	--	--	1,627	1,593	1,374	1,544	1,404	1,154	1,161	1,285	1,301	1,037	1,063	1,062	897	1,134	1,329	1,233	706	691	975	1,049	1,093	1,206	1,033	760	649	729	643	163
Revenue - Copper	US$M	$30,869	--	--	--	1,127	1,643	1,496	1,470	1,305	1,198	1,342	1,344	1,315	1,103	1,018	1,007	897	1,195	1,392	1,448	785	720	896	994	1,174	$1,363	1,254	910	760	788	774	150
Revenue - Gold	US$M	$2,846	--	--	--	130	156	154	129	108	114	119	88	98	84	79	93	54	77	109	105	64	100	77	67	80	$108	128	115	123	133	123	30
Revenue - Silver	US$M	$688	--	--	--	37	37	32	36	32	27	27	30	30	24	24	24	21	26	31	28	16	16	22	24	25	$28	24	17	15	17	15	4
Total Revenue	**US$M**	**$34,403**	**--**	**--**	**--**	**1,295**	**1,836**	**1,681**	**1,634**	**1,446**	**1,339**	**1,487**	**1,462**	**1,443**	**1,211**	**1,121**	**1,125**	**972**	**1,298**	**1,531**	**1,582**	**865**	**836**	**996**	**1,085**	**1,279**	**$1,500**	**1,407**	**1,042**	**898**	**937**	**911**	**183**
Total Offsite Costs	**US$M**	**$4,587**	**--**	**--**	**--**	**185**	**247**	**217**	**225**	**197**	**182**	**202**	**222**	**184**	**162**	**168**	**154**	**158**	**184**	**199**	**200**	**113**	**94**	**149**	**157**	**183**	**$203**	**167**	**119**	**98**	**101**	**99**	**19**
Operating Costs																																	
Total Operating Costs	**US$M**	**12,812**	**--**	**--**	**--**	**463**	**515**	**465**	**445**	**460**	**504**	**499**	**511**	**509**	**477**	**459**	**505**	**547**	**576**	**606**	**573**	**489**	**530**	**501**	**466**	**420**	**401**	**421**	**403**	**337**	**325**	**303**	**104**
Mine Operating Costs	US$M	6,317	--	--	--	239	274	223	203	218	262	257	269	267	235	217	263	306	334	365	331	287	294	265	230	184	165	186	167	101	89	67	20
Mill Processing Costs	US$M	5,847	--	--	--	200	219	219	219	219	219	219	219	219	219	219	219	219	219	219	219	181	213	213	213	213	213	213	213	213	213	213	61
G&A Costs	US$M	532	--	--	--	19	19	19	19	19	19	19	19	19	19	19	19	19	19	19	19	16	19	19	19	19	19	19	19	19	19	19	19
Co-mingle Facility Mobile Equipment Operating Costs	US$M	115.79	--	--	--	4.10	4.26	4.42	4.42	4.42	4.42	4.42	4.42	4.42	4.26	4.26	4.10	4.10	4.10	4.10	3.94	3.94	3.94	4.10	4.10	4.10	4.10	4.10	3.94	3.94	3.94	3.94	3.47
Total Cash Cost *	**US$/lb Cu**	**1.82**	**--**	**--**	**--**	**1.72**	**1.41**	**1.35**	**1.39**	**1.60**	**1.84**	**1.67**	**1.85**	**1.74**	**1.95**	**2.08**	**2.17**	**2.83**	**2.22**	**1.93**	**1.78**	**2.67**	**2.84**	**2.47**	**2.16**	**1.71**	**1.39**	**1.41**	**1.73**	**1.58**	**1.42**	**1.39**	**2.41**
All-in Sustaining Costs **	US$/lb Cu	1.91	--	--	--	1.87	1.50	1.52	1.40	1.80	1.91	1.68	1.86	1.74	2.09	2.08	2.19	2.89	2.27	2.10	2.05	2.68	2.89	2.48	2.29	1.71	1.40	1.41	1.74	1.59	1.44	1.39	8.18
Capital Expenditures																																	
Total Initial Capital	**US$M**	**2,160**	**32**	**674**	**1,454**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**
Total Sustaining Capital	**US$M**	**518**	**--**	**--**	**--**	**42**	**37**	**64**	**1**	**65**	**20**	**1**	**3**	**1**	**40**	**0**	**5**	**14**	**16**	**58**	**98**	**1**	**9**	**1**	**33**	**0**	**3**	**1**	**1**	**1**	**3**	**0**	**0**
Closure Cost	**US$M**	**216**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**216**

* Dollar figures in Real 2024 US$M unless otherwise noted.

22.5 Sensitivity Analysis

A sensitivity analysis was conducted on the base case pre-tax and post-tax $NPV_{8\%}$ and IRR of the Project using the following variables: metal price, discount rate, total operating cost, and initial capital cost. Table 22-3 shows a summary of the pre-tax sensitivity results, Table 22-4 shows a summary of the post-tax sensitivity results.

As shown in Figure 22-2 and Figure 22-3, the sensitivity analysis revealed that the Project is most sensitive to commodity price, head grade, and operating cost and less sensitive to initial capital cost.

Table 22-3: Pre-Tax Sensitivity Summary

Pre-Tax NPV Sensitivity To Discount Rate (Commodity Price (%))

Discount Rate	(20%)	(10%)	0%	10%	20%
$4,169					
3.0%	4,124	6,345	8,565	10,786	13,007
5.0%	2,902	4,624	6,346	8,068	9,790
8.0%	1,711	2,940	4,169	5,398	6,627
10.0%	1,184	2,191	3,197	4,204	5,210
12.0%	793	1,632	2,470	3,308	4,147

Pre-Tax IRR Sensitivity To Discount Rate (Commodity Price (%))

Discount Rate	(20%)	(10%)	0%	10%	20%
32.4%					
3.0%	19.3%	26.1%	32.4%	38.3%	43.9%
5.0%	19.3%	26.1%	32.4%	38.3%	43.9%
8.0%	19.3%	26.1%	32.4%	38.3%	43.9%
10.0%	19.3%	26.1%	32.4%	38.3%	43.9%
12.0%	19.3%	26.1%	32.4%	38.3%	43.9%

Pre-Tax NPV Sensitivity To Operating Cost (Commodity Price (%))

Operating Cost	(20%)	(10%)	0%	10%	20%
$4,169					
(20.0%)	2,587	3,816	5,045	6,274	7,503
(10.0%)	2,149	3,378	4,607	5,836	7,065
--	1,711	2,940	4,169	5,398	6,627
10.0%	1,273	2,502	3,731	4,961	6,190
20.0%	836	2,065	3,294	4,523	5,752

Pre-Tax IRR Sensitivity To Operating Cost (Commodity Price (%))

Operating Cost	(20%)	(10%)	0%	10%	20%
32.4%					
(20.0%)	23.8%	30.2%	36.2%	41.9%	47.3%
(10.0%)	21.6%	28.2%	34.3%	40.1%	45.6%
--	19.3%	26.1%	32.4%	38.3%	43.9%
10.0%	16.8%	24.0%	30.4%	36.5%	42.2%
20.0%	14.1%	21.7%	28.4%	34.6%	40.4%

Pre-Tax NPV Sensitivity To Initial Capital Cost (Commodity Price (%))

Capital Cost	(20%)	(10%)	0%	10%	20%
$4,169					
(20.0%)	2,126	3,355	4,584	5,813	7,042
(10.0%)	1,919	3,148	4,377	5,606	6,835
--	1,711	2,940	4,169	5,398	6,627
10.0%	1,504	2,733	3,962	5,191	6,420
20.0%	1,296	2,525	3,754	4,983	6,212

Pre-Tax IRR Sensitivity To Initial Capital Cost (Commodity Price (%))

Capital Cost	(20%)	(10%)	0%	10%	20%
32.4%					
(20.0%)	25.1%	33.1%	40.4%	47.3%	53.8%
(10.0%)	21.9%	29.3%	36.0%	42.4%	48.4%
--	19.3%	26.1%	32.4%	38.3%	43.9%
10.0%	17.1%	23.5%	29.4%	34.9%	40.1%
20.0%	15.3%	21.3%	26.8%	32.0%	36.9%

Pre-Tax NPV Sensitivity To Mill Head Grade (Commodity Price (%))

