Cañariaco Norte Project

Lambayeque Region, Peru

NI 43-101 Technical Report on Preliminary Economic Assessment

Effective Date: 8 February 2022

Amended and refiled 15 March 2022

Prepared for: CANDENTE COPPER CORP

Suite 801 - 1112 West Pender Street Vancouver, BC Canada V6E 2S1

Prepared by: Ausenco

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List of Qualified Persons:

Mr. Kevin Murray, P.Eng. — Ausenco Engineering Canada Inc.
Mr. Scott Elfen, P.E. — Ausenco Engineering Canada Inc.
Mr. Scott Weston, P.Geo. — Hemmera Envirochem Inc.
Mr. Jay Melnyk, P.Eng. — AGP Mining Consultants Inc.
Mr. David Thomas, P.Geo. — DKT Geosolutions Inc.

CERTIFICATE OF QUALIFIED PERSON
Kevin Murray, P. Eng.

I, Kevin Murray, P.Eng., certify that:

1. I am employed as a Manager Process Engineering with Ausenco Engineering Canada Inc., with an office address of 1050 West Pender Street, Suite 1200, Vancouver, BC, Canada, V6E 3T4.
2. This certificate applies to the technical report titled "Cañariaco Norte Project NI 43-101 Technical Report, Preliminary Economic Assessment, Region of Lambayeque, Peru" that has an effective date of 8 February 2022 and an amended and refiled date of 15 March 2022 (the "Technical Report").
3. I graduated from the University of New Brunswick, Fredericton NB, in 1995 with a Bachelor of Science in Chemical Engineering. I am a member in good standing of Engineers and Geoscientists British Columbia, License# 32350.
4. I have practiced my profession for 21 years. I have been directly involved in all levels of engineering studies from preliminary economic analysis to feasibility studies. I have been directly involved with test work and flowsheet development from preliminary testing through to detailed design and construction.
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 Norte Project.
7. I am responsible for sections 1.1, 1.2, 1.3, 1.9, 1.13, 1.14.1, 1.14.2, 1.14.5, 1.14.6, 1.16, 1.17, 1.18, 1.19, 1.20, 1.21, 1.22, 1.23, 2, 3.1, 13, 17, 18.1, 18.2, 18.7.1, 18.7.2, 18.8, 18.9, 18.10, 18.11, 19, 21.1, 21.2.1, 21.2.2.1, 21.2.2.3, 21.2.3, 21.3.1, 21.3.2, 21.3.4, 21.3.5, 22, 24, 25.1, 25.5, 25.8, 25.9, 25.11, 25.12, 25.13, 25.14, 25.15, 25.16, 26.1, 26.3.3, and 27 of the Technical Report.
8. I am independent of Candente Copper Corp., as independence is defined in Section 1.5 of NI 43-101. I have had no previous involvement with the Cañariaco Norte 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.
10. 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 March 15, 2022.

"*Signed and Sealed*"

Kevin Murray, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

Scott C. Elfen, P.E.

I, Scott C. Elfen, P.E., do hereby certify that:

1. I am the Global Lead Geotechnical and Civil Services of Ausenco Engineering Canada Inc., 1050 West Pender Street, Suite 1200, Vancouver, BC, Canada, V6E 3T4.

2. This certificate applies to the technical report titled “Cañariaco Norte Project NI 43-101 Technical Report, Preliminary Economic Assessment, Region of Lambayeque, Peru” that has an effective date of 8 February 2022 and an amended and refiled date of 15 March 2022 (the “Technical Report”).

3. I graduated from the University of California, Davis with a Bachelor of Science degree in Civil Engineering (Geotechnical) in 1991. I am a Registered Civil Engineer in the State of California (No. C56527) by exam since 1996 and I am also a member of the American Society of Civil Engineers (ASCE), Society for Mining, Metallurgy & Exploration (SME) that are all in good standing.

4. I have practiced my profession continuously for 24 years and have been involved in geotechnical, civil, hydrological, and environmental aspects for the development of mining projects; including feasibility studies on numerous underground and open pit base metal and precious metal deposits in North America, Central and South America, Africa and Australia.

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 Norte Project.

7. I am responsible for sections 1.1, 1.2, 1.14.3, 1.14.4, 1.17, 1.18, 1.21, 1.22, 1.23, 2.1, 2.2, 2.3, 2.6, 2.8, 18.1, 18.4, 18.5, 18.6, 18.7.3, 20.1.3, 20.1.4, 21.2.2.3.2, 21.2.4, 21.3.6, 25.1, 25.9, 25.10, 25.12, 25.13, 25.15, 25.16, 26.1, 26.3.4, 26.3.5, and 27 of the Technical Report.

8. I am independent of Candente Copper Corp., as independence is defined in Section 1.5 of NI 43-101. I have had no previous involvement with Cañariaco Norte 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.

10. 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 March 15, 2022.

“*Signed and Sealed*”

Scott C. Elfen, P.E.

CERTIFICATE OF QUALIFIED PERSON
Scott Weston

I, Scott Weston, P. Geo., certify that:

1. I am employed as Vice President of Business Development with Hemmera Envirochem Inc., an affiliate of Ausenco Engineering Canada Inc., with an office address of 4515 Central Boulevard, Burnaby, BC, Canada.
2. This certificate applies to the technical report “Cañariaco Norte Project NI 43-101 Technical Report, Preliminary Economic Assessment, Region of Lambayeque, Peru” that has an effective date of 8 February 2022 and an amended and refiled date of 15 March 2022 (the “Technical Report”).
3. I graduated from the University of British Columbia, Vancouver, BC, Canada, in 1995 with a Bachelor of Science, Physical Geography, and from Royal Roads University, Victoria, BC, Canada, in 2003 with a Master of Science, Environment and Management.
4. I am a Professional Geoscientist of Engineers and Geoscientists British Columbia (No.124888). I have practiced my profession for 25 years and have experience in geomorphic and landscape genesis interpretation, soil science, terrain stability analysis, natural hazard assessment, hydrology and environmental impact assessment.
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 Norte Project.
7. I am responsible for sections 1.1, 1.2, 1.15, 1.21, 1.22, 1.23, 2.1, 2.2, 2.3, 2.6, 2.8, 3.2, 3.3, 4.8, 4.9, 4.10, 20.1.1, 20.1.2, 20.2, 20.3, 20.4, 25.1, 25.10, 25.15, 25.16, 26.1, 26.3.6, and 27 of the of the Technical Report.
8. I am independent of Candente Copper Corp. as independence is defined in Section 1.5 of NI 43-101. I have had no previous involvement with the Cañariaco Norte 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.
10. 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 March 15, 2022.

“*Signed and Sealed*”

Scott Weston, P. Geo.

CERTIFICATE OF QUALIFIED PERSON
Jay Charles Melnyk

I, Jay Charles Melnyk, P. Eng., certify that:

1. I am employed as a Professional Engineer, with AGP Mining Consultants Inc., with a business address at #246-132 Commerce Park Dr., Unit K, Barrie, ON, L4N 0Z7.
2. This certificate applies to the technical report titled "Cañariaco Norte Project NI 43-101 Technical Report, Preliminary Economic Assessment, Region of Lambayeque, Peru" that has an effective date of 8 February 2022 and an amended and refiled date of 15 March 2022 (the "Technical Report").
3. I graduated from the Montana Tech of the University of Montana with a Bachelor of Science degree in Mining Engineering in 1988 and from the British Columbia Institute of Technology with a Diploma in Mining Technology in 1984. I am a member in good standing of the Engineers and Geoscientists of British Columbia (License #25975).
4. I have practiced my profession for 33 years. I have been directly involved in open pit mining operations, and design of open pit mining operations in Argentina, Canada, Chile, Eritrea, Indonesia, Iran, Mexico, Perú, and the United States.
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 Norte Project.
7. I am responsible for sections 1.1, 1.2, 1.12, 1.17, 1.18, 1.21, 1.22, 1.23, 2.1, 2.2, 2.3, 2.6, 15, 16, 18.3, 21.2.2.2, 21.2.4, 21.3.3, 25.1, 25.7, 25.12, 25.13, 25.15, 26.1, 26.2, 26.2.1, and 27 of the Technical Report.
8. I am independent of Candente Copper Corp. as independence is defined in Section 1.5 of NI 43-101. I have been involved with the project as a QP for the previous 43-101 technical report titled 'Cañariaco Norte Project Prefeasibility Progress Report', effective date 18 January 2011.
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.
10. 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 March 15, 2022.

"*Signed and Sealed*"

Jay Charles Melnyk, P.Eng.

AGP Mining Consultants Inc.

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

I, David Gwilym Thomas, P. Geo., certify that:

1. I am employed as a Principal Geologist with DKT Geosolutions Inc., with an office address of Suite 170 – 420 Richards Street, Vancouver, BC, Canada.
2. This certificate applies to the technical report titled "Cañariaco Norte Project NI 43-101 Technical Report, Preliminary Economic Assessment, Region of Lambayeque, Peru" that has an effective date of 8 February 2022 and an amended and refiled date of 15 March 2022 (the "Technical Report").
3. 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.
4. I have practiced my profession for 22 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).
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 between February 14th and February 16th, 2022, for a visit duration of 3 days.
7. I am responsible for sections 1.1, 1.2, 1.4, 1.5, 1.6, 1.7, 1.8, 1.10, 1.11, 1.21, 1.22, 1.23, 2.1, 2.2, 2.3, 2.6, 2.8, 3.2, 4, 5, 6, 7, 8, 9, 10, 11, 12, 14, 23, 24, 25.1, 25.2, 25.3, 25.4, 25.6, 25.15, 26.1, 26.2, 26.2.2, 26.3.1, 26.3.2, and 27 of the Technical Report.
8. 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 Cañariaco 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 and 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.
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.
10. 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 March 15, 2022.

"*Signed and Sealed*"

David Gwilym Thomas, P. Geo.

Important Notice

This report was prepared as National Instrument 43-101 Technical Report for Candente Copper Corp. (Candente Copper) by Ausenco Engineering Canada Inc. (Ausenco), AGP Mining Consultants Inc. (AGP), 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 Candente 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 is at that party's sole risk.

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1 SUMMARY

1.1 Introduction

Candente Copper Corp. (Candente Copper) requested that Ausenco Engineering Canada, Inc. (Ausenco) prepare a technical report (the Report) on the results of a preliminary economic assessment (2022 PEA) on the Cañariaco Norte deposit within the Cañariaco Project in Peru.

1.2 Terms of Reference

The Report was prepared to support disclosures in Candente Copper's news release dated February 8, 2022, entitled "Candente Copper Announces Positive PEA Results for the Cañariaco Copper Project".

The Report was amended to address a typographical error in one of the Certificates of Qualified Person that was used as a Report date and signature page. No other changes were made to the Report.

Units used in the report are metric units unless otherwise noted. Monetary units are in United States dollars (US$) unless otherwise stated. Mineral Resources are classified using the 2014 edition of the Canadian Institute of Mining and Metallurgy (CIM) Definition Standards for Mineral Resources and Mineral Reserves (the 2014 CIM Definition Standards). Mineral Resources estimates were prepared using the guidance set out in the 2019 edition of the CIM's Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (2019 Guidelines). The Report uses Canadian English.

1.3 Project Setting

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 120 km to the northeast of the city of Chiclayo.

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.

Temperatures in the Project area vary with altitude and range between approximately 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.

Elevations in the Project area range between 2,200 and 3,600 masl, as the Project is situated on the eastern side of the continental 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 for tailings disposal, mine waste disposal, and installations such as a process plant and related mine infrastructure.

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

The Project consists of 15 mining concessions totalling approximately 10,190 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 Candente Copper's case, the 10th year was 2008, and penalties have been paid since that date. Ausenco was advised by Candente 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 land owner to obtain access rights. All transactions and contracts pertaining to a mining concession must be registered with the Public Mining Registry.

Candente 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 Cañariaco Norte and Cañariaco Sur deposits are considered to be examples of porphyry copper deposits.

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 mapped. The host rocks to the mineralized intrusions are Calipuy Group andesite volcanic rocks.

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. 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-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. The main copper minerals are chalcopyrite, covellite, chalcocite, and enargite, with minor bornite, tennantite, and digenite, and trace tetrahedrite. Copper sulphide minerals are mainly associated with pyrite and other sulphide 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 Cañariaco Sur deposit is located approximately 2 km southwest of Cañariaco Norte. The deposit consists of a large area of potassic alteration with a coincident copper–gold–molybdenum soil geochemical anomaly with approximate dimensions of 1,400 x 900 m. The host rocks to the mineralized intrusions are Calipuy Group andesite volcanic rocks. Two main intrusions were noted from geological mapping, an early 750 x 500 m quartz monzonite stock, and a later diorite stock with an area of approximately 100 x 50 m. 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.

The Quebrada Verde prospect consists of a 1 km x 750 m diorite porphyry stock that 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 also considered sufficiently acceptable to reliably inform conceptual mine planning at Cañariaco Norte. The Cañariaco Sur deposit is not used in the 2022 PEA.

1.6 History

Prior to Candente 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 surveys, petrographic studies, core drilling, mineral resource estimation, and very preliminary leach testwork.

Candente Copper acquired the Project in 2002. Work conducted since acquisition included 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. A preliminary assessment was completed in 2006 on the Cañariaco Norte deposit and was updated in 2008. A mining study was completed in 2011. These studies are not considered to be current; however, some of the data and information obtained during the 2008–2011, such as environmental baseline data, are used to support the 2022 PEA.

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 Candente Copper.

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

Candente 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 Candente Copper using Pajari, Sperry Sun, or Reflex EZ-Shot instruments.

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. Candente 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.

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

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 Candente Copper programs. Actlabs is independent of Candente 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 Candente 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 Candente Copper and held ISO:9000 accreditation for the Peruvian laboratory at the time the work was performed.

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 and copper, lead, zinc, molybdenum and arsenic, 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.

Sample preparation undertaken on the Candente 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. 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. 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 used a 36-element inductively-coupled plasma optical emission spectrometry method. ALS Chemex performed a 33-element ICP atomic emission spectroscopy method.

There is no information on any quality assurance/quality control (QA/QC) programs for INGEMMET and Placer Dome. Billiton used blanks, standard reference materials (standards) and check assays. Chain-of-custody and sample preparation protocols were also part of Billiton's QA/QC program.

Candente Copper analytical programs included submission of duplicates, standards, blanks, and check assays. The QA/QC program results do not indicate any problems with the analytical programs. The copper, gold, and silver analyses from the core drilling are suitable for inclusion in Mineral Resource estimation.

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.

1.8 Data Verification

All data in the field were recorded in written form in field books, log books, 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 were verified by Candente Copper personnel.

Drill data collected from the INGEMMET, Placer Dome, and Billiton campaigns were re-logged by Candente Copper personnel, and nine of the drill holes have been re-assayed. Based on the correlations between the historical grades and the Candente Copper re-assay grades, all of the historical data were accepted into the final database. Three pairs of twinned holes were drilled by Candente 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 testwork on the Cañariaco Norte deposit was focused on leachable copper. However, as the Project has developed, and the mineralization was confirmed as primarily sulphide, conventional milling was determined to be a more appropriate process.

Several major phases of testwork were conducted on the Cañariaco Norte deposit. The first consisted of process development to define the type of processing most applicable to the mineralization. This was followed by more detailed work to optimize process conditions. 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.

Testwork has included mineralogy, QEMSCAN examination, comminution and variability comminution tests, tests on the effects of grind sizes, collectors and pH, sulphidation, cleaner flotation tests and locked cycle tests. Concentrates were also produced for proof-of-concept testing to show that arsenic and antimony levels in concentrate could be reduced by using a roaster step, resulting in an enhanced concentrate which would have low or no penalty elements and would be broadly marketable.

The latest phase of variability testwork conducted in 2011 and 2012 was designed to improve the geometallurgical understanding of the deposit. The programs consistent on mineralogical characterization of the composites, additional comminution tests, and roughing and cleaning flotation optimization tests. The program demonstrated that a lower grade concentrate could be produced with low or no penalty element concentrations without the use of a roasting step.

The testwork recommended key design parameters, as follows:

- For a feed grade of 0.40% Cu, a recovery of 89.7% is forecast at the target grind;
- Recovery will vary with feed grade;

- The concentrate grade was 26% Cu, with arsenic being lowered to low or no penalty concentrations;
- The grind was established at 200 µm with a JKSimMet breakage parameter Axb value of 68; and
- Bond ball mill work index (BWi) of 11.1 kWh/t.

Copper recoveries are expected to vary with feed grade, ranging from 93.1% at 0.6% Cu and 58.8% at 0.1% Cu. 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, gold and silver recoveries are projected to be 64.7% and 57.2% respectively.

The work conducted in 2012 focused on establishing robust improvements and proposed solutions to deal with penalty elements. This resulted in a final flowsheet generating a low-grade copper concentrate to the smelter with arsenic content controlled to a level where no impact on concentrate marketability is anticipated over the 2022 PEA LOM.

No testwork has been completed to the Report effective date on the Cañariaco Sur deposit. Due to proximity of the mineralization to the Cañariaco Norte deposit, and a similar deposit setting, the recoveries for copper, gold, and silver forecast for the LOM for the Cañariaco Norte deposit are used in the Mineral Resource estimate for Cañariaco Sur. Until deposit-specific metallurgical testwork is available, the confidence category should be limited to Inferred, however.

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 0.15% Cu; which is above the breakeven cut-off grade of 0.12% Cu. Mineral Resources for Cañariaco Sur are also reported at a cut-off grade of 0.15% 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 in Table 1-2 for Cañariaco Sur. The estimates have an effective date of 27 January 2022.

Mineral Resources from the Cañariaco Norte deposit support the 2022 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 0.15% 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	423.5	0.48	0.43	0.07	1.9	4.04	0.98	25.71
Indicated	670.7	0.39	0.36	0.05	1.6	5.25	1.16	33.72
Measured + Indicated	1,094.2	0.42	0.39	0.06	1.7	9.29	2.14	59.43
Inferred	410.6	0.32	0.29	0.04	1.4	2.66	0.55	18.09

Notes to accompany Cañariaco Norte Mineral Resource table:

1. The Mineral Resources estimate has an effective date of January 27, 2022. 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 Hexagon's MinePlan 3D™ software using:
   - A copper price of US$3.50/lb;
   - Mining cost of US$1.60/t;
   - A combined processing, tailings management and G&A cost of US$6.52/t;
   - Variable pit slope angles ranging from 36 to 39 degrees;
   - A copper process recovery of 88%;
   - Copper concentrate smelter terms: US$75/dry metric tonne treatment charge, US$0.075/lb refining charge and 96.2% payable;
   - Estimated concentrate shipping costs of US$133.00/wet metric tonne of concentrate.
5. Copper equivalent grades including contributions from gold and silver, were estimated using metal prices of copper: US$3.50/Ib, gold: US$1,650 /oz, and silver: US$21.50/oz), metallurgical recoveries of copper: 88%, gold: 65%; silver: 57%, 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

	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	0.15	384.5	0.32	0.26	0.10	1.2	22	2.22	1.18	15.02	18.91

Notes to accompany Cañariaco Sur Inferred Mineral Resource table:

1. The Mineral Resource estimate has an effective date of January 27, 2022. 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 0.15% Cu cut-off grade.
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 Hexagon's MinePlan 3D™ software using:
   - A copper price of US$3.50/lb;
   - Mining cost of US$1.60/t;
   - A combined processing, tailings management and G&A cost of US$6.52/t;
   - 45-degree pit slope angles;

- A copper process recovery of 88%;
- Copper concentrate smelter terms: US$75/dry metric tonne treatment charge, US$0.075/lb refining charge and 96.2% payable;
- Estimated concentrate shipping costs of US$133.00/wet metric tonne of concentrate.

10. Copper equivalent grades including contributions from gold, silver and molybdenum, were estimated using metal prices of copper: US$3.50/lb, gold: US$1,650/oz, silver: US$21.50/oz and molybdenum: US$11.00/lb; metallurgical recoveries of copper: 88%, gold: 65%; silver: 57% 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. There is currently no metallurgical testwork on Cañariaco Sur mineralization. Cañariaco Norte average recoveries were applied for conceptual pit shell generation and CuEq estimations.
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

The 2022 PEA is preliminary in nature and is partly based on 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 2022 PEA based on these Mineral Resources will be realized.

Mining will be conducted using conventional open pit methods and conventional equipment. 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 mineralized material to the mineralized material crusher, or short-term stockpile as required. Waste will be hauled during pre-production to the planned co-disposal (dry-stack) waste rock and tailings storage facility (WSF), 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 WSF. Based on earlier internal desktop trade-off studies, the mine plan considers an initial 40 kt/d processing rate which is later doubled to 80 kt/d in production Year 7. The peak mining capacity is 54.6 Mt/a.

The mine planning was performed using metal prices of US$3.25/lb Cu, US$1,550/oz Au and US$20/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 FS level pit slope designs for the Cañariaco Norte project.

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. An additional 2% of mining loss was applied to account for carry back and mineralization routing errors.

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, 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 Mining Consultants Inc (AGP). Process and G&A costs were provided by Ausenco based on earlier internal studies.

Four phase designs were developed for the planned single open pit. 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 ultimate pit design was based on the RF 0.55 LG shell. The overall dimensions of the ultimate pit are approximately 2,000 m in the north–south direction, 1,700 m in the east–west direction and a 930 m maximum depth. It will have a single ramp exit point at the 2,680 m elevation, providing access to the mill feed and waste crushers, as well as the truck shop and fuel bay.

Ramps widths were based on the use of 218-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 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.

Mine planning was performed based on marginal cut-offs applied to the NSR grade item. During the initial Project phase when the concentrator is operating at 40 kt/d, the marginal cut-off is the sum of the mill feed based operating costs, (processing, G&A and tailings management), which is US$7.91/t. When the process rate increases to 80 kt/d, the marginal cut-off decreases to US$6.52/t. No long-term stockpiling of low-grade material has been considered. Within the ultimate pit, at the US$6.52/t NSR cut-off the classification breakdown of the mill feed material is 54% Measured, 38% Indicated and 8% Inferred.

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 WSF, conduct road building/pioneering, and to strip 35.0 Mt of waste rock from the pit, exposing initial mill feed material, and hauling the waste directly to the base of the WSF. Pre-stripping will be performed in Years -2 and -1.

Mill feed delivery to the crusher in the first production year is forecast at 11.1 Mt. In production Year 2 through Year 6, the full 14.6 Mt (40,000 t/d) will be delivered to the crusher area. In Year 7, the production rate will increase to 26.3 Mt. During Year 8 through Year 27, the full 29.2 Mt (80,000 t/d) will be delivered. The last year of production, Year 28, will be a partial year with 8.2 Mt processed.

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.

Equipment requirements include 35 m^3 electric hydraulic face shovels, 33 m^3 front-end loaders, 218 t electric-drive haul trucks, drill rigs that can drill 270 mm and 160 mm diameter drill holes, and a fleet of track dozers, wheel dozers, and graders.

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

1.13 Recovery Methods

The processing plant will be built in two phases. Phase 1 will have a nominal throughput of 14.6 Mt/a during the first six production years. Phase 2 will double the annual production by adding a parallel line, from the mill feed stockpile to concentrate and tailing filtration and will have the same equipment and layout as the Phase 1 process line. The plant is designed to operate 24 hours per day, 365 days per year with an overall plant availability of 92%.

The process plant selected is a conventional copper concentrator and the process design is typical of a concentrator treating copper sulfide ores. The process plant feed will be supplied from the open pit mine with a LOM average feed grade of 0.39% Cu and will produce a copper concentrate containing 26% Cu.

The process plant will include the following units, processes, and facilities:

- Primary crushing of ROM mill feed and waste rock;
- Overland conveyor system to transport ore and waste, in discrete lots, to tripper conveyor and separate stockpiles near the process plant ;
- Coarse ore stockpile and reclaim system;
- Coarse waste stockpile and reclaim system;
- Mill feed material grinding including a SAG mill, pebble recycle crusher and ball mill in closed circuit with hydrocyclones;
- Copper flotation;
- 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 conveying with waste rock in the WSF;
- Reagent storage and distribution; and
- Water services and management (process water, treated water, raw water).

1.14 Project Infrastructure

1.14.1 Location

The mine site facilities are divided into four general areas: the mine, the crusher (that include buildings and structures for repair and maintenance of mine and plant equipment), the plant site, and the camp area, (which includes facilities for personnel accommodations, administration, and security). Other support facilities and services include site access, power supply and distribution, water supply, explosive storage and handling, communication systems and waste disposal facilities at the site.

The site was selected on a natural elevated area on 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 the Corral Quemado Road, which runs along the Huancabamba Valley. The main access road to the plant site and camp site will be constructed off the Corral Quemado Road. The entrance to the road will be controlled, 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. The mine site will need a network of general vehicle access roads around facilities, service roads to remote structures, and haul roads.

1.14.3 Co-disposal Waste and Tailings Storage Facility

A co-disposal facility is envisaged for waste rock and tailings storage. Approximately 697 Mm^3 of mine waste will be stored within the WSF, including 439 Mm^3 of filtered tailings and 258 Mm^3 of waste rock. The WSF is not expected to behave like a conventional mine waste facility because of the large proportion of stored tailings and will be constructed with perimeter berms of waste rock to provide containment of the filtered tailings and provide erosion protection for the overall structure. The WSF was classified as significant or moderate under both Canadian Dam Association and the British Columbia Mine Waste Rock Pile Research Committee "Investigation and Design Manual".

During pre-production and operations, mine waste will be crushed and conveyed to the plant site by conveyor. The mine waste will be transferred to the WSF conveyor to be placed in the WSF. It is assumed that the pre-production waste rock will be non-acid generating (NAG), and any PAG waste rock will be mixed to mitigate any metals leaching or acid rock drainage (ML/ARD). During operations the waste rock delivered to the plant area will be conveyed by itself to the WSF to build exterior berms or conveyed with filtered tailings for co-mingle disposal in the WSF using two conveyor systems. Based on the potential acid generation of materials, calcium carbonate will be added as part of the ARD mitigation measures. It is assumed that PAG materials will become more prevalent with depth. After Year 6, there will be insufficient waste rock to co-mingle with the tailings waste stream. Therefore, the waste rock will be used to develop exterior berms and the balance will be co-mingled with tailings near the front of the facility to improve physical stability.

The WSF will be constructed in seven phases over the 28-year mine life. The WSF will includes a contact water management pond (CWMP) downstream of the facility, surface water management structures, rock drains, access road, waste conveyor system, equipment for spreading and compacting materials, and a water reclaim pipeline.

Instrumentation and monitoring will be required to assess embankment performance. Vibrating wire piezometers will be installed to monitor pore pressure within the embankment fill materials and slope inclinometers and survey monuments will be installed to monitor slope movement and deformation.

1.14.4 Water Supply and Management

Raw water for potable and process make-up, and other uses at the mine site, will be obtained from pit stormwater pond filled by pit surface water runoff and rainwater and from the WSF CWMP.

Process water for the plant will come from three sources: the CWMP in WSF, process plant run-off water in the settling pond, and overflow from the process water tank. The thickener overflow and WSF reclaim will be the primary water sources.

The surface of the filtered tailings and waste rock within the WSF will be graded to encourage flow to defined surface water collection points. The collected surface water will be directed to the CWMP. The water stored in the CWMP will be used as a source of make-up water for the process plant. Any water in excess of the process plant requirement (if any) will be directed to the water treatment plant (WTP) and released in the drainage below the CWMP.

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 will range from 625 m^3/hr to a maximum of 2,228 m^3/hr over the life of the Project. A water treatment plant with a capacity to treat up to 144 m^3/hr will be required to treat excess effluent.

1.14.5 Built Infrastructure

The mine design includes provision for a truck shop, truck wash station, mill maintenance shop, bulk emulsion plant, an ammonium nitrate storage facility, and a detonator storage magazine, and a camp and administration area.

1.14.6 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 55 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 total connected load for operation is 119 MW in Phase 1 (Years 1–6) and 171 MW for Phase 2 (Years 7–28).

The pit dewatering pumps will be diesel powered. Power required for the water treatment plant will be produced by diesel generating sets.

1.15 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 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. In addition, AMEC conducted baseline studies for a detailed EIA from 2007 to 2014. This detailed EIA was considered to be approximately 80% complete but will now need more work to be completed. Plan are underway to meet with SENACE and get this work started.

1.15.1 Environmental Considerations

Completed baseline studies include:

- Air quality: concentrations obtained for the parameters analyzed show that they do not exceed the National Environmental Quality Standards (ECAs) established for each parameter;
- Noise: most stations complied with the ECA for daytime and night-time hours during the entire analysis period in 2012; however, in 2021, two stations had elevated levels, associated with the circulation of vehicles on the road near these stations and the presence of strong winds;
- Hydrography, hydrology and hydrogeology: 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;
- Soils: land was classified as either of low agricultural quality or having a higher-use capacity;
- Water quality: concentrations obtained for the parameters analyzed show that they generally do not exceed the National Environmental Quality Standards (ECAs) established for each parameter. During 2012, total nitrogen, ammonia and copper exceeded ECA values at some monitoring points;
- Ecosystem assessments: four ecosystems were defined in the Project area, including montane forest, grassland, shrub thicket, and coastal/Andean agriculture;
- Flora: a total of 473 species of flora were recorded during wet and dry season surveys, distributed in 41 orders, 84 families and 238 genera. Species of concern under national legislation, S. D. N° 043-2006-AG, the International Union for Conservation of Nature (IUCN) listings, and the Convention on International Trade in Endangered Species (CITES) Appendix II. None of the species of concern were endemic to the Project area;
- Fauna: information was collected on mammal, bird, reptile and fish species during wet and dry season surveys. The presence of species of concern were recorded. None of the mammal species is currently endemic to Peru. One bird species, *Thaumasius taczanowskii* "Taczanowski's Hummingbird, is endemic. No fish were present in either season, and the absence of fish was confirmed by interviews with local residents.

The Surface Water Management Plan will preserve the "no contact" status of surface waters to the maximum extent practicable. Waters 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, detour, 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 most of the facilities (the WSF, open pit and process plant, camp) will be constructed, and the Quebrada Yerma valley, which is the proposed location of the WSF. 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 WSF 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 WSF. Contact water from the facilities in the well area, the process plant, and the WSF will be collected immediately downstream of the WSF 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 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 economic analysis in the 2022 PEA assumes a closure cost of US$104 M.

1.15.3 Permitting Considerations

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;
- 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

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. 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.

1.16 Markets and Contracts

No market studies or product valuations were completed as part of the 2022 PEA. Market price assumptions were based on a review of public information, industry consensus, standard practiced and specific information from comparable operations in the region.

Copper concentrates are widely traded. Concentrates can be marketed directly from producer (mine) to smelter, or through third party concentrate trading entities. The lack of investment in new copper mines and expansions to existing operations contribute to a widely held belief that there will be increasing demand for copper concentrates in this decade and beyond. The expected supply constraint is further exacerbated by a world copper demand that is expected to increase on basis of emerging, copper-dependant technologies, such as electronic vehicles and renewable energy, and continued urbanization of the world's population.

Project economics were estimated based on long-term metal prices of US$3.50/lb Cu, US$1,650/oz Au and US$21.50/oz Ag, which was established by Candente Copper in conjunction with consensus forecasts from various financial institutions.

The QP notes that Candente Copper's pricing used in the cash flow analysis is reasonably aligned with various long-term forward-looking estimates.

No contracts for transportation or off-take of the concentrates are currently in place, but if and when they are negotiated, they are expected to be within industry norms. Similarly, there are no contracts currently in place for supply of reagents, utilities, or other bulk commodities required to construct and operate the Project.

1.17 Capital Cost Estimates

Capital costs are provided in Q4 2021 US$. The estimate uses AACE International (AACE) guidelines and is reported as an AACE Class 5 Order of Magnitude/Conceptual Study estimate with a -30% to +50% accuracy. Exchange rates used include US$1.00 = PENS/4.00, and US$1.00 = C$/1.29.

The overall capital cost estimate was developed by Ausenco with contributions from AGP for the mining cost estimates. Costs were separated into initial and sustaining capital costs, and further subdivided out into what will be needed for the first project phase (40 kt/d) and expansion phase (80 kt/d). Initial capital costs consisted of:

- Direct: 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 include contractor's distributable costs, are contained within the direct costs;

- Indirect: all costs that are necessary for project completion but not related to the direct construction cost and incurred by the Owner, engineer or consultants in project design, procurement, construction, and commissioning to support during the construction period.

A breakdown of capital cost figures by major work area is presented in Table 1-3 for the first project phase, and for the expansion phase.

Table 1-3: Capital Cost

WBS	Description	Total for Phase 1 (US$M)	Total for Phase 2 (US$M)
0200	Sitewide water services	7.4	1.1
0300	Sitewide power & lighting	7.1	-
2000	Mining	189.2	-
3000	Process plant	360.3	204.3
4000	Site services and utilities	6.7	0.6
5000	Internal infrastructure	134.8	1.5
6000	External Infrastructure	38.5	-
	Total Direct Cost	**744.0**	**207.5**
7000	Common construction facilities and	62.5	14.7
8000	Engineering, procurement &	74.6	27.8
9100	Owner's costs	13.7	5.0
9900	Contingency	148.3	50.0
	Total Indirect Cost	**299.2**	**97.4**
	Total Capital Cost	**1043.1**	**304.9**

Note: Totals may not sum due to rounding

Sustaining capital costs consist of cost allocations for mining and the WSF and are summarized in Table 1-4.

Table 1-4: Sustaining Capital Cost

Description	Total Cost (US$M)
Total mining	64.2
Total internal infrastructure	54.8
Total Sustaining Capital Cost	**119.0**

1.18 Operating Cost Estimates

Operating costs are provided in Q4 2021 US$. The estimate uses AACE guidelines and is reported as an AACE Class 5 Order of Magnitude/Conceptual Study estimate with a -30% to +50% accuracy. Exchange rates used include US$1.00 = PENS/4.00, and US$1.00 = C$/1.29.

Mine 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 are fuel and labour.

Process operating cost were developed based on the production of copper concentrate at plant feed rates and waste handling strategy as per the production schedule. The estimate includes average annual processing costs for power, consumables, maintenance consumables and labour.

The G&A operating costs were estimated based on benchmarked data from similar projects in similar locations. Costs include camp operations, G&A personnel, off-site offices, contracts, and vehicle maintenance, as well as miscellaneous project costs.

The operating costs for the WSF include the mechanical spreading and compacting equipment, water truck for dust suppression. The conveyor and stacking equipment, and CWMP pump and pipeline to the plant and water treatment plant are located in the processing operating cost. The mechanical equipment operating costs were estimated based on benchmark data from similar projects in similar locations. The costs include fuel, equipment maintenance, and labor.

A summary of the individual components that make up the LOM operating costs is presented in Table 1-5.

Table 1-5: Summary of Operating Cost Estimate

On-site Costs	LOM		Phase 1		Phase 2	
	US$/t Milled	US$/lb Cu	US$/ t Milled	US$/lb Cu	US$/ t Milled	US$/lb Cu
G&A	0.70	0.10	1.17	0.13	0.64	0.10
Mining	2.62	0.38	6.57	0.77	2.08	0.31
Processing	4.76	0.69	5.54	0.65	4.65	0.70
WSF	0.12	0.02	0.23	0.03	0.11	0.02
Total On-site Costs	8.21	1.19	13.51	1.57	7.49	1.12

Off-site Costs	US$/t Milled	US$/lb Cu	US$/ t Milled	US$/lb Cu	US$/ t Milled	US$/lb Cu
Concentrate transport	1.65	0.24	2.06	0.24	1.60	0.24
Smelting and refining	1.51	0.22	1.88	0.22	1.46	0.22
Royalties	0.12	0.02	0.15	0.02	0.11	0.02
Total off-site costs	3.32	0.48	4.10	0.48	3.21	0.48
Credits (gold, silver)	(2.67)	(0.39)	(3.72)	(0.43)	(2.52)	(0.38)
Total Operating Cost	8.86	1.28	13.89	1.62	8.17	1.23

The C1 cost of US$1.28/lb of payable copper consists of mining, processing, site G&A, off-site treatment and refining, transport, and royalties net of by-product credits (gold and silver). The C3 cost of US$1.39/lb of payable copper includes the C1 costs plus sustaining capital, expansion capital, and closure costs.

1.19 Economic Analysis

The 2022 PEA is preliminary in nature and is partly based on 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 2022 PEA based on these Mineral Resources will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

The Project was evaluated using a discounted cash flow (DCF) analysis based on an 8% discount rate. Cash inflows consisted of annual revenue projections. Cash outflows consisted of capital expenditures, including pre-production costs; operating costs; taxes; and royalties. These were subtracted from the inflows to arrive at the annual cash flow projections. Cash flows were taken to occur at the mid-point of each period. The economic analysis was run on a constant dollar basis with no inflation.

Key assumptions included:

- Construction period of three years;
- Mine life of 28 years (last year is a partial year);
- 100% ownership;
- 0.5% NSR royalty;
- All metal products are sold in the same year they are produced;
- Project revenue is derived from the sale of copper concentrate inclusive of gold and silver credits.

The taxes assumed in the economic analysis include:

- Peruvian corporate income tax of 29.5% (plus 2% during the term of an assumed stability agreement);

- Employee profit sharing of 8% of taxable income; and
- Mining taxes for the exploitation of mineral resources based on the operating profit (mining royalties on a sliding scale of 1% to 12% with a minimum of 1% of sales and special mining tax on a sliding scale of 2% to 8.4%).

Under the Peruvian mining regime, local mining companies may enter into stability agreements of guarantees and investment promotion measures that guarantee the following for 10, 12, or 15 years. A 15-year term was assumed; the requirements for this term length are either mine 15,000–20,000 t/d mineralization, or to invest a minimum amount of US$500 M in a project.

The economic analysis was performed assuming an 8% discount rate. The pre-tax NPV discounted at 8% is US$2,022.5 M; the internal rate of return (IRR) is 21.6%, and payback period is 6.1 years. On a post-tax basis, the NPV discounted at 8% is US$1,010.3 M; the IRR is 16.3%, and payback period is 7.1 years. The cashflow results are summarized in Table 1-6.

Table 1-6: Economic Analysis Summary Table

General	LOM Total / Avg.
Copper price (US$/lb)	3.50
Gold price (US$/oz)	1,650
Silver price (US$/oz)	21.50
Mine life (years)	28
Total waste tonnes mined (Mt)	465
Total mill feed tonnes (Mt)	703
Strip ratio	0.66
Production	**LOM Total / Avg.**
Mill head grade - copper (%)	0.38
Mill head grade - gold (g/t)	0.07
Mill head grade - silver (g/t)	1.69
Mill recovery rate - copper (%)	88.12
Mill recovery rate - gold (%)	64.67
Mill recovery rate - silver (%)	57.18
Total mill recovered - copper (mlb)	5,225
Total mill recovered - gold (koz)	960
Total mill recovered - silver (koz)	21,889
Total average annual payable production - copper (mlb)	173
Total average annual payable production - gold (koz)	31
Total average annual payable production - silver (koz)	704

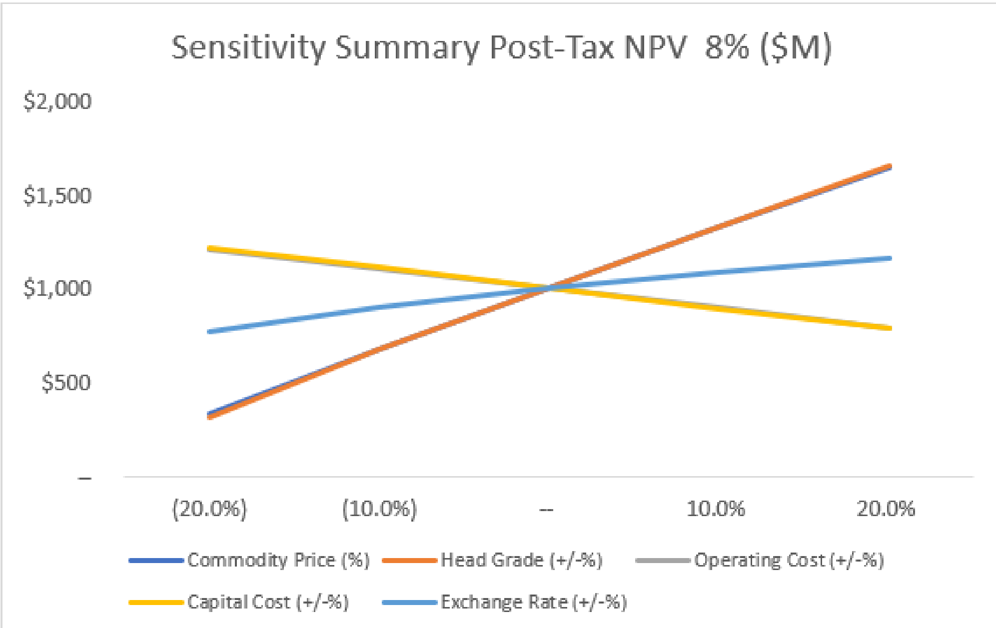
On-site Costs	LOM Total / Avg.	
	US$/t processed	US$/lb Cu
Mining	2.62	0.38
Processing	4.76	0.69
WSF	0.12	0.02
General & administration	0.70	0.10
Total on-site costs	8.21	1.19
Off-site Costs	**LOM Total / Avg.**	
	US$/t concentrate	**US$/lb Cu**
Concentrate transport	$127.47	$0.24
Smelting & refining	$119.10	$0.22
Total off-site costs	$246.57	$0.46
By-product credits (gold, silver)	-	($0.39)
Cash Costs	**LOM Total**	
C1 Cost * (US$/lb Cu)	$1.28	
C3 Cost ** (US$/lb Cu)	$1.39	
Capital Costs	**LOM Total**	
Initial capital (US$M)	$1,043	
Expansion capital (US$M)	$305	
Sustaining capital (US$M)	$119	
Closure costs (US$M)	$104	
Financials	**Pre-Tax**	**Post-Tax**
NPV (8%) (US$M)	$2,023	$1,010
IRR (%)	21.6%	16.3%
Payback (years)	6.1	7.1

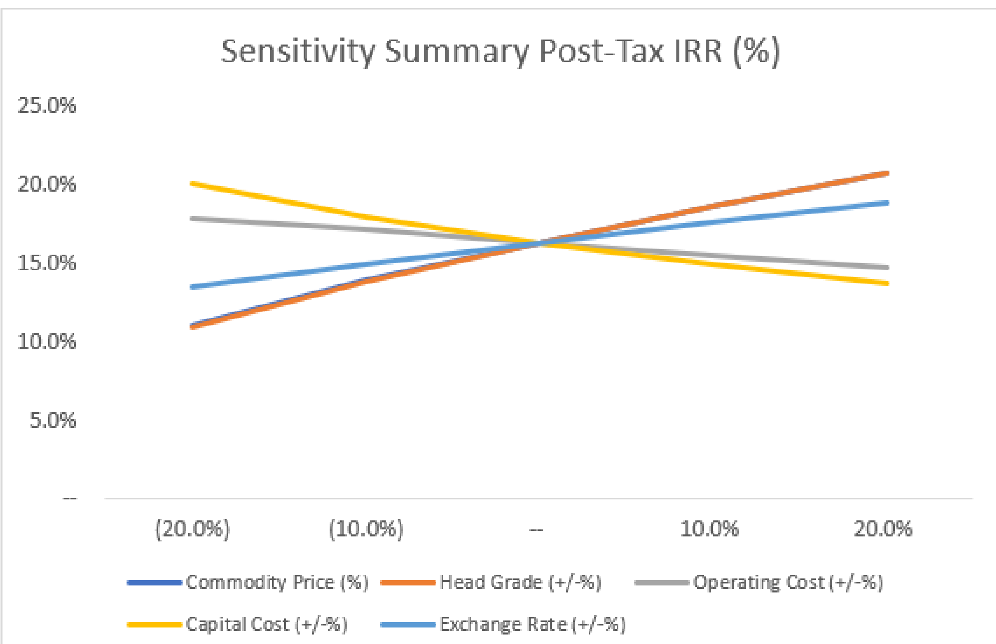
Note: * C1 costs consist of mining , processing , site G&A, off-site treatment & refining, transport, and royalties net of by-product credits (gold and silver).
** C3 costs consist of C1 plus sustaining capital, expansion capital, and closure.

1.20 Sensitivity Analysis

A sensitivity analysis was conducted on the base case pre-tax and post-tax NPV and IRR of the Project, using the following variables: commodity prices, discount rate, total operating cost, total capital cost, head grade, and foreign exchange rate. The Project is most sensitive to changes in commodity price and head grade. In order of decreasing sensitivity, the project is less sensitive to changes in total capital cost, total operating cost, and foreign exchange rate. Figure 1-1 shows the post-tax sensitivities to NPV and IRR, and Figure 1-2 shows the pre-tax sensitivities to the same variables.

Figure 1-1: Post-Tax NPV and IRR Sensitivity Results



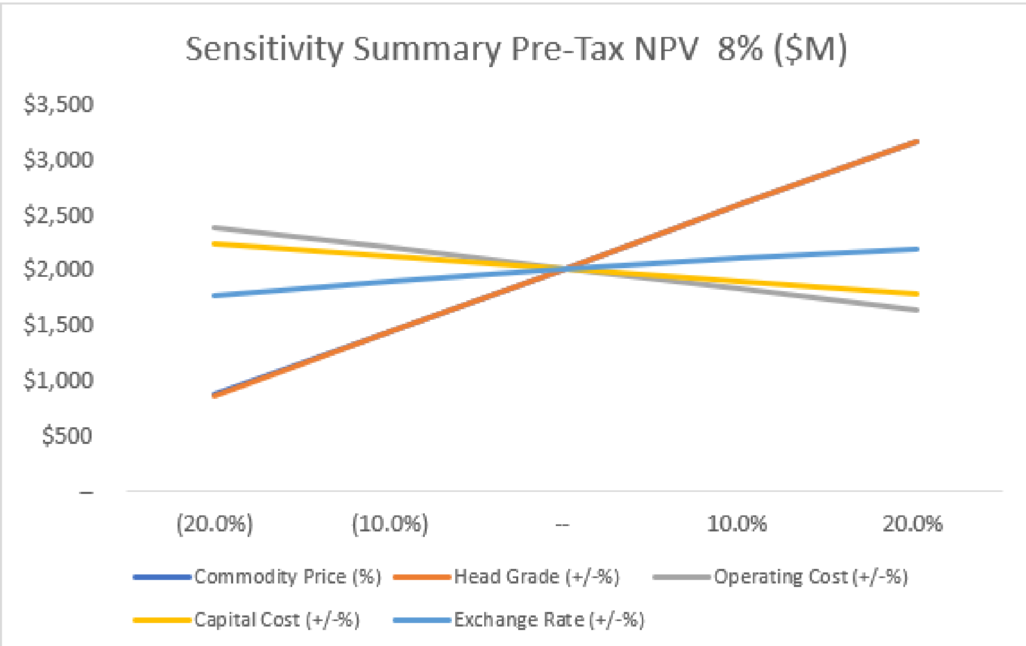


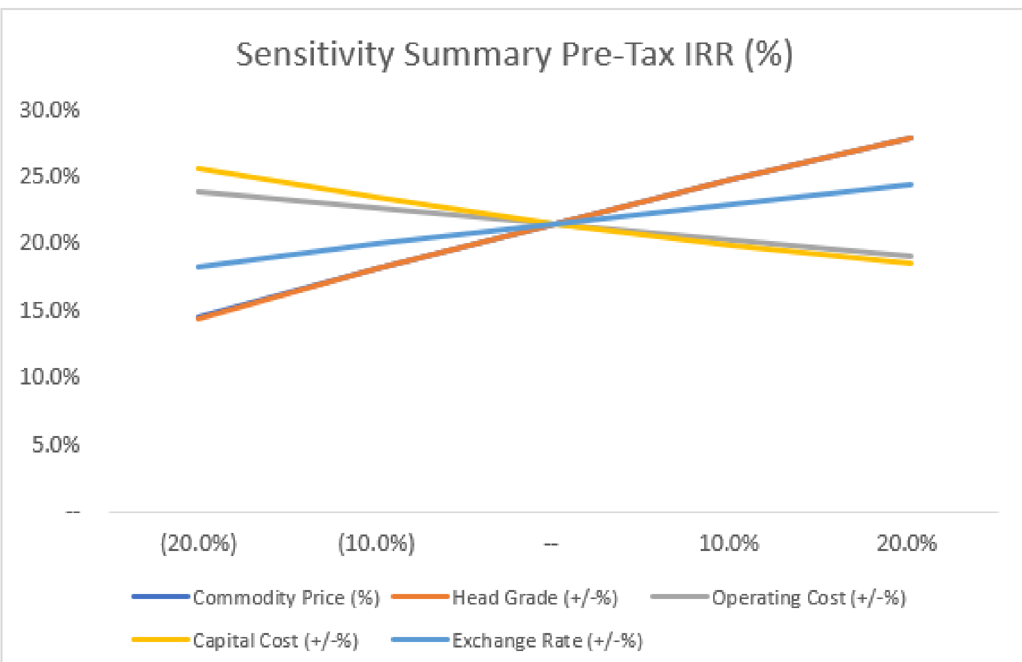
Notes: Figure prepared by Ausenco, 2022.
For the chart titled "Sensitivity Summary Post-Tax NPV 8% ($M)", chart lines overlap for:

- commodity price and head grade, and
- operating cost and capital cost.

For the chart titled "Sensitivity Summary Post-Tax IRR (%)", chart lines overlap for commodity price and head grade.

Figure 1-2: Pre-Tax NPV and IRR Sensitivity Results





Notes: Figure prepared by Ausenco, 2022.
For the chart titled “Sensitivity Summary Pre-Tax NPV 8% ($M)”, chart lines overlap for commodity price and head grade.
For the chart titled “Sensitivity Summary Pre-Tax IRR (%)”, chart lines overlap for commodity price and head grade.

1.21 Risks and Opportunities

1.21.1 Risks

Risks that may affect the Mineral Resource estimate specifically were provided in Section 1.10. Risks that may affect the 2022 PEA that is based on that estimate include:

- Geotechnical design of +900 m pit highwall: the ultimate pit highwall is over 900 m in vertical height, making the proposed Cañariaco Norte pit one of the world's deepest. The current study design basis for the overall pit and mine bench slope angles is pending completion of a geotechnical drilling program that was halted in 2013. The program is required to be completed to provide additional information on the rock quality in the pit wall areas. The current design is based on assessment of scoping level information, visual inspection of available core, and limited physical testing of available material. If future geotechnical investigations result in shallower overall pit slope angles, then the stripping ratio will increase and have a negative impact on the project mining costs;

- Main access road routing: the conceptual design of the main access road is based on low-quality contour data. This may lead to a less-than-optimum design with a risk that the road may not be constructible in some sections. Aerial or land reconnaissance followed by more detailed geotechnical investigations along the proposed road routing is recommended to decrease the risk in the design of this road;

- Landslides: 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 valley slopes will be required to assess the severity of this risk and what mitigation can be undertaken to reduce such risk;

- Coarse mill feed material stockpile and dusting: the open-air design of the coarse ore stockpile is typical of many operating mines; however, it may result in non-compliant dust loading during dry periods. Dust suppression or a stockpile cover may be required, which will incur additional capital/operating costs for mitigation;

- There is insufficient data too provide a proper 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 any ARD risk;

- There is no geotechnical information of the WSF foundation. There may be unknow 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 WSF will be required to assess any potential risk to the facilities and develop any potential mitigation measures;

- Inability to meet filtered tailings moisture content that could create trafficability issues for the conveyor system. An additional filtering test program should be completed to validate that the filter design moisture content can be achieved;

- 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.;

- Arsenic in concentrate: Arsenic content in the copper concentrate will attract penalties. Further marketing studies are required to confirm what penalties may be imposed on the low-grade copper concentrate;

- Delivery of long-lead capital equipment: market conditions may increase delivery schedules for long-lead capital equipment items. Equipment may need to be secured and purchased earlier than anticipated. This may result in increases to the capital costs as envisaged in the 2022 PEA;

1.21.2 Opportunities

Opportunities include:

- Mineral resource estimation: additional drilling should be undertaken in the vicinity of Cañariaco Norte with the objective of providing sufficient data to support potential upgrading of mineralization that is currently classified as Inferred to higher confidence categories. The potential exists to include the mineralization at the Cañariaco Sur deposit, which is also currently classified as Inferred, with further drilling, in more detailed studies. The Quebrada Verde prospect retains exploration potential;

- Mine plan: there is potential to optimize the mine plan in future more detailed studies. This may potentially result in a decrease in the forecast Project payback period, and an increase in the IRR;

- Flowsheet: there is potential to optimize the process flowsheet plan in future more detailed studies. This may potentially result in a decreased capital the forecasted Project payback period, and an increase in the IRR;

- Geo-metallurgy/ metallurgy: increased testwork and database has the potential to optimize the process parameters in future more detailed studies. This may potentially result in an increase of payables recoveries and/ or concentrate quality and/ or decreased deleterious element recoveries. This may potentially result in increase revenue, NPV and IRR;

- WSF there is a potential to optimize the deposition program in future studies. This may result in decrease capital, sustaining capital, and operating costs.

1.22 Interpretation and Conclusions

Under the assumptions and parameters presented in this Report, the 2022 PEA shows a positive economic outcome. Additional mining and technical studies are warranted.

1.23 Recommendations

A two-phase work program is recommended.

Recommendations Phase 1 will consist of a geotechnical drill program to provide additional geotechnical support for pit designs, and an exploration program to better delineate the mineralization at Cañariaco Sur and provide core for an initial metallurgical testwork program. The recommended budget for this program is approximately US$6 M.

A portion of the second recommendations phase is dependent on the drill campaign for provision of fresh drill core from Cañariaco Sur; however, the majority of the recommended studies can be conducted concurrently with the Cañariaco Sur drill program. The second phase includes updated Mineral Resource estimates for Cañariaco Norte and Cañariaco Sur. Metallurgical testwork consisting of variability tests for Cañariaco Norte is recommended using core in cold storage from the 2013 metallurgical drilling program. Initial metallurgical testwork on Cañariaco Sur, using the core from the first work phase is proposed. Other recommendations in the second work phase that can be conducted concurrently with the first phase include additional mining and technical studies for support of process plant, infrastructure, and WSF and water management designs. A CIRA must be obtained for all areas that are planned to or could host infrastructure. The recommended budget for this program is about US$2.7 M.

2 INTRODUCTION

2.1 Introduction

Candente Copper Corp. (Candente Copper) requested Ausenco Engineering Canada Inc. (Ausenco) to prepare a technical report (the Report) on the results of a preliminary economic assessment (the 2022 PEA) on the Cañariaco Norte deposit within the Cañariaco Project in Peru (Figure 2-1).

2.2 Terms of Reference

The Report was prepared to support disclosures in Candente Copper's news release dated 8 February 2022, entitled "Candente Copper Announces Positive PEA Results for the Cañariaco Copper Project".

The Report was amended to address a typographical error in one of the Certificates of Qualified Person that was used as a Report date and signature page. No other changes were made to the Report.

Units used in the report are metric units unless otherwise noted. Monetary units are in United States dollars (US$) unless otherwise stated. Mineral Resources and Mineral Reserves are classified using the 2014 edition of the Canadian Institute of Mining and Metallurgy (CIM) Definition Standards for Mineral Resources and Mineral Reserves (the 2014 CIM Definition Standards). Mineral Resource and Mineral Reserve estimates were prepared using the guidance set out in the 2019 edition of the CIM's Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (2019 Guidelines). The Report uses Canadian English.

2.3 Qualified Persons

The following 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 43-101F1:

- Mr. Scott Elfen, P.E., Global Lead Geotechnical and Civil Services, Ausenco Engineering Canada Inc.;
- Mr. Kevin Murray, P.Eng., Manager Process Engineering, Ausenco Engineering Canada Inc.;
- Mr. Scott Weston, P.Geo., Vice President of Business Development, Hemmera Envirochem Inc.;
- Mr. Jay Melnyk, P.Eng. Principal Mining Engineer, AGP Mining Consultants Inc.;
- Mr. David Thomas, P.Geo., Principal, DKT Geosolutions Inc.

Figure 2-1: Project Location Map

Note: Figure prepared by Candente Copper, 2011

2.4 Site Visits and Scope of Personal Inspection

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 Candente Copper staff and reviewed geological interpretations in sections and plans.

2.5 Effective Dates

There are several effective dates pertinent to information used in the Report, as follows:

- Effective date of the Mineral Resource estimate for Cañariaco Norte: 27 January 2022;
- Effective date of the Mineral Resource estimate for Cañariaco Sur: 27 January 2022;
- Effective date of the financial analysis that supports the 2022 PEA: 8 February 2022.

The overall Report effective date for the Report is taken to be the date of the financial analysis that supports the 2022 PEA and is 8 February 2022.

2.6 Information Sources and References

Reports and documents listed in Section 3 and Section 27 of this Report were used to support preparation of the Report. Additional information was provided by Candente Copper personnel as requested. Supplemental information was also provided to the QPs by third-party consultants retained by Candente Copper in their areas of expertise.

Ms Joanne Freeze P.Geo., CEO of Candente Copper, has visited the site many times during the Project history; the most recent visit being in December, 2019. Ms Freeze provided information to the QPs on the site access conditions, community relations, and care-and-maintenance status of the exploration-related Project infrastructure, such as the camp.

Candente Copper maintains a small, manned camp in the Cañariaco Norte area that is staffed by local employees. These employees also provided information to the QPs on the seasonal site access conditions, general site conditions, and community relations.

2.7 Previous Technical Reports

Candente Copper has previously filed the following technical reports on the Project:

- 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;

- Bonson, C., Nowak, M., Doerksen, G., Johnston, A., and van Egmond, R., 2008: Technical Report Cañariaco Norte Project, Department of Lambayeque, Peru: technical report prepared by SRK Consulting Ltd for Candente Resource Corp., effective date 11 July 2008;

- Stone, D.M.R., Godden, S., Van Egmond, R., and Tosney, J.R., 2007: Updated Technical Report and Preliminary Economic Assessment on the Cañariaco Norte Porphyry Copper Project Starter Pit Option: technical report prepared by Minefill Services Inc. for Candente Resource Corp., effective date 18 April 2007;

- Collins, J., McCrea, J., and Rokosh, J., 2006: Cañariaco Copper Project, Peru, Preliminary Assessment and Economic Evaluation Report: technical report prepared by Merit Consultants for Candente Resource Corp., effective date 28 June 2006;

- 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.

2.8 Unit and Name Abbreviations

Table 2-1: 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
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.
IDF	inflow design flood
INGEMMET	Peruvian Instituto Geológico, Minero y Metalúrgico
IP	induced polarization
LG	Lerchs-Grossmann
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
NXWL	NXWL core size
NWA	National Water Authority Peru (Autoridad Nacional de Agua)
QP	Qualified Person
PLS	programmable logic controllers
PMA	particle mineralogical analysis
PWPT	potable water treatment plant
RQD	rock quality designation
ROM	run of mine
RF	revenue factor
SUNARP	Superintendencia Nacional de los Registros Publicos (Peru)
UIT	*Unidad Impositiva Tributaria* (imposed fiscal unit)
WMT	wet metric tonne
WSF	co-disposal waste rock and tailings storage facility
WTP	water treatment plant

Table 2-2: Unit of Measurement

Unit	Description
PEN S/	Peruvian nuevo sol
US$	United States dollars
°C	degrees Celsius
cm	centimetre
g	gram
gr/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 level
Min	minute
Mm^3	Million cubic metres
Mt/y	million tonnes per year
t	tonnes
t/y	tonnes per year
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 licence, 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 Candente Copper and legal experts retained by Candente 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 used in Section 4 of the Report and is used to support the Mineral Resource estimate in Section14 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 Candente 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 used in Section 20 of the Report and in support of the Mineral Resource estimate in Section 14 and the economic analysis in Section 22.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Introduction 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 approximately 120 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 10,189.50 ha in 15 concessions. Three porphyry copper targets, Cañariaco Norte, Cañariaco Sur, and Quebrada Verde, have been identified. Candente 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 Property Agreements

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

4.4 Mineral Tenure

The government of Peru auctioned the right to explore the Project in 2001 under "Public Auction (Bid) PRI-51-2000, Private Investment Promotion – Mining Prospects". Candente 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. Candente has since acquired additional claims. Some of the new claims are held by Candente Copper's other subsidiary, Cobriza Metals Peru S.A. ("CZA") and the most recent seven claims were acquired by Peru Fortescue S.A.C. ("F") and are currently being transferred to CCPSA. The current mineral tenure is summarized in Table 4-1 and the claims are shown in Figure 4-1.

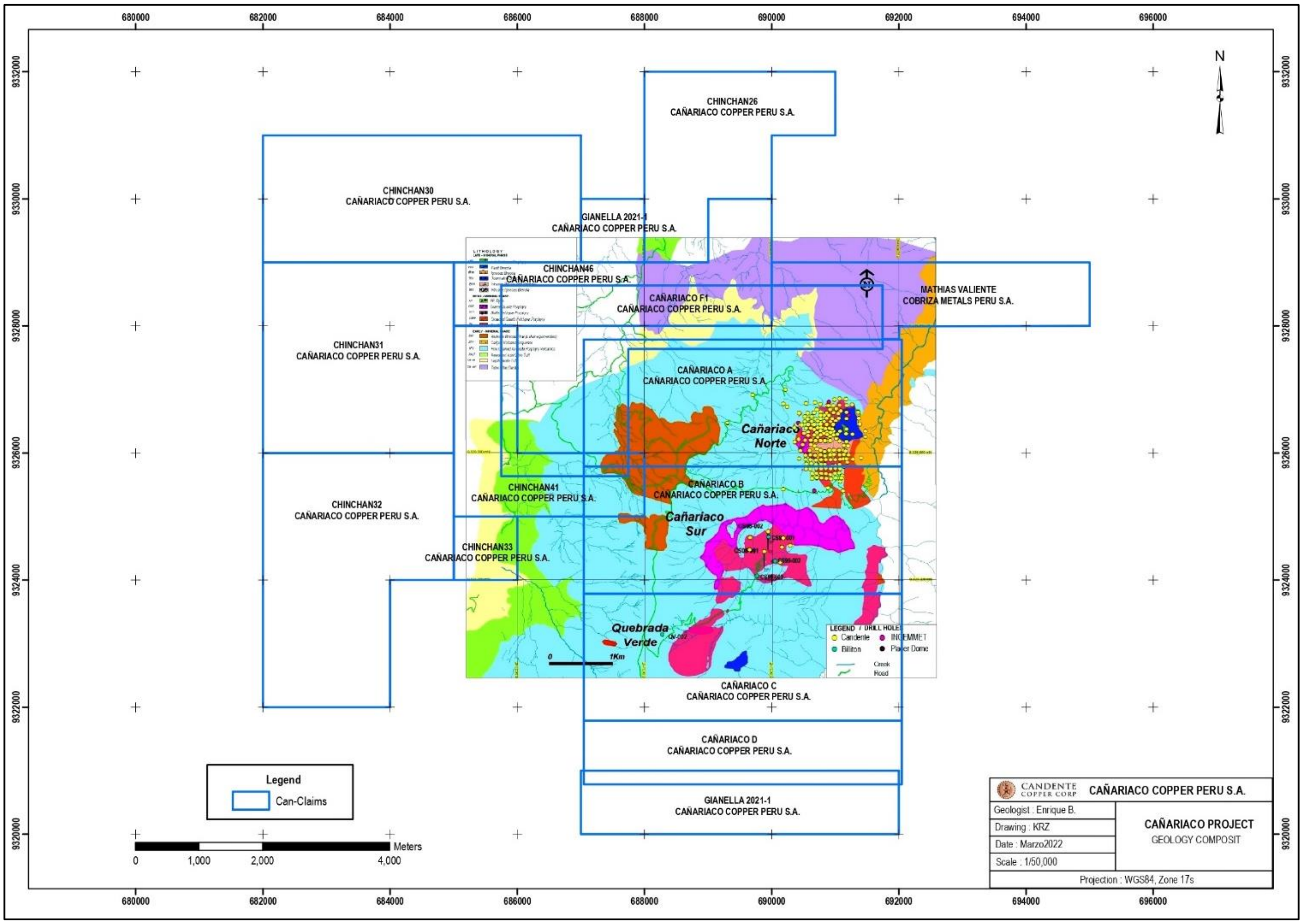
Table 4-1: Mineral Tenure Table

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	2021
2	Cañariaco B	15000005Y02	CCPSA	1992	1,000	3,000	20,000	2021
3	Cañariaco C	15000006Y01	CCPSA	1992	1,000	3,000	20,000	2021
4	Cañariaco D	15000007Y01	CCPSA	1992	500	1,500	10,000	2021
5	Cañariaco F1	01-00312-04	CCPSA	2004	790	2,369	PS 67,897	2021
6	Mathias Valiente	01-01981-20	CZA	App 2020	600	1,800	N/A	2021
7	Gianella 2021-1	01-02100-21	CCPSA	App 2021	100	300	N/A	2021

No.	Claim Name	Code	Holder	Granted	Hectares	Validity Fee* (US$)	Penalty (US$ or Peru Sole)	Years Penalty Owed
8	Gianella 2021-2	01-02101-21	CCPSA	App 2021	500	1,500	N/A	2021
9	Chinchan26	01-00234-20	F to CCPSA	2022	600	1,800	N/A	2021
10	Chinchan30	01-00205-20	F to CCPSA	2022	1,000	3,000	N/A	2021
11	Chinchan31	01-00217-20	F to CCPSA	2021	900	2,700	N/A	2021
12	Chinchan32	01-00204-20	F to CCPSA	2022	1,000	3,000	N/A	2021
13	Chinchan33	01-00233-20	F to CCPSA	2021	100	300	N/A	2021
14	Chinchan41	01-00188-20	F to CCPSA	2021	500	1,500	N/A	2021
15	Chinchan46	01-00219-20	F to CCPSA	2021	600	1,800	N/A	2021

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



Note: Figure prepared by Candente 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 Candente 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 2021 are currently owing and are to be paid June 30, 2022.

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 years 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 Candente 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.

4.5 Surface Rights

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. 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 rights. 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 usually increases with the level of impact on both the land and 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 therefor 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 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 of the communities to allow CCPSA to make agreements for non-invasive activities directly with the posesionarios but all invasive activities (drilling, trenching etc.) require an agreement with the community.

The Company currently has agreements with the *posesionarios* holding rights to lands where activities are currently underway 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 in 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 activities 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 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 EIA baseline study has recently been completed for drilling permits which are being re-applied for from MINEM. Given this, a new servidumbre agreement for drilling is under discussions with the community and the posesionarios. The Chamber of Commerce of Lambayeque may act as the presiding Institution for the obligations of the new agreement as they have recently signed a new Memorandum of Commitment for the evaluation, monitoring and support in development projects for the Community.

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

Candente 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 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 Candente Copper as required by Peruvian law when trenching, drilling, or other major work is performed. 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. Additional permits will be 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.

4.9 Environmental Considerations

AMEC, on behalf of Candente Copper, commenced environmental baseline studies and monitoring in 2007. A social and environmental impact assessment study framework was prepared but will need updating. Studies included determining the waste rock and mineralization geochemical characteristics of, potential water supply sources, and reviewing social and community settings.

Baseline data collected in 2007 indicated that a diverse flora and fauna were present in the Project area. AMEC noted that in many areas of Peru biodiversity assessments are not yet complete and therefore it is not uncommon for detailed environmental impact investigations to identify new species. Candente Copper instigated two conservation projects that encompass the relocation of protected orchids found at the site and the identification of an area for conservation and environmental management.

At this stage of the Project, environmental liabilities are limited to those expected for an exploration-stage project and include drill pads and access roads.

4.10 Social License Considerations

Candente Copper received a certificate 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.

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

- 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 Comunidad Campesina San Juan Bautista de Cañaris. Access has been negotiated with the community as required for exploration to date;
- 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 Candente 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).

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE, AND PHYSIOGRAPHY

5.1 Accessibility

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

Figure 5-1: Existing Cañariaco Property Access Routes and Regional Infrastructure

Note: Figure prepared by Candente 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. 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	21,660	Paved road
Pucara – Cañaris	2,447	Unimproved road
Cañaris – Cañariaco	631	Horse trail/New Mountain Road
Total	**294**	
Chiclayo – H. da. Batan Grande	50	Paved road
H. da. Batan Grande – Incahuasi – Uyllurpampa	687	Secondary gravel road
Uyllurpampa – Road Terminus	30	Drivable track
Road Terminus – Cañariaco	2	4x4 access road to camp
Total	**150**	
Chiclayo – Motupe	81	Paved road
Motupe – Road Terminus	97	Unimproved road
Road Terminus – Cañariaco	2	4x4 access road to camp
Total	**180**	

5.2 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 about 3°C to 20°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 2022 PEA have been scheduled year-round, 24 h/d, with no allowance for shutdowns due to weather-related events.

5.3 Local Resources and Infrastructure

Peru has a long history of mining. Consequently, mining professionals and machine operators are generally available in most population centres. Candente 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 usually contracted out of Lima.

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

The 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 2022 PEA is provided in Section 18.

5.4 Physiography

The Cañariaco Norte Project area lies along the eastern side of the continental divide at elevations ranging between 2,700 and 3,600 masl. The topography varies from steep incised valleys at lower elevations to open grassy highlands (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 zone.

Figure 5-2: View of Cañariaco Norte Project Mineral Resource and Pit Area – Looking West

Note: Figure prepared by Candente 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 in the forested valleys.

5.5 Comments on Accessibility, Climate, Local 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 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 INGEMMET, in cooperation with a British Geological Survey team. This work identified a copper anomaly A-2. Subsequently, from 1971 to 1974, INGEMMET undertook a detailed geochemical study of the Río Cañariaco valley, which delineated 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). 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) took up an option 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. Candente Copper, through its wholly owned Peruvian subsidiary, Exploraciones Milenio (subsequently CCPSA), acquired 100% ownership of the Project in February 2002. Since that date, Candente Copper has completed geological mapping, prospecting, IP/resistivity and ground magnetic geophysical surveys, rock chip sampling, petrographic studies, bulk sampling and drilling for metallurgical testwork, 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 Candente 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 erratic, and a substantial increase in the extent of the sulphide mineralization warranted a more robust metallurgical process such as a conventional process plant consisting of 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. Candente 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 geotechnical drilling 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 lack of interest in the equity markets which affected Candente 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	7 Geological mapping at 1:1,000 scale at Cañariaco Norte; preliminary geological mapping, scale not known, at Cañariaco Sur 8 Soil and rock chip sampling over an area of about 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 9 Trenching (2,200 m) and pitting (80 test pits) at Cañariaco Norte and Cañariaco Sur 10 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 11 Three core holes (853.91 m) at Cañariaco Norte 12 Petrographic studies on selected drill core samples 13 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	Candente 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 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, 72,189,7m 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 - Yaku conducted environmental work for both OEFA concerns re drill platform closures and a new baseline for an EIAsd was conducted; • Currently focused on obtaining authorizations for additional exploration drilling in Cañariaco Sur and initial drilling in Quebrada Verde; • Desktop Engineering Studies by Ausenco Feb to June 2021; • Current updated PEA Engineering Studies for a smaller start-up by Ausenco underway since September 2021. • In 2020, the progress of the Project began again with the Yaku Environmental work, culminating in the Baseline for a new EIAsd for drilling permits

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

A regional geological plan for the project area is shown in 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.

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 7-2.

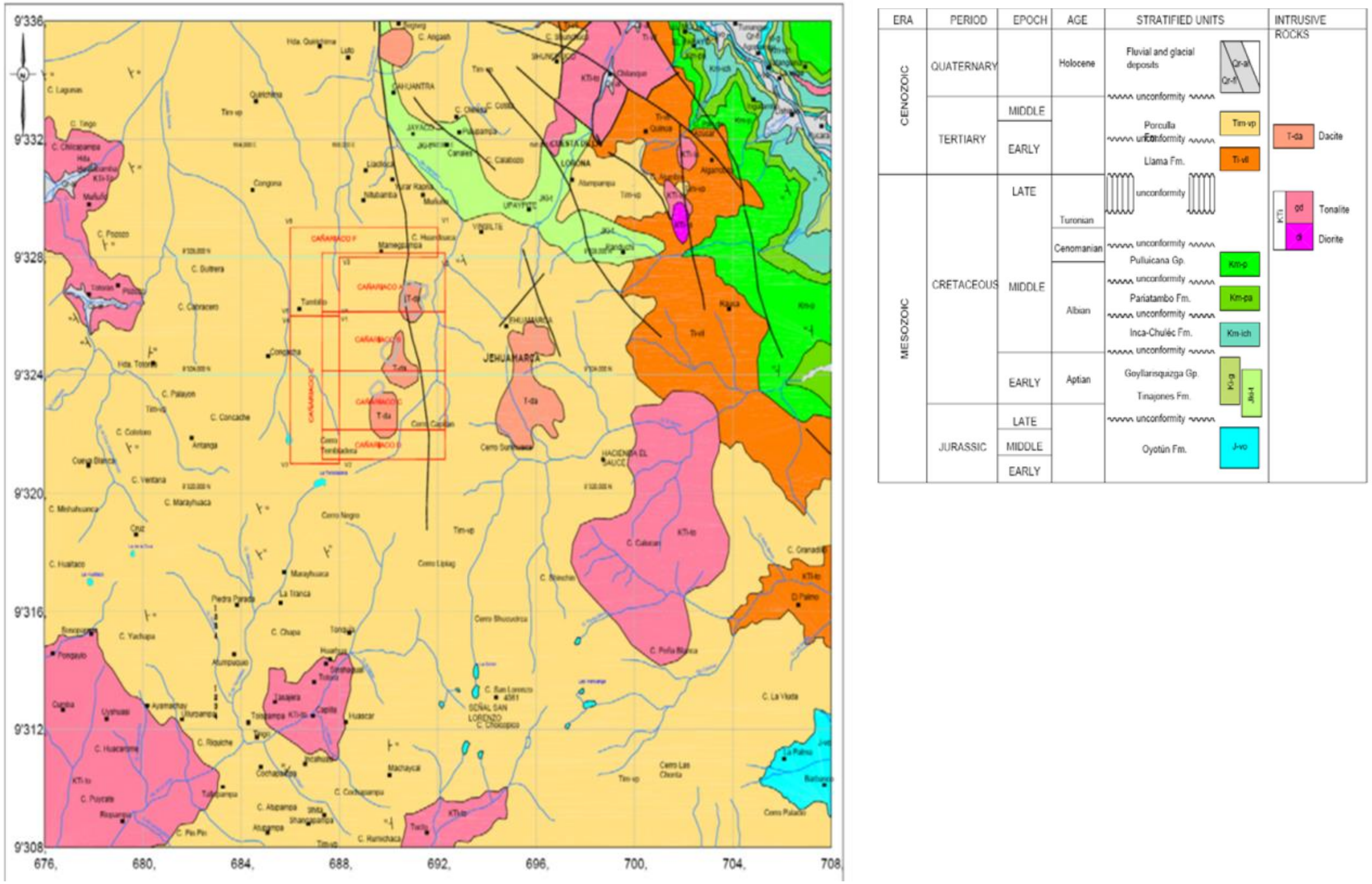
In the Early Tertiary, volcanic units of the Llama and Porculla Formations of the Calipuy Group were erupted, followed by uplift and erosion. Renewed volcanic activity commenced with the deposition of volcanic rocks of the Huambos Formation during the Late Tertiary. Porphyry stocks, breccias, and dykes 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 centred on, or close to, major fault intersections, suggesting that the emplacement of the intrusive complex was localized by fault intersections (Murphy, 2004).

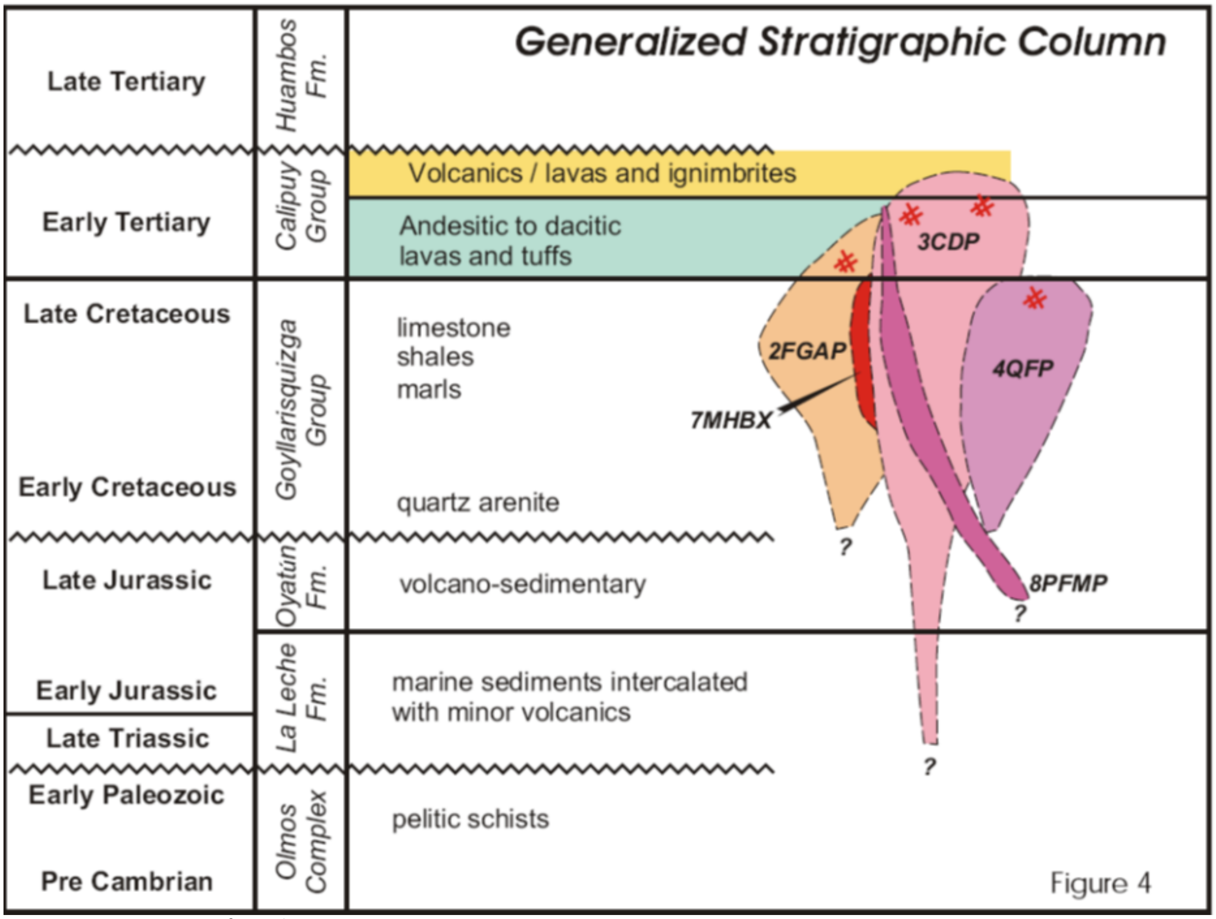
Faults that have been identified at a regional scale consist of two, parallel, long range, northwest–southeast-trending district-scale faults. One bisects the Project area, and the second fault is approximately 7 km to the northeast. The northwest–southeast faults have a probable conjugate northeast–southwest 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: Regional Geology Map



Note: Figure courtesy of Candente Copper Corp., 2011. 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 7-2: Regional Stratigraphic Column



Note: Figure courtesy of Candente Copper, 2011.

7.2 Project Geology

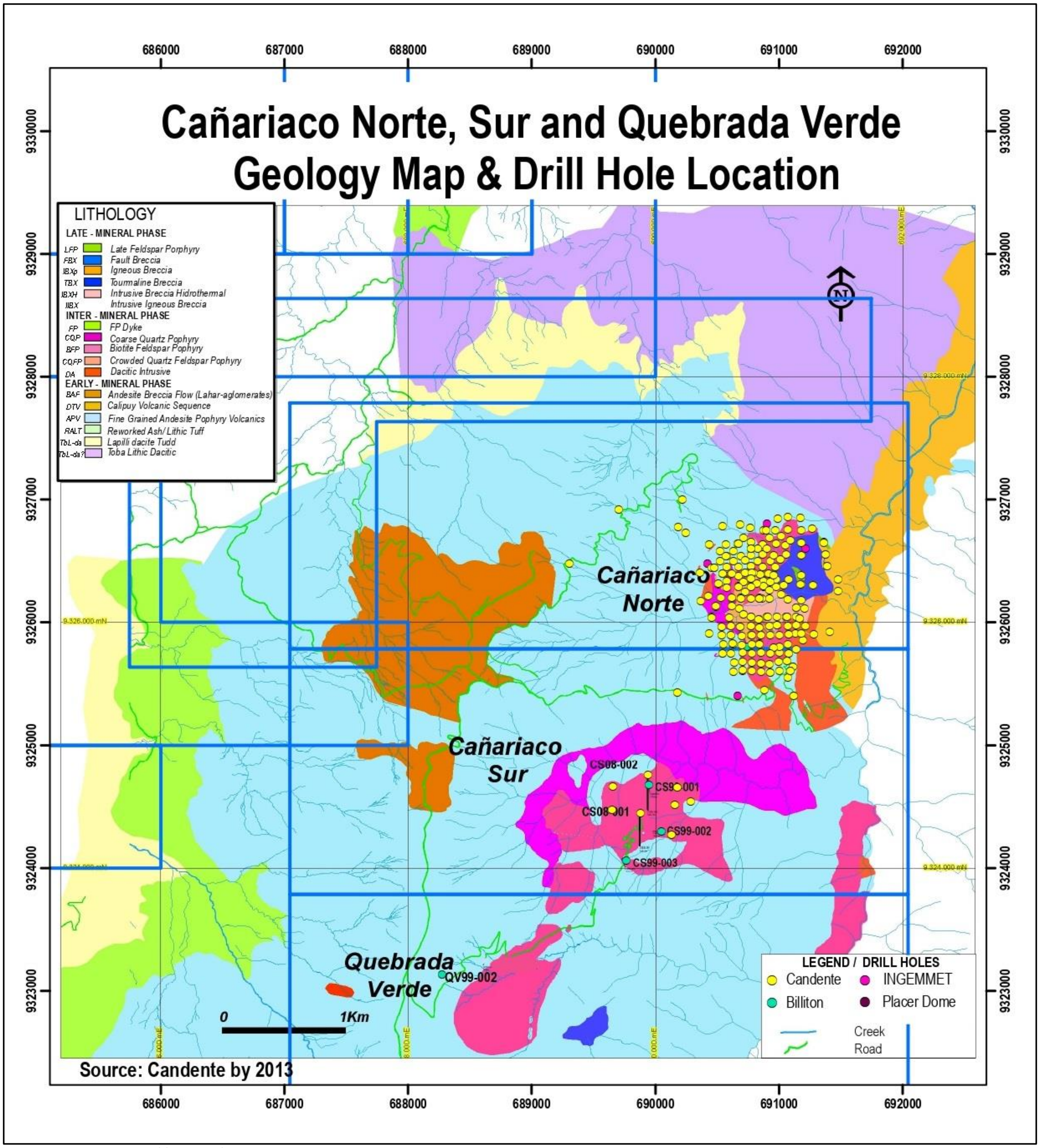
The Project covers a north-easterly 4–5 km long structural trend that hosts three porphyry centres, Cañariaco Norte, Cañariaco Sur and Quebrada Verde, as delineated by geophysics, geochemistry and geological mapping and drilling.

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.

A geology map is provided in Figure 7-3.

Figure 7-3: Project Geology Map



Note: Figure prepared by Candente Copper, 2013.Deposit Descriptions

7.3 Cañariaco Norte

7.3.1 Deposit Descriptions

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 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-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).

7.3.1.1 Lithologies

Table 7-1 presents the Cañariaco Norte stratigraphy.

Table 7-1: Cañariaco Norte Stratigraphy

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.