Mill Head Grade	(20%)	(10%)	0%	10%	20%
$4,169					
(20.0%)	79	1,062	2,045	3,028	4,011
(10.0%)	895	2,001	3,107	4,213	5,319
--	1,711	2,940	4,169	5,398	6,627
10.0%	2,527	3,879	5,231	6,583	7,935
20.0%	3,344	4,819	6,293	7,768	9,243

Pre-Tax IRR Sensitivity To Mill Head Grade (Commodity Price (%))

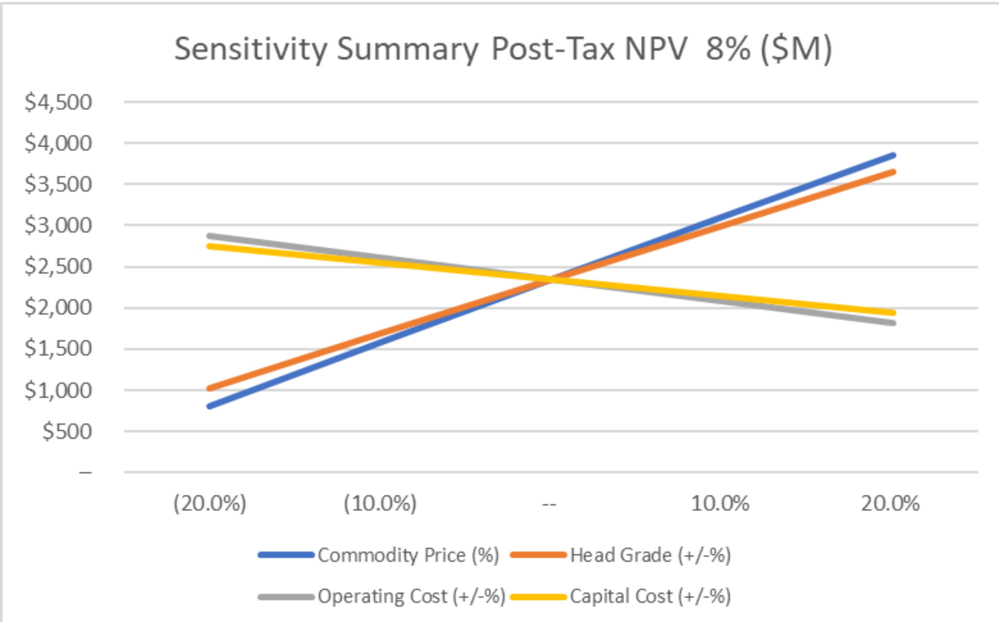
Mill Head Grade	(20%)	(10%)	0%	10%	20%
32.4%					
(20.0%)	8.6%	15.4%	21.2%	26.6%	31.6%
(10.0%)	14.3%	21.0%	27.0%	32.6%	37.9%
--	19.3%	26.1%	32.4%	38.3%	43.9%
10.0%	23.9%	31.0%	37.5%	43.7%	49.6%
20.0%	28.2%	35.6%	42.4%	48.9%	55.1%

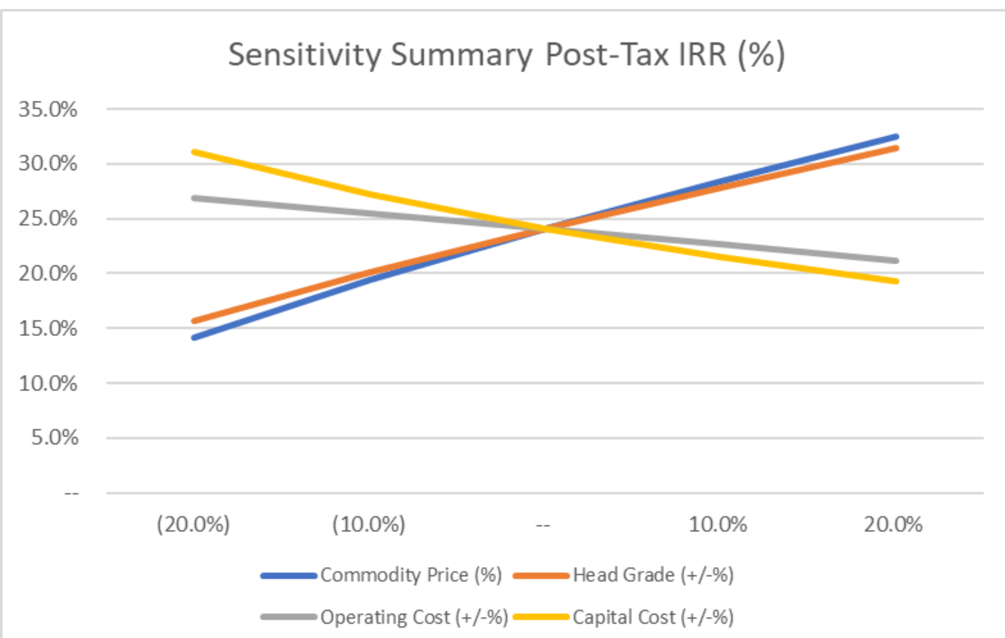
Table 22-4: Post-Tax Sensitivity Summary

Post-Tax NPV Sensitivity To Discount Rate						
		Commodity Price (%)				
Discount Rate	$2,346	(20%)	(10%)	0%	10%	20%
	3.0%	2,347	3,743	5,128	6,505	7,871
	5.0%	1,568	2,652	3,725	4,791	5,848
	8.0%	803	1,579	2,346	3,106	3,858
	10.0%	462	1,100	1,728	2,351	2,967
	12.0%	209	742	1,266	1,785	2,298
Post-Tax NPV Sensitivity To Operating Cost						
		Commodity Price (%)				
Operating Cost	$2,346	(20%)	(10%)	0%	10%	20%
	(20.0%)	1,349	2,113	2,871	3,623	4,371
	(10.0%)	1,080	1,849	2,610	3,366	4,118
	--	803	1,579	2,346	3,106	3,858
	10.0%	519	1,306	2,079	2,842	3,599
	20.0%	231	1,030	1,809	2,577	3,337
Post-Tax NPV Sensitivity To Initial Capital Cost						
		Commodity Price (%)				
Capital Cost	$2,346	(20%)	(10%)	0%	10%	20%
	(20.0%)	1,210	1,982	2,747	3,505	4,258
	(10.0%)	1,007	1,781	2,547	3,305	4,059
	--	803	1,579	2,346	3,106	3,858
	10.0%	600	1,377	2,145	2,904	3,660
	20.0%	396	1,175	1,945	2,705	3,459
Post-Tax NPV Sensitivity To Mill Head Grade						
		Commodity Price (%)				
Mill Head Grade	$2,346	(20%)	(10%)	0%	10%	20%
	(20.0%)	-272	382	1,016	1,634	2,248
	(10.0%)	274	988	1,684	2,373	3,057
	--	803	1,579	2,346	3,106	3,858
	10.0%	1,320	2,166	3,002	3,832	4,660
	20.0%	1,832	2,748	3,654	4,558	5,453

Post-Tax IRR Sensitivity To Discount Rate						
		Commodity Price (%)				
Discount Rate	24.1%	(20%)	(10%)	0%	10%	20%
	3.0%	14.2%	19.4%	24.1%	28.4%	32.5%
	5.0%	14.2%	19.4%	24.1%	28.4%	32.5%
	8.0%	14.2%	19.4%	24.1%	28.4%	32.5%
	10.0%	14.2%	19.4%	24.1%	28.4%	32.5%
	12.0%	14.2%	19.4%	24.1%	28.4%	32.5%
Post-Tax IRR Sensitivity To Operating Cost						
		Commodity Price (%)				
Opex	24.1%	(20%)	(10%)	0%	10%	20%
	(20.0%)	17.7%	22.4%	26.8%	30.9%	34.9%
	(10.0%)	16.0%	21.0%	25.5%	29.7%	33.7%
	--	14.2%	19.4%	24.1%	28.4%	32.5%
	10.0%	12.2%	17.7%	22.6%	27.1%	31.3%
	20.0%	10.0%	16.0%	21.1%	25.7%	30.0%
Post-Tax IRR Sensitivity To Initial Capital Cost						
		Commodity Price (%)				
Capex	24.1%	(20%)	(10%)	0%	10%	20%
	(20.0%)	19.5%	25.5%	31.0%	36.1%	40.9%
	(10.0%)	16.6%	22.2%	27.2%	31.9%	36.3%
	--	14.2%	19.4%	24.1%	28.4%	32.5%
	10.0%	12.2%	17.1%	21.5%	25.5%	29.3%
	20.0%	10.6%	15.2%	19.3%	23.1%	26.6%
Post-Tax IRR Sensitivity To Mill Head Grade						
		Commodity Price (%)				
Mill Head Grade	24.1%	(20%)	(10%)	0%	10%	20%
	(20.0%)	5.6%	11.1%	15.7%	19.8%	23.5%
	(10.0%)	10.2%	15.5%	20.1%	24.2%	28.2%
	--	14.2%	19.4%	24.1%	28.4%	32.5%
	10.0%	17.7%	23.0%	27.8%	32.4%	36.7%
	20.0%	21.0%	26.4%	31.4%	36.2%	40.6%