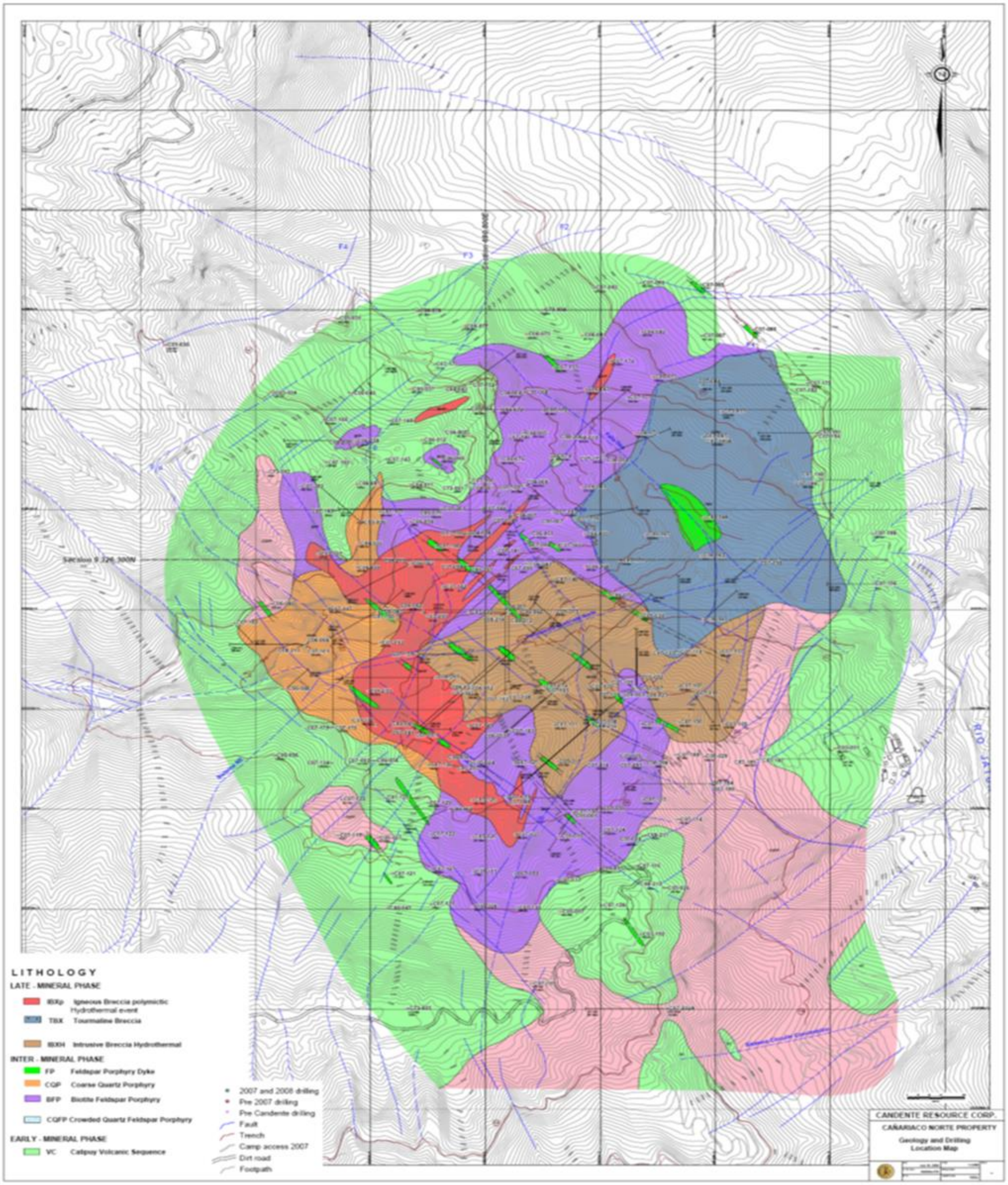
Mineralization occurs primarily as disseminations and in fractures, sulphide and quartz veins, faults, and breccias. Fracture density is the single most important factor influencing copper grades and alteration intensity, although breccias and faults can locally be important.

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.

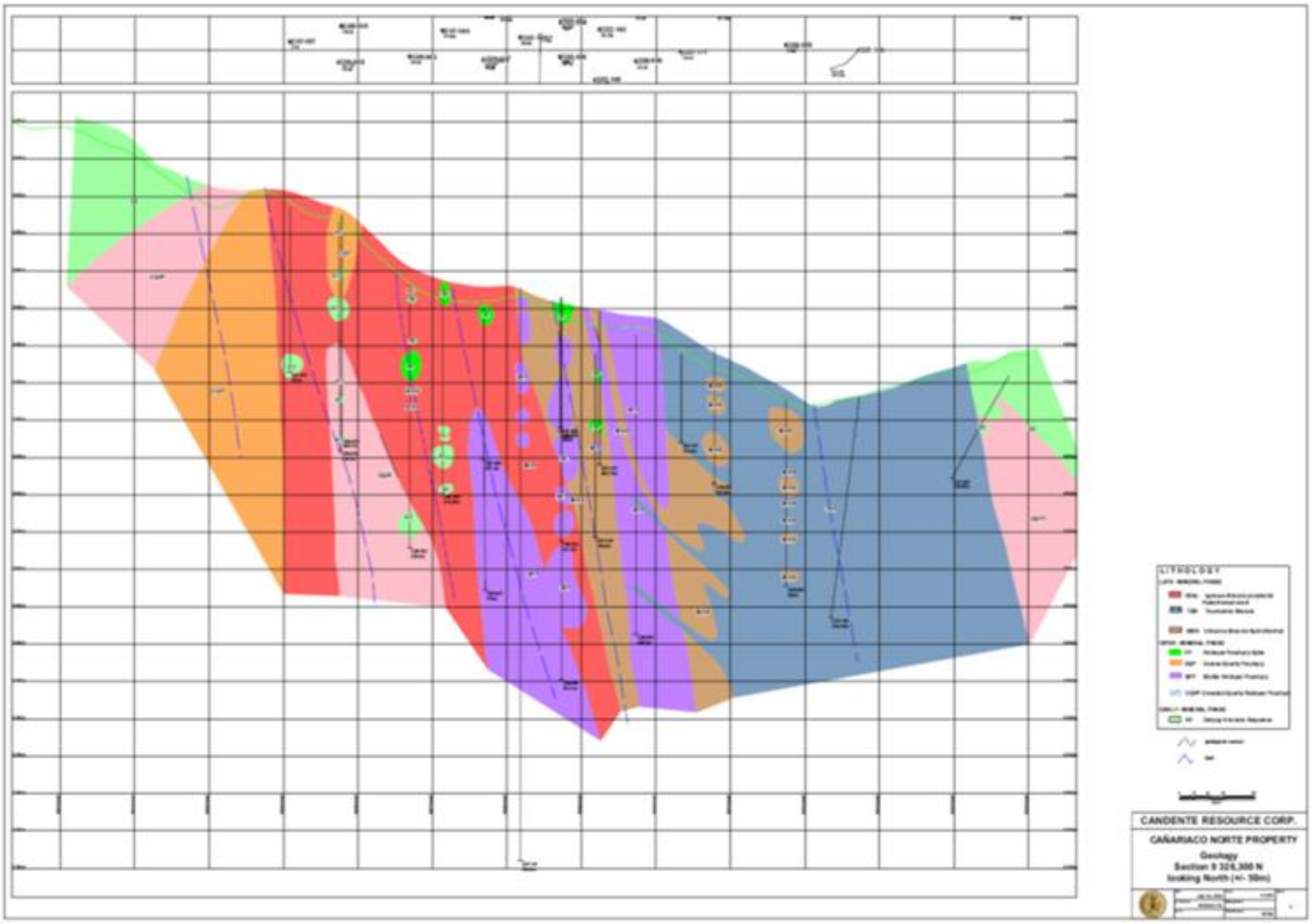
A geological map for the deposit is presented in Figure 7-4. Figure 7-5 and Figure 7-6 are drill sections through the deposit.

Figure 7-4: Cañariaco Norte Geology Map



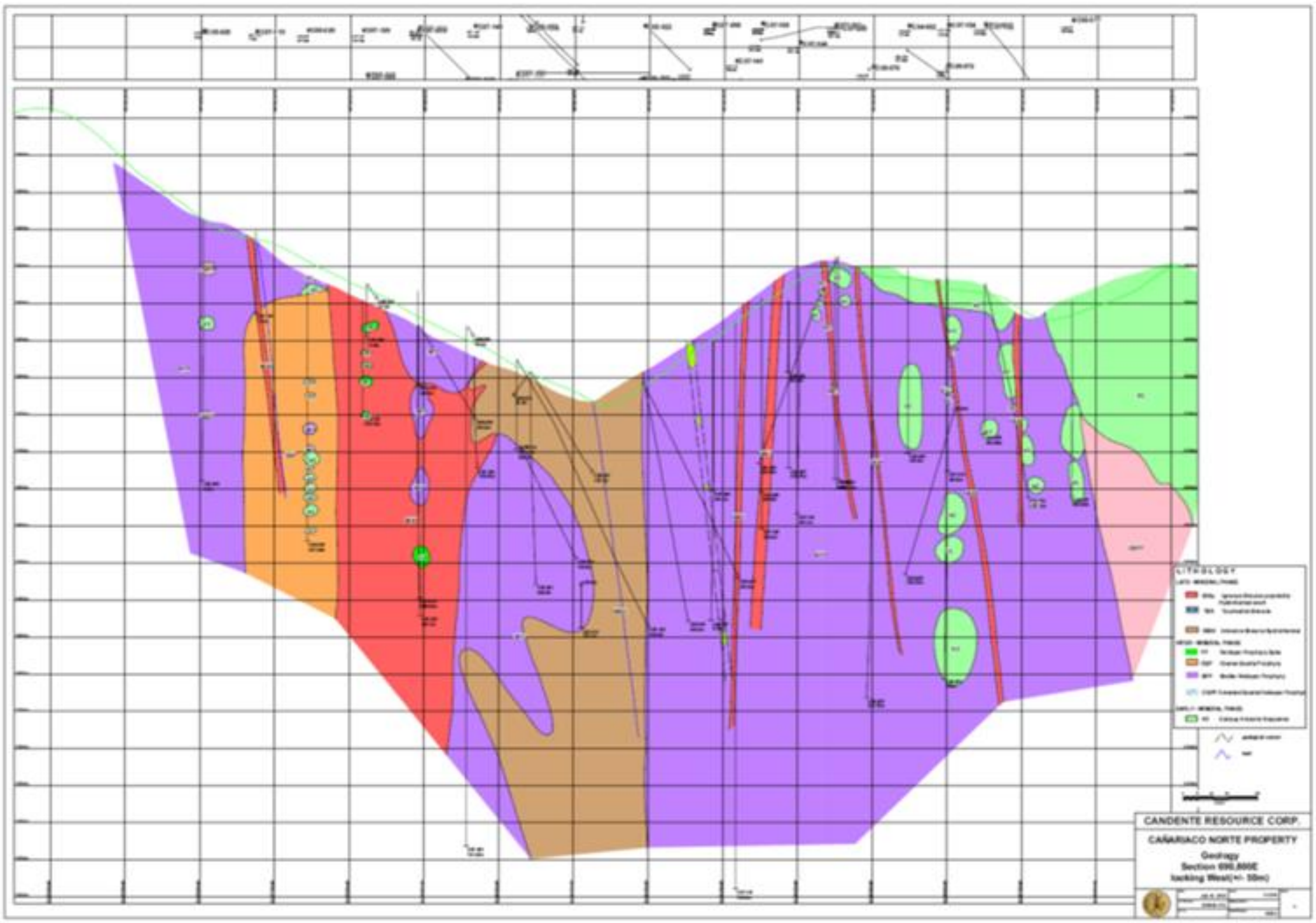
Note: Figure prepared by Van Egmond et al., 2008

Figure 7-5: Deposit Cross-Section 9,326,300N



Note: Figure prepared by Van Egmond et al., 2008

Figure 7-6: **Deposit Cross-Section 690,800E**



Note: Figure prepared by Van Egmond et al., 2008

7.3.1.2 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.

7.3.1.3 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).

7.3.1.4 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 Quemscan 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 sulphide minerals are mainly associated with pyrite and other sulphide gangue minerals;
- Non-metallic gangue minerals – quartz and micas/biotite/chlorite/clay group;
- Sulphide gangue mineral – pyrite, which is present as liberated pyrite or in association with other sulphide 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 Candente Copper staff, between the water table being at or near the present topographic surface and the active erosive 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.

7.3.2 Cañariaco Sur

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

Fifteen holes were 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.

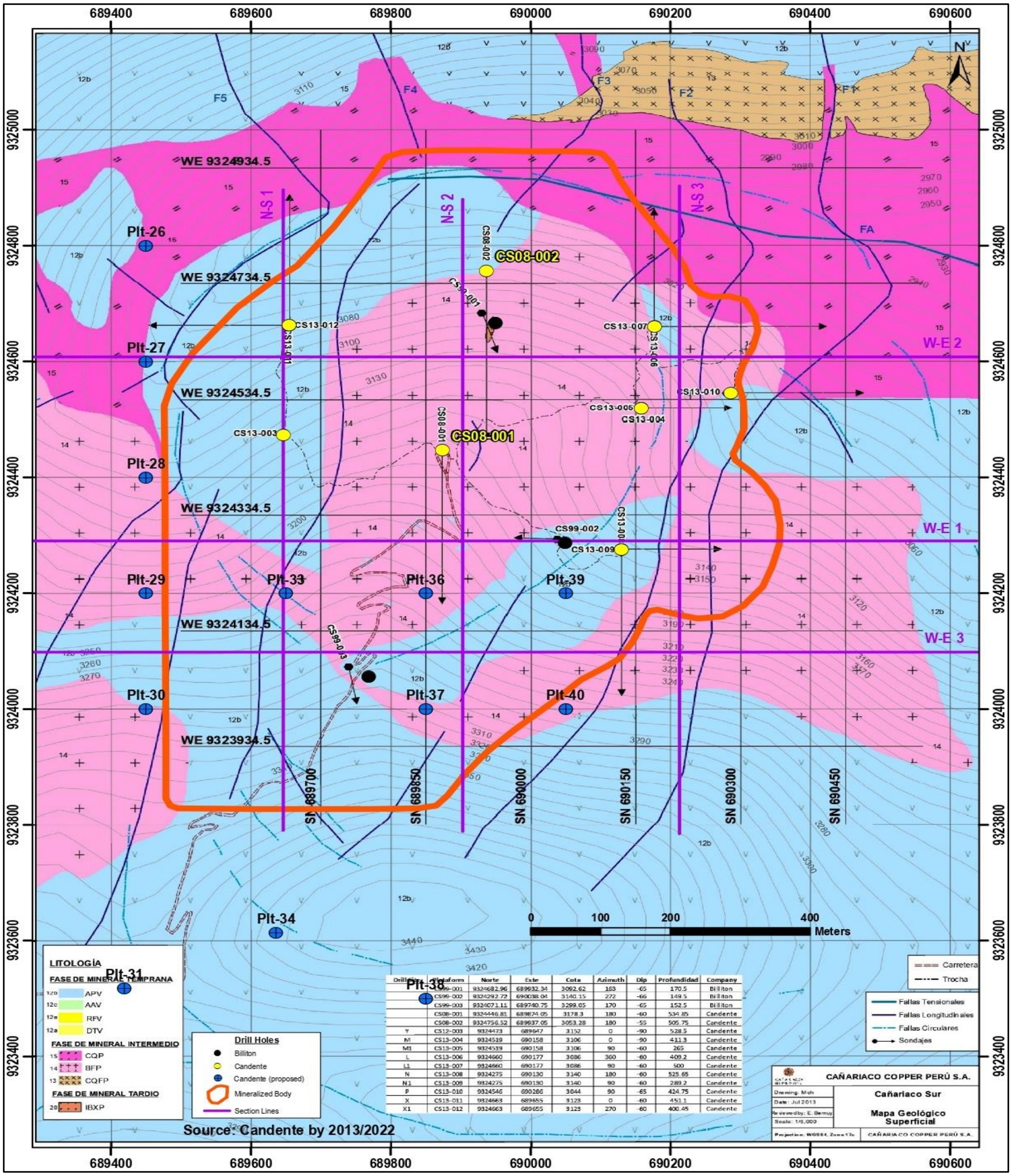
The Cañariaco Sur deposit is located approximately 2 km southwest of Cañariaco Norte. The deposit consists of a large area of potassic alteration with a coincident copper–gold–molybdenum soil geochemical anomaly with approximate dimensions of 1,400 x 900 m. A large, coincident magnetic high, IP/resistivity low, and chargeability high was identified coincident with the soil geochemical anomaly and area of potassic alteration. Candente 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 host rocks to the mineralized intrusions are Calipuy Group andesite volcanic rocks. Two main intrusions were noted from geological mapping, an early 750 x 500 m quartz monzonite stock, and a later diorite stock with an area of approximately 100 x 50 m. A third intrusive phase may also exist (Stillitoe, 1999).

Mineralization at Cañariaco Sur is predominantly chalcopyrite and bornite with minor pyrite. The depth of oxidation averages 40 m to 50 m and contains copper oxides in the form of copper wads. Meter-scale supergene enrichment is present in some drillholes.

Figure 7-7 is a geology map of the Cañariaco Sur deposit. Figure 7-8 shows the alteration types in the deposit area. Figure 7-9 and Figure 7-10 show the locations of the exsiting and propose drill holes (see discussion in Section 26.3) in relation to copper and gold soil anomalies, respectively. Figure 7-11 to Figure 7-16 are illustrative geological cross-sections through the deposit showing the relationship of the alteration and copper grades to the drill hole intercepts.

Figure 7-7: **Geology Map with Section Lines**



Drill Site	Plataform	Norte	Este	Cota	Azimuth	Dip	Profundidad	Company
	CS99-001	9324682.96	689932.34	3092.62	163	-65	170.5	Billiton
	CS99-002	9324292.72	690038.04	3140.15	272	-66	149.5	Billiton
	CS99-003	9324071.11	689740.75	3299.05	170	-65	152.5	Billiton
	CS08-001	9324446.81	689874.05	3178.3	180	-60	534.85	Candente
	CS08-002	9324756.52	689937.05	3053.28	180	-55	505.75	Candente
Y	CS12-003	9324473	689647	3152	0	-90	528.5	Candente
M	CS13-004	9324519	690158	3106	0	-90	411.3	Candente
M1	CS13-005	9324519	690158	3106	90	-60	265	Candente
L	CS13-006	9324660	690177	3086	360	-60	409.2	Candente
L1	CS13-007	9324660	690177	3086	90	-60	500	Candente
N	CS13-008	9324275	690130	3140	180	-60	525.65	Candente
N1	CS13-009	9324275	690130	3140	90	-60	289.2	Candente
P	CS13-010	9324546	690286	3044	90	-65	424.75	Candente
X	CS13-011	9324663	689655	3123	0	-60	451.1	Candente
X1	CS13-012	9324663	689655	3123	270	-60	400.45	Candente

Figure 7-8: Alteration Map with Section Lines

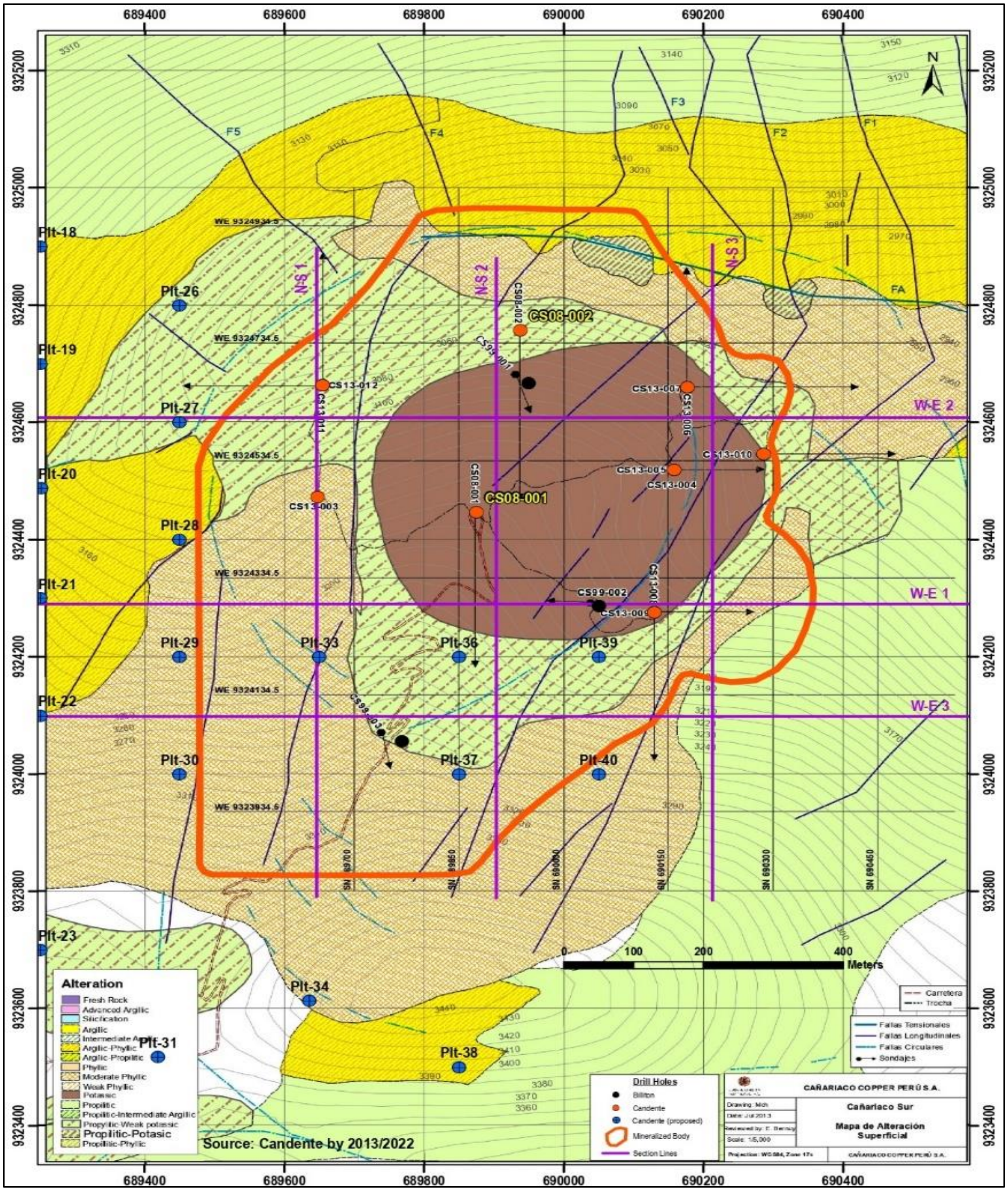


Figure 7-9: Copper Mineralization at or above 0.15% copper within copper soil anomalies. Drilled holes in black, proposed holes in blue.

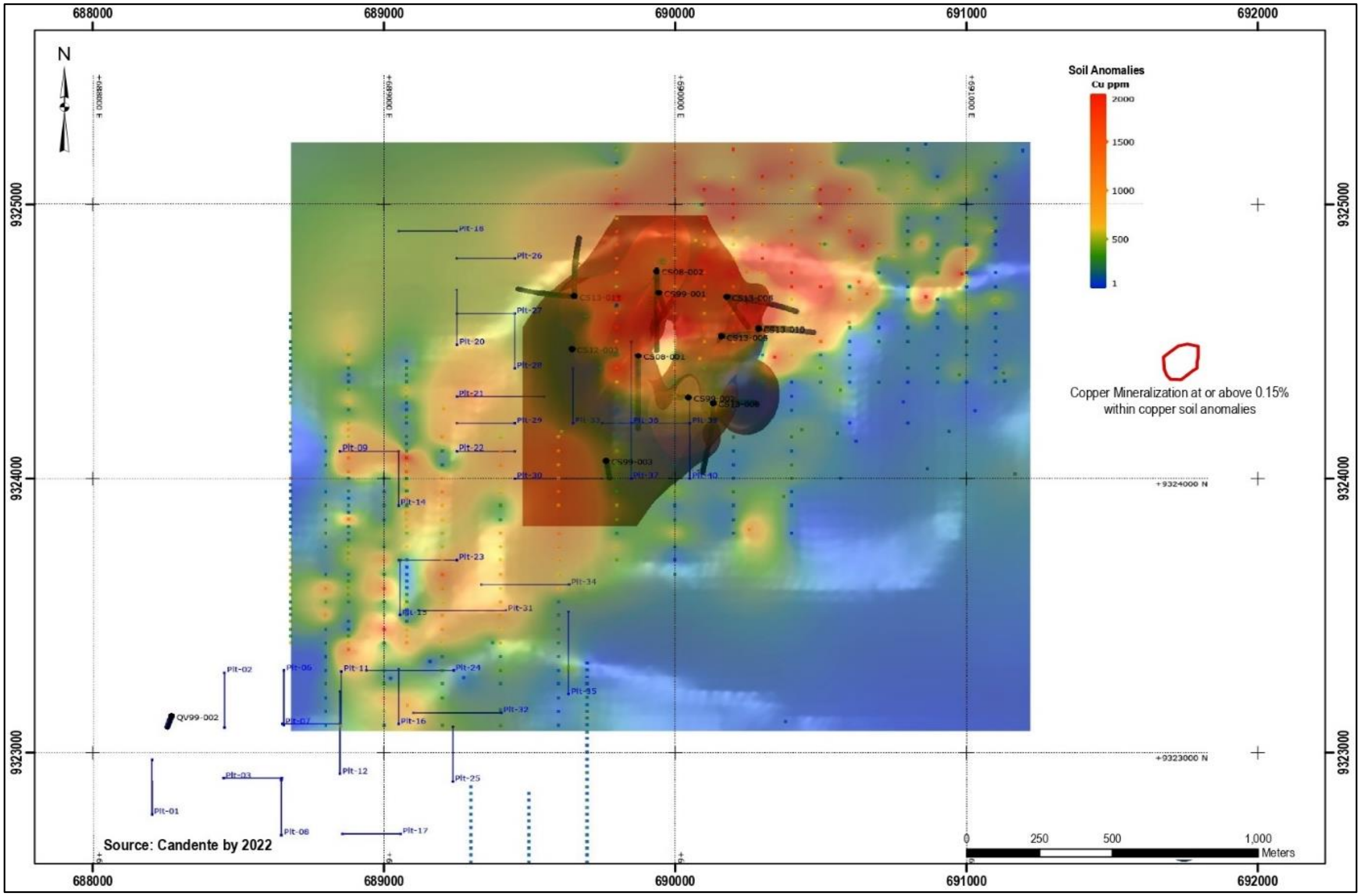


Figure 7-10: Copper Mineralization at or above 0.15% copper within gold soil anomalies. Drilled holes in black, proposed holes in blue.

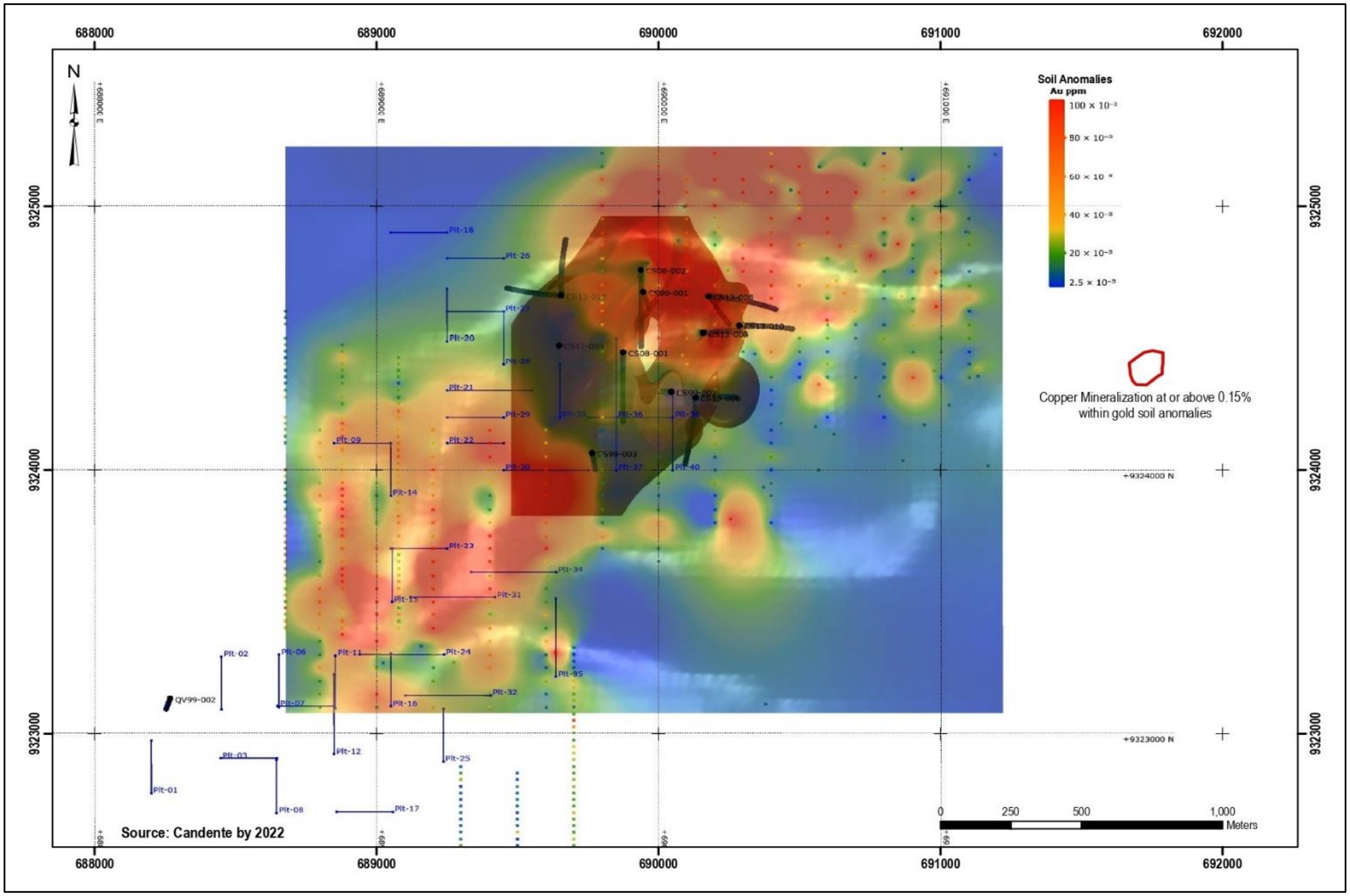


Figure 7-11: Cañariaco Sur Section W-E 1

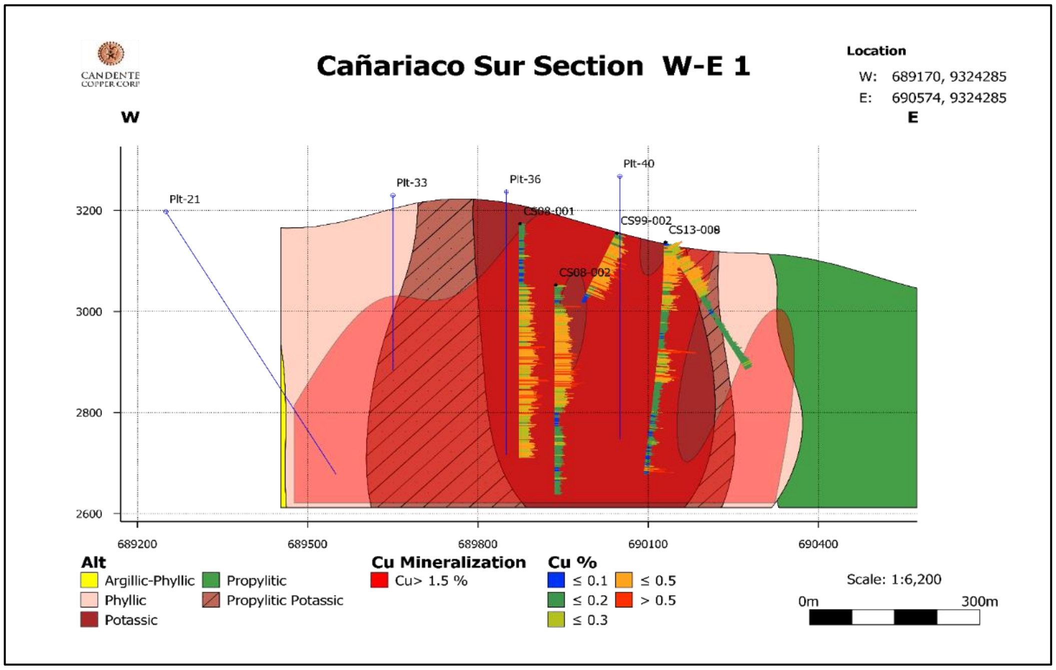


Figure 7-12: Cañariaco Sur Section W-E 2

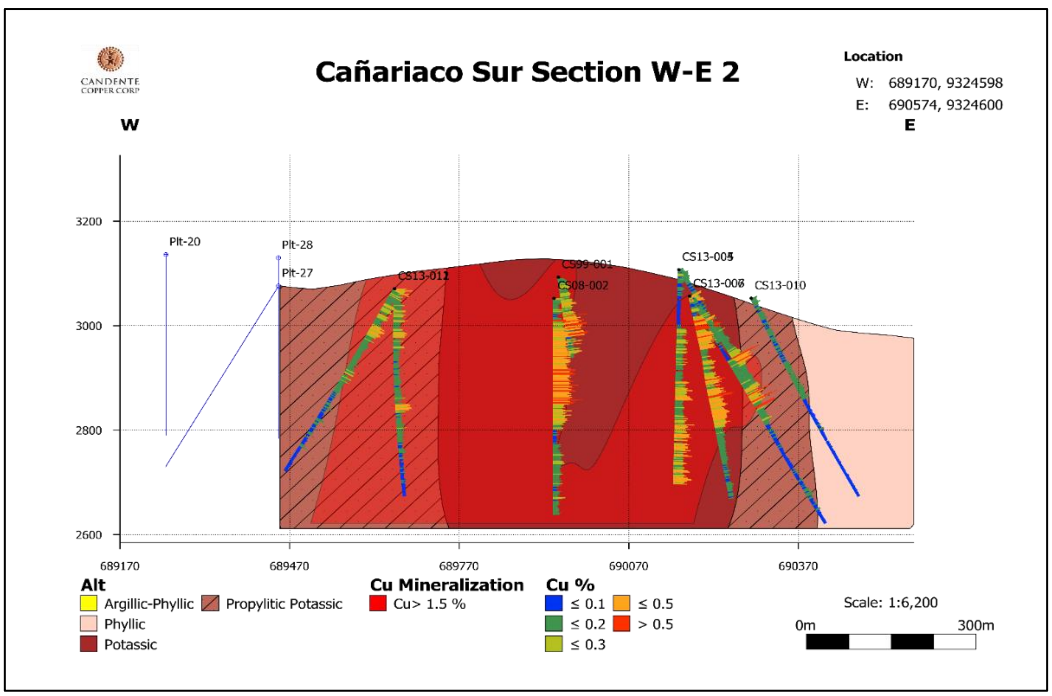


Figure 7-13: Cañariaco Sur Section W-E 3

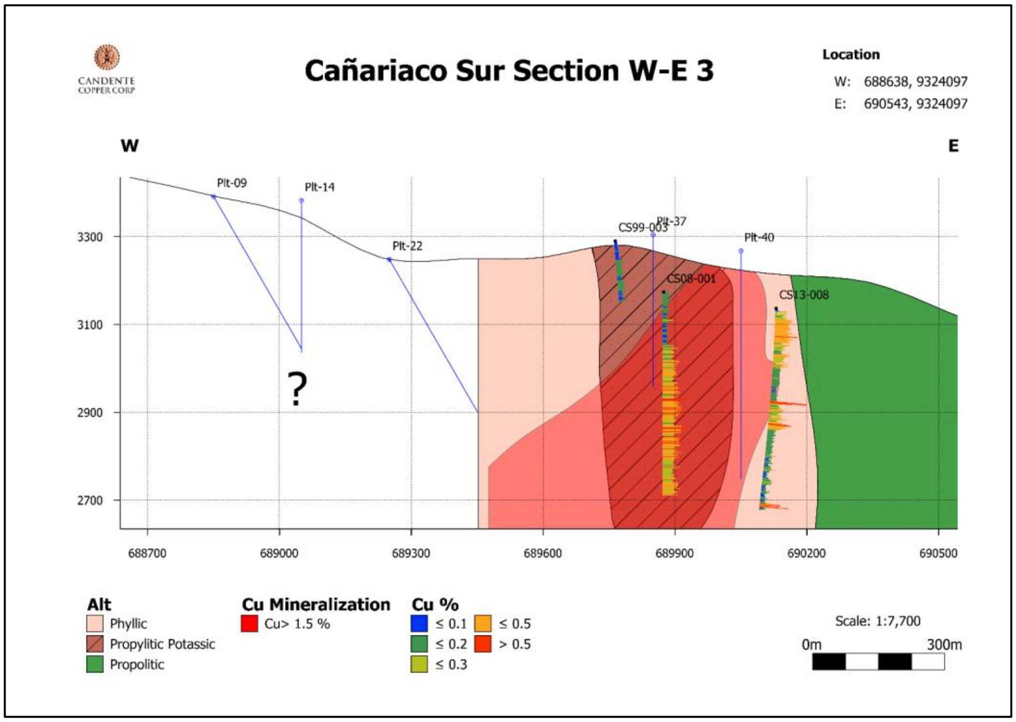


Figure 7-14: Cañariaco Sur Section N-S 1

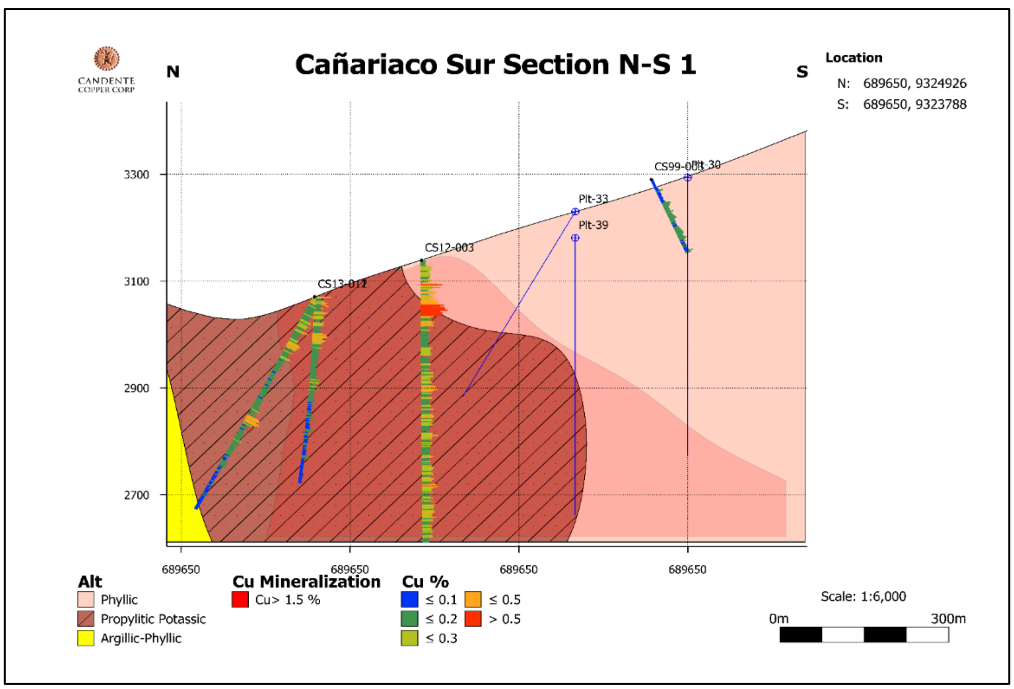


Figure 7-15: Cañariaco Sur Section N-S 2

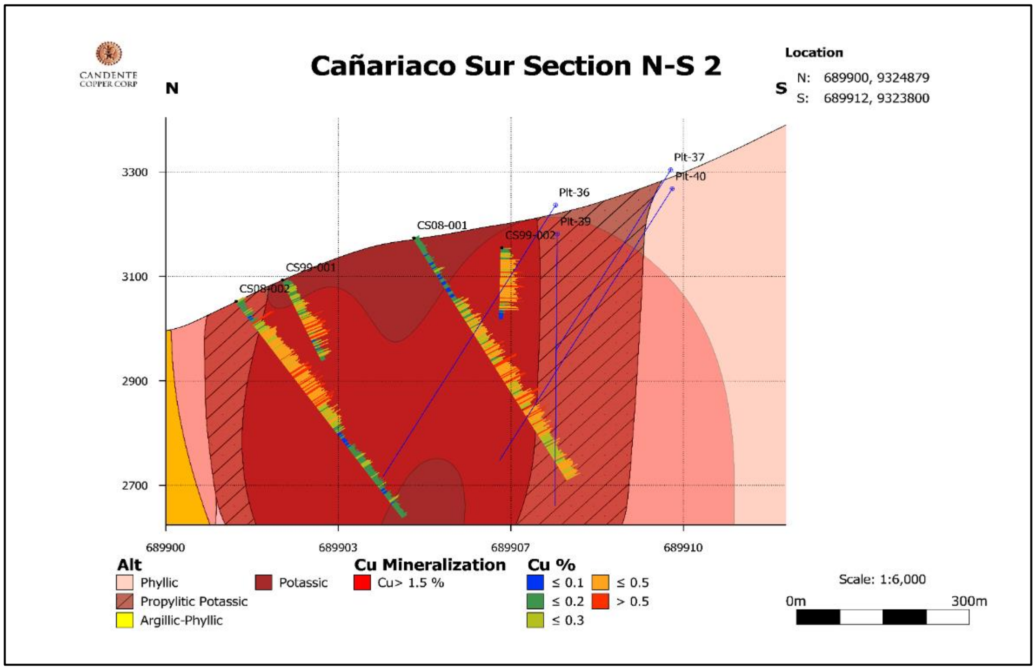
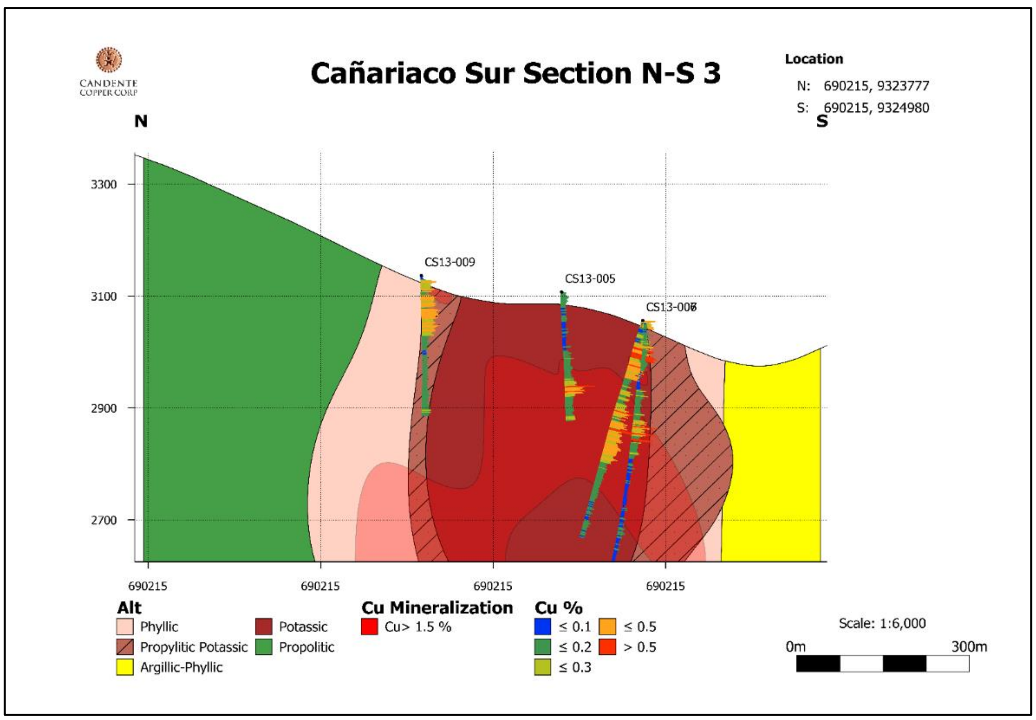


Figure 7-16: Cañariaco Sur Section N-S 3



7.3.3 Quebrada Verde Prospect

The Quebrada Verde prospect is 3.2 km south of Cañariaco Norte and 1.5 km south of Cañariaco Sur (Figure 7-3). Strongly anomalous levels of copper and gold in soil samples cover an area of 0.7 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. Candente 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 consist of a 1 km x 750 m diorite porphyry stock that intruded Calipuy Group andesite volcanic rocks (Wilson, 1999). A satellite stock of the same diorite porphyry measuring 400 m x 400 m is located 200 m to the northeast. An east–west striking, post-mineralization granodiorite dyke, with dimensions of approximately 1 km x 100 m, intruded the volcanic rocks and the larger diorite porphyry stock.

7.4 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 is not used in the 2022 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 examples of 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, ore-grade material 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-sulphidation epithermal deposits may occur in lithocaps above porphyry-copper deposits, where massive sulphide 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.

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 ores, the most common ore–sulphide 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, USA) porphyry deposits. In supergene-enriched ores, a typical assemblage can comprise chalcocite + covellite ± bornite, whereas, in oxide ores, a typical assemblage could include malachite + azurite + cuprite + chrysocolla, with minor amounts of minerals such as carbonates, sulphates, phosphates, and silicates. Typically, the principal copper sulphides consist of millimetre-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 sulphur- 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 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 deposits. The alteration assemblages are consistent with the physico-chemical conditions of a porphyry environment;

- Mineralization is focused in well-developed quartz–sulphide 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 physico-chemical conditions of a porphyry environment;
- Mineralization is focused in well-developed quartz–sulphide 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 Overview

Exploration activities such as geological mapping and geochemical sampling have been performed by Candente 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 9-1.

Table 9-1: Exploration Summary Table

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 km2 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 about 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

Year	Operator	Work Undertaken
		chargeability and/or resistivity. It was also noted that some of the strongest IP targets had not been drilled by INGEMMET • 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—present	Candente 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

Year	Operator	Work Undertaken
		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 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; • EIAsd2012 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 - Yaku conducted environmental work for both OEFA concerns re drill platform closures and a new baseline for an EIAsd was conducted; • Currently focused on obtaining authorizations for additional exploration drilling in Cañariaco Sur and initial drilling in Quebrada Verde; • Desktop Engineering Studies by Ausenco Feb to June 2021; • Current updated PEA Engineering Studies for a smaller start-up by Ausenco underway since September 2021.

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. 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 were 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 Candente 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 sulphide 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

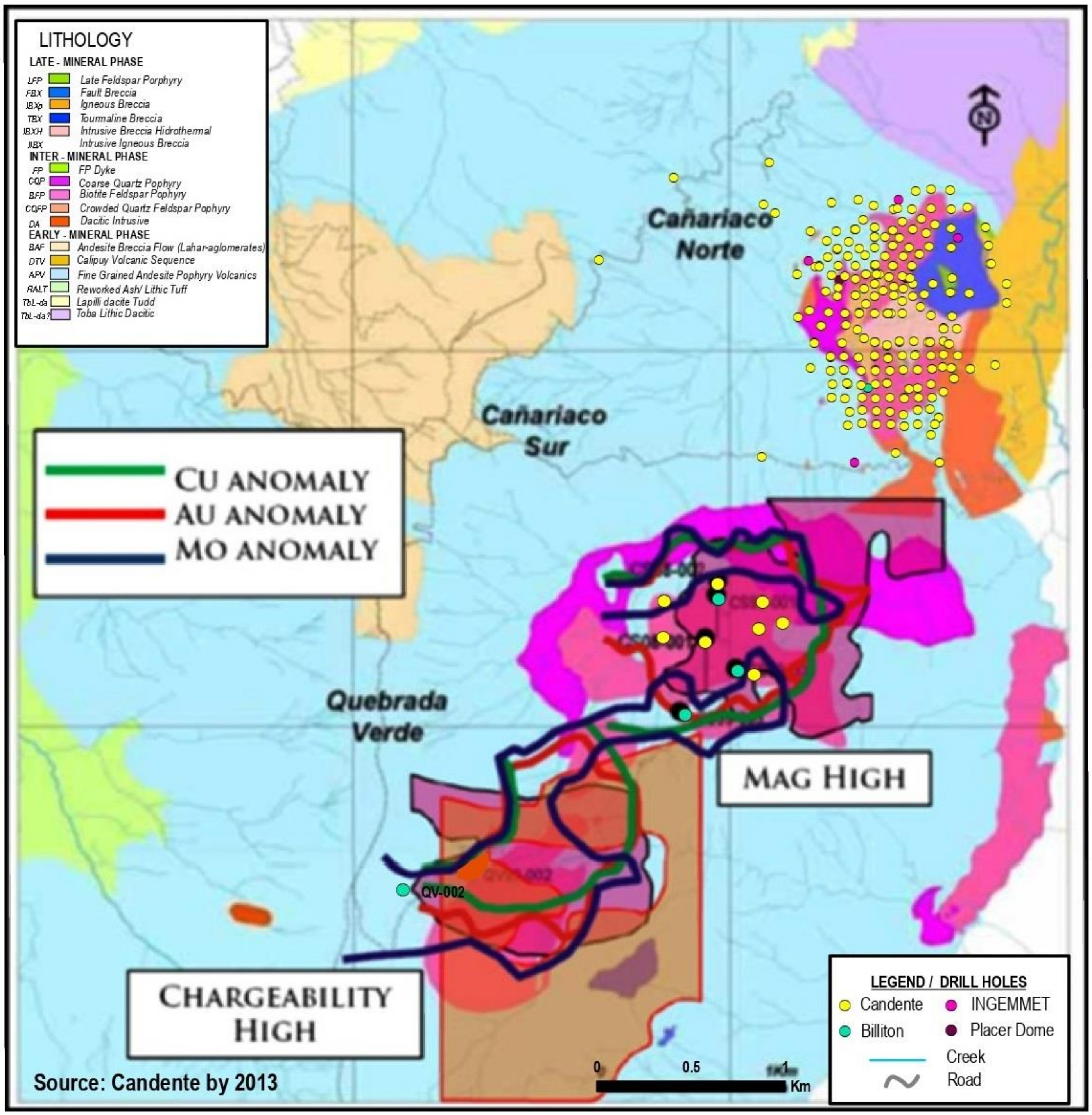
Stream sediment, soil, and rock chip sampling was completed to evaluate the mineralization potential and generate targets for core drilling at Cañariaco Norte, Cañariaco Sur, and Quebrada Verde. Sampling was performed by INGEMMET, Placer Dome, Billiton, and Candente Copper.

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.

Soil 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 soils cover an area of approximately 0.7 x 0.9 km at Quebrada Verde (Figure 9-1).

Figure 9-1: Geochemical and Ground Geophysical Anomalies



Note: Figure prepared by Candente Copper, 2022

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 Candente Copper.

Results of the surveys prior to the Candente Copper work were considered inconclusive due to wide line spacings. Candente Copper completed 20-line-kms of combined IP/resistivity and ground magnetic surveys. The ground magnetic surveys at Cañariaco Norte outlined areas about 0.7 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 centred on a resistivity anomaly and covering an area of 0.8 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 7-9 and Figure 7-10.

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

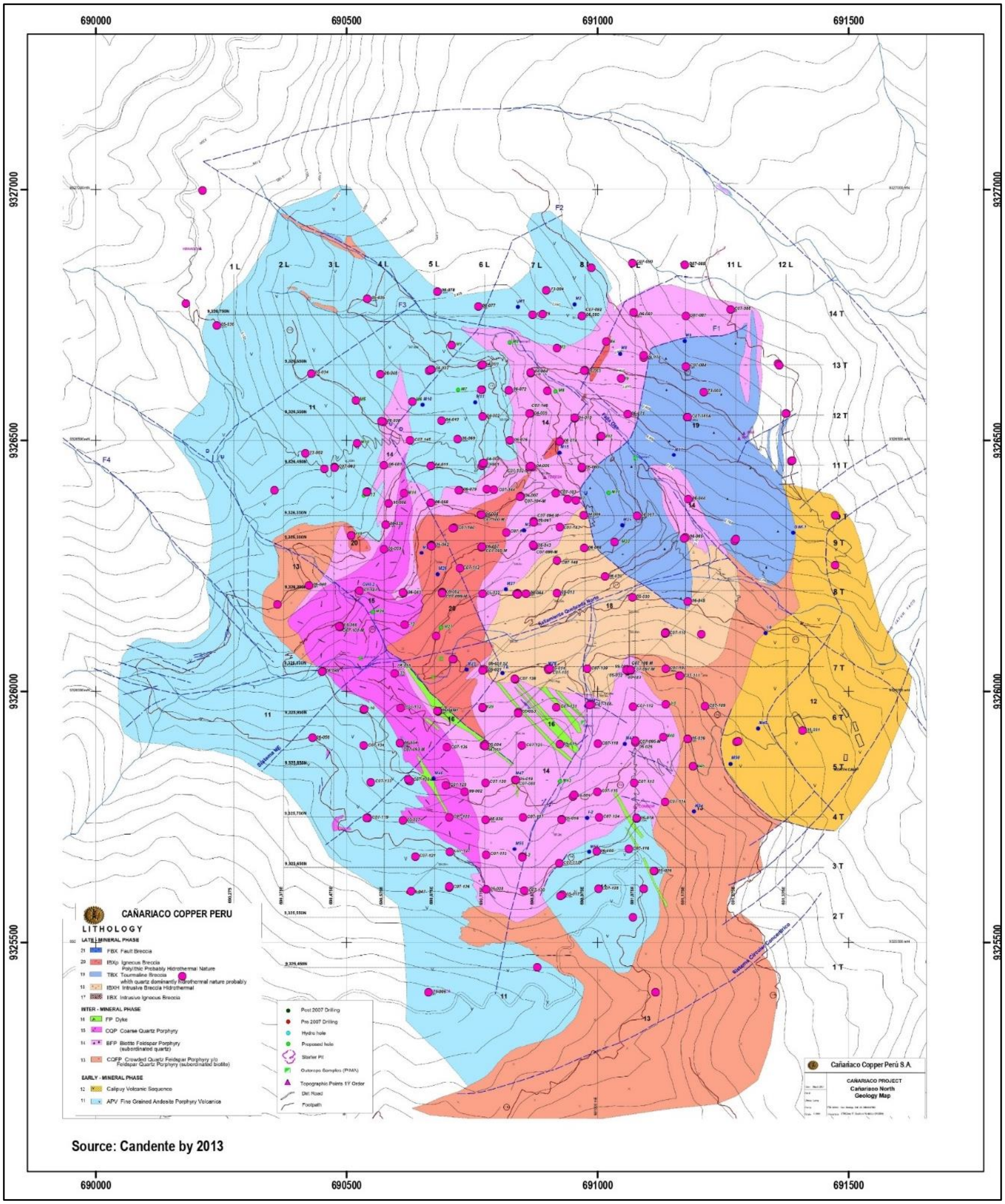
Drilling on the Project consists of 289 core holes (85,183.16 m), including geotechnical, metallurgical, and hydrogeological drilling.

Table 10-1 is a drill summary by operator and area. Drill hole locations for the Cañariaco Norte deposit are shown in Figure 10-1. Figure 10-2 shows the locations of the drill holes completed at Cañariaco Sur. In this section, all drilling completed prior to Candente Copper's involvement in the Project is termed "legacy" data.

Table 10-1: Drill Summary Table

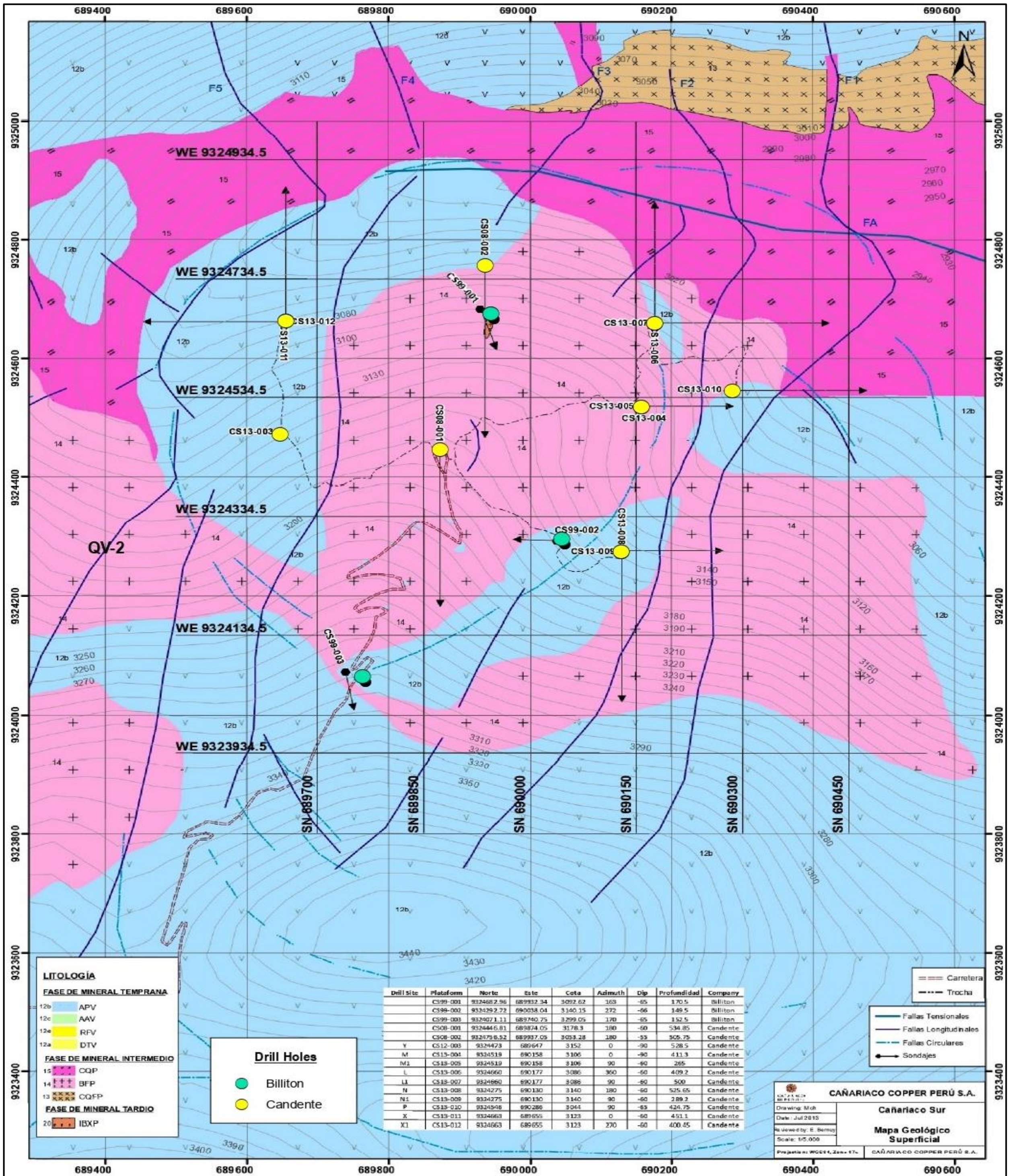
Year	Project Operator	Deposit or Prospect	Core	
			Number of Holes	Metres
1971–1974	INGEMMET	Cañariaco Norte	5	1,500.0
1994–1997	Placer Dome	Cañariaco Norte	3	853.91
1999–2000	Billiton	Cañariaco Norte	3	555.7
		Cañariaco Sur	3	472.5
		Quebrada Verde	1	100.0
2004	Candente Copper	Cañariaco Norte	12	2,647.38
2005		Cañariaco Norte	24	7,387.87
2006		Cañariaco Norte	46	15,983.55
2007		Cañariaco Norte	123	31,350.80
2008		Cañariaco Norte Cañariaco Sur	41	13,771.75 1040.55
2013		Cañariaco Norte Met Cañariaco Norte R-Geo Cañariaco Sur	 10 6 10	2,761 2,553 4,205.15
Total by Candente Copper			**289**	**85,183.16**

Figure 10-1: Drill Collar Location Plan and Geology Cañariaco Norte



Note: Figure prepared by Candente Copper, 2013.

Figure 10-2: Cañariaco Sur Drill Hole Locations and Surface Geology



Drill Site	Plataform	Norte	Este	Cota	Azimuth	Dip	Profundidad	Company
	CS99-001	9324682.96	689932.34	3092.62	163	-65	170.5	Billiton
	CS99-002	9324292.72	690038.04	3140.15	272	-66	149.5	Billiton
	CS99-003	9324071.11	689740.75	3299.05	170	-65	152.5	Billiton
	CS08-001	9324446.81	689874.05	3178.3	180	-60	534.85	Candente
	CS08-002	9324756.52	689937.05	3053.28	180	-55	505.75	Candente
Y	CS12-003	9324473	689647	3152	0	-90	528.5	Candente
M	CS13-004	9324519	690158	3106	0	-90	411.3	Candente
M1	CS13-005	9324519	690158	3106	90	-60	265	Candente
L	CS13-006	9324660	690177	3086	360	-60	409.2	Candente
L1	CS13-007	9324660	690177	3086	90	-60	500	Candente
N	CS13-008	9324275	690130	3140	180	-60	525.65	Candente
N1	CS13-009	9324275	690130	3140	90	-60	289.2	Candente
P	CS13-010	9324548	690288	3044	90	-65	424.75	Candente
X	CS13-011	9324663	689655	3123	0	-60	451.1	Candente
X1	CS13-012	9324663	689655	3123	270	-60	400.45	Candente

Source: Candente by 2013

Note: Figure prepared by Candente 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 Candente 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 Candente 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 Candente 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, Candente 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 Candente 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 Candente Copper.

10.4 Recovery

Drill core recovery data from the legacy drill campaigns are not available. Candente 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 Surveys

Candente 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 to confirm true vertical holes. 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 Grade Control

Ore control is covered in Section 16.

10.8 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.

10.8.1 Cañariaco Norte

Drill spacing varies in the Cañariaco Norte deposit averaging less than 75 x 75 m for most of the deposit and up to 100 x 100 or 100 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 but were drilled predominately at -60 to -70. The 273 holes averaged 291 m in depth and varied from 90 to 667 m with 102 holes drilled between 200 and 300m; 68 drilled between 300 and 400m and 51 holes drilled from 100 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.

10.8.2 Cañariaco Sur

Drill spacing varies in the Cañariaco Sur deposit from 100 x 100 m to 220 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 90–360; 5 were at 163 to 180; 4 at 90; 4 at 360; 2 at 27. Depths of the holes averaged 364 m in depth and varied from 150–535m.

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 Candente 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.

Candente 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 Candente 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

	Bulk Density	
	Oxide (g/cm^3)	Sulphide (g/cm^3)
VC	2.13	2.51
CQFP	2.33	2.50
BFP	2.19	2.48
CQP	2.17	2.50
FP	2.03	2.56
IBXH	2.17	2.50
TBX	2.17	2.45
IBXP	2.17	2.50

11.3 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 Candente Copper programs. Actlabs is independent of Candente 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 Candente 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 Candente Copper and held ISO:9000 accreditation for the Peruvian laboratory at the time the work was performed.

11.4 Sample Preparation and Analysis

11.4.1 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.

11.4.2 Candente Copper

11.4.2.1 Sample Preparation

Sample preparation undertaken on the Candente 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.

11.4.2.2 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 sulphuric 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.

11.4.2.3 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.

11.4.2.4 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).

11.5 Quality Assurance and Quality Control

11.5.1 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.

11.5.2 Candente Copper

11.5.2.1 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 10th sample.

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

Candente 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, Candente 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 Candente Copper. All independent checks included standards, blanks, and duplicates.

11.5.2.2 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.

11.5.2.3 Standard Reference Materials

Candente 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 were 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,

11.5.2.4 Re-sampling

Candente 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 Candente 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 Candente Copper assays from the re-sampling program were excluded because of the composite lengths.

11.5.2.5 Twin Holes

Candente 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.

11.6 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 Candente 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.

11.7 Sample Security

Sample security at the Cañariaco Project during the Candente 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 Candente Copper or laboratory personnel using Candente 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.

11.8 Sample Storage

Prior to the construction of an access road into the Candente 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 Candente Copper's regional offices, and thereafter core boxes were transported directly to the new core storage facility for detailed logging and permanent storage.

11.9 Comments 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.

- Drill sampling has been adequately spaced to first define, then infill, copper and gold anomalies to produce prospect-scale and deposit-scale drill data. Drill hole spacing varies with depth. Drill hole spacing in the central core of the deposit can be about 50 m and is 75 m x 75 m on average, due to drill pad location restrictions as a result of the topography. Drill hole spacing increases with depth as the number of holes decreases and holes deviate and is wider on the edges of the deposit.

- 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; 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, Candente 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 Candente 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 Candente 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 Candente 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 Candente 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 Candente 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

External Verification

The QP 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 Visits

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	Database Northing	Database Elevation	GPS Easting	GPS Northing	GPS Elevation	Difference in Easting	Difference in Northing	Difference in Elevation
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 (see discussion in Section 2.4).

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 Candente 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 completed a site visit. 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

13.1.1 Cañariaco Norte Testwork

Three major phases of testwork were conducted. The first consisted of process development to define the type of processing most applicable to the mineralization. This was followed by more detailed work to optimize process conditions. 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 ore variability testing campaign conducted 2011 and 2012 was designed to improve the geometallurgical understanding of the deposit and robustness in the proposed process flowsheet.

The testwork recommended key design parameters, as follows:

- For a feed grade of 0.40% Cu, a recovery of 89.7% is forecast at the target grind;
- Recovery will vary with feed grade;
- The concentrate grade was 26% Cu, with arsenic being lowered to salable limits;
- The grind was established at 200 µm with a JKSimMet breakage parameter Axb value of 68; and
- Bond ball mill work index (BWi) of 11.1 kWh/t.

13.1.2 Phase 1 (2005–2007)

Initial exploration identified potentially leachable copper mineralization. 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 sulphide samples.

Drilling in 2007 indicated the presence of a significantly larger copper deposit with predominantly sulphide copper mineralization. The testwork program was revised to assess the metallurgical response of the sulphide copper mineralization to flotation. In late 2007, SGS Chile re-composited untreated samples remaining from the 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 sulphide 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 sulphide mineralization was warranted.

Principal development composites were:

- CPY: primary sulphides composite;
- CN: secondary sulphides 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.

13.1.3 Phase 2A (2008)

Based on the positive initial sulphide 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 impurities (As, Sb) 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 sulphide 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.

13.1.4 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 and antimony 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 latter 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 sulphide (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.

13.1.5 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 programs consistent on mineralogical characterization of the composites, additional comminution tests, and roughing and cleaning flotation optimization tests.Principal development composites were:

- IBxP: breccia mineralization
- TBX: Cu:S ratio suggested secondary minerals present;
- IBxH: a predominantly hypogene mineralization type that is primarily chalcopyrite;
- CQP: mineralization type that contains predominantly enriched minerals such as covellite, bornite, enargite, and tennantite;
- BFP: selection of samples containing an average of Cu:As ratio equal to 15; and
- CQFP: 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 Metallurgical Testwork

This sub-section outlines the testwork programs completed between 1999 to 2012. The metallurgical testwork campaigns are summarized in Table 13-1.

Table 13-1: **Metallurgical Testwork Summary Table**

Year	Laboratory	Testwork Performed
Phase 1		
1999	Geomet S.A., Santiago, Chile	Sequential leach tests
2004	Kappes, Cassiday & Associates, Reno, Nevada	Column leach tests
2006	GS Metallurgical Laboratory, Santiago, Chile	Leachability 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 sulphidation 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
Variability Testwork		
2011	G & T Metallurgical Services LTD	Bulk Mineral Analysis with Liberation (BMA) Effects of grind sizes Flotation variability testing
2012	Candente Copper Corp.	Bulk Mineral Analysis with Liberation (BMA) Effect of grind sizes Flotation variability testing

13.2.1 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 as 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 605 and 930 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 sulphide 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 about 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. On average, pyrite was the most abundant copper sulphide mineral 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 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 about 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 sulphide interlocking was complex and intricate. Fine regrind sizes for the rougher concentrate will be required to efficiently separate copper sulfides from pyrite.

13.2.2 Comminution Tests

The feasibility study 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

13.2.3 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).

13.2.3.1 Particle Size

Several tests were performed at different particle sizes (K80 of 75, 106, 150, 212 and 250 µm) and at a pH 10. A P80 of 150 µm was selected for subsequent flotation testing on all 2006 composites.

In 2008, the effect of grind size (P80 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 P80 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 (P80 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.

13.2.3.2 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 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.

13.2.3.3 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.

13.2.3.4 Modifiers

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

In 2008, sulphidation 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.

13.2.3.5 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 P80 of 37 µm (80% passing 400 mesh).

Regrinds were evaluated at 4, 8, and 12 minutes instead of a specific P80 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 was required for the other three composites. This corresponds to a P80 of 50 µm for chalcopyrite-rich and arsenic-rich mineralization, and a P80 of 35 µm for secondary copper mineral-rich and mixed sulphide 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 approximately 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.

13.2.3.6 Variability testwork

Two additional testing campaigns were performed in 2011 and 2012 at G & T Metallurgical Services LTD and Candente Copper Corp. to evaluate the effect of ore 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 about 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 an excelled 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 the better performance can be achieved by reducing the grind size. At a regrind size of 16 to 17 μm K_{80}, 80 percent of the copper was recovered in a concentrate grading about 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 cooper in the feed was recovered into a high-grade concentrate grading 33.6 percent copper.

13.3 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.

13.4 Recovery Estimates

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

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

Table 13-4: 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	$89.3*(1-EXP(Cu\%_{feed}*-10))$	$3.499 \times Cu_{Recovery}^{0.7194}$	$0.0173 \times Cu_{Recovery}^{1.7542}$	$3E-05 \times Cu_{Recovery}^{3.2099}$

13.5 Deleterious Elements

As part of the final conclusions of the testing campaign carried out in various testwork programs it was noted that the presents of arsenic and antimony 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. This resulted in a final flowsheet generating a low-grade copper concentrate to the smelter with arsenic content controlled to a level where no impact on concentrate marketability is anticipated over the LOM.

13.6 Cañariaco Sur

There has been no metallurgical testwork completed on Cañariaco Sur

Fifteen holes were 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.

Due to the proximity of Cañariaco Sur to Cañariaco Norte and the similar porphyry copper mineralization style, LOM metallurgical recoveries from Cañariaco Norte were applied to Cañariaco Sur to support reasonable prospects of eventual economic extraction:

- Cu Recovery 88%
- Au Recovery 65%
- Ag Recovery 57%

However, based on this analogue assumption for the metallurgical recoveries, the Mineral Resource confidence category can support Inferred Mineral Resources only.

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 2022 Cañariaco Norte Mineral Resource estimate has been updated by using more current metal prices, costs and metallurgical recoveries.

14.2 Cañariaco Norte

14.2.1 Geological Models

Candente 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 Candente. 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 in to 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 10, 20, 30, 40, 80 15, 25, 35, 45, 85	FP, IBXH, TBX BFP, VC, CQFP, CQP, IBXP BFP, VC, CQFP, CQP, IBXP	 North South
High-Grade (> 25 ppm As)	5, 6, 7 10, 20, 30, 40 15, 25, 35, 45 80 85	FP, IBXH, TBX BFP, VC, CQFP, CQP BFP, VC, CQFP, CQP IBXP IBXP	 North South North 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

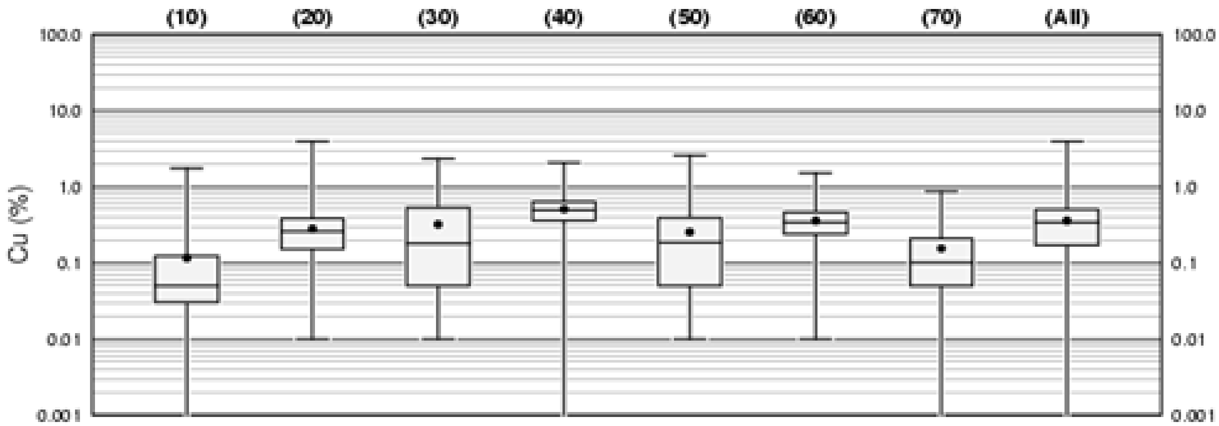
Element	Domain code	Threshold
Copper (%)	10	1.10
	20	2.00
	30	2.50
	40	2.50
	50	1.10
	60	1.20
	70	1.10
Gold (ppb)	10	-
	20	130
	30	530
	40	-
Silver (ppm)	10	12
	20	12
	30	17
	40	17

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 Table 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	Number of data
Mean	0.115	0.28	0.32	0.504	0.254	0.357	0.154	0.357	Mean
Maximum	1.73	3.97	2.31	2.06	2.56	1.5	0.87	3.97	Maximum
Upper quartile	0.122	0.38	0.53	0.63	0.39	0.46	0.21	0.5	Upper quartile
Median	0.05	0.26	0.18	0.49	0.186	0.34	0.1	0.34	Median
Lower quartile	0.03	0.15	0.05	0.36	0.05	0.24	0.05	0.17	Lower quartile
Minimum	0.0	0.01	0.01	0.0	0.01	0.01	0.0	0.0	Minimum
Variance	0.0323	0.0364	0.1127	0.0503	0.0653	0.0309	0.0231	0.0615	Variance
CV	1.557	0.68	1.049	0.445	1.007	0.493	0.989	0.695	CV
Skewness	4.271	3.677	1.522	0.782	2.33	0.825	1.611	1.274	Skewness

Note: Figure prepared by 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 back-tagged 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 large bulk mining open pit operation is envisioned for Cañariaco Sur, featuring large rope shovels working on 15 m benches. A single pit with internal phases is projected to provide mill feed at a conceptual rate of 95,000 t/d.

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$/t mined	1.60
Process Cost	US$/t milled	3.24
G&A Cost	US$/t milled	1.15
Tailings Management Cost	US$/t milled	2.13
Total Mineralization Based Costs	US$/t milled	6.52
Cu Price	US$/lb	3.50
Au Price	US$/oz	1,650.00
Ag Price	US$/oz	21.50
Selling Cost	US$/lb	0.40
Cu Recovery	%	88
Au Recovery	%	65
Ag Recovery	%	57
Overall Pit Slope(s)	Degree	40

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.

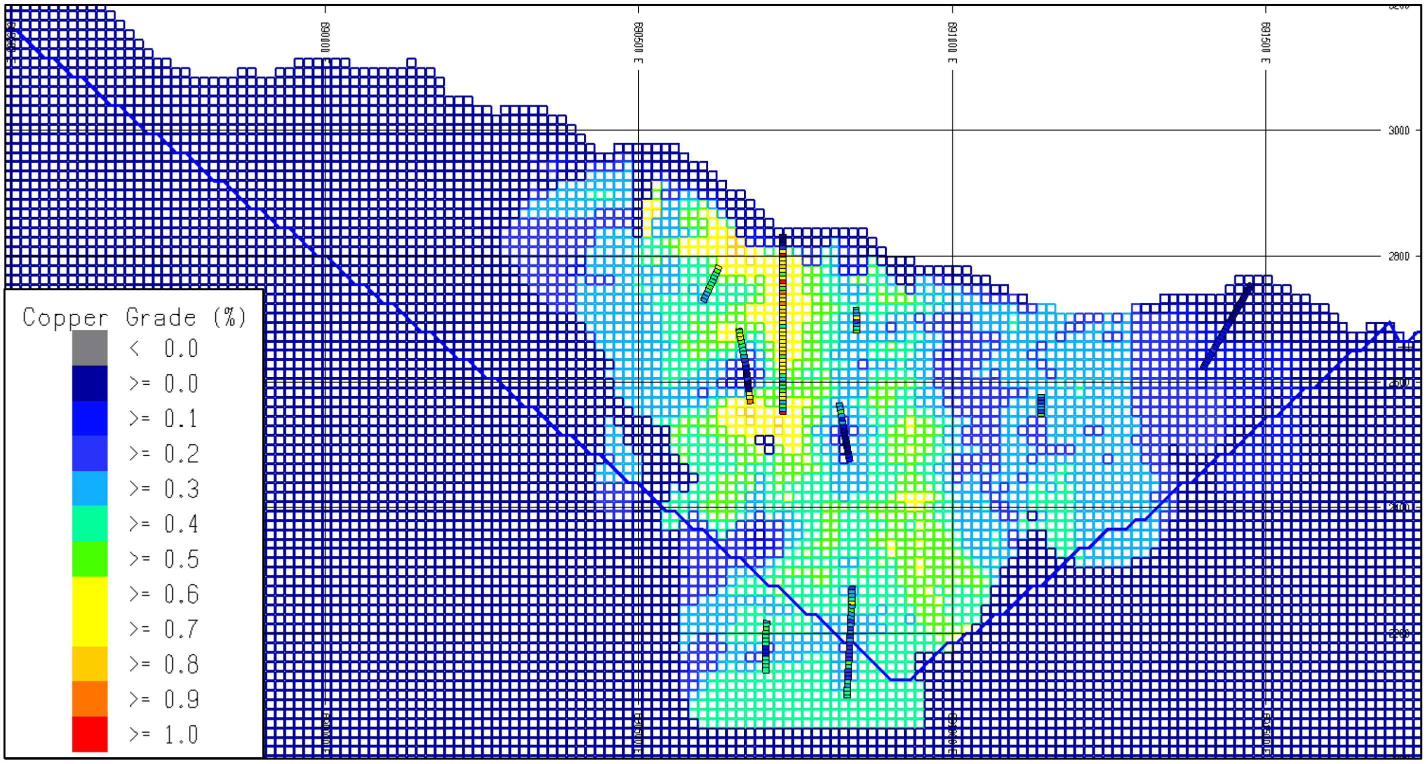
A marginal cut-off of 0.12% Cu (Table 14-8) was defined. However, Candente Copper selected a cut-off grade of 0.15% Cu for Mineral Resource reporting.

Table 14-8: Marginal Cut-Off Calculation

Parameters	Value
Processing Cost (US$/t)	6.52
Recovery (%)	88
Price (US$/lb)	3.50
Selling Cost (US$/lb)	0.46
Cut-Off Cu (%)	0.12

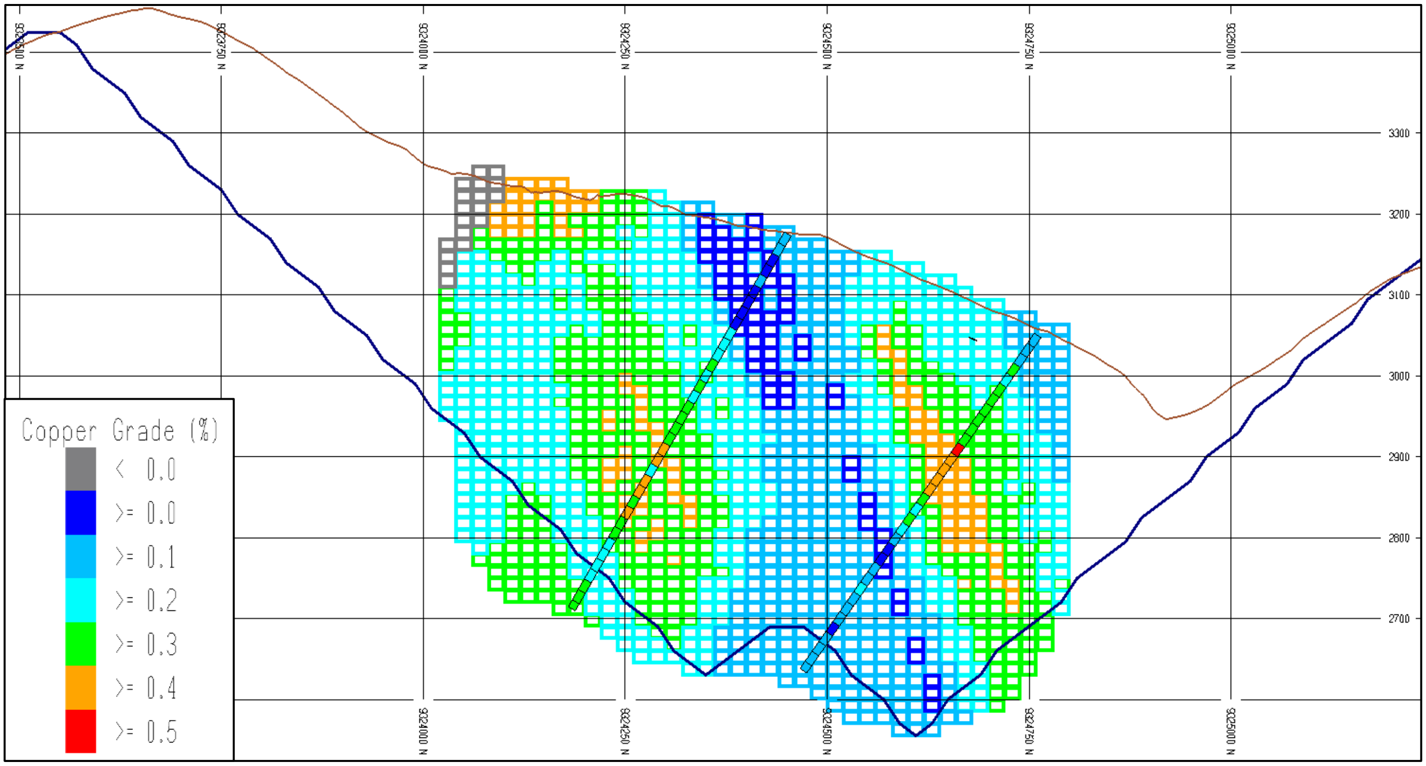
Figure 14-2 and Figure 14-3 show cross-sections displaying the resource constraining pit shells, block models and composites for Cañariaco Norte and Cañariaco Sur respectively.

Figure 14-2: Cañariaco Norte East–West Section 9,326,250 N Showing Resource Constraining Pit Shell, Block Model and Composites



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

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 27 January 2022. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Table 14-9: Mineral Resource Statement for Cañariaco Norte at a 0.15% 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	423.5	0.48	0.43	0.07	1.9	4.04	0.98	25.71
Indicated	670.7	0.39	0.36	0.05	1.6	5.25	1.16	33.72
Measured + Indicated	1,094.2	0.42	0.39	0.06	1.7	9.29	2.14	59.43
Inferred	410.6	0.32	0.29	0.04	1.4	2.66	0.55	18.09

Notes to accompany Cañariaco Norte Mineral Resource table:

1. The Mineral Resources estimate has an effective date of January 27, 2022. 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 Hexagon's MinePlan 3D™ software using:
   - A copper price of US$3.50/lb;
   - Mining cost of US$1.60/t;
   - A combined processing, tailings management and G&A cost of US$6.52/t;
   - Variable pit slope angles ranging from 36 to 39 degrees;
   - A copper process recovery of 88%;
   - Copper concentrate smelter terms: US$75/dry metric tonne treatment charge, US$0.075/lb refining charge and 96.2% payable;
   - Estimated concentrate shipping costs of US$133.00/wet metric tonne of concentrate.
5. Copper equivalent grades including contributions from gold and silver, were estimated using metal prices of copper: US$3.50/lb, gold: US$1, 650 /oz, and silver: US$21.50/oz), metallurgical recoveries of copper: 88%, gold: 65%; silver: 57%, 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.

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								
0.10	433.6	0.46	0.43	0.07	1.9	4.07	1.00	26.17
0.15	423.5	0.48	0.43	0.07	1.9	4.04	0.98	25.71
0.20	406.7	0.48	0.44	0.07	1.9	3.98	0.96	24.96
Indicated								
0.10	704.7	0.37	0.34	0.05	1.5	5.35	1.18	34.72
0.15	670.7	0.39	0.36	0.05	1.6	5.25	1.16	33.72
0.20	598.2	0.41	0.38	0.06	1.6	4.97	1.08	31.17
Measured + Indicated								
0.10	1,138.3	0.41	0.38	0.06	1.7	9.42	2.18	60.89
0.15	1,094.2	0.42	0.39	0.06	1.7	9.29	2.14	59.43
0.20	1,005.0	0.44	0.40	0.06	1.7	8.95	2.04	56.12
Inferred								
0.10	473.4	0.29	0.27	0.04	1.3	2.84	0.60	19.91
0.15	410.6	0.32	0.29	0.04	1.4	2.66	0.55	18.09
0.20	305.5	0.36	0.33	0.05	1.5	2.25	0.46	14.94

Note: Footnotes to apply to this table as well.

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 14-11: Mineral Resource Statement for Cañariaco Sur at a 0.15% 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	433.2	0.30	0.25	0.09	1.2	22	2.36	1.26	16.39	20.88
0.15	384.5	0.32	0.26	0.10	1.2	22	2.22	1.18	15.02	18.91
0.2	290.0	0.35	0.29	0.11	1.3	22	1.85	0.98	11.88	14.25

Notes to accompany Cañariaco Sur Inferred Mineral Resource table:

1. The Mineral Resource estimate has an effective date of January 27, 2022. 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 0.15% 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 Hexagon's MinePlan 3D™ software using:
   - A copper price of US$3.50/lb;
   - Mining cost of US$1.60/t;
   - A combined processing, tailings management and G&A cost of US$6.52/t;
   - 45 degree pit slope angles;
   - A copper process recovery of 88%;
   - Copper concentrate smelter terms: US$75/dry metric tonne treatment charge, US$0.075/lb refining charge and 96.2% payable;
   - Estimated concentrate shipping costs of US$133.00/wet metric tonne of concentrate.
10. Copper equivalent grades including contributions from gold, silver and molybdenum, were estimated using metal prices of copper: US$3.50/lb, gold: US$1,650/oz, silver: US$21.50/oz and molybdenum: US$11.00/lb; metallurgical recoveries of copper: 88%, gold: 65%; silver: 57% 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. There is currently no metallurgical testwork on Cañariaco Sur mineralization. Cañariaco Norte average recoveries were applied for conceptual pit shell generation and CuEq estimations.
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 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.