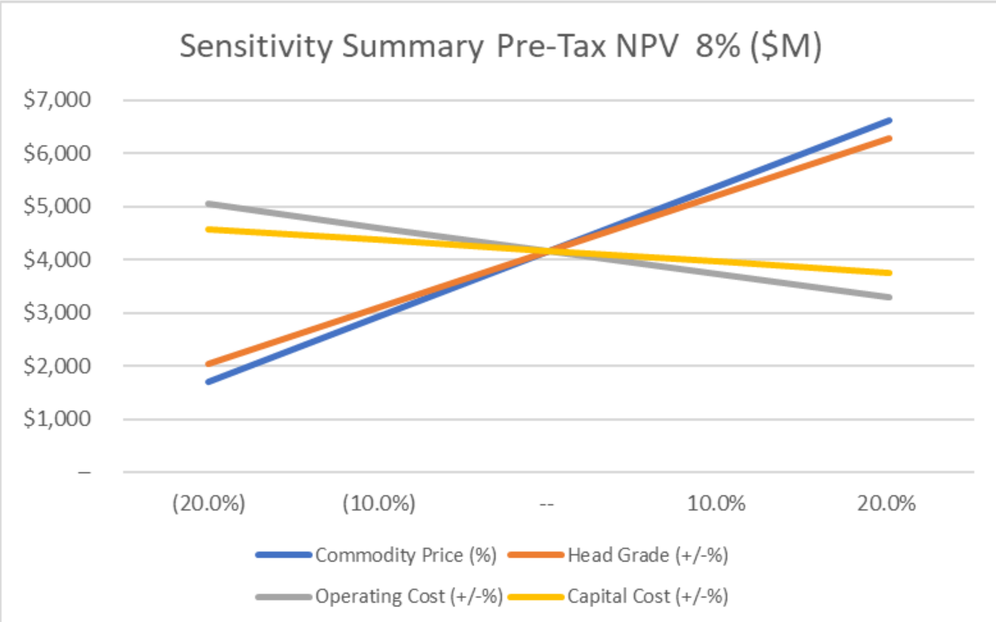
Figure 22-2: Post-Tax NPV and IRR Sensitivity Analysis

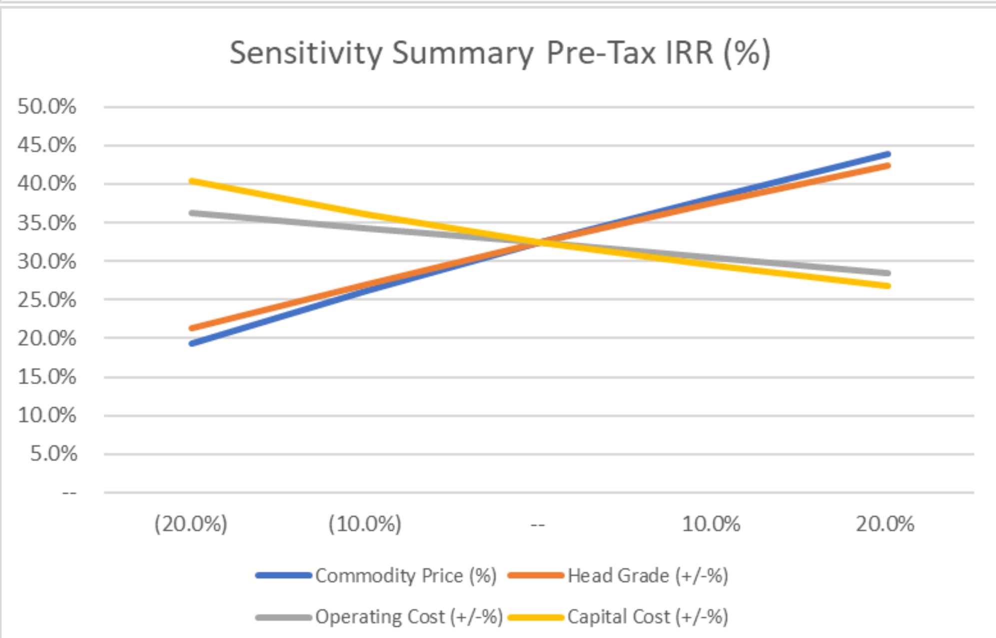




Source: Ausenco, 2024.

Figure 22-3: Pre-Tax NPV and IRR Sensitivity Results





Source: Ausenco, 2024.

23 ADJACENT PROPERTIES

This section is not relevant to this technical report.

24 OTHER RELEVANT DATA AND INFORMATION

This section is not relevant to this technical report.

25 INTERPRETATION AND CONCLUSIONS

25.1 Introduction

The QPs note the following interpretations and conclusions in their respective areas of expertise, based on the review of data available for this Report.

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

The Cañariaco Project consists of 15 mining concessions totaling approximately 9,725.12 ha. Alta Copper, through its Peruvian subsidiaries Cañariaco Copper Peru S.A (CCPSA) and Cobriza Metals Peru S.A. ("CZA"), holds a 100% interest in the Cañariaco Project. Details are included in Table 4-1. Peruvian mining legislation does not require location of concession boundaries on the ground. To maintain the concessions, payments of $3.00 per hectare must be paid when the minimum production or investment (exploration) has not been met by the 10th year since, starting on the 11th year. The 11th year started in 2009, and penalties have been paid annually since that date to stake the concessions. The mining concessions will also require agreements with the registered landowner to obtain access rights and register all future transactions and contracts to the Public Mining Registry. Alta Copper will need to obtain authorization from the National Water Authority (ANA) to use water for exploration/or mining purposes, including for domestic and industrial use.

The project is subject to a 0.5% royalty payable to Anglo Pacific, a third party on the Cañariaco A, B, C, D and F1 concessions. Profit based taxes are due to the Government of Peru. It is expected that the project will incur a net profits interest (NPI) royalty.

25.3 Geology and Mineralization

The Cañariaco Norte and Cañariaco Sur deposits are considered to be examples of porphyry-copper systems. The mineralization style and setting are well understood and can support declaration of Mineral Resources.

The geological understanding of the settings, lithologies, and structural and alteration controls on mineralization in the Cañariaco Norte deposit is sufficient to support estimation of Mineral Resources. The geological knowledge of the area is also considered sufficiently acceptable to reliably inform conceptual mine planning. The knowledge of the Cañariaco Sur deposit is at an earlier stage, however an Inferred Mineral Resource estimate has been completed and it has been included in this PEA.

Exploration potential remains within the Cañariaco Sur deposit and Quebrada Verde prospect. Cañariaco Norte is still open at depth in at the central–western portion of the deposit.

25.4 Exploration, Drilling and Analytical Data Collection in Support of Mineral Resource Estimation

Prior to Alta Copper's involvement, exploration was conducted by INGEMMET, Placer Dome, and Billiton, from 1967 to 2000. Work completed by these companies included stream sediment sampling, geological mapping, rock chip and grab sampling, trenching and pitting, IP, resistivity, and ground magnetic geophysical surveys, petrographic studies, core drilling, mineral resource estimation, and very preliminary leach testwork.

Alta Copper acquired 100% ownership of the Project in February 2002. Since that date, Alta Copper has completed geological mapping, prospecting, ground magnetic, resistivity and magnetic geophysical surveys, rock chip sampling, petrographic studies, bulk sampling for metallurgical testing, re-logging and re-sampling of historic drill core, core drilling, mineral resource estimation, metallurgical testwork and mining studies.

The exploration programs discovered the Cañariaco Norte and Cañariaco Sur copper deposits, and the Quebrada Verde prospect.

The exploration programs completed to date are appropriate for the style of the deposits on the Project.

Drilling on the Project consists of 289 core holes (85,183.16 m), including geotechnical, metallurgical, and hydrogeological drilling.

Core was logged for geological and geotechnical parameters. Drill collar locations were picked up by a surveyor, using a total station instrument. Down-hole surveys were performed using either a Pajari, Sperry Sun, or Reflex EZ-Shot instrument.

Drill core generated by INGEMMET and Placer Dome was halved; there is no information as to the typical sample intervals. The Billiton and Alta Copper drill core was halved and sampled on 2 m intervals.

A total of 9,424 bulk density readings were taken by Alta Copper personnel on core, and an additional 550 specific gravity determinations were performed by ALS Chemex.

Several primary assay laboratories were used during the legacy campaigns for sample preparation and analysis. INGEMMET used Plenge and an internal INGEMMENT laboratory. Placer Dome used SGS, and Billiton used ALS Chemex. INGEMMET samples were analyzed for copper and molybdenum, and more rarely gold and silver, using a colorimetric analytical method. Billiton samples assayed for gold (fire assay with AA finish, 10 ppb detection limit) and copper, lead, zinc, molybdenum, and arsenic (multi-acid, total digest), with an AA finish for each element. SGS completed check assays on a split of one in 20 pulps using the same analytical procedures as the initial analysis performed by ALS Chemex.

Actlabs performed all of the sample preparation and the majority of the analyses for the Alta Copper programs. ICP analyses were performed by Actlabs. Some analyses for the re-analysis of pre-2008 core samples for gold and ICP were undertaken by ALS Chemex. ACME was used as a check laboratory for pulp analyses.

Each Alta Copper sample was subject to total copper and sequential copper leaching analysis, using a three-acid digest and AA finish. Gold used an aqua regia digest, with a fire assay and AA finish. Depending on the sample, a 36-element suite was analysed by Actlabs using ICP-OES, or a 33element suite was analysed by ALS Chemex using an ICP–AES method.

There is no information on QA/QC programs for INGEMMET and Placer Dome. Billiton used blanks, standards, and check assays. The Alta Copper QA/QC program used field, pulp and coarse reject duplicates, blanks, and standards.

All data in the field were recorded in written form in field books, logbooks, sample sheets, logging forms or shipping forms. All field data was hand-entered into Excel tables. Data from third parties such as laboratories or survey contractors were generally supplied in digital and printed form. All data was verified by Alta Copper personnel.

Drill data collected from the INGEMMET, Placer Dome, and Billiton campaigns were re-logged by Alta Copper personnel, and nine of the drill holes have been re-assayed. Based on the good correlation between the historical grades and the Alta Copper re-assay grades, all of the historical data have been inserted into the final database.

Three pairs of twinned holes were drilled by Alta Copper to verify grade uniformity at short distances. In general, similar average grades were noted over the same depth intervals.

Sampling methods are acceptable for Mineral Resource estimation.

Sample preparation, analysis and security were generally performed in accordance with exploration best practices and industry standards. Sample preparation for samples that support Mineral Resource estimation has followed a similar procedure since 2004.

The quantity and quality of the lithological, geotechnical, collar and down-hole survey data collected during the exploration and delineation drilling programs are sufficient to support Mineral Resource estimation. The collected sample data adequately reflect deposit dimensions, true widths of mineralization, and the style of the deposits. Sampling is representative of the copper, gold, silver, and molybdenum grades in the deposits, reflecting areas of higher and lower grades.

The QA/QC programs completed by Alta Copper adequately address issues of precision, accuracy and contamination. Drilling programs typically included blanks, duplicates, and CRM samples. QA/QC submission rates meet industry-accepted standards.

The collected data were subject to validation by built-in program triggers that automatically checked data on upload to the database. Verification was performed on all digitally collected data on upload to the main database, and includes checks on surveys, collar coordinates, lithology data, and assay data. The checks are appropriate and consistent with industry standards. Sample security relied upon the fact that the samples were always attended or locked in the on-site sample preparation facility. Chain-of-custody procedures consisted of filling out sample submittal forms that were sent to the laboratory with sample shipments ensure that all samples were received by the laboratory.

A number of data verification programs and audits were performed over the Project history, primarily in support of technical reports. No errors or omissions were noted during these reviews.

The QP reviewed the findings of the external data review programs to confirm that no significant issues were found with the databases or data collected at the time. The QP was a member of the AMEC team verifying data and information in 2010. The QP is satisfied that the data are suitable to support mineral resource estimation.

The QP's data verification at Cañariaco Sur consisted of comparison of the assay database with original assay certificates, verification of drillhole collar positions in the field and verification of the geological models by inspection of outcrops and drill core.

The QP concludes that the data collected from the Project adequately support the geological interpretations and constitute a database of sufficient quality to support the use of the data in Mineral Resource estimation.

25.5 Metallurgical Testwork

Four major phases of testwork were conducted. The first three phases focused on the Norte Deposit. The first consisted of process development to define the type of processing most applicable to the mineralization. The second phase followed with more detailed work to optimize process conditions. The third phase focused on mill feed variability testing to improve the geometallurgical understanding of the deposit and robustness in the proposed process flowsheet established in the previous studies. The fourth phase of testing assessed comminution and metal recoveries for the Sur deposit.

Testwork included: mineralogy, QEMSCAN examination, comminution and variability comminution tests, flotation tests on the effects of grind size, variable collectors, pulp pH and sulfidization as well as cleaner flotation tests and locked cycle tests.

Metallurgical test work and associated analytical procedures are appropriate to the mineralization type, appropriate to establish the conceptual process routes and were performed using samples that are typical of the mineralization within the proposed mining area.

Samples selected for testing were representative of the various types and styles of mineralization. Samples were selected from a range of depths within the Cañariaco Norte and Cañariaco Sur deposits and featured various types of lithologies present in the deposit. Sufficient samples were taken for tests to be performed on sufficient sample mass for reliable results.

Recovery factors estimated for Cañariaco Norte are based on variability samples tested in open and lock cycle flotation tests conducted under the process criteria and configuration used for the concentrator design. The recoveries of the Sur deposit are based on samples tested using the same process criteria and rougher configuration established for the Norte deposit. No cleaner flotation tests were conducted on Sur material, so the cleaner flotation recovery factor established from the Norte testwork was applied to the Sur results. For a feed grade of 0.35% Cu, copper recovery is expected at 88.2%.

The presence of arsenic exists in the Norte deposit, but with the current flowsheet the LOM arsenic concentration will average slightly over 1% which will be marketable to copper smelters but will incur penalties. The Sur deposit will produce copper concentrate with arsenic contents below the penalty limit of 0.2%. Overall, arsenic content should be controlled to a level where no impact on concentrate marketability is anticipated over the LOM.

25.6 Mineral Resources Estimates

Mineral Resources are reported using the 2014 CIM Definition Standards and are based on open pit mining methods.

Factors that may affect the Mineral Resource estimate include: metal price and exchange rate assumptions; changes to the assumptions used to generate the copper grade cut-off grade; changes in local interpretations of mineralization geometry and continuity of mineralized zones; changes to geological and mineralization shape and geological and grade continuity assumptions; density and domain assignments; changes to geotechnical, mining and metallurgical recovery assumptions; and changes to the input and design parameter assumptions that pertain to the conceptual pit constraining the estimates.

The Mineral Resource Estimate for the Cañariaco deposit includes Measured, Indicated and Inferred resources. The mineral resource estimate is well supported by an extensive data base and comprehensive evaluation. The most recent mineral resource estimate completed in support of this study achieved increased resources due to inclusion of resources at a lower cutoff grade of 0.1% copper to reflect current global copper prices, versus 0.15% copper in the 2022 resource estimate. The current Norte Mineral Resource Estimate includes 73% classified as Measured and Indicated and 27% as Inferred. The high percentage of Measured and Indicated resources at Norte provides a high level of confidence in the resource estimate.