14.7 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 this Report.

16 MINING METHODS

Readers are cautioned that the 2022 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 2022 PEA will be realized.

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

16.1 Overview

The Cañariaco Norte deposit is a large, near surface, bulk mineable porphyry copper deposit located in the Lambeyeque region of northern Peru. 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 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 WSF, 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 WSF. Based on earlier internal desktop trade-off studies, the mine plan considers an initial 40 kt/d processing rate which is later doubled to 80 kt/d in production Year 7. The peak mining capacity is 54.6 Mt/a.

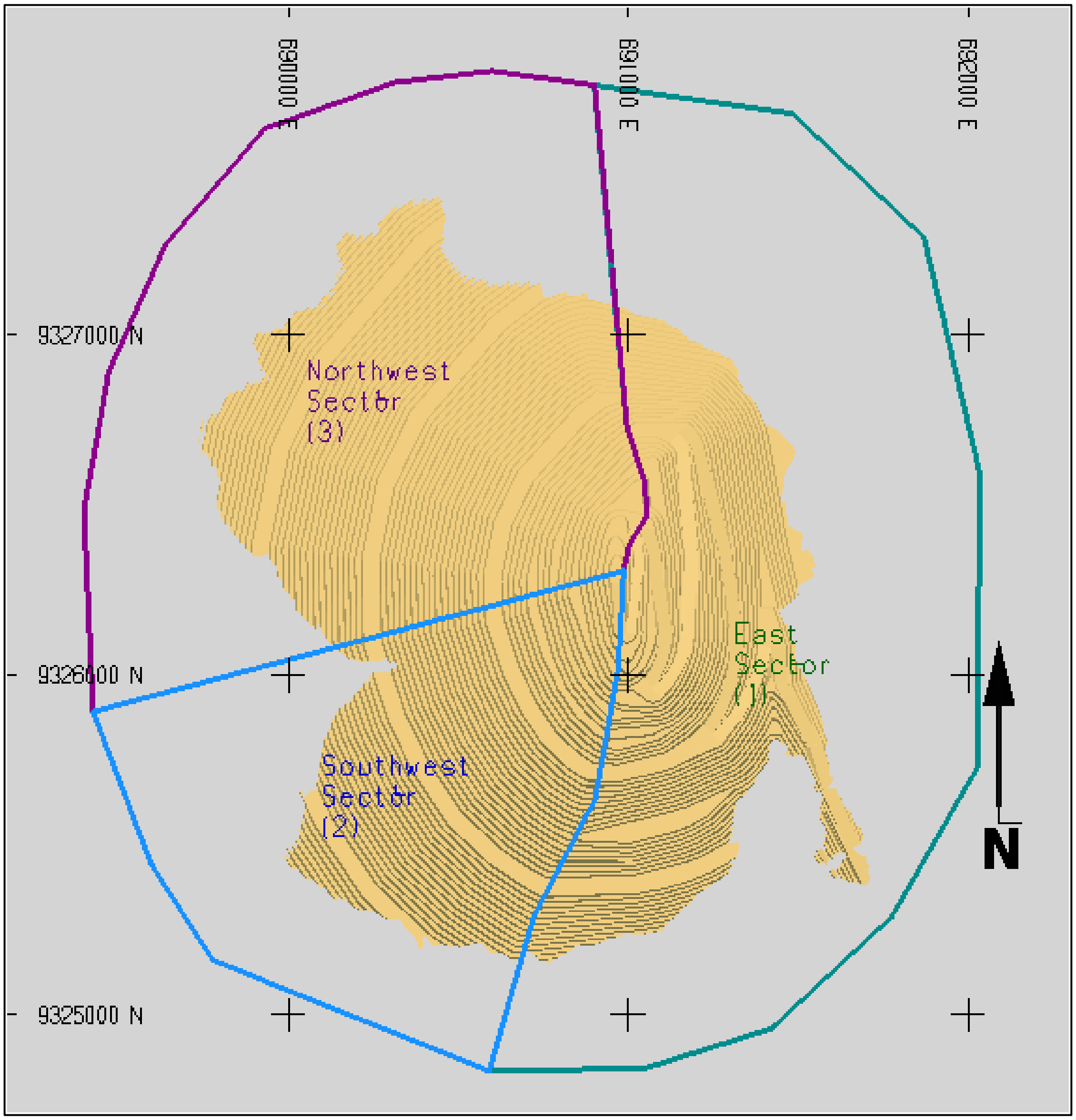
The mine planning was performed using metal prices of US$3.25/lb Cu, US$1,550/oz Au and US$20/oz Ag. Measured, Indicated, and Inferred Mineral Resources were considered for processing. The mine plan used the resource block model described in Section 14. AGP's mine planning was performed using Hexagon's 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.

The geotechnical domains are shown with the 2022 PEA ultimate pit design in Figure 16-1.

Figure 16-1: Geotechnical Domains



Note: Figure prepared by AGP, 2022.

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

Table 16-1: **Recommended Pit Slope Angles**

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

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 Norte project.

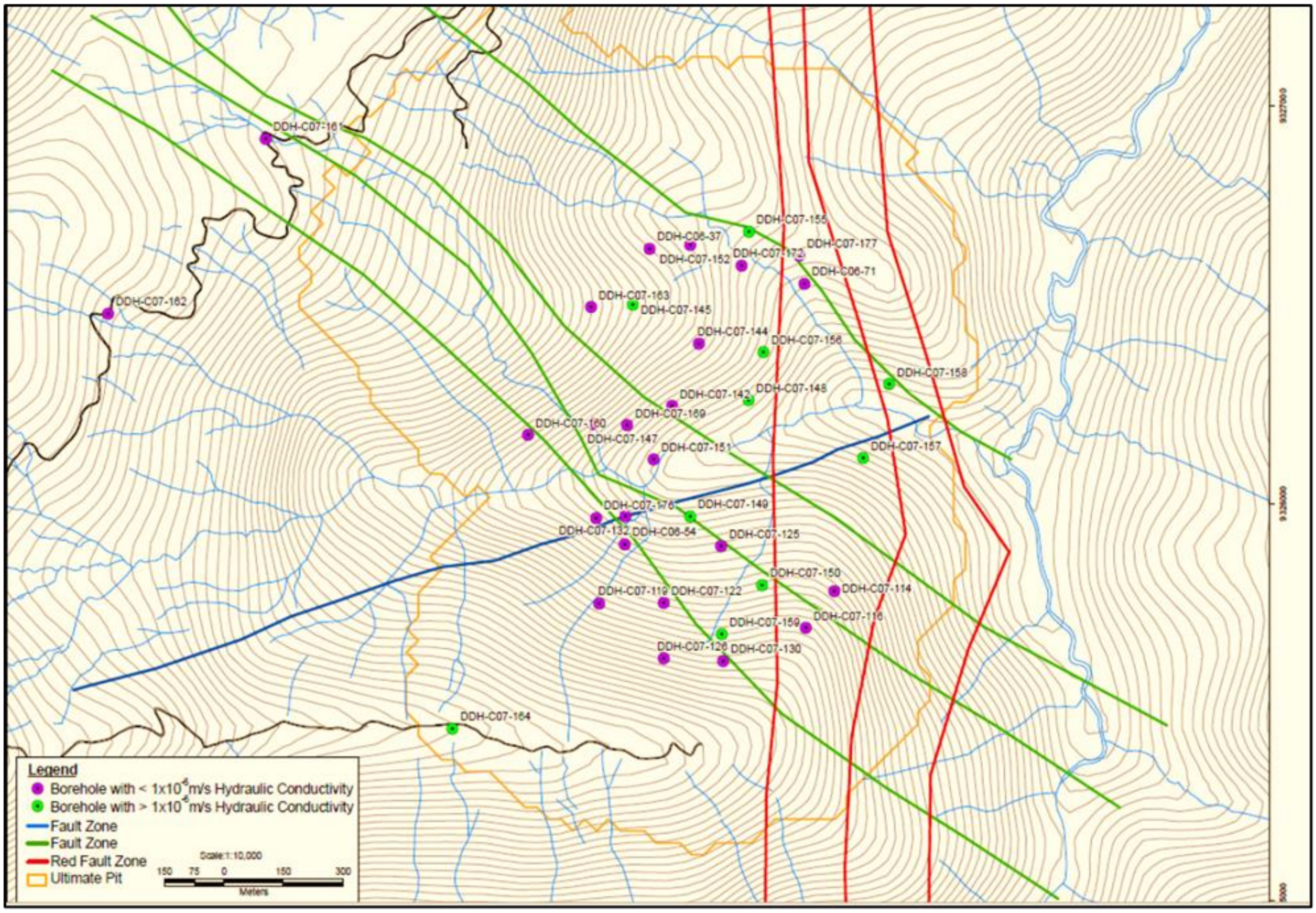
16.3 Hydrogeological Considerations

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



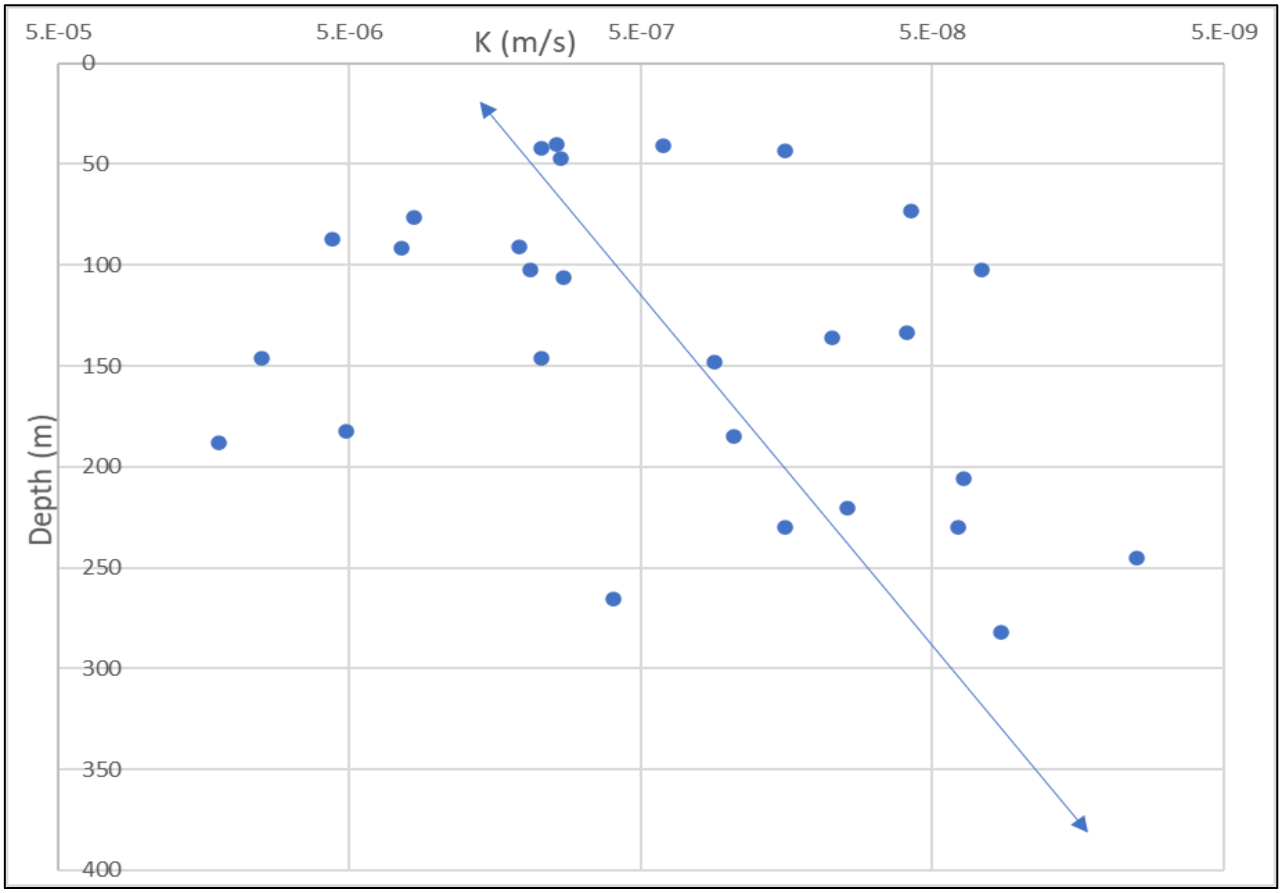
Note: Figure prepared by GWI 2008

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 2022 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 approximately 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 suggest about an order of magnitude decrease in K every 100 m from surface.

Figure 16-3: Bedrock Hydraulic Conductivity Variation with Depth



Note: n = 43, Figure prepared by AMEC 2010.

Figure 16-3 indicates that an average K at around 200 m depth is approximately 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.

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. After the initial native model handover, a revised arsenic grade item was provided by DKT Geosolutions Inc. and imported. 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.

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, 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. Process and G&A costs were provided by Ausenco based on earlier internal studies.

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 37.6 º, 36.0 º, 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 6.0. Using a similar calculation, the resulting starter pit overall slopes used were 37.6 º, 39.4 º, and 38.1 º for the east, southwest and northwest sectors respectively.

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

Metal	Price (US$/lb)
Copper	3.25
Gold	1,550
Silver	20

Table 16-4: Smelter Terms

Smelter Terms	Amount
Payable Metals and Deductions	
Cu Deduction (unit)	1
Cu Payable (%)	96.5
Au Deduction (g/t)	1
Au Payable (%)	98
Ag Deduction (g/t)	30
Ag Payable (%)	98
Penalties	
As Penalty ($/DMT conc.)	
Increment (%)	0.1
Upper Hurdle (%)	1
Upper Penalty ($/increment %)	10

Smelter Terms	Amount
Middle Hurdle (%)	0.5
Middle Penalty ($/increment %)	8
Lower Hurdle (%)	0.2
Lower Penalty ($/increment %)	2.5
Treatment Charges	
Conc. TC ($/DMT)	75
Cu RC ($/Payable lb)	0.075
Au RC ($/Payable Oz)	5
Ag RC ($/Payable Oz)	0.5

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 and are different from those developed later and presented in Section 22. The final metal prices in the financial model are higher and the smelter terms are slightly more favorable for gold and silver payables. 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 80 kt/d milled and are shown in Table 16-7.

Table 16-7: Process and G&A Costs

Process	Quantity	Amount
Processing	(US$/t milled)	3.24
Tailings Management	(US$/t milled)	2.13
G&A	(US$/t milled)	1.15
Total Ore Based Costs	(US$/t milled)	6.52

The mining operating costs were developed using a mining cost model used in earlier internal studies and considering haul destinations envisioned early in the 2022 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	Cost
Waste	Base Cost (1550)	US$/t	1.72
	above 1550	US$/t/15 m bench	0.012
	below 1550	US$/t/15 m bench	0.050
Mill feed	Base Cost (1550)	US$/t	1.11
	above 1550	US$/t/15 m bench	0.025
	below 1550	US$/t/15 m bench	0.026

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 materially. The final mill feed hauls are now somewhat longer than originally envisioned, and the waste hauls are somewhat shorter than originally envisioned. 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 Value 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 revenue factor that was varied from 0.25 to 1.0 by 0.025 increments. RF shells 0.3, 0.375, 0.475 and 0.55 were selected for pit phase guidance. The volumetric results of the set of nested shells are shown in Table 16-9 and Figure 16-4.

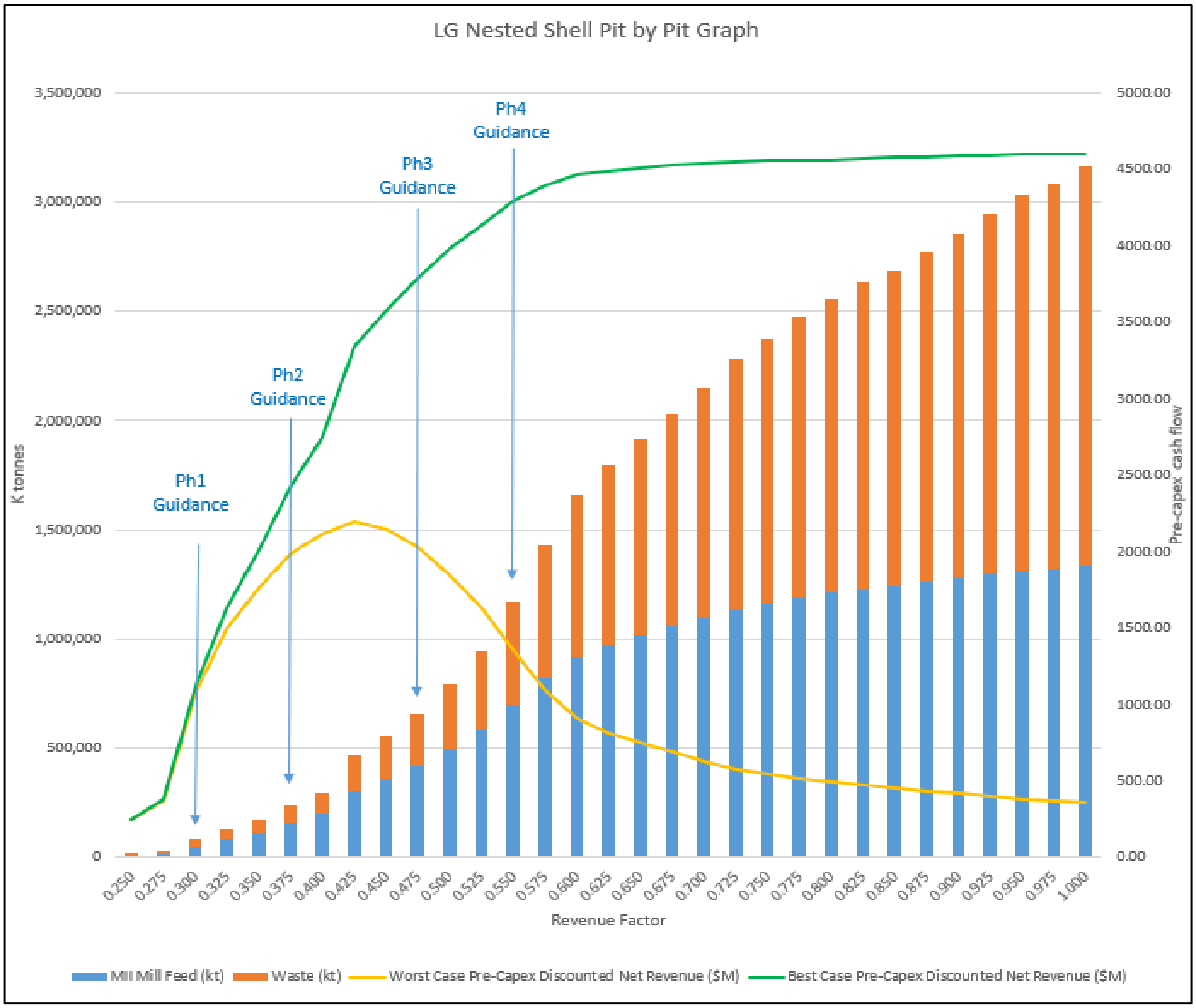
Table 16-9: LG Shell Volumetric Results

Revenue Factor	Mill Feed (kt)	NSR ($/t milled)	Cu (%)	Au (ppb)	Ag (ppm)	As (ppm)	Waste (kt)	Total Material (kt)	Strip Ratio
0.250	8,409	37.05	0.65	130	3.0	372.40	8,713	17,122	1.04
0.275	14,094	34.95	0.61	120	2.9	316.31	11,803	25,896	0.84
0.300	52,347	30.88	0.54	110	2.2	231.77	31,268	83,614	0.60
0.325	86,259	28.99	0.52	100	2.2	258.48	46,212	132,471	0.54
0.350	116,841	27.67	0.50	90	2.1	243.76	57,349	174,189	0.49
0.375	160,282	26.24	0.47	90	2.1	235.59	75,737	236,019	0.47
0.400	197,954	25.31	0.46	90	2.0	236.11	95,217	293,172	0.48
0.425	303,568	23.73	0.44	80	1.9	220.15	166,649	470,217	0.55
0.450	359,698	23.02	0.42	80	1.9	212.91	198,060	557,757	0.55
0.475	419,488	22.39	0.41	80	1.8	210.07	237,534	657,022	0.57
0.500	499,807	21.64	0.40	70	1.8	211.01	294,883	794,691	0.59
0.525	583,355	21.02	0.39	70	1.7	207.02	360,720	944,075	0.62
0.550	**697,502**	**20.39**	**0.38**	**70**	**1.7**	**201.20**	**470,952**	**1,168,455**	**0.68**
0.575	821,936	19.89	0.38	60	1.6	196.80	608,072	1,430,008	0.74
0.600	918,854	19.61	0.37	60	1.6	194.97	738,483	1,657,336	0.80
0.625	972,659	19.48	0.37	60	1.6	194.68	822,098	1,794,758	0.85
0.650	1,015,092	19.39	0.37	61	1.6	194.99	895,125	1,910,217	0.88
0.675	1,057,244	19.30	0.37	60	1.6	195.34	974,173	2,031,418	0.92
0.700	1,097,500	19.21	0.37	60	1.6	195.58	1,054,025	2,151,525	0.96
0.725	1,136,082	19.14	0.36	60	1.6	195.88	1,142,193	2,278,275	1.01
0.750	1,162,482	19.10	0.36	60	1.6	196.13	1,210,871	2,373,353	1.04
0.775	1,190,419	19.06	0.36	60	1.6	196.35	1,286,458	2,476,877	1.08
0.800	1,210,884	19.03	0.36	60	1.6	196.49	1,347,336	2,558,220	1.11
0.825	1,229,232	19.00	0.36	60	1.6	196.50	1,404,836	2,634,068	1.14
0.850	1,241,674	18.98	0.36	60	1.6	196.63	1,445,330	2,687,004	1.16
0.875	1,259,218	18.95	0.36	60	1.6	196.80	1,509,529	2,768,747	1.20
0.900	1,276,345	18.93	0.36	60	1.6	196.83	1,574,075	2,850,420	1.23
0.925	1,295,287	18.91	0.36	60	1.6	197.19	1,653,318	2,948,605	1.28

0.950	1,309,805	18.90	0.36	60	1.6	197.42	1,719,252	3,029,056	1.31
0.975	1,319,151	18.90	0.36	60	1.6	197.41	1,765,512	3,084,663	1.34
1.000	1,333,057	18.88	0.36	60	1.6	197.29	1,828,876	3,161,933	1.37

Note: shell volumetrics reported at a $7.00/t NSR cut-off, which was an approximate weighted average of the 40 kt/d and 80 kt/d breakeven cut-offs.

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



Note: Figure prepared by AGP, 2022.

The shells were selected based on the following considerations:

- A desire to have four pushbacks with equipment appropriate minimum mining widths and practical ramp access; and
- A LOM in the range of 25 to 30 years.

16.7 Pit Design

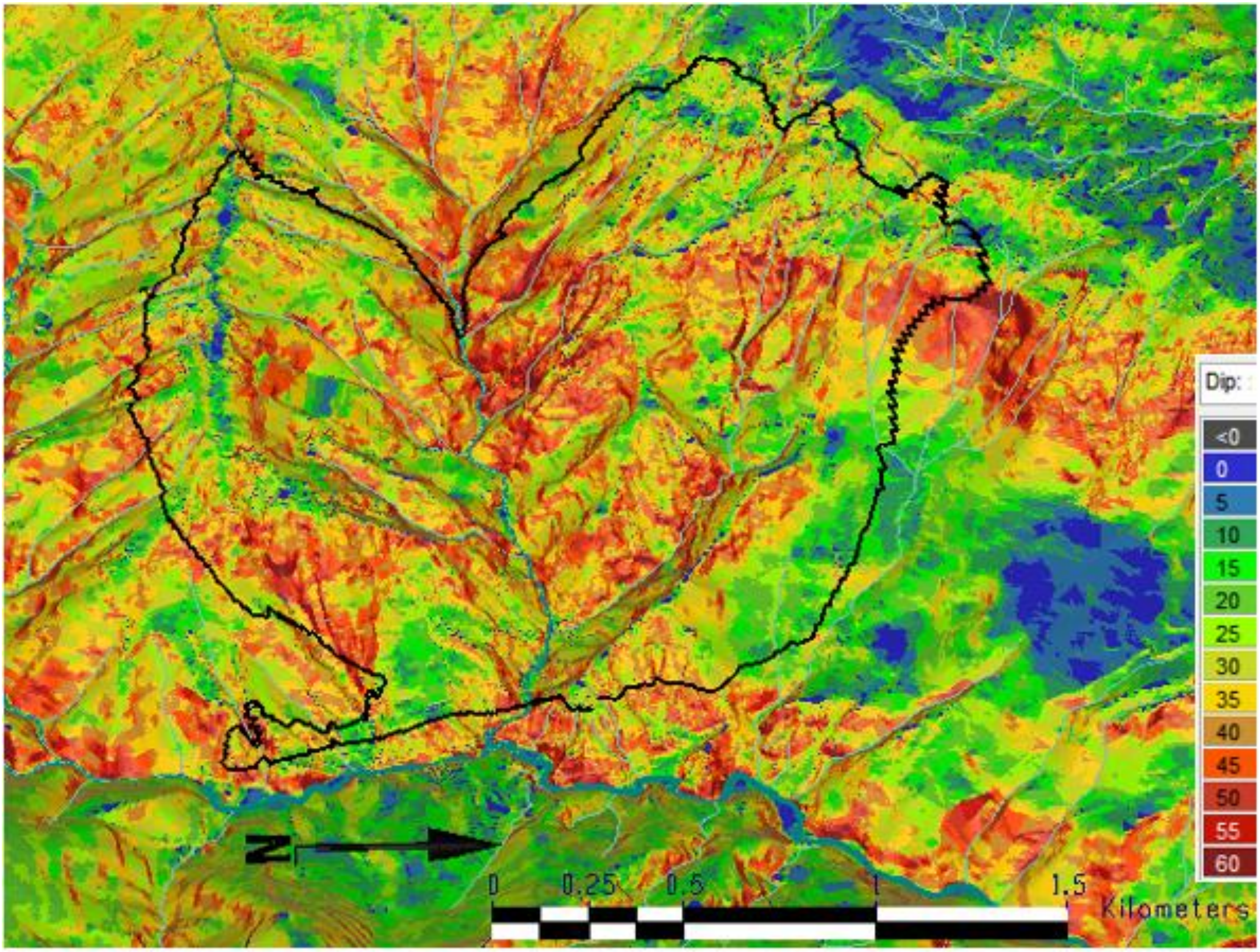
Four phase designs were developed for the planned single open pit. Multiple phases were designed to release mineralized material in a timely manner and to smooth out stripping requirements on an annual basis. The nested pit optimization shells used to determine the ultimate pit were also used to outline areas of higher value for targeted early mining and phase development.

Geotechnical parameters outlined in Section 16.2Table 16-1 were applied to the pit phase designs. The starter pit used the steeper 'starter pit stage' slope parameters. The other phases used the final stage' parameters. The 'interim pit stage' slope parameters were not used as phases 2 and 3 will be mined back to the ultimate pit wall at their upper elevations.

Ramps widths were based on the use of 218-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 a 35 m^3 electric rope shovel, which was later changed to a 35 m^3 electrified hydraulic shovel to reduce initial capital costs.

The topography of the pit area is generally steep with eroded drainage features. A 3D view of the pit area topography, colour coded by dip angle (degrees) is shown in Figure 16-5Figure 16-5. Pioneering road development will be significant, and pit phase designs require internal ramps in their highwalls to access later pit phases.

Figure 16-5: West looking 3D View of Pit Area Topography, Colour Coded by Dip

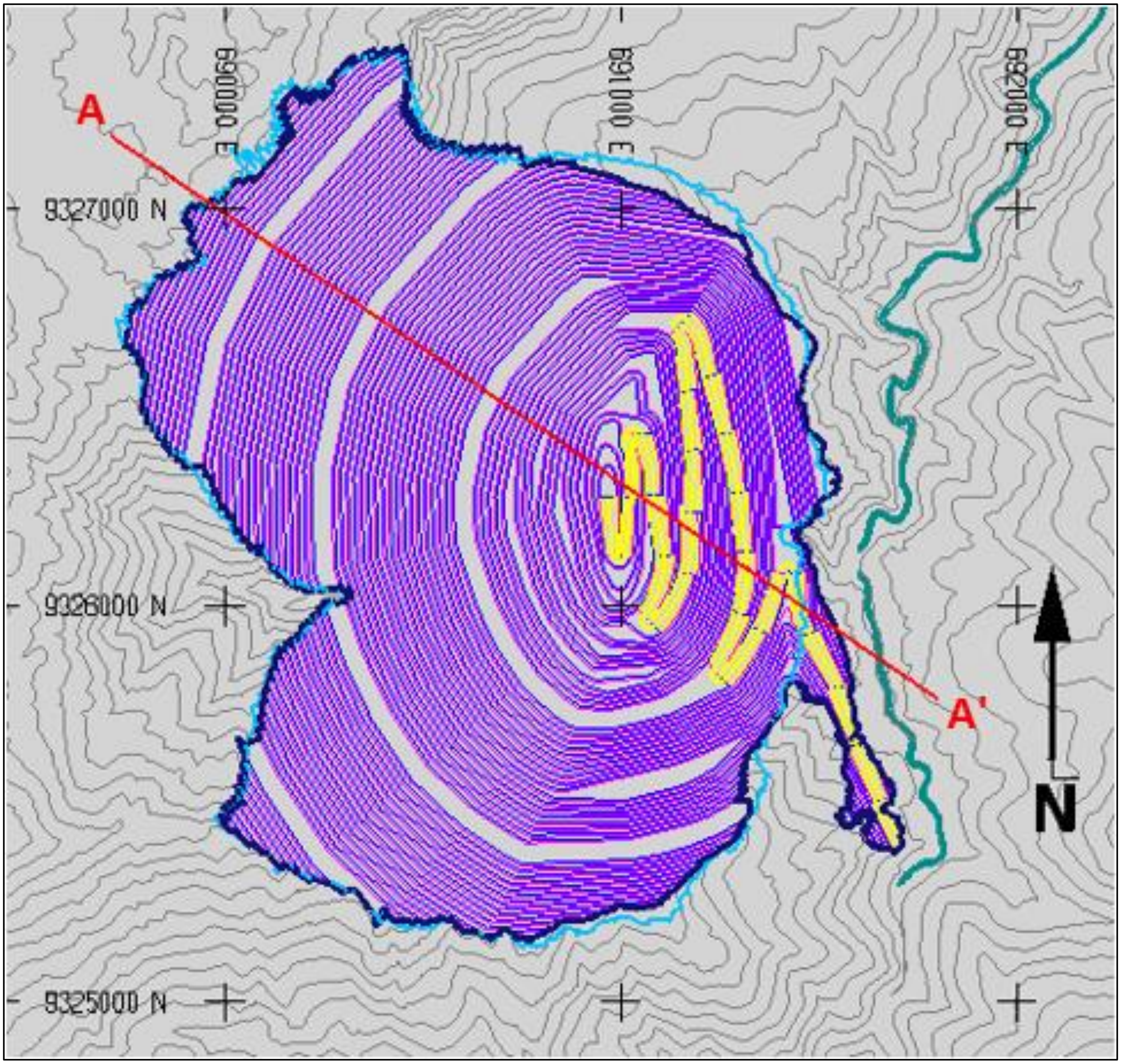


Note: Figure prepared by AGP 2022.

16.7.1 Ultimate Pit Design

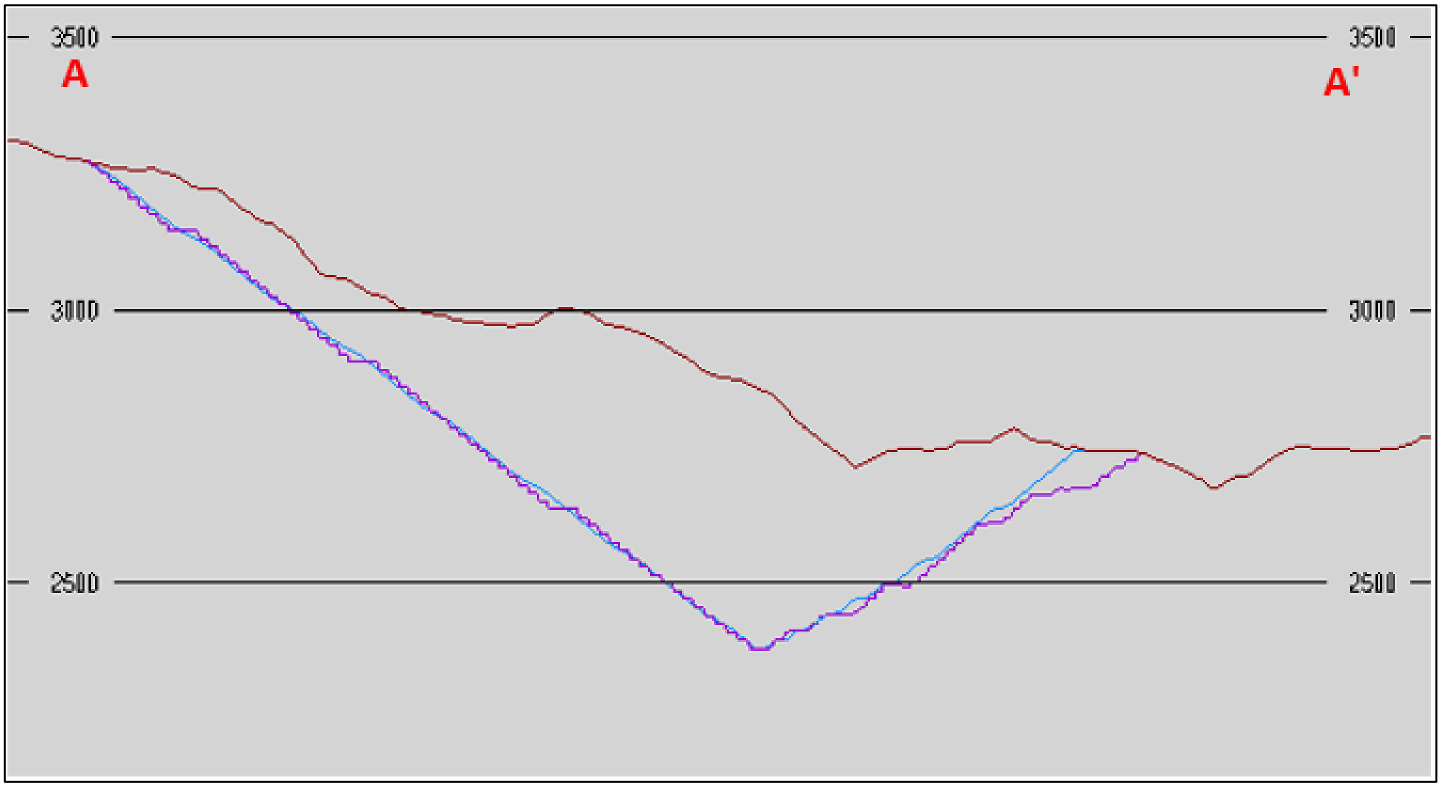
The ultimate pit design was based on the RF 0.55 LG shell. The overall dimensions of the ultimate pit are approximately 2,000 m in the north–south direction, 1,700 m in the east–west direction and a 930 m maximum depth. It will have a single ramp exit point at the 2,680 m elevation, providing access to the mill feed and waste crushers, as well as the truck shop and fuel bay. The ultimate pit design is shown with the RF 0.55 shell (in light blue) in plan and vertical section view in Figure 16-6 and Figure 16-7, respectively.

Figure 16-6: Final Pit Design



Note: Figure prepared by AGP, 2022. Section line A–A' is location of Figure 16-7.

Figure 16-7: Section View of Ultimate Pit Design and RF 0.55 Shell



Note: Figure prepared by AGP, 2022. Section line location is provided on Figure 16-6.

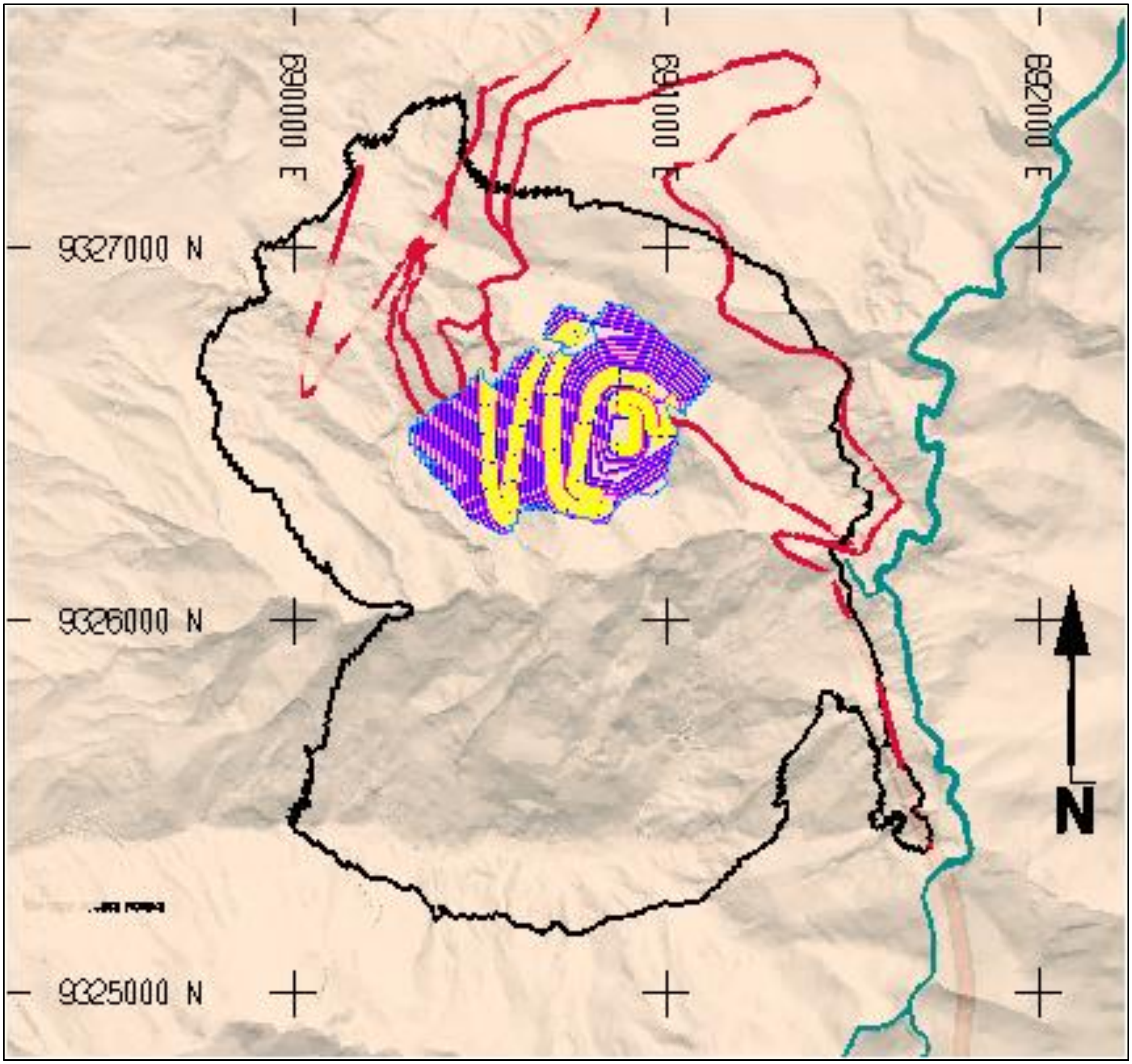
Volumetrically, the ultimate pit contains 1.2% more mill feed tonnes, 0.4% less NSR grade, 1.7% less waste tonnes and 0.8% more total NSR value than the RF 0.55 guiding LG shell.

16.7.2 Internal Phase Designs

The phasing strategy was to limit the first two pushbacks to the north 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 Phase 1 design is a small starter pit, in the northern portion of the ultimate footprint. It will have a crest elevation of 3,155 m, a bottom elevation of 2,755 and a ramp exit elevation of 2,790 m. 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 and the pit pioneering roads needed to access Phases 1 and 2 in red in Figure 16-8.

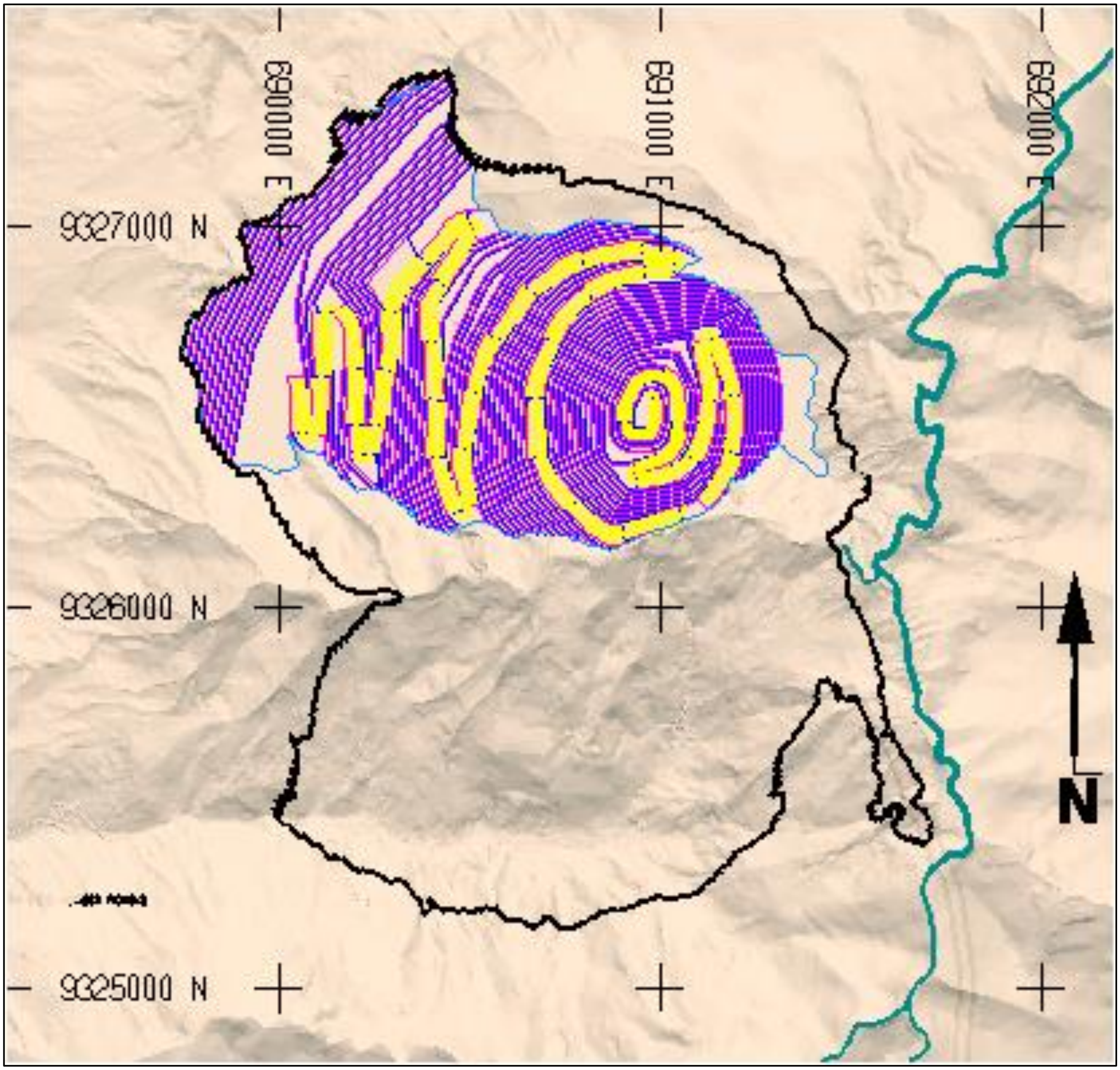
Figure 16-8: Phase 1 Design



Note: Figure prepared by AGP 2022

The Phase 2 design will mine deeper in the northern portion of the ultimate pit. Due to the steep and incised topography, it was necessary for the Phase 2 design to mine back to the ultimate pit limit at the top, to establish a practical minimum mining width. 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 3,320 m, a bottom elevation of 2,575 and a ramp exit elevation of 2,725 m. Phase 2 will be mined from Year -2 through Year 14. The Phase 2 design is shown Figure 16-9.