The Cañariaco Sur Mineral Resource Estimate is based on 15 drill holes and includes Inferred resources only. Additional drilling and evaluation is required to provide data to enable reclassification of the Sur mineral resource to Measured and Indicated resources. Drilling to date has encountered copper, gold, silver and molybdenum mineralization in each drill hole, starting at surface and generally extending to the final depth of each hole. Based on the results to date it is anticipated that additional drilling has a high probability of increasing the volume of mineralization and potentially increasing the quantity of the mineral resources in Sur.

25.7 Mine Plan

The 2024 PEA mine plan is based on a subset of the Mineral Resource estimate. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.

A conventional open pit truck and shovel operation is planned. Production forecasts are achievable with the proposed equipment and plant.

The mine plan considers a 120 kt/d processing rate with the peak mining capacity of 130 Mt/a. Two pit areas are mined: Norte and Sur. Norte has been divided into ten phases while Sur has six phases. The mine life will run for 28 years, with the last year being a partial year. Three years of pre-stripping will be required.

Three years of pre-production mining activities are required. Mill feed delivery to the crusher in the first production year is forecast at 32.9 Mt and rises to the full 43.8 Mt (120,000 t/d) in Year 2 onwards. The last year of production, Year 28, will be a partial year with 12 Mt processed.

The mine plan will deliver 1,176.2 Mt to the process plant with grades of 0.35% copper, 0.07 g/t gold, 1.59 g/t silver, and 171.7 ppm arsenic. Waste mined over the life of mine is 1,562.9 Mt for a strip ratio of 1.33:1.

The mine planning has been performed based on marginal cut-offs applied to the NSR grade item. Feed and stockpile material was routed by period, and the cut-off for each period is different due to location of mill feed in the pit, head grades, geometallurgical parameters, and overall mill feed availability. The marginal cut-off is the net value (NV), which is the NSR minus the sum of the mill feed based operating costs, which are processing, G&A, and tailings management. The global average for mill feed operating costs is US$7.26/t. These costs vary by material type, so the cut-off in terms of NV can vary by location and period.

Stockpiling of low-grade material adjacent to the Norte crusher has been considered and will be reclaimed completely at various points in the mine life. Within the ultimate pit the classification breakdown of the mill feed material is 33% Measured, 39% Indicated and 28% Inferred.

The steep topography in the upper elevations of the pit designs has resulted in designs and a mining schedule that has front loaded significant quantities of waste stripping. Refinements from future iterations of road access and phase designs may be able to improve upon the current designs and reduce capitalized stripping costs. Smaller equipment has been considered in the pre-stripping period to prepare the working faces for the larger mining equipment later in the mine life.

25.8 Recovery Methods

The processing plant will be designed to process material at a rate of 120 ktpd. The process plant features the following:

- Mill feed primary gyratory crusher near the Norte pit with a dedicated overland mill feed conveyor to the Mill feed stockpile at the process plant;
- Waste rock mineral sizer near the Norte pit with a dedicated overland waste rock conveyor to the CPSF;
- Primary gyratory crusher near the Sur Norte pit with a dual purpose, mill feed and waste, conveyor to the Norte crushing plant that will be deferred until Sur mining commences in year 17;
- Single SAG mill with pebble crushing and screen classification;
- Two ball mill lines with cyclone classification;
- Rougher flotation followed by secondary regrind with cyclone classification;
- Three stage cleaners followed by cleaner scavenger flotation cells for concentrate production;
- Concentrate thickener and filtration for loadout; and
- Tailing thickener and filtration for dry stack tailings disposal.

The process plant flowsheet designs were based on testwork results, financial evaluations, and industry-standard practices. The flowsheet was developed for optimum value recovery while minimizing capital expenditure and life-of-mine operating costs. The process plant is a conventional copper concentrator, and the process design is typical of a concentrator treating copper sulfide mill feed. The comminution and recovery processes are conventional and well-established in the mining industry with no elements of significant technological innovation.

25.9 Infrastructure

The project infrastructure for the Alta Copper project includes on-site infrastructure and its civil structural and earthworks development associated with, such as:

- Mine facilities, including mining administration offices, a mine fleet truck shop and wash bays, and a mine workshop;
- Process facilities to include crushing, grinding, flotation, reagent mixing and process water distribution along with the assay laboratory and warehouse;
- Overland conveyors to transport the mineralized material and waste materials;
- Tunnel between the Norte Pit and Sur Pit, deferred until Sur deposit development is initiated in year 16;
- Camp and administration buildings along with its supporting infrastructure such as the potable water treatment plant and sewage treatment plant; and
- Co-placement waste and tailing storage facility, designed in accordance to international standards for storage of tailings and waste rock.

Offsite infrastructures are also developed:

- Access roads to reach the mine;
- Raw water supply; and
- Power supply.

The administration and concentrator site was selected on a natural elevated area, close to an existing road and away from watersheds. The mine infrastructure area (MIA) and crusher/ROM pad are close to the mine pit to minimize the hauling distance, and to keep the ROM pad activities away from the administration area.

A site-wide water balance was completed to estimate the quantity of mine site contact water expected to be managed during operations. The process plant water demands 902 m^3/hr which will be made up by contact water and reclaim water from the CPSF.

25.10 Environmental, Permitting and Social Considerations

The Project is located on the surface land of the Community of San Juan de Cañaris. Baseline studies, investigations and field work were carried out in 2021 through 2023 by Yaku Consultants for the development of the semi-detailed Environmental Impact Assessment (EIAsd 2021). Earlier studies supported the development of the 2012 EIAsd prepared by AMEC and approved by Directorial Resolution No. 177-2012-MEM/AAM in May 2012 as well as other permits in 2014.

Completed baseline studies evaluated air quality, noise, hydrography, hydrology and hydrogeology, soils, water quality, ecosystems, flora, and fauna. Environmental baseline studies are well advanced for the Project and will help to form the basis for Environmental Impact Assessment and future permitting applications. There remain a few areas where baseline studies need to be initiated and/or advanced including groundwater studies and modelling, geochemistry studies, and further refinement of the mine water balance. These studies will support future Project design, mine water effluent predictions and effluent treatment requirements.

The Surface Water Management Plan will preserve the "no contact" status of surface waters to the maximum extent practicable. Water will be impounded upstream of the CPSF to supply the freshwater replenishment requirements of the process plant.

The economic analysis in the 2024 PEA estimates a closure cost of US$216M based on typical closure parameters of comparable projects.

A number of permits will be required in support of construction and operations. For the purposes of the 2024 PEA, Alta Copper has identified the critical permits that must be obtained. No permits are currently held for construction or operations activities. Review of the final project footprint and activities is required to ensure that permit requirements are identified for all of the planned operational areas and activities. Sufficient work has been undertaken on the permitting and environmental aspects of the project to gain an understanding of the regulatory requirements that will need to be met to construct, operate, and close the mine.

In 2007, 2010, 2011, 2014, 2021 and 2023 archaeological evaluations were carried out. No archaeological sites of significance were identified in the surveys.

The community public consultation and engagement process for the preparation of the 2012 EIAsd and the 2023 DIA drilling application was carried out in accordance with Peruvian regulatory requirements. Continued socio-economic studies and community engagement efforts will help to identify community needs and provide a basis for targeted community investment in local development projects, training, education, and employment. These activities will help to gain community support that will be required to progress the Project on a timely basis through the regulatory process.

25.11 Markets and Contracts

In support of this PEA, Alta Copper contracted Open Mineral, an independent metal marketing specialist consultant, to carry out an updated assessment of the marketability of the copper concentrate to be produced at Cañariaco Project. The assessment included:

- A comparison of similar concentrates currently in the market and how they are assessed.
- Treatment & Refining Charges (TCRCs): An updated assessment of the likely TCRCs for the concentrate, relative to prevailing market conditions.
- Payable Metal: An overview of the payable metals contained in the concentrate and the various thresholds at which the payable metal may change.
- Penalties: An updated assessment of any penalties incurred on any penalties incurred on deleterious elements (e.g. arsenic) and an overview of the different thresholds.

Based on the Open Mineral study, net metal payables, precious metal credits and penalties for metals in concentrate are summarized as:

- Copper in concentrate is payable at 96.5% subject to a minimum deduction of 1 tonne of copper per 100 tonnes of concentrate.