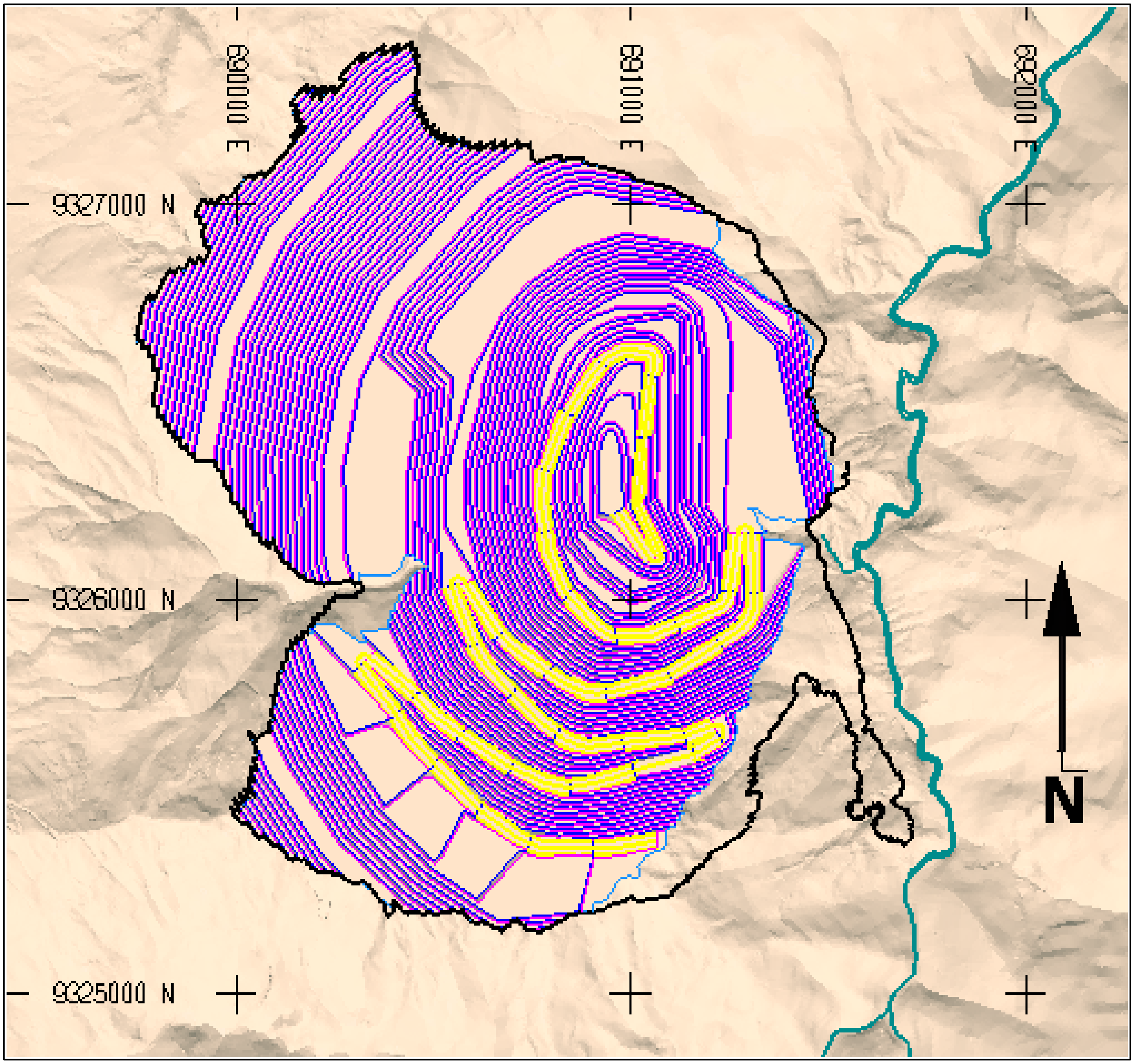
Figure 16-9: Phase 2 Design



Note: Figure prepared by AGP, 2022

The Phase 3 design will cross Quebrada Norte and mine the southern portion of the ultimate pit. As with Phase 2, Phase 3 will mine back to the ultimate pit limit at the top, to establish a practical minimum mining width. A ramp will be left in the highwall to provide access for the ultimate pit phase. Phase 3 will have a crest elevation of 3,255 m, a bottom elevation of 2,500 m and a ramp exit elevation of 2,710 m. Phase mining will start in production Year 2 and end in Year 20. The Phase 3 design is shown Figure 16-10.

Figure 16-10: Phase 3 Design



Note: Figure prepared by AGP 2022.

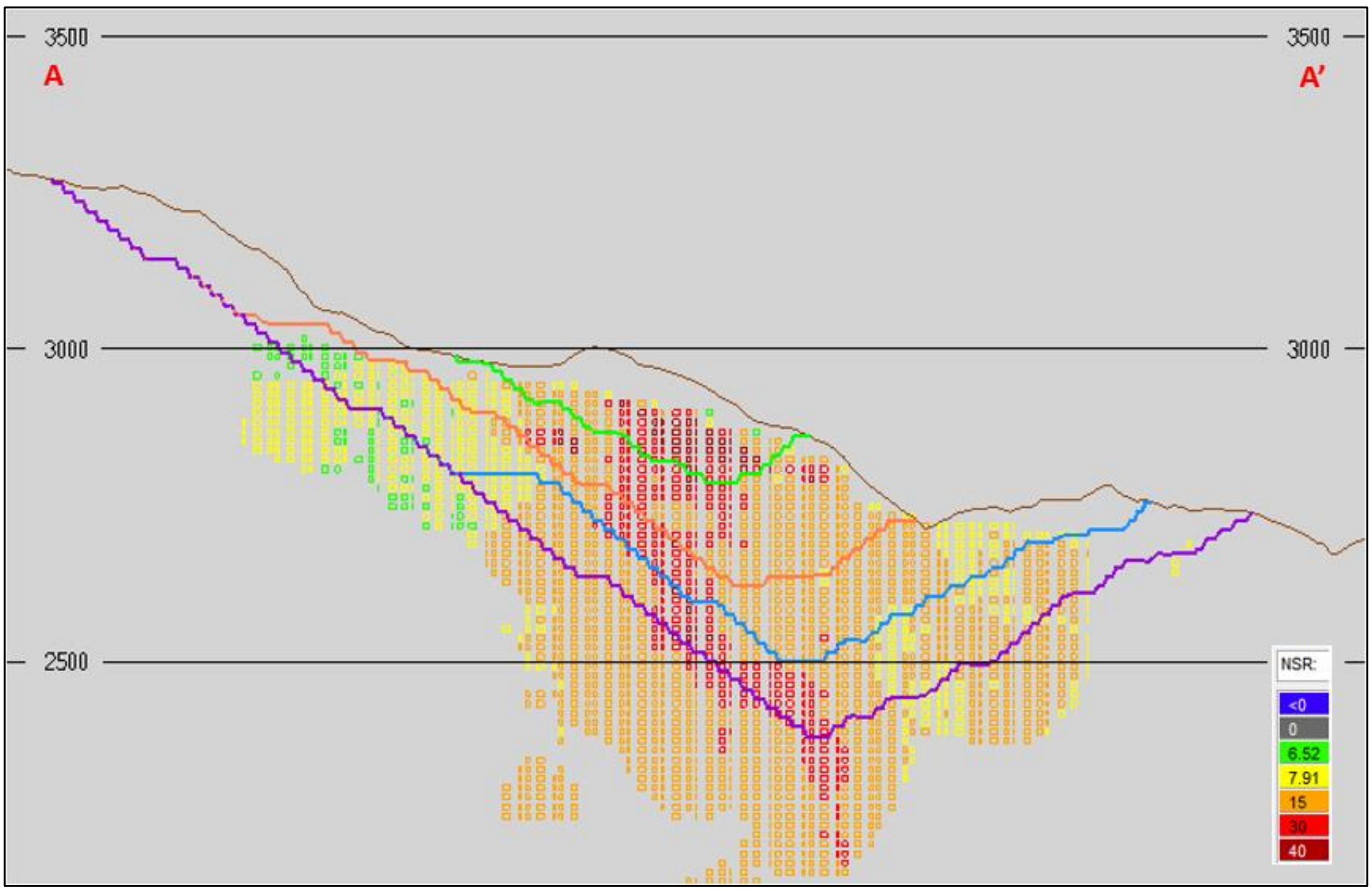
The pit phase volumetrics are shown in Table 16-10. The four pit phases are shown with NSR grade blocks in vertical section view in Figure 16-11.

Table 16-10: Pit Phase Volumetrics

	Mill Feed Above $6.52/t NSR Cut-off						Waste (kt)	Total Material (kt)	Strip Ratio
	kt	NSR (US$/t)	Cu Dil (%)	Au Dil (ppb)	Ag Dil (ppm)	As Dil (ppm)			
PH1	20,200	31.87	0.57	110	2.6	329.44	20,800	41,000	1.03
PH2	115,600	25.09	0.45	90	1.9	189.17	89,900	205,500	0.78
PH3	248,800	19.98	0.38	70	1.7	197.68	250,400	499,100	1.01
PH4	320,800	18.10	0.35	50	1.5	196.21	100,800	421,600	0.31
Grand Total	705,300	20.30	0.38	70	1.7	199.39	461,900	1,167,200	0.65

Note: the total pit phase volumetrics does not match the total mine plan quantities due to an elevated cut-off ($7.91/t NSR) being used during the first six years of production while the concentrator is operating at 40 kt/d. Total material mined differs slightly from that shown in the mine plan due to scheduling software differences. 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 16-11: Pit Phases with NSR Block Grades



Note: Figure prepared by AGP 2022.

16.7.3 Cut-off Grades

The mine planning has been performed based on marginal cut-offs applied to the NSR grade item. During the initial Project phase when the concentrator is operating at 40 kt/d, the marginal cut-off is the sum of the mill feed based operating costs, (processing, G&A and tailings management), which is US$7.91/t. When the process rate increases to 80 kt/d, the marginal cut-off decreases to US$6.52/t. No long-term stockpiling of low-grade material has been considered.

Within the ultimate pit, at the US$6.52/t NSR cut-off the classification breakdown of the mill feed material is 54% Measured, 38% Indicated and 8% Inferred.

16.8 Production Schedule

The mine plan presented in this Report was developed using Hexagon MinePlan 3D's Schedule Optimizer. 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.

16.8.1 Pre-Production

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

- 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 WSF. Year -3 activities consist of road building/pioneering.
- Strip 35.0 Mt of waste rock from the pit, exposing initial mill feed material, and hauling the waste directly to the base of the WSF. The pre-stripping will be performed in Years -2 and -1.

16.8.2 Production

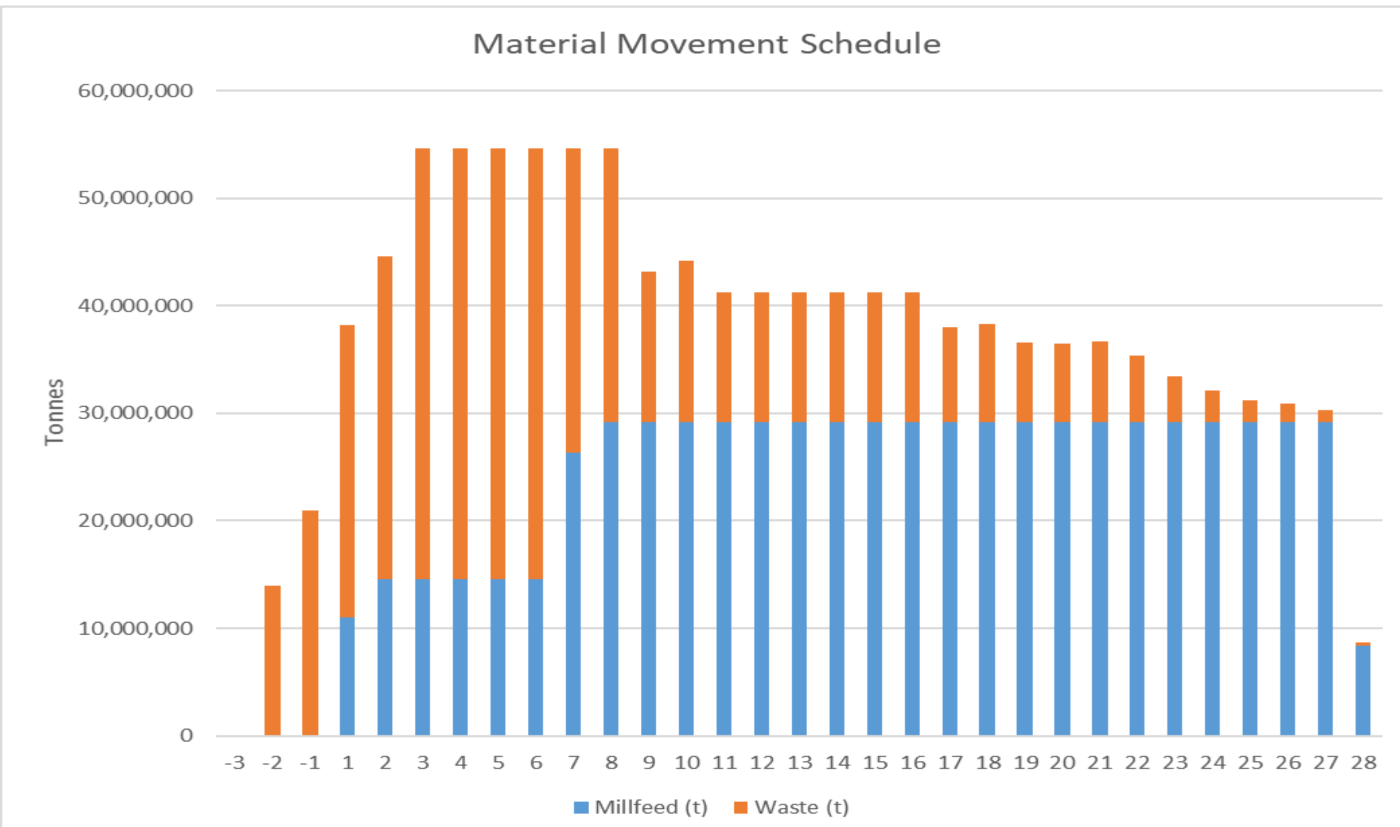
Mill feed delivery to the crusher in the first production year is forecast at 11.1 Mt. In production Year 2 through Year 6, the full 14.6 Mt (40,000 t/d) will be delivered to the crusher area. In Year 7, the production rate will increase to 26.3 Mt. During Year 8 through Year 27, the full 29.2 Mt (80,000 t/d) will be delivered. The last year of production, Year 28, will be a partial year with 8.2 Mt processed. The mine material movement is shown in Table 16-11 and Figure 16-12. Mill feed tonnes and grades are shown in Table 16-12. Select end-of-period surfaces are shown in Figure 16-13 to Figure 16-18.

Table 16-11: Mine Material Movement Summary

Year	Mill Feed (kt)	Waste (kt)	Total Material (kt)	Year	Mill Feed (kt)	Waste (kt)	Total Material (kt)
-3	0		0	14	29,200	12,000	41,200
-2	0	14,000	14,000	15	29,200	12,000	41,200
-1	0	21,000	21,000	16	29,200	12,000	41,200
1	11,008	27,239	38,247	17	29,200	8,797	37,997
2	14,600	30,000	44,600	18	29,200	9,059	38,259
3	14,600	40,000	54,600	19	29,200	7,405	36,605
4	14,600	40,000	54,600	20	29,200	7,307	36,507

Year	Mill Feed (kt)	Waste (kt)	Total Material (kt)	Year	Mill Feed (kt)	Waste (kt)	Total Material (kt)
5	14,600	40,000	54,600	21	29,200	7,440	36,640
6	14,600	40,000	54,600	22	29,200	6,141	35,341
7	26,300	28,300	54,600	23	29,200	4,236	33,436
8	29,200	25,400	54,600	24	29,200	2,928	32,128
9	29,200	14,000	43,200	25	29,200	2,013	31,213
10	29,200	15,000	44,200	26	29,200	1,745	30,945
11	29,200	12,000	41,200	27	29,200	1,042	30,242
12	29,200	12,000	41,200	28	8,428	246	8,673
13	29,200	12,000	41,200	Grand Total	702,736	465,296	1,168,032

Figure 16-12: Mine Material Movement

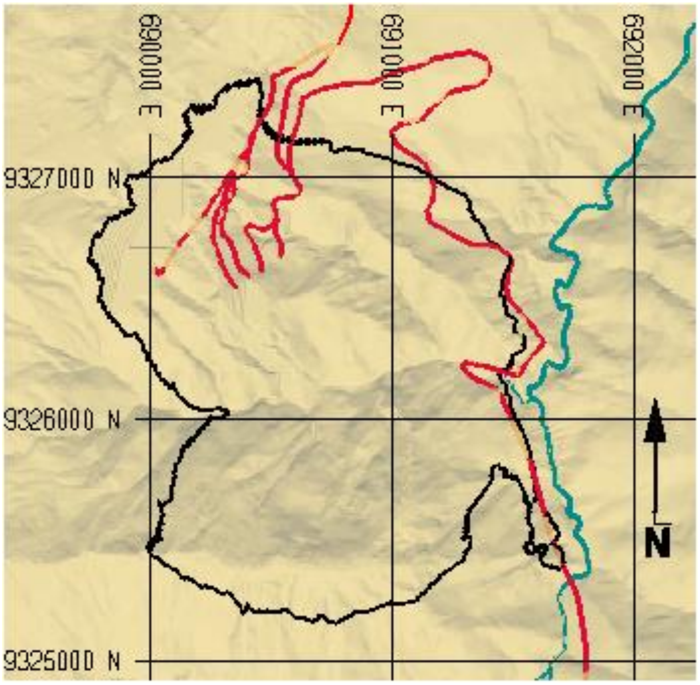


Note: Figure prepared by AGP 2022.

Table 16-12: Mill Feed Summary with Diluted Grades

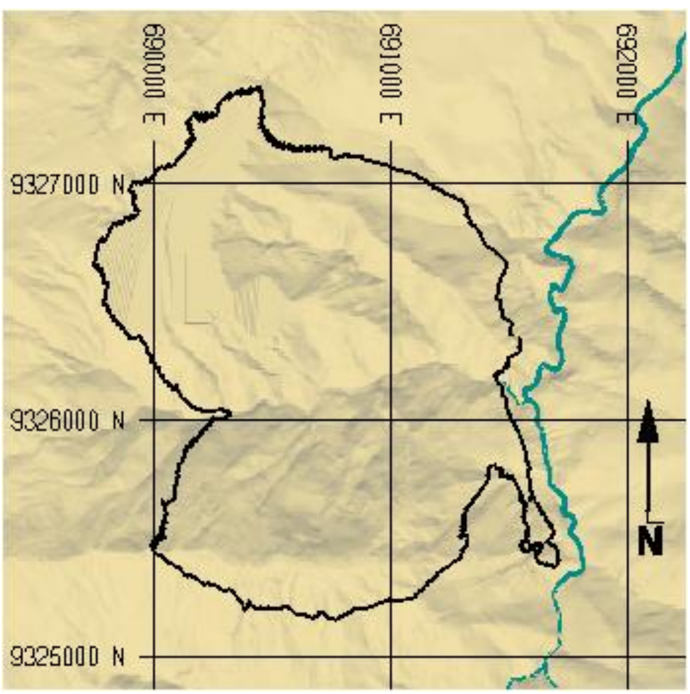
Year	Mill Feed (kt)	NSR ($/t)	Cu (%)	Au (ppb)	Ag (ppm)	As (ppm)
1	11,008	30.35	0.55	110	2.8	391.34
2	14,600	26.91	0.50	90	2.4	316.84
3	14,600	19.37	0.38	60	1.9	318.57
4	14,600	26.63	0.48	80	2.0	234.12
5	14,600	27.71	0.50	90	2.1	212.29
6	14,600	22.92	0.42	80	1.8	183.14
7	26,300	21.56	0.39	80	1.6	150.25
8	29,200	15.10	0.30	60	1.5	217.19
9	29,200	18.30	0.35	60	1.7	254.51
10	29,200	24.72	0.44	90	1.6	133.25
11	29,200	20.36	0.40	60	2.0	289.23
12	29,200	23.35	0.43	70	2.0	207.69
13	29,200	23.06	0.43	70	1.9	206.88
14	29,200	20.86	0.38	70	1.7	134.93
15	29,200	21.95	0.40	80	1.7	146.50
16	29,200	21.53	0.39	70	1.5	115.68
17	29,200	16.87	0.30	60	1.2	89.84
18	29,200	10.37	0.23	30	1.1	221.02
19	29,200	12.79	0.28	30	1.3	276.89
20	29,200	15.86	0.34	40	1.6	313.83
21	29,200	18.89	0.38	50	1.7	301.25
22	29,200	19.77	0.39	50	1.7	235.43
23	29,200	19.52	0.37	60	1.6	156.12
24	29,200	19.78	0.37	60	1.	140.47
25	29,200	23.16	0.42	70	1.9	167.15
26	29,200	21.64	0.39	70	1.8	131.25
27	29,200	22.21	0.40	70	1.5	82.31
28	8,428	24.32	0.45	70	1.8	187.48
Grand Total	702,700	20.35	0.38	70	1.7	199.07

Figure 16-13: Year -2 End of Period Surface with Pioneering Roads



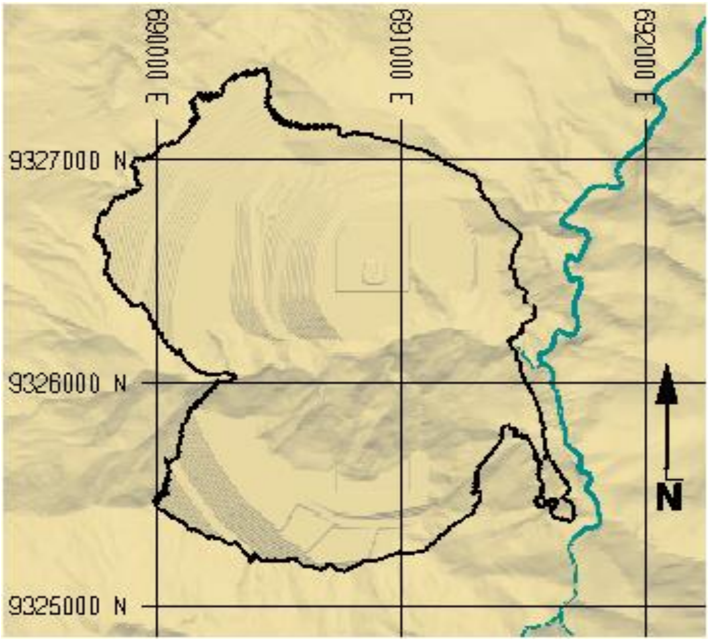
Note: Figure prepared by AGP 2022.

Figure 16-14: Year -1 End of Period Surface



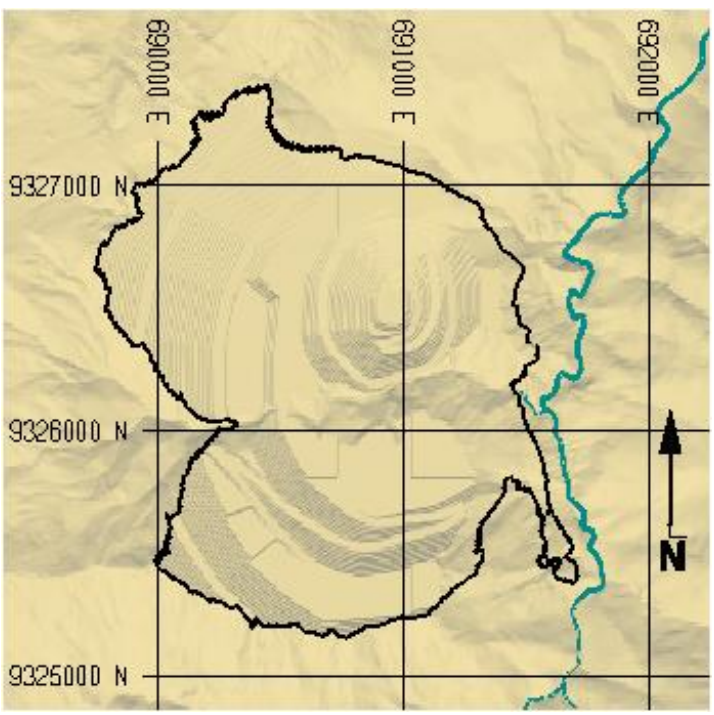
Note: Figure prepared by AGP 2022.

Figure 16-15: Year 5 End of Period Surface



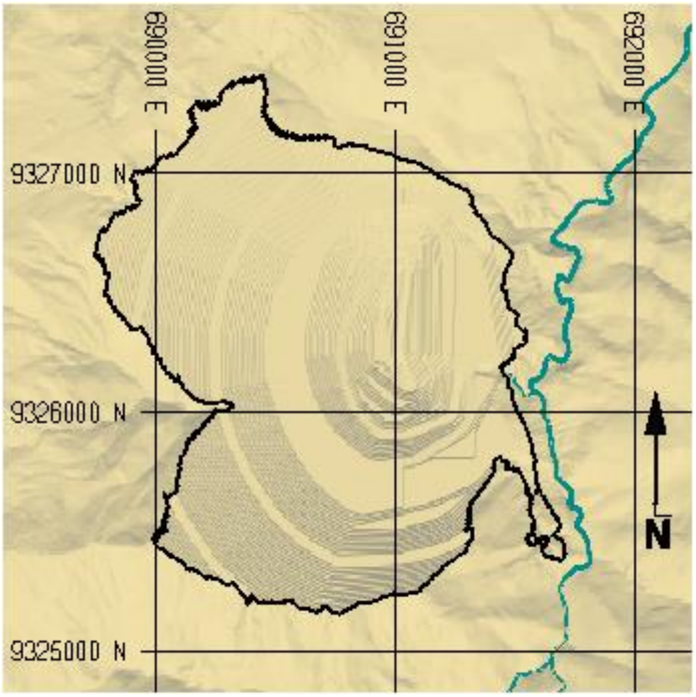
Note: Figure prepared by AGP 2022.

Figure 16-16: Year 10 End of Period Surface



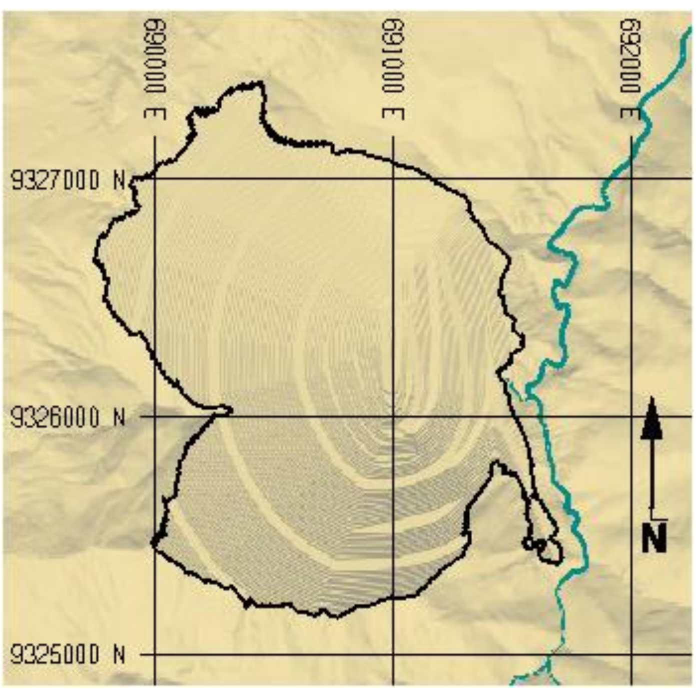
Note: Figure prepared by AGP 2022.

Figure 16-17: Year 20 End of Period Surface



Note: Figure prepared by AGP 2022.

Figure 16-18: Year 28 End of Period Surface



Note: Figure prepared by AGP 2022.

16.9 Mining Operations

The Cañariaco Norte deposit is 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 16-13.

Table 16-13: Primary Production Equipment

Equipment Type	Equipment Class	Maximum Fleet Size
Haul truck	218t	18
Hydraulic shovel	35 m^3	2
Front-end loader	33 m^3	1
Track dozer	4.7-m blade	5
Grader	4.3-m blade	3
Blast hole drill	38 t pulldown	3
Secondary drill	25 t operating wt., 160 mm bit	1

16.9.1 Loading

Production loading duties will be performed by 35 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 equipment is well matched to the 15-m bench height. The peak loader requirements are two shovels and one front end loader.

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 a high percentage of downhill loaded hauling vs uphill loaded hauling. Electric drive haul trucks were selected over mechanical drive as a better fit for the significant downhill hauling requirements. A 218-t truck was selected as it matched well to the loading tools. A peak of 18 trucks will be required in Year 1.

An opportunity exists to investigate autonomous haulage during the next stage of project planning.

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 three primary drills and one secondary drill.

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 16-14.

Table 16-14: 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 1.2 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 228 hourly operators and maintenance workers and 54 staff. Additionally, there will be nine blasting contractors on site at all times. The peak total mine operations workforce in camp is 155 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 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.

17 RECOVERY METHODS

17.1 Introduction

The processing plant will be built in two phases. Phase 1 will have a nominal throughput of 14.6 Mt/a during the first six production years. Phase 2 will double the annual production by adding a parallel line, from the mill feed stockpile to concentrate and tailing filtration and will have the same equipment and layout as the Phase 1 process line. The plant is designed to operate 24 hours per day, 365 days per year with an overall plant availability of 92%.

The process plant selected is a conventional copper concentrator and the process design is typical of concentrator treating copper sulfide ores. The process plant feed will be supplied from the open pit mine with a LOM average feed grade of 0.38% Cu and will produce a copper concentrate containing 26% Cu.

The process plant includes the following units, processes, and facilities:

- Primary crushing of ROM mill feed and waste rock;
- Overland conveyor system to transport mill feed and waste, in discrete lots, to tripper conveyor and separate stockpiles near the process plant;
- Coarse ore stockpile and reclaim system;
- Coarse waste stockpile and reclaim system;
- Mill feed material grinding including a SAG mill, pebble recycle crusher and ball mill in closed circuit with hydrocyclones;
- Copper flotation;
- 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 conveying with waste rock in the WSF;
- Reagent storage and distribution; and
- Water services and management (process water, treated water, raw water).

17.2 Plant Design

The process plant has been designed in accordance with established good engineering practices for traditional grinding and flotation plants. Where data were not available at the time of flowsheet development, Ausenco's criteria for the sizing and equipment selection are based on comparable industry applications, benchmarking, and the use of modelling and simulation techniques. Selection and sizing of the crushing and grinding circuits were determined through analysis of comminution test work performed in the previous study stage. Test work provided CWi, BWi and Bond rod mill indexes, Ai, SMC, and JK Axb values for selected samples.

The process plant is designed to treat a nominal 40,000 tpd initially, increasing to 80,000 tpd (or 29.2 Mtpa) total throughput at Year 7.

Key design criteria used in the plant design are summarized in Table 17-1, which also summarizes the forecast feed grade and recovery data.

Table 17-1: Key Design Criteria

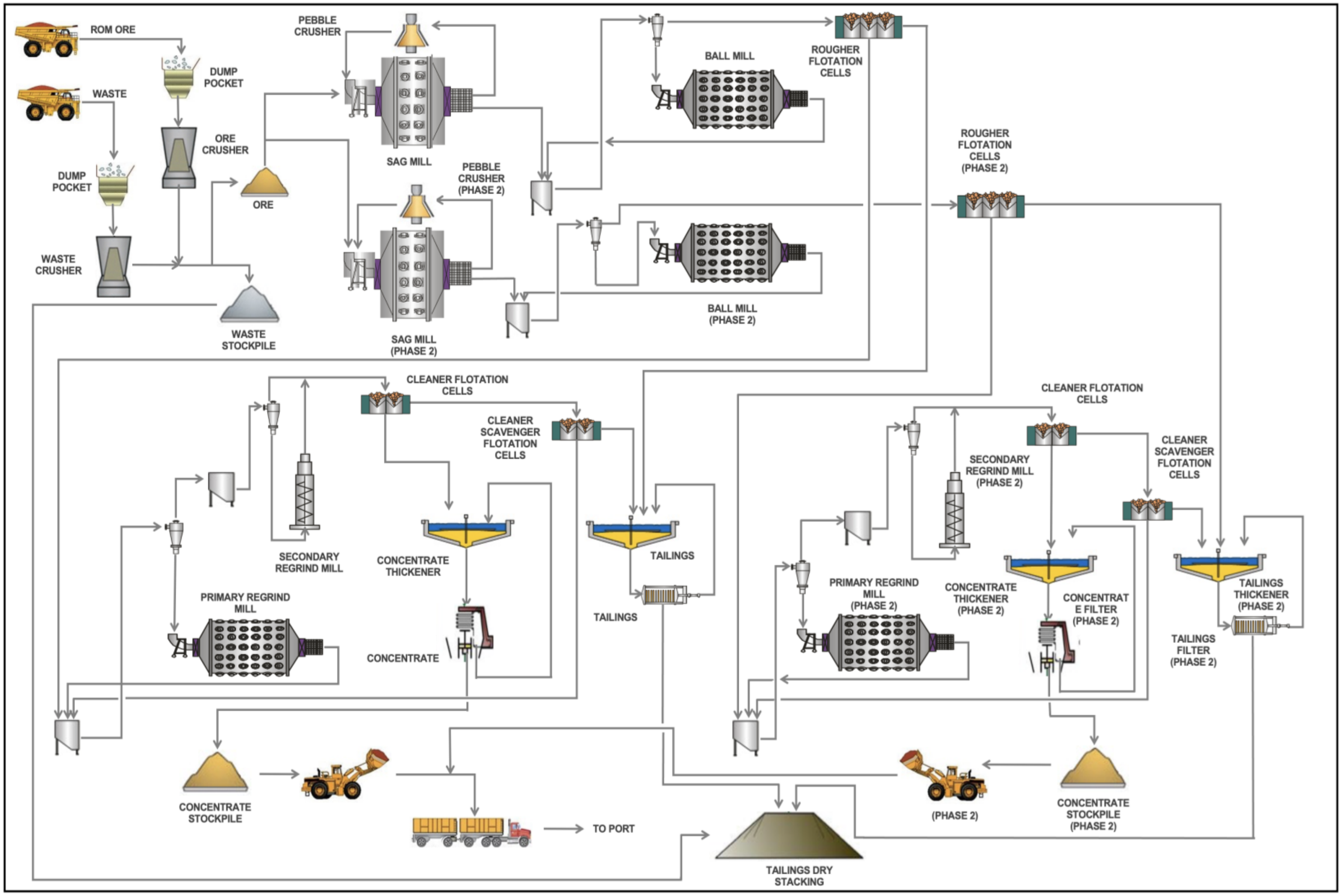
Description	Units	Phase 1	Phase 2
Plant design capacity	Mt/y	14.6	29.2
Life of mine (LOM)	years	1–6	7–28
Crushing availability	%	75	
Grinding and flotation availability	%	91.3	
Concentrate filters availability	%	84	
Tailings filters availability	%	98.6	
Copper feed grade (average)	%	0.47	0.37
Gold feed grade (average)	g/t	0.08	0.06
Silver feed grade (average)	g/t	2.14	1.63
Copper recovery, nominal	%	89.9	87.6
Gold recovery, nominal	%	70.1	62.9
Silver recovery, nominal	%	63.2	55.9
Concentrate grade, Cu	%	26	26
Concentrate grade, Au	g/t	3.6	3.2
Concentrate grade, Ag	g/t	83.5	73.2
Impact crushing work index (CWi)	kWh/t	12.2	12.2
JK SMC test parameters Axb	–	68	68
Bond ball mill work index	kWh/t	11.1	11.1
Primary crusher F_{80}	mm	529	529
Primary crusher type	–	gyratory	gyratory
Primary ore crusher P_{80}	mm	74	74
Primary waste crusher P_{80}	mm	155	155

Description	Units	Phase 1	Phase 2
Grinding circuit configuration	–	SABC	SABC
Grinding P_{80}	µm	200	200
Pebble crusher type	–	cone crusher	cone crusher
Pebble recycle rate, normal operating	% of new feed	22	22
Ball mill	–	overflow	
Ball mill circulating load, nominal	%	300	
Rougher recovery, mass	%	15.5	13.0
Primary regrind mill	–	overflow ball mill	overflow ball mill
Primary regrind mill circuit	open / closed	closed	closed
Overflow product size, P80	µm	50	50
Secondary regrind mill	–	High intensity grinding mill	High intensity grinding mill
Secondary regrind mill circuit	open/closed	open	
Overflow product size, P80	µm	25	open
Cu Cleaner 1 recovery, mass	%	18.9	25
Cu Scavenger recovery, mass	%	5.3	21.5
Cu Cleaner 2 recovery, mass	%	58.2	7.4
Cu Cleaner 3 recovery, mass	%	82.7	50.9
Filtered Tailings moisture	%	15	15

17.3 Process Flow Sheet

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

Figure 17-1: **Process Flowsheet**



Note: Figure prepared by Ausenco 2021.

17.4 Processing Description

17.4.1 Crushing/Conveying

The primary crushing circuit will consist of a single gyratory crusher per crushing line for mill feed and waste material, treating 14.2 Mt/a (dry) each. Run-of-mine (ROM) material will be delivered into the dump pocket of the crusher by mine haul trucks with a live capacity of up to 352 t.

Each primary crusher will receive ROM (F80 of 529 mm) and reduce it to a P80 of 74 mm. Crushed material will fall into the crusher discharge vault below the crusher, which will discharge into an apron feeder. The crusher discharge feeder will draw mineralization from the crusher discharge vault at a controlled rate using a variable-speed drive and will discharge to the crusher sacrificial conveyor.

Crushed mineralization and waste will be conveyed in discrete lots on the overland conveyor to a tripper conveyor where the mill feed material and waste will be split and deposited in their respective stockpiles.

The overland conveyor No.1 and No.2 were sized for a design capacity of 8760 t/h to be able to handle the waste and mill feed tonnages as outlined in Table 16-11 over the LOM.

17.4.2 Coarse Mill Feed/Waste Stockpile and Reclaim

The coarse mill feed and the waste from the primary crushers will be fed to separate stockpiles that each will have a total storage capacity of approximately 250,000 t. The nominal live capacity of the stockpiles will be 62,500 t. The stockpile has been sized to provide surge capacity between the mining and crushing operations and the process plant so that the plant can continue to receive mineralization from the stockpile when there are intermittent stoppages of feed from the crusher or the mine. A track dozer will push mill feed material into the feeders when mineralization needs to be reclaimed from the dead section of the stockpiles during upstream maintenance activities.

Mill feed will be reclaimed from the mineralized stockpile with two in-line apron feeders installed in a concrete tunnel below the stockpile. Two additional in-line apron feeders will be installed once the plant is expanded to the 80 kt/d throughput and will deliver crushed mill feed material to new concentrator train. The reclaim feeders will discharge crushed mineralization at a controlled rate onto the semi-autogenous grind (SAG) mill feed conveyor. The feeder will have the capacity to provide 100% of the full base case tonnage rate (1,825 t/h) to the SAG mill but will generally operate at 50% capacity (913 t/h) for even draw down of the stockpile.

The SAG mill feed conveyor will deliver the reclaimed mineralization from the stockpile to the SAG mill and the oversize material from the pebble crusher to the SAG mill.

Crushed waste will be reclaimed from the waste stockpile with two in-line apron feeders installed in a concrete tunnel below the stockpile. The reclaim feeders will discharge waste at a controlled rate onto the crushed waste conveyor and will be blended with flotation tails on the conveyor and transported to the WSF.

17.4.3 Grinding and Classification

The final grinding circuit will consist of two parallel lines with the second line being installed for the expansion. Each grinding line consist of a single SAG mill, followed by a single ball mill operating in closed circuit with cyclone cluster and will treat, on average, a total throughput of 40 kt/d. The product of the griding circuit will have a nominal particle size of 80% passing 200 µm.

Each SAG mill will operate in closed circuit with a screen and a pebble crushing station. The screen will be fitted with water sprays to assist the separation of fine slurry from coarse rock or pebbles. Screen undersize will gravitate to the ball mill cyclone feed hopper where it will be combined with the discharge from the ball mill. Screen oversize (pebbles) from each SAG mill will be transported by conveyor to dedicated pebble crushing stations.

The SAG mill will be a 36-ft (11 m) diameter x 19-ft (5.8 m) effective grinding length (EGL) mill. Lime will be added to the SAG mill feed to suppress the flotation of pyrite while recovering the copper, gold, and silver minerals in the rougher flotation circuit.

SAG mill discharge at a pulp density of 70% w/w solids will pass over a vibrating screen with an aperture size of 12 x 50 mm and spray water. Undersize from the vibrating screen will report to the ball mill cyclone feed hopper where it will be combined with the ball mill discharge and pumped to the ball mill cyclones. Screen oversize, consisting of pebbles greater than 12 mm, will be washed of fines on the screen. The washed pebbles in the vibrating screen oversize will feed the pebble crusher, with an optional by-pass that will allow the oversized material to recycle back to the SAG mill feed conveyor. The pebble crusher will consist of a cone crusher with a circulating load of 22% of the fresh feed rate. The conveyors are designed to handle peak loads of up to 30% of fresh feed.

Each grinding line will be equipped with a dedicated cyclone feed hopper and single variable-speed cyclone feed pump that will transport slurry to its associated ball cyclone cluster. Process water for dilution will be added to the cyclone feed hoppers before the slurry is pumped to the respective cyclone cluster for classification. The ball mills in each grinding line will operate in closed circuit with the primary cyclone clusters.

The overflow ball mill will be 22-ft (6.7 m) diameter x (10.7 m) 35-ft EGL. A ball mill trommel screen will protect each cyclone feed pump from large rocks or balls that may be discharged from the mills during upset conditions. Ball mill discharge will join the undersized SAG mill discharge in the ball mill cyclone feed hopper.

The classification circuit was designed for a nominal circulating load of 300%, operating with a target overflow of 80% passing particle size of 200 µm. The cyclone overflow will gravitate to the trash screen. The cyclone underflow streams will report to the respective ball mill feed chute.

17.4.4 Copper Flotation

The final flotation circuit will consist of two parallel lines with the second line being installed for the Year 7 expansion.

The flotation circuit is designed to recover the minerals containing copper, gold, and silver into a bulk concentrate. Concentrate will subsequently be transported to port facilities for commercial sale. The design of the flotation circuit maximizes the recovery of these valuable minerals while minimizing the overall capital and operating costs.

Depending on the lithology and mineralogy, the mill feed material could contain amounts of clay minerals and pyrite. Copper minerals and some pyrite, along with entrained and locked non-sulphide gangue, will be recovered in rougher flotation and then be reground to further liberate the minerals from each other. The rougher tailings will be sent to the tailing thickener.

Flotation feed from the trash screen will report to the rougher flotation circuit. MIBC frother will be added to the rougher feed stream, and additional lime will be added to the roughers to achieve a pH of approximately 10. A mixture of A3894 and potassium amyl xanthate (PAX) will be added as a collector. A similar reagent scheme will be used in the cleaning stage.

The rougher flotation will produce a low-grade copper concentrate that will require further mineral liberation for upgrading. The average 80% passing particle size in the rougher circuit will be 200 µm with a feed pulp density of 34% w/w solids. At 40 kt/d, the total rougher residence time will be 27 minutes.

The copper rougher tailings stream will be combined with the cleaner scavenger tailings and thickened.

Rougher concentrate will be combined with the cleaner scavenger concentrate in the primary regrind cyclone feed pumpbox and pumped to the primary regrind cyclones for classification. The underflow from the primary regrind cyclones will report to the primary regrind mill, which will overflow back into the primary regrind cyclone feed pumpbox. The overflow from the primary regrind cyclone will report to the secondary cyclone feed pumpbox. Secondary regrind cyclone overflow will report to the first stage of cleaner flotation while the underflow will report to the secondary regrinding mill prior to reporting to the first stage of cleaner flotation.

The reground rougher concentrate will be cleaned in a bank of 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 product, will be pumped to the concentrate thickener. The third cleaner tailings will return to the second cleaner flotation to maximize copper recovery.

17.4.5 Copper Concentrate Thickening and Filtration

The final concentrate thickening and filtration circuit will consist of two parallel lines with the second line being installed for the Year 7 expansion.

The copper concentrate thickening and filtration circuit will consist of a single 79 ft (24 m) diameter high-rate thickener and a pressure filter. Copper concentrate will gravitate from the copper flotation 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.

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.

17.4.6 Tailings Thickening and Filtration

The final tailings thickening and filtration circuit will consist of two parallel lines with the second line being installed for the Year 7 expansion.

The tailings thickening circuits will consist of 213 ft (65 m) diameter high compression thickeners (HCT) to thicken the slurry to 60% solids (w/w) and will recover process water to the process water tank.

The copper rougher tailings streams will be combined with the copper cleaner scavenger tailings and pumped to the tailings thickener. Flocculant will be added to the thickener feed to enhance settling. Thickener overflow will gravitate to the process water tank for subsequent use in the grinding, copper flotation and tailing filtration circuits. The thickened underflow will be pumped to the tailings filter feed.

The tailings filter feed tank will feed the tailings filter presses. The tailings will be dewatered to 15% moisture and conveyed to a transfer station where they are mixed with crushed waste rock on the overland conveyor to the WSF.

A dedicated air compressor will supply high pressure air for the tailings filter. Filter membrane pressing will be supplied by the filter pressing water pump. The filtrate will then be recycled to the tailing thickeners.

17.4.7 Reagents and Consumables

The reagents used within the process plant will include:

- Dialkyl thionocarbamate (A3894) - liquid flotation collector and promoter;
- MIBC - frother reagent;
- Quicklime – used for pH control;
- PAX – collector reagent;
- Flocculant – reagent used for solids settling purposes; and
- Sodium carbonate – reagent used for pH buffer purposes.

All reagent solutions will be prepared and stored in a bermed containment area in a separate reagent preparation and storage facility. The reagent storage tanks will be equipped with level indicators and instrumentation to ensure that spills do not occur during preparation or operation. Ventilation, fire and safety protections will be provided at the facilities.

The liquid reagents (including MIBC and A3894) will be added in undiluted form to various process circuits via individual metering pumps. The solid reagents, including flocculant and PAX, will be mixed with fresh water to their solution strengths setpoints, respectively, in separate mixing tanks and stored in holding tanks before being added into the process circuits at various points using metering pumps. Quicklime will be slaked on site from bulk pebble quicklime, diluted to a 20% strength milk of lime and distributed to various addition points from a circulating loop.

Flocculant will be dissolved, diluted to the target strength, and added to various thickener feeds using metering pumps.

The major consumables used within the process plant will include:

- Crusher liners
- SAG and Ball Mill liners
- SAG and Ball Mill Media

17.4.8 Services

To the greatest extent possible the process plant will re-use process water recovered from the WSF to meet process plant requirements. Raw water will only be used where water quality with low dissolved solids is required and as make-up in the process water circuit.

Raw water will be supplied from the water treatment plant for the following applications:

- Fire water for emergency use;
- Cooling water for mill motors and mill lubrication systems;
- Reagent preparation; and
- Gland seal water.

Air systems for the milling operation will be as follows:

- A high-pressure air compressor will supply the primary crushing area to provide air for dust collection systems;
- High pressure air for various plant services will be supplied by dedicated air compressors;
- High pressure air for filter pressing and drying of copper and pyrite concentrates will be supplied by dedicated air compressors;
- Low pressure air for flotation cells will be supplied by blowers; and
- Instrument air will be dried and stored for use at the main process plant site.

17.5 Power

The power requirements for the project are discussed in Section 18.10

18 PROJECT INFRASTRUCTURE

18.1 Introduction

The mine site facilities are divided into four general areas: the mine, the crusher (that include buildings and structures for repair and maintenance of mine and plant equipment), the plant site, and the camp area, (which includes facilities for personnel accommodations, administration, and security). Other support facilities and services include site access, power supply and distribution, water supply, explosive storage and handling, communication systems and waste disposal facilities at the site.

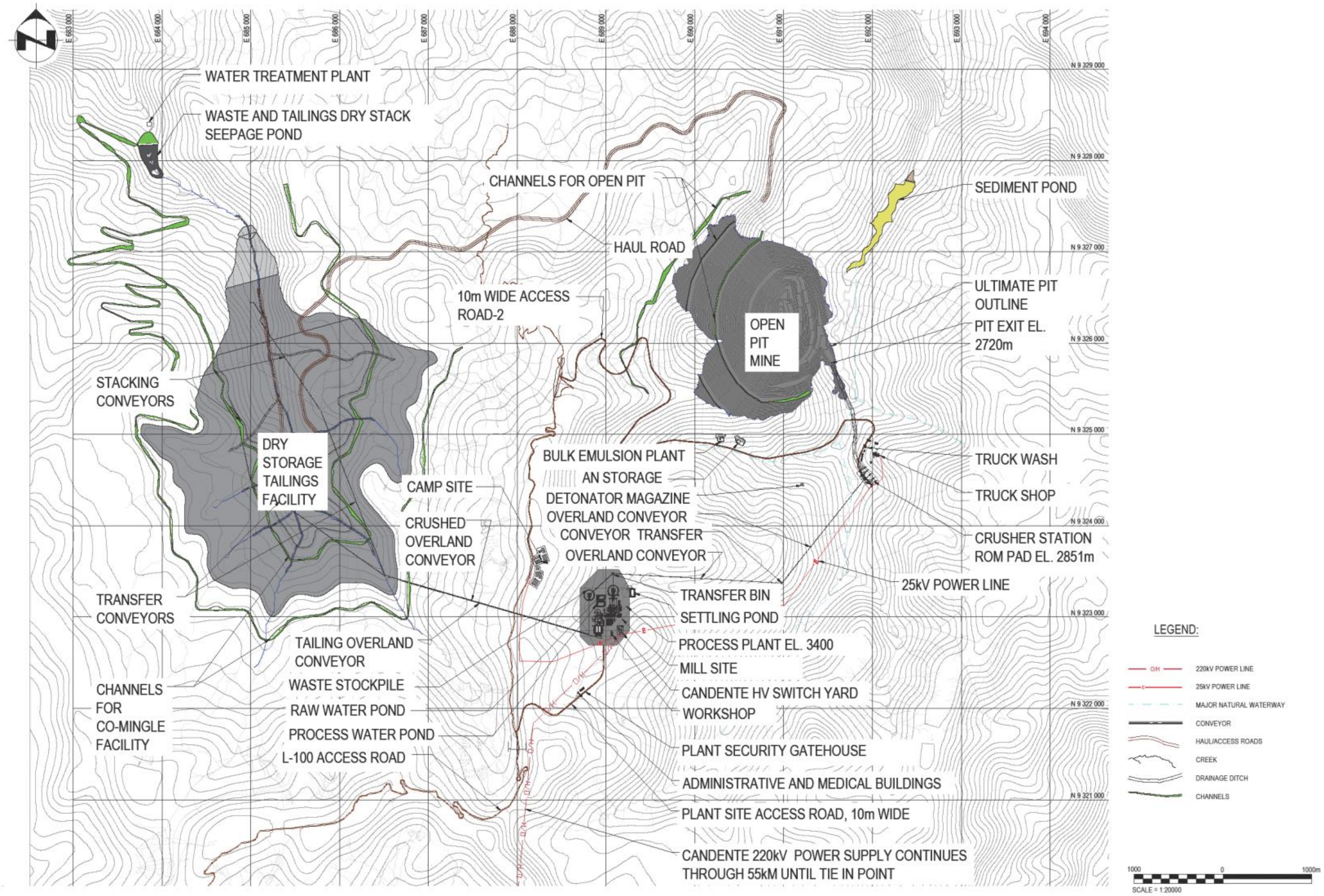
The plant and camp sites were selected on a natural elevated area close to existing road and away from watersheds. The mine infrastructure area (MIA) and crusher/ROM pad are located near the ultimate pit to minimize the hauling distance.

Key factors considered for layout and site selection included:

- Process plant to be on a safe place at flooding period;
- Manned facilities located out of blasting radius;
- The location of the crusher/ROM pad near the mine pit;
- Separating heavy mine vehicle traffic from non-mining light-vehicle traffic; and
- Locate permanent camp close to access road and process plant.

An overall general arrangement of the development sites is shown in Figure 18-1.

Figure 18-1: Overall Site Plan



Note: Prepared by Ausenco 2022.

18.2 Roads and Logistics

18.2.1 Plant Access Road

Road access to the site will be via the Corral Quemado Road, which runs along the Huancabamba Valley. The main access road to the plant site and camp site will be constructed off the Corral Quemado Road. The 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, 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, service roads to remote structures, and haul roads. During the life of the mine, roads and lay down areas will change as needed The general access roads will be two-way and 8 m wide, the service roads will be one-way and 5 m wide with pullouts, and the haul roads will be one-way or two-way depending on requirements and have a travel width that will accommodate the Komatsu 830E-5 or similar haul truck.

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.

18.2.3 Existing Roads

The site is currently accessible via an existing road from Chiclayo that generally parallels the Pan-American Highway. The routing from Chiclayo to the Cañariaco Project includes a combination of paved and gravel roads for approximately 150 km, passing through numerous small villages. This access can be particularly difficult during heavy rains because of the poor drainage and low-quality road conditions.

The road is currently unsuitable for heavy trucks and therefore cannot be used for construction access. Construction traffic will need to wait until the new access road connecting the site to the Corral Quemado Road is completed.

18.3 Stockpiles

The mine plan does not utilize long term stockpiling of low-grade mill feed material. Mine stockpiling is limited to a small short-term stockpile near the crushers which acts as a buffer between the mine and the crushers.

18.4 Topsoil Stockpiles

The topsoil stockpiles will be strategically placed around the mine site. The stockpiles will contain stripped materials from all excavations from the project development. The stockpiled materials will be used for rehabilitation applications at closure. Collection ditches and sedimentation ponds will be built to capture suspended solids in runoff from these piles before releasing the runoff into the environment.

18.5 Co-disposal Waste Rock and Tailings Storage Facility

18.5.1 Design Basis

The primary design objectives for the co-disposal waste rock and tailings storage facility (WSF) are secure confinement of the waste materials (both waste rock and filtered tailings in a co-mingling, mixed together, and co-placement, placed separately, manner) and protection of the regional groundwater and surface water during both mine operations and in the long-term (after closure). The design of the WSF and water management facilities has taken into account the following:

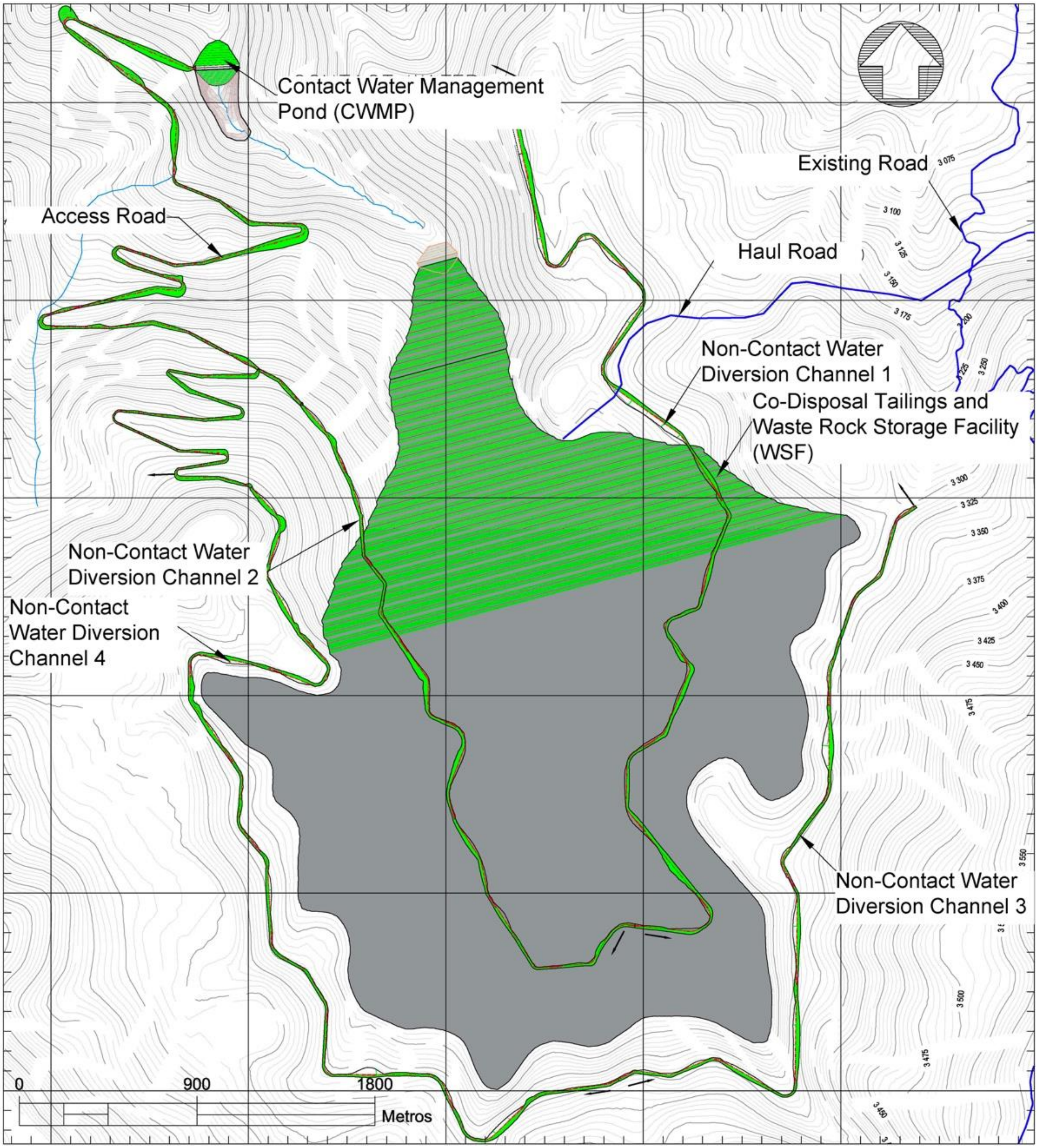
- Staged development of the facility over the life of the Project;
- Flexibility to accommodate operational variability in the filtered tailings (filter plant shutdowns and mill feed variability, moisture content variability, along with placement during variable climate conditions;
- Co-mingle and co-placement of waste stream to improve overall physical stability of the waste storage facility and sediment management;
- Reduction, control, collection, and removal of water from the facility during operations for recycle as process water to the maximum practical extent; and
- Integration of metal leaching and acid rock drainage (ML/ARD) mitigation measures to provide long term geochemical stability for sulfide containing waste rock and tailings.

Approximately 697 Mm^3 of mine waste will be stored within the WSF, including 439 Mm^3 of filtered tailings and 258 Mm^3 of waste rock. The WSF is not expected to behave like a conventional mine waste facility because of the variability of tailings and waste rock placed on an annual basis, In the initial years the waste rock generation exceeds tailings production so the materials will be mixed together along with exterior berms of waste rock. In the latter years the tailings production exceeds the waste rock generations and perimeter berms constructed of waste rock to provide containment of the filtered tailings and provide erosion protection for the overall structure. A portion of the tailings will be co-mingled with waste near the exterior slope. The construction of perimeter berms will provide a number of benefits:

- Filtered tailings that do not meet moisture content or density targets will not have an impact on overall stability of the facility. The primary requirement for the filtered tailings will be the ability to transport the material to the facility and trafficability for subsequent placement; and
- The perimeter embankment will provide freeboard for collection of annual run off, along with directing runoff to contact water channels located along the perimeter built within the facility.

The general arrangement of the WSF is shown on Figure 18-2.

Figure 18-2: Waste Storage Facility General Arrangement



Note: Figure prepared by Ausenco 2022.

18.5.2 Hazard Classification

The design standards for the WSF 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
- British Columbia Mine Waste Rock Pile Research Committee "Investigation and Design Manual" (BCMWRPRC, 1991).

The WSF was classified as significant or moderate under both CDA and BCMWRPRC guidelines. The recommended IDF during operations is defined as between the 24-hour 1/100 to 1/1000-year return period flood for significant dam classification. The IDF chosen based on our experience was the 1/475-year return period, which is 113 mm. EDGM parameters were determined for the WSF using estimates from the seismic hazard assessment (Knight Piésold, 2012). The design earthquake is characterized as between the 1/100 to 1/1000-year return period for significant dam classification. The subsequent peak ground acceleration utilized was the 1/475-year return period (PGA), which is 0.44 *g*.

18.5.3 Tailings and Waste Rock Characteristics

Tailings and waste rock characterizations are based on historical testing programs and are shown in Table 18-1.

Table 18-1: Tailings and Waste Rock Characteristics

Description	Units	Phase 1
Tailings		
Specific gravity	unitless	2.6
Deposition void ratio	unitless	0.5
Deposition dry density	t/m^3	1.60
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	0.2
Deposition dry density	t/m^3	1.83
Moisture content of waste rock	%	varies
Waste rock gradation (P_{80})	mm	155
Classification	USCS	GP (Poorly Grade Gravel)
Friction angle	°	32

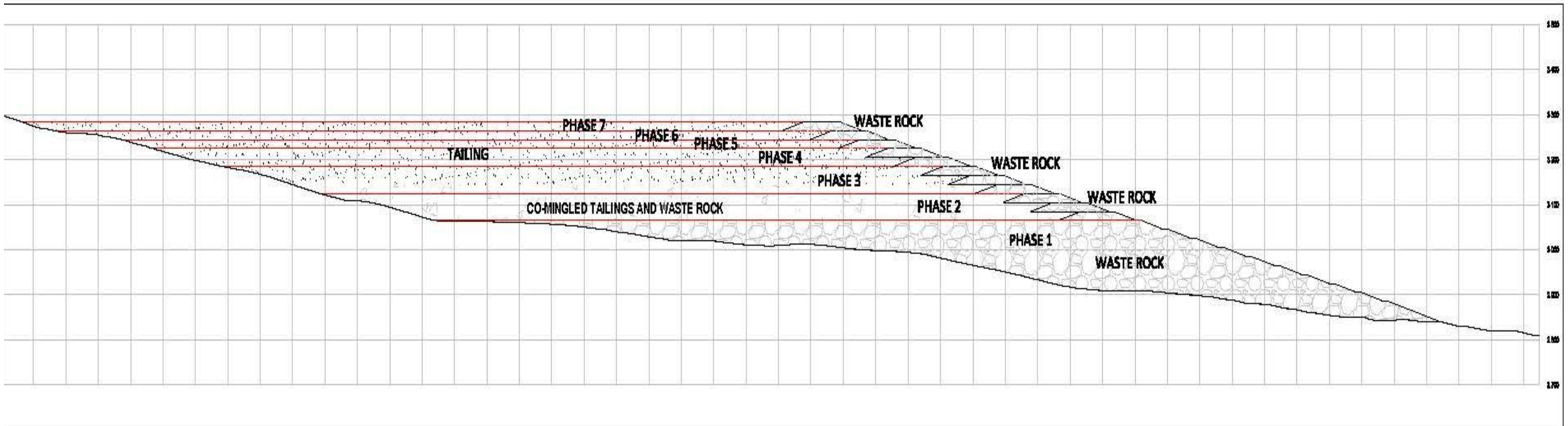
Note: USCS = Unified Soil Classification System; ML = inorganic silt with low to medium compressibility.

The tailings are classified as a non-plastic inorganic silt with a low permeability when compacted at the proposed filtered moisture content. The waste rock is classified as a poorly graded gravel with a moderate permeability.

18.5.4 Facility Design

During pre-production, mine waste rock will be trucked to the waste storage facility. During operations, mine waste will be crushed and conveyed to the plant site by conveyor. The mine waste will be transferred to the WSF conveyor to be placed in the WSF. It is assumed that the pre-production waste rock will be non-acid generating (NAG), and any PAG waste rock will be mixed to mitigate ML/ARD. During operations the waste rock delivered to the plant area will be conveyed by itself to the WSF to build exterior berms or conveyed with filtered tailings for co-mingle (mixed) disposal in the WSF using two conveyor systems (refer to Figure 18-3Figure 18-3: Waste Storage Facility Schematic Section). Based on the potential acid generation of materials, calcium carbonate will be added as part of the ARD mitigation measures. It is assumed that PAG materials will become more prevalent with depth. After Year 6, there will be insufficient waste rock to co-mingle with the tailings waste stream. Therefore, the waste rock will be used to develop exterior berms and the balance of waste rock will be co-mingled with tailings near the front of the facility to improve the facility physical stability.

Figure 18-3: Waste Storage Facility Schematic Section



Note: Figure prepared by Ausenco 2022..

The WSF will be constructed in seven phases over the 28-year mine life. The WSF will includes a contact water management pond (CWMP) downstream of the facility, surface water management structures, rock drains, access road, waste conveyor system, mechanical equipment for spreading (dozers) and compacting (compactors) materials, and a water reclaim pipeline for the CWMP. The WSF and CWMP footprint areas he will be cleared of organic material and topsoil for foundation preparation. Basin preparation will include removal of any unsuitable materials (soft soils and saturated soils) and compaction of the subgrade in the CWMP footprint and the first 200 m of the WSF starter platform. It is assumed that less than an average 1 m of overburden removal will be required over the footprint of the CWMP and toe area of the WSF.

The WSF will initially be constructed during pre-production using 25.6 Mm^3 of NAG waste rock to build the rock drains and the WSF platform to support the deposition of waste rock and tailings during operations. The platform will be constructed to an elevation of approximately 3,065 masl. The platform will also promote drainage of the tailings and waste rock at the front of the WSF, maintaining a low phreatic head in the downstream portion of the facility.

A foundation drainage network (rock drains with high-density polyethylene (HDPE) pipe) will be developed within the base of the facility using selective placement of NAG waste rock (see Section 18.6). In addition, a non-contact water diversion channel will be constructed to intercept surface runoff and convey it around the WSF, which remains functional till the end of Year 5. Between the rock platform and the Initial non-contact water diversion channel, topsoil and unsuitable soil removal will on an as needed basis (approximately 3 months in advance) for stack waste rock and tailings, to minimize the cleared area footprint and sediment production. In addition, within the WSF around the perimeter contact water channels will be constructed to convey surface runoff from the WSF to the CWMP.

The ultimate non-contact water diversion channel will be constructed in Year 2. Again, the topsoil and unsuitable soil removal and contact water diversion channel will be constructed as needed based on the stacking plan.

18.5.5 Waste Placement

During production, crushed waste will be conveyed to the plant site where it will either be sent directly to the WSF for berms or rock drains and the tailings placed alone as co-placement or combined with dewatered tails as co-mingled and stacked in the WSF. The waste rock and tailings production rates are shown in Table 16-11.

Crushed waste will be reclaimed from the waste stockpile with two in-line apron feeders installed in a concrete tunnel below the stockpile. The reclaim feeders will discharge waste at a controlled rate onto the crushed waste conveyor. The waste materials will be stacked in 5-meter lifts and the surface spread and compacted to reduce infiltrations and promote runoff to the contact water diversion channels.

18.5.6 Monitoring

Instrumentation and monitoring will be required to assess embankment performance. Vibrating wire piezometers will be installed to monitor pore pressure within the embankment fill materials and slope inclinometers and survey monuments will be installed to monitor slope movement and deformation.

18.6 Waste Storage Facility Water Management

The surface of the filtered tailings and waste rock within the WSF will be graded and compacted to encourage flow to defined surface water collection points. 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. Any water in excess of the process plant requirement (if any) will be directed to the water treatment plant (WTP) and released.

Water that infiltrates the surface of the facility will fill any void space remaining within the waste materials and may end up contributing to elevation of 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 underdrain and HDPE pipe 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 WSF to capture surface water above the facility and convey it below the CWMP to discharge into the watershed below the facility. The channels are design to convey up to the 1:100-year storm event. The channel has a light vehicle service road next to the channel.

18.7 Water Supply

18.7.1 Fresh Water

Raw water for potable and process make-up, and other uses at the mine site, will be obtained from the pit stormwater pond filled by pit surface water runoff and rainwater and from the WSF CWMP. The water from stormwater pond will be pumped to crusher area water tank and to the raw water pond in the process plant area. The WSF CWMP downstream of this facility. The water from raw water pond will be pumped from a barge to the fresh/fire gravity head tank.

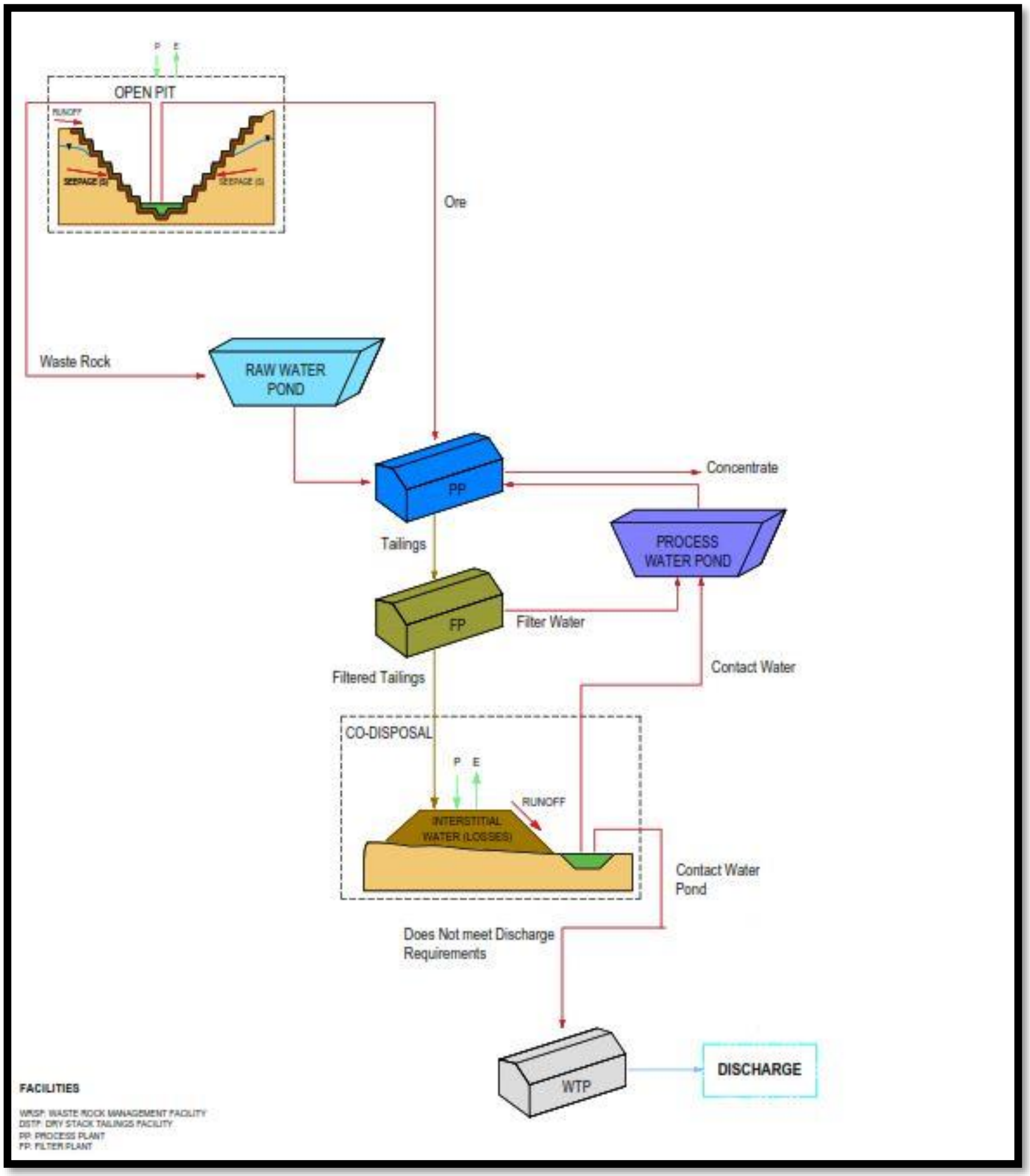
18.7.2 Process Water

The process plant will require 625 m^3/hr to a maximum of 2,228 m^3/hr over the proposed 28 year mine life. Process water for the plant will come from three sources: the CWMP below the WSF, reclaimed water from the tailings filter plant, and from the open pit. The open pit, filter plant and CWMP will be the primary of process water sources.

18.7.3 Site Wide Water Balance

A site-wide water balance was completed to estimate the quantity of mine site contact water expected to be managed during operations (Figure 18-4).

Figure 18-4: Site Wide Water Balance Schematic



Note: Figure prepared by Ausenco 2022..

The site-wide water balance used a model developed in GoldSim® on a monthly timestep basis for the operational phase (i.e. the LOM excluding the pre-production phase). For stochastic analysis, GoldSim® uses the Monte Carlo method to generate monthly precipitation and evaporation lake time series for the water balance model.

The water balance model considered the precipitation and groundwater gains, and evaporation and infiltration losses (where applicable) of the following mine components: open pit, WSF, process plant and associated infrastructure. The site-wide water balance study provided a water management strategy that was focused on the following main objectives:

- Estimating the effluent flows of the main mine components;
- Estimating the capacity of the required water treatment plant (WTP);
- Estimating the volume of the water storage ponds.

The model's main inputs were obtained from meteorology reports and other available information regarding hydrogeology, operation strategy, open pit water management, and WSF water management at a PEA level.

The model considered over 100 simulations to obtain acceptable probabilistic estimates of the effluent flows for the mean (monthly average) and 95% non-exceedance probability (wet condition) scenarios:

- Mean (monthly average): the average monthly flow of all simulations.
- 95% non-exceedance probability (p95%): maximum monthly flow with 5% of risk considering all simulations. This means that there is a 5% probability that the reported value will be exceeded in one month of the simulation period. The p95% scenario may independently occur for each mine component, e.g. the WSF experiences a wet condition (p95%), while the low-grade mineralization stockpile experiences the Mean scenario.

Year 1, years 2 to 6, years 7, years 8 to 27, and year 28 were modelled separately due to changes in water demands. The following conclusions were reached:

- The process plant water demands range from 625 m^3/hr to a maximum of 2,228 m^3/hr;.
- A water treatment plant with a capacity to treat up to 144 m^3/hr is required to treat excess effluent;
- The raw water pond, the process water pond and the contact water management pond were sized to meet the Project water supply needs;
- No off-site water supply is required.

18.8 Built Infrastructure

18.8.1 Mine Truck Shop

The mine truck shop will be the main services complex on site. The complex will contain maintenance facilities for the mine mobile equipment fleet, warehouse space, first aid, lockers, lunchroom facilities and a machine shop. The maintenance area will have four large bays for trucks and front-end loaders plus smaller bays for support equipment, welding and light vehicle maintenance. A change room and dispatch area for the mine operations crew will be provided on the second floor and offices on the third floor above one bay of the building.

The mine truck shop will also be used to service all light- and medium-duty vehicles used on the site, including personnel buses, pickup trucks, and other vehicles.

18.8.2 Truck Wash Station

A separate truck wash station will be provided in an open area adjacent to the truck shop for manual truck washing. The facility will consist of a heavy slab on grade with water cannons at the sides, and no roof. A water tank and a pump system will be supplied as part of the package.

18.8.3 Mill Maintenance Shop

The mill maintenance shop will service all stationary mill equipment such as pumps, instruments, and motors. The building will contain the following facilities:

- Electrical and instrumentation shops;
- Machine and millwright shops;
- Welding shop;
- Tool crib and warehouse; and
- Space allowed for a tire repair shop.

18.8.4 Blasting Agent Facilities

The location of the blasting agent facilities including a bulk emulsion plant, an ammonium nitrate storage facility, and a detonator storage magazine will be south of the pit (see Figure 18-1). A bulk emulsion plant and an ammonium nitrate storage facility will be located north of access road and the detonator magazine will be located south of the access road and west of the truck shop. The three sites, and the necessary utilities for the sites, will be provided to a licensed blasting contractor to establish and manage. The contractor will supply packaged explosives, initiation systems, cast primers, and accessories. Each site will be designed with increased security and in accordance with industry standards and guidelines for safe distances of explosives from significant structures. The terrain provides a natural barrier between the blasting facilities and the mine and camp infrastructure. The location, just off the main access road, is convenient for both the delivery of ammonium nitrate and the transport of explosives to the mine site.

18.9 Accommodation

The camp and administration area is 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.

18.9.1 Camp

The camps will be constructed to initially house the approximately 1,140-person construction workforce.

The camps 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.

18.9.2 Administrative Building

The administrative and medical buildings 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. With adequate planning, the facility will be built early in the construction phase and used for construction administration.

18.10 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 55 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 load for operation is:

Phase 1 (Year 1–6):

- Mine = 6MW
- Process Plant and Infrastructure (including tailings and waste rock, surface water management) = 108MW
- Buildings including Camp = 5MW
- Total Connected Load = 119MW

Phase 2 (Inclusive of Phase 1 and Year 7 - 28):

- Mine = 6MW
- Process Plant and Infrastructure (including tailings and waste rock, surface water management) = 160MW
- Buildings including Camp = 5MW
- Total Connected Load = 171MW

Power will be supplied to the various mine facilities through radial feeders originating at the main substation and routed on site through cable tray in pipe racks, 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.

The pit dewatering pumps will be diesel powered.

18.11 Fuel Storage and Distribution

18.11.1 Diesel Fuel Tank

The main diesel fuel storage tanks will be stored in single-walled, above-ground tanks within HDPE-lined, bermed enclosures to provide secondary containment in the event of the sudden tank failure. Spillage resulting from a broken valve or hose will also be contained within the secondary containment.

The refuelling station will be used to refuel mine haul trucks and to fill the fuel tanker, which will refuel the drills and support equipment in the pit. The dispensing station will also accommodate the smaller vehicles and equipment on site.

The fuel dispensing facility will consist of a fuel module and a concrete pad for secondary containment to avoid leakage of fuel to the environment.

18.11.2 Lubricating Oils and Hydraulic Fluids

Lubricating oils, hydraulic fluids, and other potentially hazardous substances will be stored in large on-site storage tanks. Mobile or portable oil storage tanks, drums, and other small containers will be stored in secondary containments capable of holding the contents of the largest container. Empty containers will be returned to suppliers for re use.

19 MARKET STUDIES AND CONTRACTS

19.1 Market Studies

No market studies or product valuations were completed as part of the 2022 PEA. Market price assumptions were based on a review of public information, industry consensus, standard practiced and specific information from comparable operations in the region.

Copper concentrates are widely traded. Concentrates can be marketed directly from producer (mine) to smelter, or through third party concentrate trading entities. The lack of investment in new copper mines and expansions to existing operations contribute to a widely held belief that there will be increasing demand for copper concentrates in this decade and beyond. The expected supply constraint is further exacerbated by a world copper demand that is expected to increase on basis of emerging, copper-dependant technologies, such as electronic vehicles and renewable energy, and continued urbanization of the world's population.

Candente's management were provided with indicative smelter terms. The net payable for a metal is calculated as the payable content of the contained metal, less a minimum deduction of 1% for copper, 0% payable for less than 1 g/t gold and less than 30 g/t silver. The treatment charges (TC) of US$75/ dry tonne and refining charges (RC) of US$0.075/ payable lb Cu, US$5.0/ payable oz Au and US$0.4/ payable oz of silver are deducted from the payable value of the concentrates to account for the costs of smelting and refining. The TC/RCs are influenced by global supply and demand and governed by mine and smelter economics based on metal prices and operating costs. TC/RCs may be based on variable annual negotiations, fixed rates and/or market benchmarks. The following penalty charges were applied per tonne of concentrate with the following Arsenic grades in the concentrate.

- 0.5% to 1.5% As- US$4.0/01% above 0.5%
- 1.5% to 3% As- US$6.0/01% above 1.5% + US$40.0

19.2 Commodity Price Projections

Project economics were estimated based on long-term metal prices of US$3.50/lb Cu, US$1,650/oz Au and US$21.50/oz Ag, which was established by Candente Copper in conjunction with consensus forecasts from various financial institutions.

The QP notes that Candente Copper's pricing used in the cash flow analysis is reasonably aligned with various long-term forward-looking estimates from major Canadian and international banks.

19.3 Contracts

No contracts for transportation or off-take of the concentrates are currently in place, but if and when they are negotiated, they are expected to be within the industry norms. Similarly, there are no contracts currently in place for 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 Environmental Considerations – Existing Environmental Setting

20.1.1 Baseline and Supporting Studies

The information contained in this section is derived from baseline studies, investigations and field work carried out in 2021 by Yaku Consultants for the development of the semi-detailed Environmental Impact Assessment (EIAsd 2021).

The following environmental components were studied:

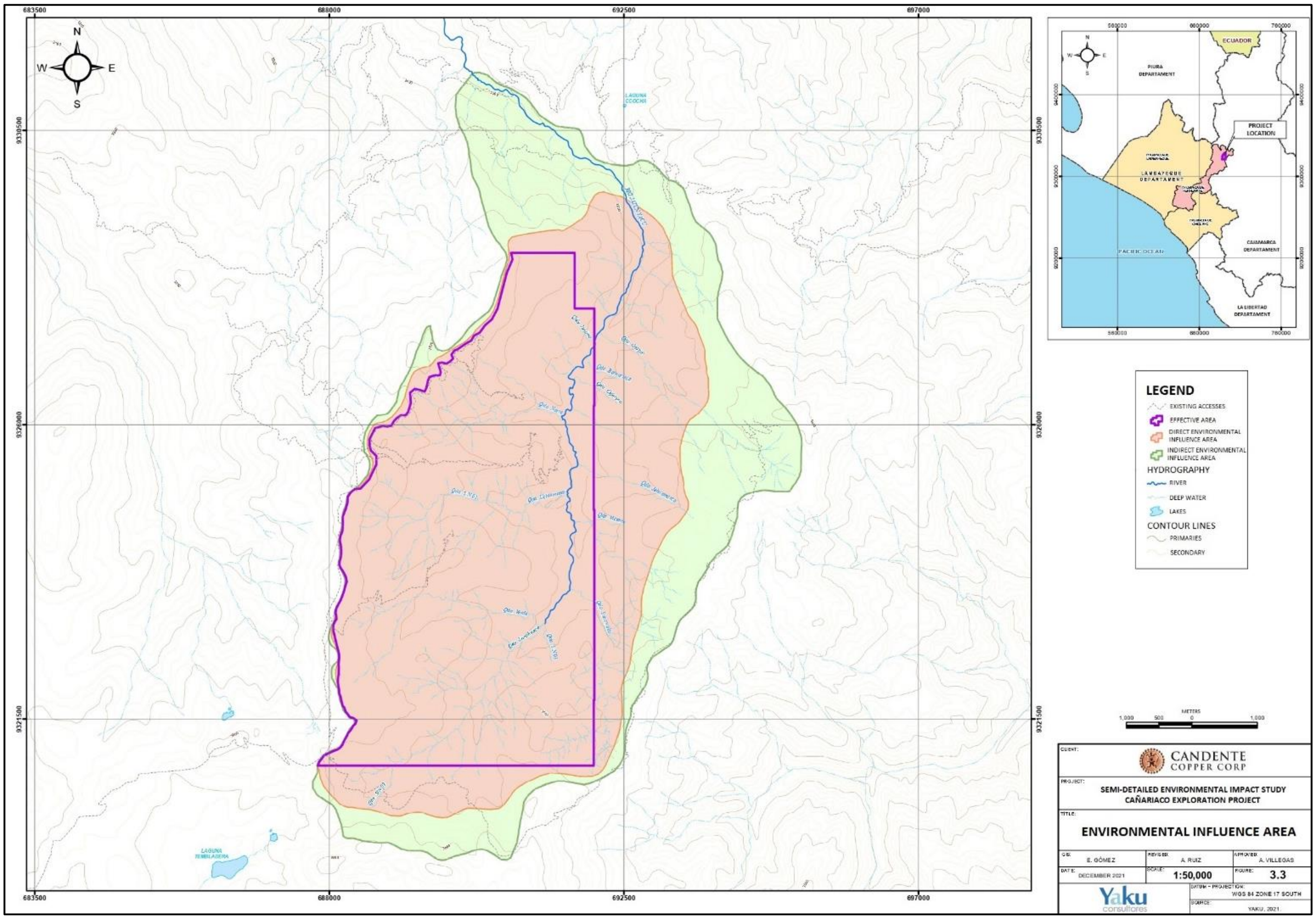
- Physical components: climate and meteorology, air quality and environmental noise, vibrations, soils, geology, geomorphology, hydrology and hydrogeology, and water quality;
- Biological components: ecosystems, flora, fauna and hydrobiology; and
- Socioeconomic and archaeological components.