Gold in concentrate is payable at:

- 0% for Au grade below 1 g/t.
- 90% for Au grade above or equal to 1 g/t and below 3 g/t.
- 92% for Au grade above or equal to 3 g/t and below 5 g/t.
- 93% for Au grade above or equal to 5 g/t.

Silver in concentrate is payable at:

- 0% for Ag grade below or equal to 30 g/t.
- 90% for Au grade above 30 g/t.

Arsenic in concentrate results in a penalty calculated on the following progressive scale:

- US$2.50/t per 0.1% for As grades above 0.2% and less than 0.5%.
- Then, US$5.50/t per 0.1% for As grades above 0.5% and less than 3.0%.
- Then, US$9.00/t per 0.1% for As grades above 3.0%.

The Open Mineral study identified other mines currently producing copper concentrate containing arsenic and confirmed that the Cañariaco concentrate is within the range of the arsenic content in these other concentrates.

Copper concentrates are widely traded and are marketed globally with significant optionality regarding the ultimate customer base. Based on the findings of this study and previous studies the concentrate produced is of sufficient quality to be marketable to smelters globally.

25.12 Capital Cost Estimates

The capital cost estimate conforms to Class 5 guidelines of the Association for the Advancement of Cost Engineering International (AACE International) for a PEA-level estimate with a +50%/-30% accuracy. The capital cost estimate includes a contingency of 21% on initial project capital.

The overall capital cost estimate was developed by Ausenco, with contributions from AGP for the mining cost estimates. The total initial capital cost is estimated at US$ 2,160.2 M. The total sustaining capital cost is estimated at US$ 518.3 M. The capital costs are summarized in Section 21.

25.13 Operating Cost Estimates

The operating cost estimate was developed in Q1 2024 dollars from budgetary quotations and Ausenco's in-house database of projects and studies as well as experience from similar operations. Mine operating costs have been estimated from first principals using quotations from local mine equipment vendors plus local supply consumables. The accuracy of the operating cost estimate is +50%/-30%. The estimate includes mining, processing, and general and administration (G&A) costs. For more details, refer to Section 21.

The overall life-of-mine operating cost is US$12,812.2 M or US$10.89/t of material milled.

25.14 Economic Analysis

Based on the assumptions and parameters in this report, the preliminary economic assessment shows positive economics including a post-tax NPV8% of US$2,346M and 24.1% post-tax IRR. The NPV is most sensitive to metal prices, followed by resource grade, overall operating cost and lastly capital cost.

Readers are cautioned that the PEA is preliminary in nature. It includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the PEA will be realized.

25.15 Risks

25.15.1 Metallurgical Test Work and Recovery Method

The Norte deposit has undergone extensive metallurgical testwork over a number of testwork programs and the metallurgical characteristics for comminution and rougher flotation metal recoveries are well understood. Additional testwork is warranted to confirm rougher concentrate regrind criteria, cleaner flotation metal recoveries and final concentrate grades. This information will support development of the process flowsheet design for the rougher concentrate regrind and cleaner flotation circuits. Based on the results of this testwork the risk exists that the metal recoveries, final concentrate grades and the future design of the regrind and cleaner flotation circuits may vary from that presented in this study, which may increase the capital and operating costs of these components relative to what has been presented in this study.

There is limited metallurgical testwork to date on the Sur deposit. Preliminary metallurgical testwork has achieved positive results. Sur rock hardness is moderately soft and comparable to Norte. Rougher flotation metal recoveries are slightly higher than Norte. Additional metallurgical testwork including mineralogical analysis, rougher concentrate regrind and cleaner flotation testing is warranted to further define the metallurgical characterization of the Sur deposit. Based on the results of future testwork, rock grindability, metal recoveries and the process design may vary from what has been proposed in this study which could result in lower revenue and/or higher operating costs. Limited thickening and filtration testwork has been completed on concentrate and tailings for Norte, which could result in modifications to the process design and potentially an increase in capital and/or operating costs.

There has been no thickening or filtration testing performed on samples of Sur material. Additional testwork will enable finalization of process design criteria and determination of capital and operating costs for these components of the process. The risk associated with limited testwork can be mitigated with testwork in the next phase.

25.15.2 Infrastructure

25.15.2.1 Main Access Road

The conceptual design of the main access road is based on low-quality contour data along the proposed route. This may lead to less-than optimal design with risks that the road may not be constructible in some of the proposed sections, potentially requiring rerouting which could affect the capital cost. Aerial and land route reconnaissance followed by more detailed geotechnical investigations along the proposed road routing is recommended to decrease the risk in the design of this road.

25.15.2.2 Overland Conveyors

The routes selected for the overland conveyors for the waste and mineralized material from the Norte crushing facility is preliminary in nature. The route is designed to follow the topography to reduce earthworks and conveyor support requirements with a minimum radius of 200 m and a maximum incline angle of 15 degrees. Due to the lower quality of contour data, the risks exist that the placement of transfer tower location and the conveyor route may change in subsequent design stages which may have an impact on capital and/or operating costs.

25.15.2.3 Water Diversion Channels

Naturally occurring landslides were observed to have taken place in the vicinity of the location of the proposed water diversion channel. Further assessment of the stability of the valley slopes will be required to assess the severity of this risk, and where risk exists, what mitigation can be undertaken to reduce such risk.

25.15.2.4 Co-Placement Storage Facility

There is no geotechnical information of the CPSF foundation. There may be unknown geotechnical and/or geohazards within the footprint of the facility that might affect the stability of this facility and supporting infrastructure. Further geotechnical and geohazard assessment of the CPSF will be required to assess potential risk to the facilities and to develop potential mitigation measures.

Inability to achieve targeted tailings moisture content could also cause adverse effects for the operation of the CPSF. Additional filtering and dewatering test programs are required to validate that the proposed design moisture content can be achieved.

ARD testwork has been completed in past testwork programs however the quantity of data is insufficient to provide a comprehensive ARD management program for waste rock and tailings. However, based on preliminary data from the project a mitigation program was developed for the PEA. Further assessment of ARD potential of waste materials will be required to assess the significance of ARD risk.

25.15.3 Environmental Studies, Permitting, and Social or Community Impact

The main risks associated with the environmental, permitting, and social aspects of the Project include:

- Maintaining support for the Project from local communities, rightsholders and stakeholders;
- Maintaining regulatory compliance and ongoing implementation of social commitments during the exploration phase of the project, and future start-up construction and operational phases;
- Potential for baseline data gaps in the areas of hydrogeology, geochemistry, and mine water balance determinations that may impact critical path design components for the Project and required regulatory approvals;
- Potential delays in obtaining approvals for the purpose of construction and operation of key infrastructure;
- Potential mine effluent characteristics that may require water treatment throughout the mine life; and
- The timely implementation of the recommendations presented in Section 26.8 will help to quantify, qualify, and mitigate these risks to permitting, construction and operations schedules.

25.15.4 Market Studies and Contracts

This study demonstrates that the Cañariaco project is feasible at current metal prices as well as at metal prices below current prices. In addition, current global supply and demand parameters indicate the potential for relatively strong copper prices in the foreseeable future. However risks existthat global metal prices may vary significantly which could positively or negatively impact the financial feasibility of this project.

The Cañariaco Norte concentrate will contain arsenic in the copper concentrate which will incur treatment penalty charges. These charges have been accounted for in this study. The concentrate produced from the Sur deposit will not incur arsenic penalties. Preliminary independent studies by specialists in global concentrate markets have indicated the Cañariaco Norte/Sur concentrate is marketable. However additional detailed marketing studies including contract discussions with smelters are required for future studies to confirm potential purchasers and to confirm the arsenic penalties that may be imposed on the copper concentrate.

25.16 Opportunities

25.16.1 Mineral Resource Expansion and Estimation

Additional drilling on several parts of the Cañariaco property including the Norte and Sur deposits as well as to the south of Sur covering the Quebrada Verde prospect have potential to delineate additional mineral resources; to support both an increase and/or expansion of known resources and also potentially upgrade the portions of the mineral resources currently classified as Inferred to Indicated and/or Measured resources.

The current mineral estimation could be strengthened with:

- A more robust support for the mineralized envelope and the definition of grade estimation domains;
- A mineralization hardness model is recommended to provide support for the metallurgical throughput assumptions; and
- Density values were assigned to blocks during Mineral Resource estimation based on the lithological codes. Additional bulk density determinations are recommended to be collected from core samples to provide a statistical database of density determinations for each major mineralization and waste lithology.

25.16.2 Mine Plan

Additional geotechnical drilling along with rock quality analysis and ground water assessment is required to enable complete design of pit slopes. Although geotechnical data is available from previous programs it is not sufficient for optimal pit slope design for Feasibility or construction. Specifically, for this study the highwall on the north side of the Norte pit has an overall slope of only 35 degrees. The availability of more geotechnical data may allow the wall to be designed at a steeper slope, potentially in the range of 45 degrees thereby reducing waste stripping requirements.

25.16.3 Metallurgical Test Work and Recovery Methods

Increasing the overall testwork database will provide more data to optimize the process parameters in future studies. This may result in increased recoveries, lower operating costs and improved concentrate quality. Further flotation testwork should consider alternative technologies such as course particle flotation and/or air induction style flotation machines.

25.16.4 Infrastructure

Potential to optimize the CPSF in future studies. This may result in decrease of capital, sustaining capital and operating costs.

25.16.5 Environmental, Permitting, and Community

Opportunities, as listed below, should be considered as the Project continues along the development and permitting path:

- The timely and sustained efforts in the area of community and regulatory engagement regarding proposed project, anticipated impacts (both positive and adverse) and proposed impact mitigation, including discussions with communities on potential benefits of the project.
- The timely baseline gap filling for any subject areas that require additional environmental and socio-economic baseline information that will inform impact mitigation and risk reduction measures associated with infrastructure footprint, and adoption of appropriate low impact and sustainable technologies.
- Regarding hydrological, hydrogeological, and geochemical studies, there are opportunities to work closely and collaborate with the geotechnical, water resources, and mineralized material processing engineering teams and hence, reduce effort and costs.

26 RECOMMENDATIONS

26.1 Overall Recommendations

The Cañariaco Project demonstrates positive economics, as shown by the results presented in this technical report. Continuing to develop the project through to pre-feasibility study is recommended. Table 26-1 summarizes the proposed budget to advance the project through the pre-feasibility stage.

Table 26-1: Recommended Work Program

Program Component	Estimated Budget (US$M)
Exploration, Drilling and Resource Estimation Updates	5.0
Metallurgical Testwork	0.3
Mining Methods	0.8
Process and Infrastructure Engineering	1.5
Site-wide Assessment and CPSF Geotechnical Field and Lab Program	1.5
Environmental, Permitting, Social, and Community	1.0
Total	**10.1**

26.2 Exploration, Drilling and Resource Estimation Updates

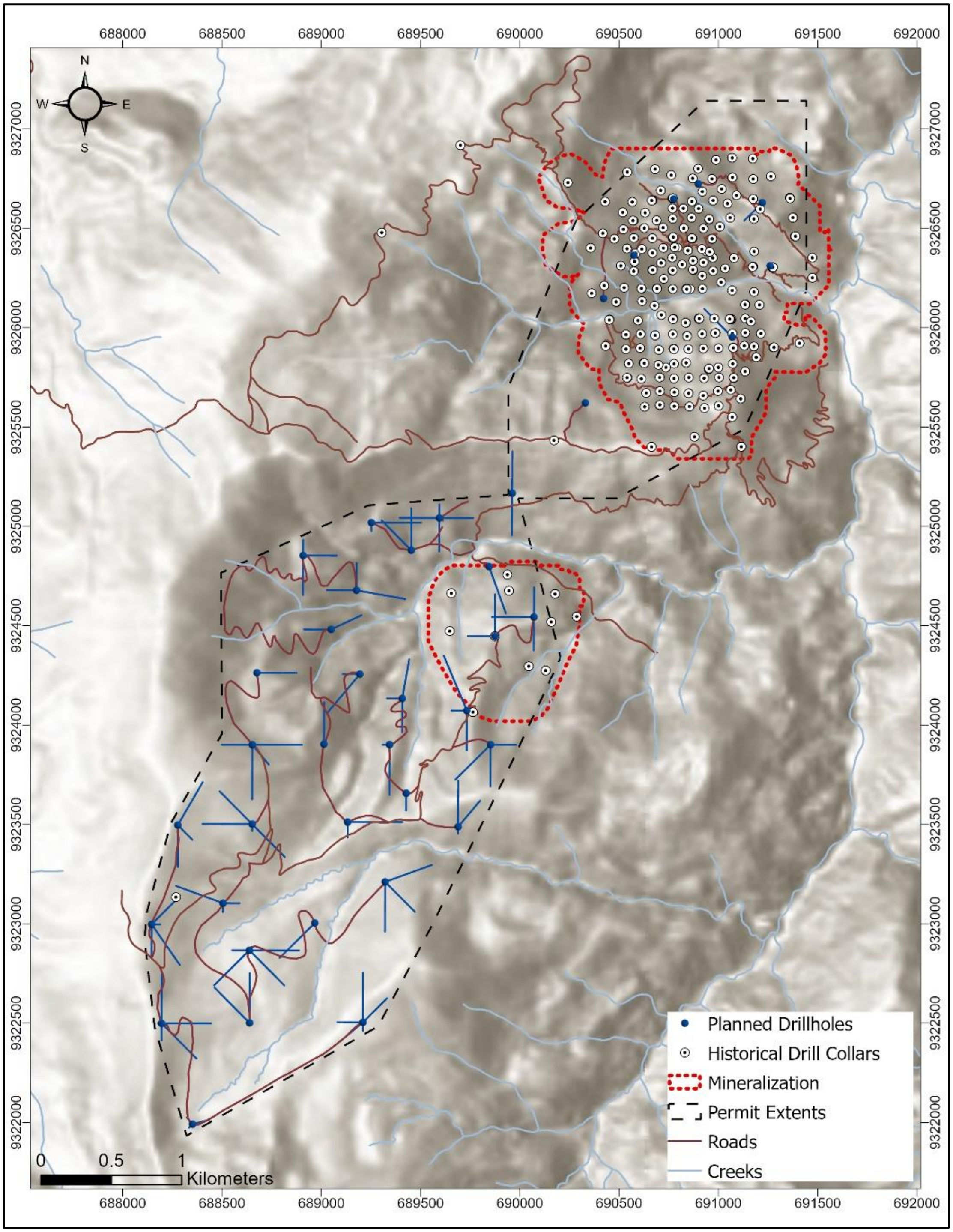
Additional drilling is recommended and planned to be undertaken in several parts of the Cañariaco property especially deeper on and laterally to the Norte deposit, laterally to and deeper on the Sur deposit as well as to the south of Sur on the Quebrada Verde prospect. The objective of the drilling at Norte is to extend higher grade zones as well as to provide sufficient data to support potential upgrading of the portions of the mineral resources currently classified as Inferred to Indicated and/or Measured resources.

The objective at Sur is to identify lateral and depth extensions of the known mineralization which may increase the mineral resources substantially. The lateral and depths limits of the Sur deposit are completely unknown with only 15 holes drilled to date.

The objective at Quebrada Verde is to discover a third mineral deposit on the Cañariaco property and delineate a mineral resource there. The Quebrada Verde target currently comprises extensive altered intrusive rocks and geochemical - geophysical anomalies typical of porphyry deposits completely untested by drilling to date.

A DIA drilling permit application was presented in December 2023 to allow the Company to drill up to 50,000 metres in up to 109 holes from 40 platforms. Drilling is planned to be conducted in stages of approximately 10,000 metres each.

Figure 26-1: Planned Drill Hole for Potential Expansions of Mineral Resources



Source: Alta Copper, 2024.

Once drilling as recommended in Section 26.2 above is completed then the following is also recommended to update the mineral resource estimate for the Cañariaco deposits:

- Mineral zonation models are recommended to be constructed, based on all available drill data. A combination of lithology, alteration and mineral zones should provide a more robust support for the mineralized envelope and the definition of grade estimation domains;
- Mineralization hardness models are recommended to provide support for the metallurgical throughput assumptions; and
- Density values were assigned to blocks during Mineral Resource estimations based on the lithological codes. Additional bulk density determinations are recommended to be collected from core samples to provide a statistical database of density determinations for each major mineralization and waste lithology.

The total recommended budget for each 10,000 metres of drilling with update to Resource Estimation is estimated at US$5.0M.

26.3 Metallurgical Testwork

Additional comminution tests (e.g. crushing work index, rod work index, SMC and abrasion index) and flotation tests (e.g. coarse particle flotation and Jameson style machine flotation) are recommended, particularly for material representative of the first 3-5 years of the planned operation, to provide more confidence in initial equipment selection as well as to ensure that there is comminution information that is spatially representative of the variability within the various mineralized zones.

Metallurgical testing for the Sur deposit needs to be developed to the same detail as the Norte deposit. Cleaner testing will help confirm the response of the various mineralized zones from the Sur deposit to the current flowsheet.

More variability testing of the deposits is recommended to increase the understanding of the responses of various mineralized zones with respect to flotation kinetics and contaminant correlations. This work should not require new metallurgical drill programs as drill cores from the 2013 metallurgical drill program from the Norte deposit have been maintained in cold storage and are available. A proposed future drill program at the Sur deposit will provide core samples for metallurgical testing.

Tailings dewatering and concentrate dewatering testwork are recommended. Rheology of the tailings slurry stream is required and testwork is recommended on the combined composite samples, without prior treatment, to build understanding and help design the CPSF.

The above scope of metallurgical testing is estimated to be US$0.3M.

26.4 Mining Methods

26.4.1 Geotechnical

The Norte pit highwall is over 1,100 m in vertical height which is comparable to other large mines including Morenci in Arizona (1,356 m) Bingham Canyon in Utah (1,200 m); Cuajone in Peru (1,290 m); Buenavista in Mexico (1,200 m) and

Climax in Colorado (950 m). Additional information is required to support the geotechnical assumptions in the 2024 PEA at a Feasibility level.

The Sur pit has a highwall in excess of 750 m vertically. No geotechnical work has been completed to date in this pit area. The work program will need to bring this pit area to a Feasibility level of study.

26.4.2 Hydrogeological

The Norte pit has grown in size since previous hydrogeological work has been completed. In addition the Sur pit has been added to the overall project plan. Interactions between the hydrogeological regmines of the two pits needs to be understood.

Additional hydrogeological work needs to be completed for both Norte and Sur to bring them to a Feasibility level of study.

26.4.3 Mining

With the inclusion of Sur into the Cañariaco Project material transportation from the pit area needs to be further considered. The use of a tunnel/conveyor system has been applied but other technologies may work including RailVeyor. The mining production rate has also increased from previous studies which increases the scale of production.

AGP recommends a detailed materials handling study to examine the overall project. As part of that study, decarbonization strategies need to be considered as vendors advance new technologies on larger mining equipment. This equipment study needs to be tied to FS level mine optimization to assisting in reducing initial stripping needs while maintaining or improving project economics.

The optimization and material handling study is expected to cost US$0.3M.

Studies for feasibility level work including blast analysis, detailed dewatering costing, mine infrastructure design (truck shop design interaction), detailed cost discussions with vendors, stockpile design and management, and scheduling. This is expected to cost a further US$0.5.

The total recommended budget for the mining aspects of the FS are estimated at US$0.8M.

26.5 Process and Infrastructure Engineering

The estimated cost for process and infrastructure engineering for the PFS is US$1.5M. Engineering deliverables would include:

- PFS trade off studies targeting NPV and IRR improvement scenarios;
- Process plant engineering, through criterion, list, drawings, MTOs and cost estimates;
- PFS cost estimating; and
- PFS project execution planning.

26.6 Site Wide Assessment and CPSF Geotechnical Field and Laboratory Program

Additional geotechnical drilling and sampling is recommended around the infrastructure, excluding the open pit, to better define the composition and expected variability in material depths. Geotechnical and hydrogeological drill holes are recommended, along with test pits within footprint of facilities. Laboratory testing is required to refine characterization of the material, including strength and hydraulic parameters for use in stability and seepage analyses.

Determination of the filtered tailings critical state line is recommended to support selection of an appropriate undrained shear strength ratio for the material (or to support modelling as a drained material).

Evaluation of the expected chemistry of the surface water run-off from the filtered tailings and PAG waste rock surface and water quality predictions in the water management pond downstream of the CPSF will be important for site water management planning (i.e. determination of the requirement for water treatment prior to discharge, if the overall site is in a water surplus).

Continue site-specific meteorological and hydrology data collection to support refinement of seasonal run-off and design storm estimates. Additional evaluation of long-term meteorological data is recommended.

A long-term synthetic climate (temperature, precipitation, and evaporation) record for the site will support evaluation of the CPSF water balance (along with water quality) based on historic dry and wet periods for the Project area, along with refinement of the overall site water management plan.

The estimated cost to perform this work is US$1.5M.

26.7 Environmental, Permitting, Social and Community Recommendations

The following recommendations are made regarding reducing risk and uncertainty in the areas of environmental studies, permitting schedule, and community. Qualified professionals should be retained to design and oversee the implementation of each of these studies. Implementation of these recommendations will help to mitigate risk and will support the project to the PFS stage and provide a strong basis for future EIS preparation and permitting.

26.7.1 Water Resources

Building upon the results of hydrographic studies, develop and implement a multi-year seasonal hydrological and meteorological monitoring plan for key areas within the study area to further characterize the hydrological conditions and to develop a future water balance model. The water balance model will be used as a predictive tool regarding the quantity and quality of water available to support mineral processing as well as prediction of effluent quality and quantity. Consideration should be given to establishing a site-specific meteorological station, based on the adequacy of continuing to use data from regional stations.

Development and implementation of a multiyear seasonal groundwater monitoring, sampling, and testing plan focusing on areas that will be potentially affected by mine infrastructure based on current infrastructure plans. Coordinating with planned exploration and geotechnical drilling programs may help to reduce effort and cost of the hydrogeological program.

A conceptual hydrogeological model should be developed for the site and study area, based on groundwater monitoring and testing results and should provide the basis for the future development of a three-dimensional numerical groundwater model that will support feasibility level design and Environmental Impact Assessment / permitting requirements. The model should provide emphasis on seasonal recharge of the freshwater aquifers within and near the project area and the potential drawdown from future pit development and dewatering activities as well as pit water inflows.

Estimated cost for the above recommendations is $0.5M. Cost savings may be realized for hydrogeological characterization work by coordinating closely with geotechnical and exploration teams and their drilling programs.

26.7.2 Geochemistry

A geochemical assessment of the ARD/ML risk for the project should be implemented utilizing the existing geological model for the site and sampling of fresh drill core sampled intervals, if available. Generally, the program should consist of the collection adequate waste rock, overburden, and tailings samples to be considered representative of the mine rock to be stored on site. The identification of quantity and location of the samples should be based on the site geological and structural model, and mineralogical considerations.

Range of analytical tests should include elemental analysis, acid-base accounting, shake flask extraction (short term leach), NAG pH, minerology, and humidity cell testing (minimum 40 weeks).

Development of preliminary source terms for the weathering of waste rock, mineralized material, tailings, and pit walls for use in water balance modelling.

Preliminary interpretation of results and assessment of requirement for site-specific mine rock management practices and water treatment.

The estimated costs for the above are US$0.2M.

26.7.3 Other Environmental Baseline Studies

A gap analysis should be conducted on feasibility level proposed infrastructure footprint that includes all proposed Project facilities (excluding proposed pit areas which have received archaeological clearance) to ensure that baseline studies are adequate for assessing archaeological resources and for obtaining the required archaeological clearances.

Develop and implement a seasonal multi-year baseline vegetation/ecosystem and wildlife/wildlife habitat survey plan for areas of disturbance within the Project area with special emphasis on listed and threatened species under Peruvian legislation. The plan should build upon baseline work completed to date.

Baseline conditions for air quality and noise should be established for near field and further afield operations.

Near surface soil textures and chemistry should be established for the project area as part of the baseline program.

The estimated costs for the above is US$0.2M.

26.7.4 Socio-Economic, Cultural Baseline Studies and Community Engagement

Socio-economic studies and community engagement effort should continue, building on previous work, to help identify community needs and provide a basis for targeted community investment in local development projects, training, education, and employment.

The estimated cost for this program is US$0.1M.

26.7.5 Environmental Constraints Mapping

To assist in the development of the project at the feasibility design phase, environmental constraints mapping should be developed and periodically updated, based on the results of historical and future baseline environmental and land use studies. This mapping should be utilized to limit risks at the design stages of the project.

The estimated cost for this program is US$0.01M.

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