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.

20.1.1.1 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, disturbed areas are observed in the study area.

Figure 20-1: Environmental Study Area



Source: Figure prepared by Yaku Consultants, 2021.

20.1.2 Environmental Monitoring

20.1.2.1 Physical Setting

20.1.2.1.1 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
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.

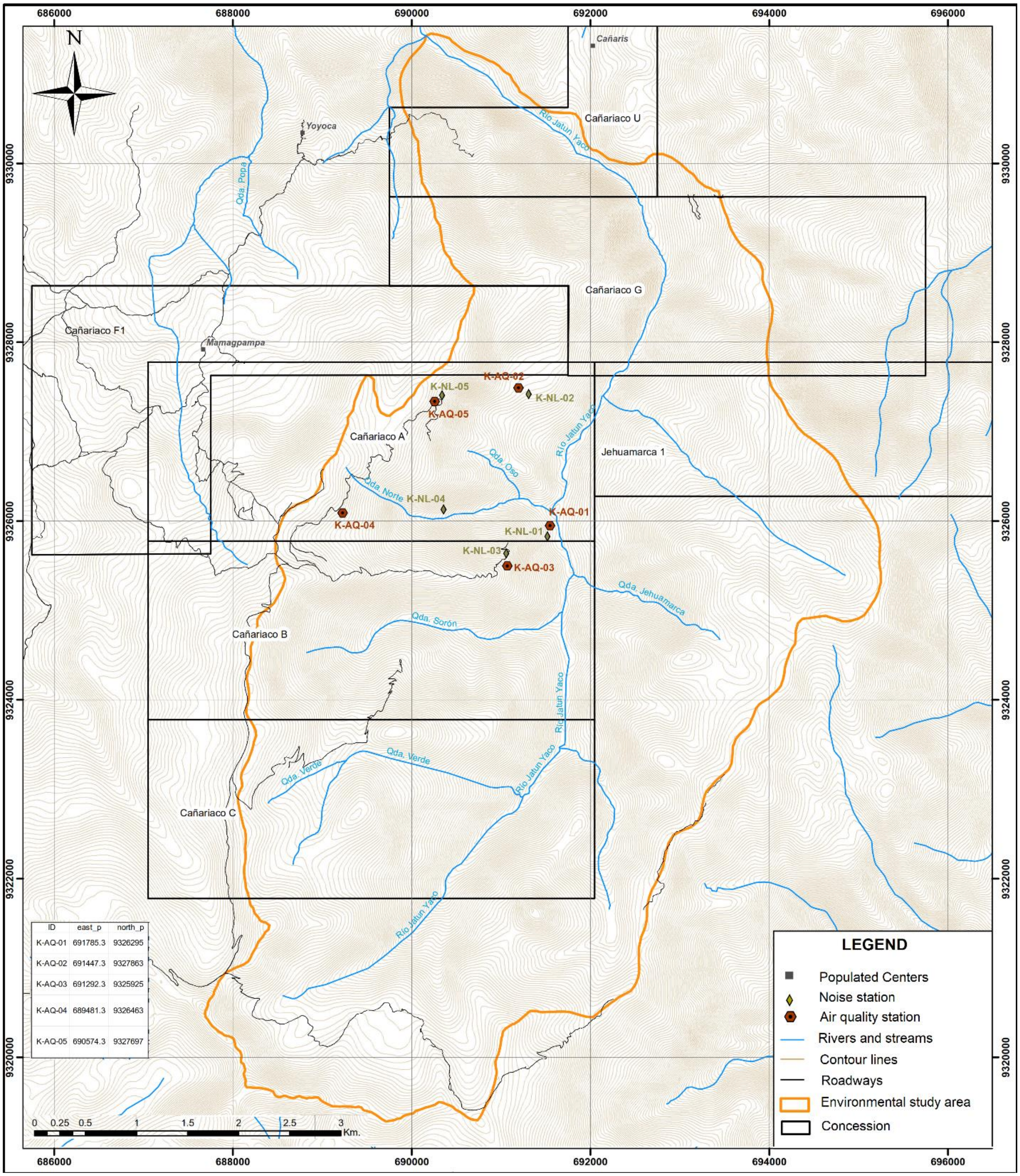
Table 20-2: Ambient Noise Sampling Stations.

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: Figure prepared by 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.

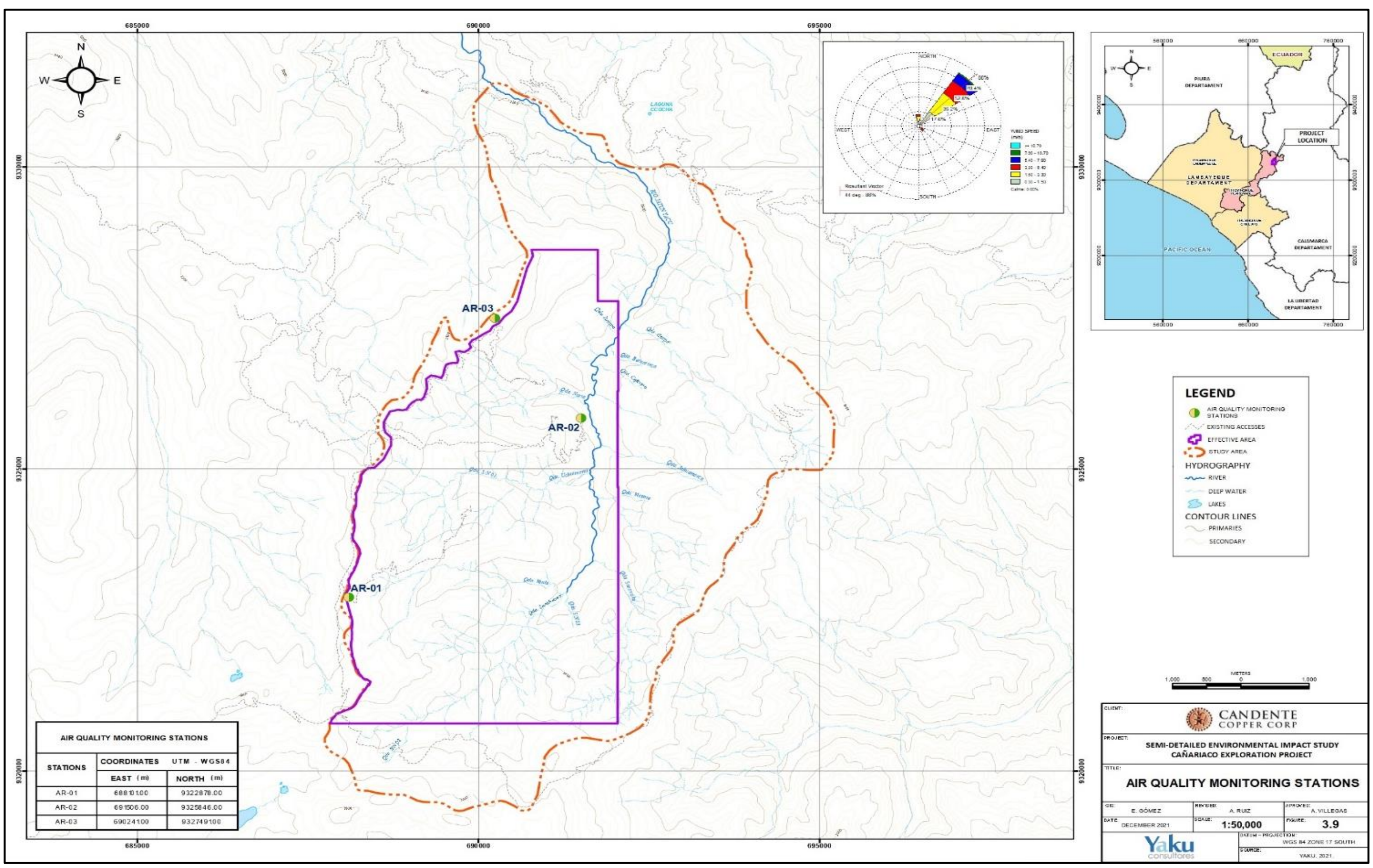
Table 20-3: Air Quality and Noise Monitoring Stations.

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
AR-03	RU-03	690,241	932,7491	3,256	Located southwest of the project area

Note: Coordinates in Datum WGS84, Zone 17S

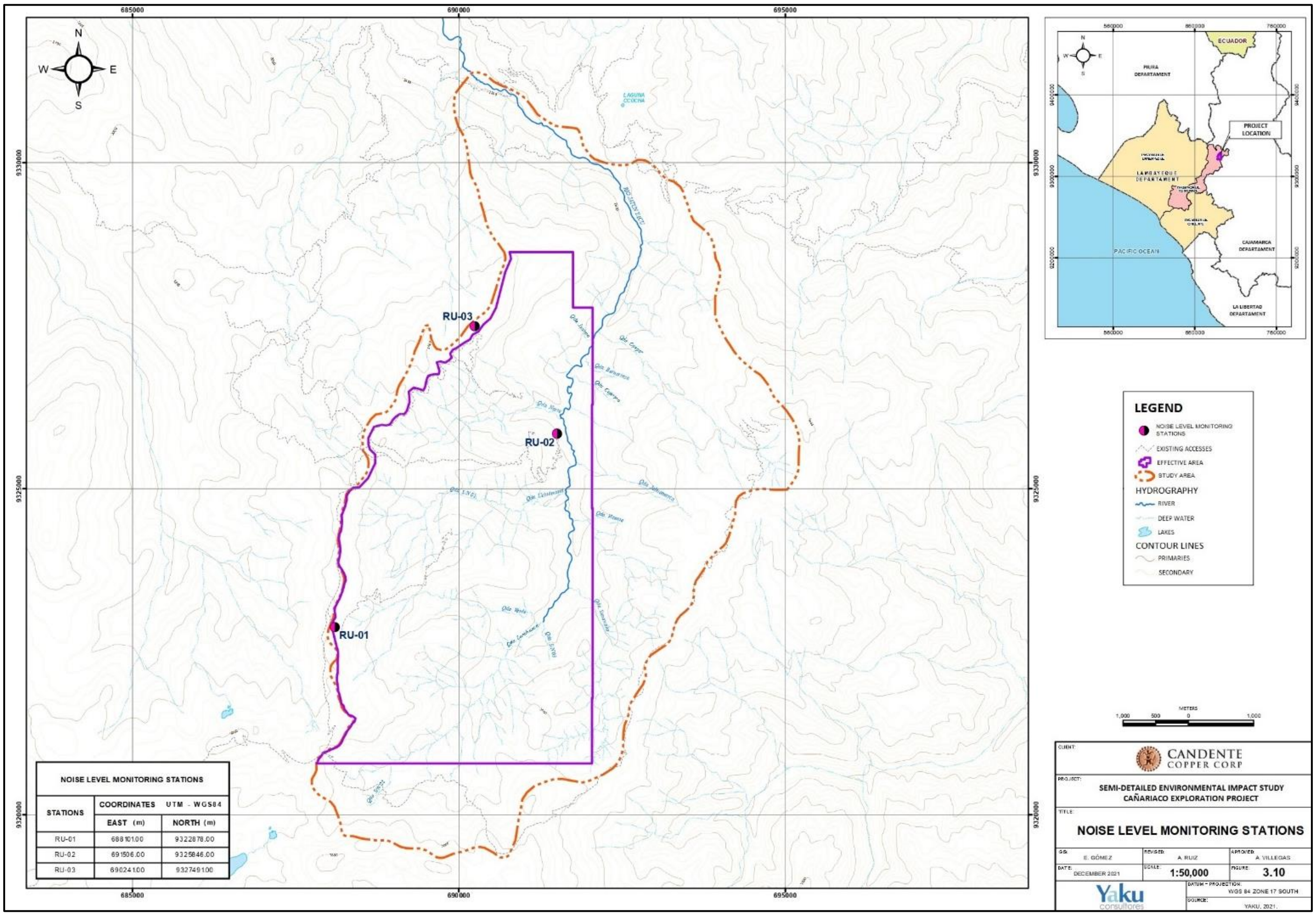
Figure 20-3 and Figure 20-4 show the location map of the air quality and ambient noise monitoring stations for the 2021 survey.

Figure 20-3: Location of Air Quality Monitoring Stations for the 2021 Survey.



Source: Figure prepared by Yaku Consultants, 2021.

Figure 20-4: Location of Ambient Noise Monitoring Stations for the 2021 Survey.



Source: Figure prepared by Yaku Consultants, 2021.

20.1.2.1.2 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, SO_2, H_2S, CO and NO_2) 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.

20.1.2.1.3 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 (NPS_{AeqT}), 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 NPS_{AeqT} 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:

- NPS_{AeqT} 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.

- NPS_{AeqT} 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.

20.1.2.1.4 Hydrography

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.

20.1.2.1.5 Hydrology

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.

20.1.2.1.6 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.

20.1.2.1.7 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:

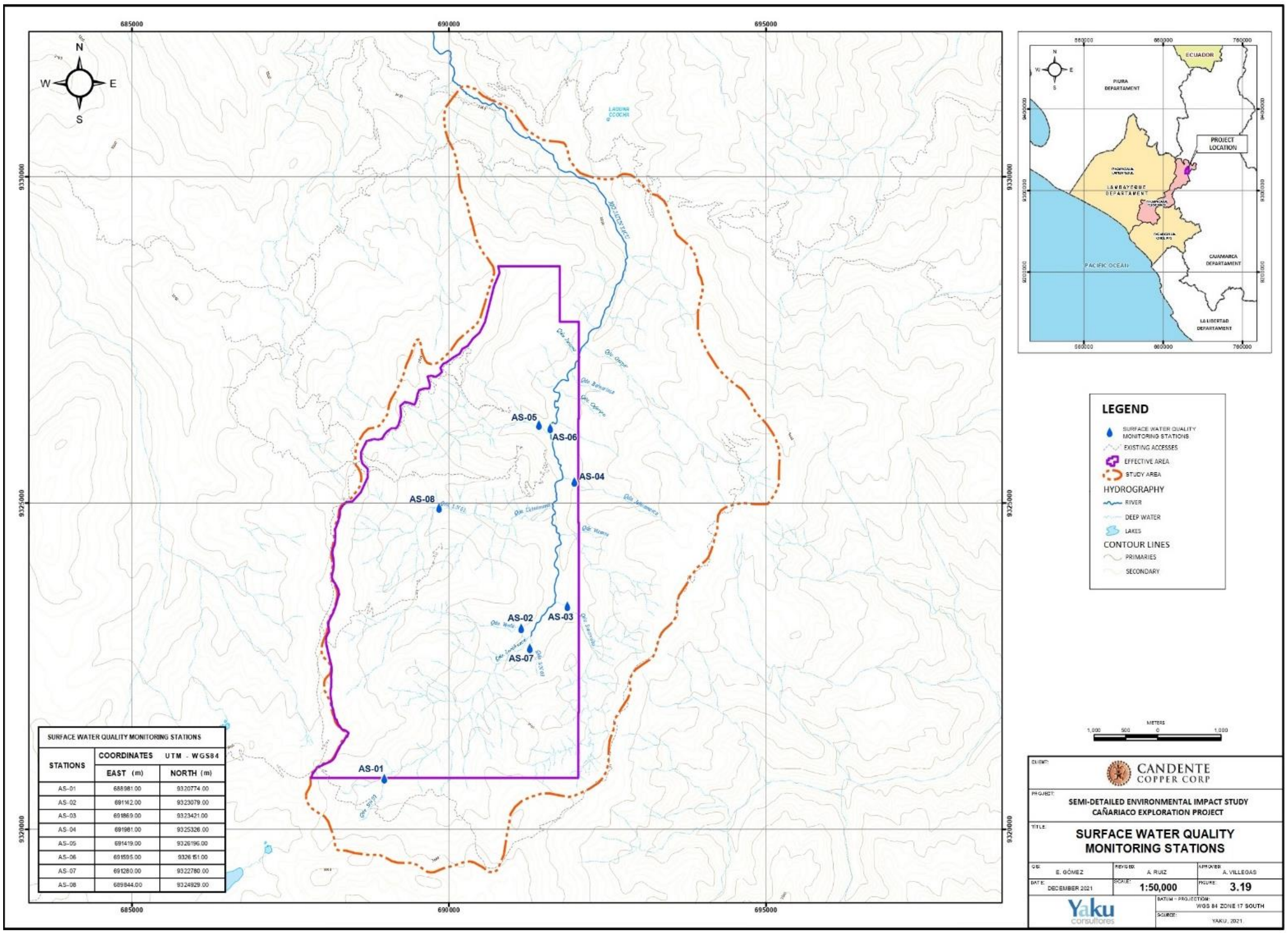
- Hydrogen potential (pH): The pH results at the surface water quality monitoring stations, for the most part, registered neutral characteristics and in some specific cases acidic characteristics.

- 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.

Figure 20-5: Location of Surface Water Quality Monitoring Stations.



Source: Figure prepared by Yaku Consultants, 2021.

20.1.2.1.8 Soils

Two soil types were defined:

- Land suitable for pasture, of low agricultural quality, soil limitations and erosional risk. Covers about 445 ha (22%) of the Project area; and
- Land with a higher use capacity, protection lands, soil limitations and erosional risk. Covers about 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.

20.1.2.2 Biological Setting

20.1.2.2.1 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 3000 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.

20.1.2.2.2 Flora

A flora sampling 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).

20.1.2.2.3 Taxonomic Composition

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.

20.1.2.2.4 Sensitive Flora Species

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).

20.1.2.2.5 Fauna

A sampling 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.

20.1.2.2.6 Mammals

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.

None of the species is currently endemic to Peru.

20.1.2.2.7 Birds

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 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 species were on the CITES Appendix II listing (2021).

In the baseline study, the only endemic species reported is *Thaumasius taczanowskii* "Taczanowski's Hummingbird"; all other species have a wide geographic distribution.

20.1.2.2.8 Amphibians and Reptiles

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 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.

20.1.2.2.9 Fish

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.

20.1.3 Water Management

The project design includes a sediment control facility (SCF) for facilities on the east side of the water shed divide and contact water management pond (CWMP) for WSF will be constructed downstream of all other Project facilities, including the open pit, WSF, and mill site. Every effort will be made to minimize mine contact water by diverting as much water as possible around these facilities to a location downstream of the SCF and CWMP, and all contact runoffs will be collected at these facilities for operational use and excess water will be treated and discharge. Therefore, the SCF and CWMP are considered some of the most important facilities associated with the Cañariaco Norte Project.

The SCF would be required to contain runoff from a drainage area of approximately 41.7 km^2 without considering the diversion of non-contact runoff water. This comprises approximately 32% of the entire Río Cañariaco drainage area above its confluence with the Huancabamba River. With effective diversions, this percentage is reduced to approximately 9% of the drainage area. The CWMP contains runoff from the WFS with an area of approximately 740 ha, which is approximately 6% of the Yerma Creek drainage basin.

The primary objective for water management at the Cañariaco Norte Project is to minimize the volume of runoff contributing to the SCF and the CWMP 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 and the CWMP into the head waters of Yerma Creek.
- Collect: all surface runoff (contact water) and seepage from WSF, mill site and surrounding haul roads.
- Recycle: as much water as possible will be recycled from the CWMP as make-up water to minimize the amount of freshwater make-up required from the SCF.
- Treat: all water collected at the SCF and the CWMP 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.

20.1.4 Tailings Management

A co-disposal facility, i.e. waste storage facility (WSF), for waste rock and filtered tailings storage will be constructed over the life of the mine to contain approximately 698 Mm^3 of mine waste and 439 Mm^3 of filtered tailings and 258 Mm^3 of waste rock. The WSF is not expected to behave like a conventional mine waste facility because of the large proportion of stored tailings combined with waste rock. The WSF was classified as significant or moderate under both Canadian Dam Association and the British Columbia and Mine Waste Rock Pile Research Committee "Investigation and Design Manual".

During pre-production and operations, mine waste will be crushed and conveyed to the plant site by conveyor. The mine waste will be transferred to the WSF conveyor to be placed in the WSF. It is assumed that the pre-production waste rock will be non-acid generating (NAG), and any PAG waste rock will be mixed to mitigate any metals leaching or acid rock drainage (ML/ARD). During operations the waste rock delivered to the plant area will be conveyed by itself to the WSF to build exterior berms or conveyed with filtered tailings for co-mingle disposal in the WSF using two conveyor systems. Based on the potential acid generation of materials, calcium carbonate will be added as part of the ARD mitigation measures. After Year 5, there will be insufficient waste rock to co-mingle with the tailings waste stream. Therefore, the waste rock will be used to develop exterior berms and the balance will be co-mingled with tailings near the front of the facility to improve physical stability.

The WSF includes a contact water management pond (CWMP) downstream of the facility, surface water management structures, rock drains, access road, waste conveyor system, equipment for spreading and compacting materials, and a water reclaim pipeline.

Instrumentation and monitoring will be required to assess embankment performance. Vibrating wire piezometers will be installed to monitor pore pressure within the embankment fill materials and slope inclinometers and survey monuments will be installed to monitor slope movement and deformation.

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20.2 Closure and Reclamation Planning

20.2.1 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
- 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.

20.2.1.1 Site Facilities Subject to Closure

The open pit, plant site, camp site and ancillary infrastructure, WSF, 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.

20.2.1.2 Decommissioning and Reclamation Measures

Candente 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.

20.2.2 Closure Cost Estimate

The closure costs for the purposes of the 2022 PEA were set at 10% of the initial capital costs, and totalled US$104 M.

20.3 Permitting Considerations

In accordance with the requirements of the Peruvian Legal Framework and the Ministry of Energy and Mines, Candente 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;
- 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.

Candente Copper is committed to keeping abreast of changes in Peruvian environmental and mining regulations, which are currently in the process of being updated. 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.

20.3.1 Pre-construction Permits

This sub-section describes some of the high priority permits that Candente 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.

20.3.1.1 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, Candente Copper must enter into an agreement with Community of San Juan de Cañaris representatives to obtain permission to use the surface land. Candente 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 construction and mining operations is pending. 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 Candente 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.

20.3.1.2 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 Ministry of Culture (MINCUL, in the Spanish acronym; formerly the National Institute of Culture) or by an archaeologist registered in the National Registry of Archaeologists, which is administered by the MINCUL. 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, Candente Copper has carried out the Archaeological Evaluation Project for the planned open pit area. The results showed that this area and the surrounding wooded areas have no archaeological sites or remains. It is suspected that other forested areas within the Project footprint will have similar results. All other Project components are in pasture or cultivated areas. As a result, MINCUL issued CIRA 2008-403 to Candente Copper for the open pit and surrounding area; the CIRA covers a 121 ha plus a 7 km perimeter zone. In order to start construction activities, Candente Copper must obtain CIRA(s) for the other planned Project facilities area. Because these site areas are primarily pasture or cultivated areas, it is not possible to predict whether they contain archaeological sites or remains without further study.

20.3.1.3 Environmental Impact Assessment and Closure Plan

In accordance with the Environmental of Ministry (MINAM, in the Spanish acronym) and Environmental Protection Standard on Mining and Metallurgical Activities, Candente 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. Currently Yaku Consultants is preparing an updated EIAsd (2021).

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.

20.3.1.4 Others

Once the EIA is submitted to MINAM through National Service of Environmental Certification (SENACE, in the Spanish acronym), applications can be prepared and submitted for other permits as listed below; however, approvals cannot be granted until the EIA 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 these authorizations is subject to various conditions and procedures.

20.3.1.4.1 Powerline Easement

Candente 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.

20.3.1.4.2 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, for its acronym in Spanish) is responsible for granting water use rights through decentralized offices, known as Local Water Administrators (ALA, in the Spanish acronym), 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 EIA and/or engineering studies.

20.3.1.4.3 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 the Provincial Municipality of Ferreñafe as Urban Housing Development or Urban Expansion Area, then Candente 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. . 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.

20.3.2 Construction and Operations Permits

20.3.2.1 Construction Phase

Once all required permits for the pre-construction phase have been obtained, the following licenses and permits are considered most critical.

- Registration in the hydrocarbon's registry of direct consumers, bottling plants and points of sale.
- Authorization for the private provision of telecommunications services.
- License for the operation of an ammunition depot.
- Authorization for the eventual use of explosives and related materials.
- Explosives handler's license.
- Issuance of the explosives transit guide.

20.3.2.2 Operations Phase

- Authorization to start exploitation activities in metallic mining concessions.
- Mining Operation Certificate.
- Multidisciplinary Civil Defense Technical Safety Inspection.
- Permits and licenses from municipalities (Ferreñafe and Cañaris).
- 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.

20.4 Social Considerations

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 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 6,808 inhabitants grouped into 1,792 families of which 3,800 are adults. The families live in 48 or more caserios (hamlets), many of which do not have road access. The village of Cañaris is adjacent to an unnamed tributary of the Cañariaco River on the east side of the Cañariaco valley, about 2.5 km from the Project site. According to the Instituto Nacional de Estadística (INEI), the village of Cañaris has 323 inhabitants grouped into 81 families. A program is being planned to determine the number of primary social groups within Cañaris. 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 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 Assembly consisting of all community members.

Road infrastructure and access to the Community of San Juan de Cañaris and surrounding farms is limited, with the nearest paved road leading to the nearest intermediate cities, such as Chiclayo (Lambayeque Region) and Jaén (Cajamarca Region) more than 40 km away.

Except for the access road and power line, the Project components will be constructed within the Cañariaco River and Yerma Creek watersheds, covering altitudes ranging from 2,400 to 3,600 masl and within different archaeological zones. The territory includes villages, agricultural areas or individual plots, collective use pastures and the Cañaris forest zone. In the latter case, the forest is 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 about 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.

20.4.1 Archeological Artefacts

The Cañariaco Project area was granted CIRA N° 2008-403.

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.

20.4.2 Public Consultation and Engagement

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.

21 CAPITAL AND OPERATING COSTS

21.1 Introduction

The following basic information pertains to the estimate of both capital and operating costs:

- Base date for these estimates is Q4 – 2021;
- All costs are expressed in United States dollars (US$);
- United States to Peruvian (S/) currency exchange rate used is US$1.00 = S/4.00;
- United States to Canadian (C$) currency exchange rate used is US$1.00 = C$/1.29;
- Estimate accuracy is reflective of the stage of project development and classified as an AACE International (AACE) Class 5 Order of Magnitude/Conceptual Study estimate with a -30% to +50% accuracy;
- Unit of measurement is metric (unless otherwise indicated); and
- Operating and sustaining capital costs are based on an estimated mine life of 27.4 years.

21.1.1 Exclusions

The following items were not considered in the cost estimates:

- Taxes and duties;
- Senior finance charges;
- Residual value of temporary equipment and facilities;
- Cost to client of any downtime;
- Environmental approvals;
- Any further Project studies;
- Force majeure issues;
- Future scope changes;
- 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;
- Management reserve;
- Escalation costs; and
- Owner's foreign exchange exposure.

21.2 Capital Costs

21.2.1 Overview

The overall capital cost estimate was developed by Ausenco with contributions from AGP for the mining costs. The capital cost estimate included historical pricing for main equipment supplemented by factored, scaled values for major disciplines from Ausenco's database of costs for similar projects in the region. Bulks such as concrete and steel were factored from the cost of major equipment. Unit rates, material costs, and electrical costs were based on typical values for comparative sites (not contractor quotations or bids).

Sustaining capital was limited to mining equipment additions and replacements, and the WSF over the LOM. The cumulative total LOM sustaining capital cost is US$119.0 M. A breakdown of capital cost figures by major work area is presented in Table 21-1 for Phase 1 (the initial 40,000 t/d) and Phase 2 (increasing total throughput at Year 7 to 80,000 t/d).

Table 21-1: Initial Capital Cost

WBS	Description	Total for Phase 1 (US$M)	Total for Phase 2 (US$M)
0200	Sitewide water services	7.4	1.1
0300	Sitewide power & lighting	7.1	0
2000	Mining	189.2	0
3000	Process plant	360.3	204.3
4000	Site services and utilities	6.7	0.6
5000	Internal infrastructure	134.8	1.5
6000	External infrastructure	38.5	0
	Total Direct Cost	**744.0**	**207.5**
7000	Common construction facilities and services	62.5	14.7
8000	Engineering, procurement & construction management	74.6	27.8
9100	Owner's costs	13.7	5.0
9900	Contingency	148.3	50.0
	Total Indirect Cost	**299.2**	**97.4**
	Total Capital Cost	**1,043.1**	**304.9**

*Note: Totals may not sum due to rounding

21.2.2 Direct Costs

21.2.2.1 Overview

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.

21.2.2.2 Mining Capital Costs

The mining capital cost estimate was grouped into initial and sustaining costs in five main categories as shown in Table 21-2.

Table 21-2: Mining Capital Cost Breakdown

Mining Capital Description	Initial Cost (US$ M)	Sustaining Cost (US$ M)	Total Capital Cost (US$ M)
Haul roads and heavy civil works	16.5	-	16.5
Open pit mine equipment	25.6	46.1	71.7
Mine services	4.8	18.0	22.8
Mine infrastructure	23.2	-	23.2
Mine pre-production stripping	119.2	-	119.2
Total	**189.2**	**64.1**	**253.4**

*Note: Totals may not sum due to rounding

The costs associated with those categories are discussed in the following sub-sections.

21.2.2.2.1 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 WSF will be 12 km long with a 36 m wide running surface. This includes the overpass on the existing municipal road. The access road to the truck shop area will be 7.3 km long and 15 m wide. Within the mining area, an initial 4 km of haul road is also included.

21.2.2.2.2 Open Pit Mine Equipment

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 21-3.

Table 21-3: 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	1.4	1.5	0.3
Production drill	mm	270	3.3	3.5	0.7
Production loader	m^3	33	7.8	8.2	1.6
Electric hydraulic shovel	m^3	34	12.5	13.2	2.5
Haulage truck	t	221	4.3	4.5	0.9
Crusher loader	m^3	20	4.0	4.2	0.8
Track dozer	kW	474	1.3	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. At the peak of mining, 18 units of 221 t capacity will be necessary to maintain mine production. This is required from Year 1 onwards. 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 21-4.

Table 21-4: Mine Equipment on Site

Equipment	Yr-2	Yr-1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 10	Yr 15	Yr 20	Yr 25
Production drill (160 mm)	1	1	1	1	1	1	1	1	1	1	1
Production drill (270 mm)	2	2	2	3	3	3	3	3	2	2	2
Production loader (33 m^3)	1	1	1	1	1	1	1	1	1	1	1
Electric hydraulic shovel (34 m^3)	1	1	2	2	2	2	2	3	3	2	2
Haulage truck (221 t)	9	13	18	18	18	18	18	18	13	13	13
Crusher loader (20 m^3)	-	-	1	1	1	1	1	1	1	1	1
Track dozer (455 kw)	5	5	5	5	5	5	5	5	5	4	4
Grader (146 kW)	3	3	3	3	3	3	3	2	2	2	2

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): 45,000 hrs;
- Production loader: 60,000 hrs;
- Electric hydraulic shovel: 72,000 hrs;
- Haul truck: 50,000 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 each four years. The integrated tool carrier for site support will be purchased at the Project start and will be replaced each 10 years over the mine life.

21.2.2.2.3 Mine Services

The mine services estimate included the mine dewatering system, dispatch and engineering office equipment including mine design software.

21.2.2.2.4 Mine Infrastructure

Mine infrastructure included the radio communication network, pit ring powerline and the truck shop sufficient for the mining fleet proposed. The bulk of the initial cost is the truck shop.

21.2.2.2.5 Mine Pre-production Stripping

Mining activity will commence in advance of the process plant achieving commercial production. This includes the movement of 35 Mt of waste prior to the primary crusher being ready for mill feed material. The mine operating costs in Year -2 and Year -1 were included in the capital cost estimate with the total expected cost to be US$119.2 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.

21.2.2.3 Process and Infrastructure Capital Costs

Process and infrastructure costs for phase 1, phase 2, and LOM are summarized in Table 21-5.

Table 21-5: Process Plant and Infrastructure Direct Costs

WBS	Sitewide Water Services	Phase 1 (US$M)	Phase 2 (US$M)	Total Cost (US$M)
0210	Sitewide fresh water	2.6	1.1	3.7
0220	Sitewide fire water	0	0	0
0230	Sitewide potable water	4.8	0	4.8
0200	**Total Sitewide Water Services – Direct Cost**	**7.4**	**1.1**	**8.5**
	Sitewide Power and Lighting			
0310	Power poles, cabling and lighting	0	0	0
0320	Grounding and lighting protection	0	0	0
0330	Backup power generator	4.1	0	4.1
0340	MV process plant switching station	0	0	0
0350	MV distribution	3.0	0	3.0
0300	**Total Sitewide Power and Lighting – Direct Cost**	**7.1**	**0**	**7.1**
	Process Plant			
3100	Primary crushing	91.7	0	91.7
3300	Copper plant	183.9	151.7	335.6
3400	Reagent	3.0	2.2	5.2
3500	Plant services	8.4	7.0	15.4
3600	Tailings filter plant	73.3	43.5	116.8
3000	**Total Process Plant – Direct Cost**	**360.3**	**204.3**	**564.6**
	Site Services and Utilities			
4100	Site stormwater pond	6.7	0.6	7.3
4000	**Total Site Services and Utilities – Direct Costs**	**6.7**	**0.6**	**7.3**
	Internal Infrastructure			
5100	Buildings and facilities	4.3	1.5	5.8
5200	Roads	1.4	0	1.4
5300	Permanent camp	2.3	0	2.3
5400	Plant site earthworks and drainage	10.8	0	10.8
5500	WSF stacking system	84.5	0	84.5
5600	WSF heavy civil works	31.5	0	31.5
5000	**Total Internal Infrastructure – Direct Cost**	**134.8**	**1.5**	**136.3**
	External Infrastructure			
6500	External power supply	38.5	0	38.5
6000	**Total External Infrastructure – Direct Cost**	**38.5**	**0**	**38.5**
	Total Direct Costs (US$)	**554.7**	**207.5**	**762.2**

*Note: Totals may not sum due to rounding

Direct costs included all contractors' direct and indirect labour, permanent equipment, materials, freight, and mobile equipment associated with the physical construction of the areas.

21.2.2.3.1 Process Plant

The definition of process equipment requirements was based on conceptual process flowsheets and process design criteria (refer to Section 17). The estimate was developed based on a compiled priced mechanical equipment list using a combination of recent quotations and historical costs for similar type equipment. Field installation and freight costs were applied to the mechanical equipment supply costs of each equipment list item.

Each major process area was built up with costs by separately addressing the following additional disciplines where applicable:

- Earthworks;
- Civil (concrete);
- Structural steel;
- Piping;
- Platework;
- Architectural (buildings);
- Electrical equipment;
- Electrical bulks; and
- Instrumentation.

Costs for the above disciplines were developed by applying historical factors (percentages of total installed cost of mechanical equipment) to each. The factors (percentages) are based on Ausenco's historical data for similar type work.

21.2.2.3.2 Infrastructure and WSF

Internal and external infrastructure costs were developed based on a PEA-level design of plant infrastructure and WSF. Ausenco developed material take-offs and estimated costs from in-house database and labour rates that included the following:

- Site development
    - Process plant earthworks;
    - Plant workshop and warehouse; and
    - Access road.

- Power supply & distribution:
  - Backup power supply and generator;
  - Internal medium voltage powerline; and
  - External high voltage powerline.
- Water supply
  - Sitewide fresh water supply; and
  - Sitewide potable water supply
- WSF
  - Earthworks associated with foundation preparation, material processing and embankment construction for the WSF;
  - Supply and installation of geomembrane liner and anchor trenches; and
  - WSF water management included construction cost of rock drains, temporary contact water collection ditches, non-contact water diversion channels, and a contact water management pond.

21.2.3 Indirect Costs

Indirect costs included all costs that are necessary for Project completion but not related to the direct construction cost and incurred by the Owner, engineer or consultants in Project design, procurement, construction, and commissioning to support during the construction period. Ausenco estimated a total of US$396.6 for Phase 1 and Phase 2, as shown in Table 21-6.

Table 21-6: Distribution of Indirect Costs

Indirect Cost Category	Phase 1 (US$M)	Phase 2 (US$M)	Total Cost (US$M)
Total Common Construction Facilities and Services	62.5	14.6	77.1
Total Engineering, Procurement & Construction Management	74.6	27.8	102.4
Total Owners Costs	13.7	5.0	18.7
Total Contingency	148.3	50	198.3
Total Project Indirect Costs	**299.2**	**97.4**	**396.6**

*Note: Totals may not sum due to rounding

21.2.3.1 Common Construction Facilities and Services Costs

Common construction facilities and services costs were based on Ausenco's historical project costs of similar nature and included, but not limited to the following:

- Temporary construction facilities and services: temporary site facilities, work areas and bays, roads, walk and parking areas, temporary buildings, temporary utilities, power, sewage, and other minor temporary facility;
- Construction support: operation and maintenance of temporary facilities, material handling, on site services, labour enhancement
- Construction equipment, tools and supplies: construction equipment, project scaffolding, fuel and lubricants, mobilisation/demobilisation;
- Pre-commissioning: contractor assistance, labours to support pre-commissioning;
- Workforce transport: flights and busing;
- Construction camp accommodation and services: construction camp to accommodate 1,140 persons for 24 months; and
- WSF: indirect cost associated with QA/QC and site investigation to support detailed design.

Ausenco estimated a total of US$77.1 M for LOM common construction facilities and services costs, which represented an average of 6% of the total direct costs as shown in Table 21-7.

Table 21-7: Distribution of Common Construction Facilities and Services Cost

Description of Work Breakdown Structure	Phase 1 (US$M)	Phase 2 (US$M)	Total Cost (US$M)
Temporary construction facilities	8.3	3.1	11.4
Construction support	4.4	1.7	6.1
Construction equipment, tools and supplies	11.1	4.1	15.2
Pre-commissioning	2.2	0.8	3.0
Workforce transport – flights and bussing	2.8	1.0	3.8
Camp accommodation catering	30.0	3.9	33.9
WSF indirect cost	3.7	0	3.7
Total Common Construction Facilities and Services Cost	**62.5**	**14.6**	**77.1**

*Note: Totals may not sum due to rounding

21.2.3.2 Engineering, Procurement and Construction Management Costs

Engineering, procurement and construction management (EPCM) services costs cover such items as engineering and procurement services (home office based), construction management services (site based), project office facilities, information technology (IT), staff transfer expenses, secondary consultants, field inspection and expediting, commissioning, corporate overhead and fees. Ausenco estimated a total of US$102.4 M for Phase 1 and Phase 2, which represented an average of 15% of the total direct costs as shown in Table 21-8.

Table 21-8: Distribution of EPCM Costs

Description of Work Breakdown Structure	Phase 1 (US$M)	Phase 2 (US$M)	Total Cost (US$M)
Execution - EPCM	66.4	24.9	91.3
First fills, pre-operational spares and fuel	5.5	1.9	7.4
Vendor reps	2.7	1.0	3.7
Total EPCM Cost	**74.6**	**27.8**	**102.4**

*Note: Totals may not sum due to rounding

21.2.3.3 Owners and Contingency Costs

Ausenco estimated a total of US$217.1 for the LOM Owner's cost and contingency costs which represent 52% of the total indirect costs as shown in Table 21-9.

Table 21-9: Distribution of Owners and Contingency Costs

Description of Work Breakdown Structure	Phase 1 (US$M)	Phase 2 (US$M)	Total Cost (US$M
Owners costs	13.7	5.0	18.7
Contingency	148.3	50.0	198.3
Total Owners and Contingency Costs	**162.1**	**55.0**	**217.1**

*Note: Totals may not sum due to rounding

Owner's costs are costs borne by the Owner in Project support and execution. Ausenco assumed an allowance of US$M18.7 for Owner's costs, which equated to approximately 2% of direct costs. Key items included staffing and expenses, pre-production labour, home office project management, home office financial, legal, insurance, bonds, licenses, and fees.

The total contingency amount of US$M198.3 was applied to the individual work areas based on the level of detail and construction cost risk associated with each area. The estimated contingencies excluded 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.

21.2.4 Sustaining Capital

The Project includes sustaining capital cost estimates for mining and the WSF. The mining sustaining costs consist of mobile fleet additions and replacements when equipment units reach their end of life as described in Section 21.2.2.2.2. The infrastructure costs consist of the WSF and WSF mobile equipment lease payments. The WSF cost estimate include the expansion of the WSF over the life of the project and the WSF mobile equipment lease payment. The WSF expansion is divided into 6 phases as the facility grows over the life of mine. The costs include clearing and grubbing, topsoil removal, unsuitable soil removal, extension of the underdrains along with development of surface water management infrastructure for the WSF. The total sustaining capital for the development of the WSF is S$48.6M. In addition, it includes equipment payments from years 2 through 6 for the WSF mobile equipment, i.e. 4 dozers, 2 compactors, 1 grader, and 1 water truck. The total sustaining capital for mobile equipment is US$6.2M. The total sustaining cost is estimated at US$119.0 M as shown in Table 21-10.

Table 21-10: Sustaining Capital Cost

Description	Total Cost (US$M)
Total mining	64.2
Total internal infrastructure	54.8
Total Sustaining Capital Cost	**119.0**

21.3 Operating Costs

21.3.1 Overview

A summary of the individual components that make up the LOM operating costs is presented in Table 21-11.

Table 21-11: Summary of Operating Cost Estimate

On-site Costs	LOM		Phase 1		Phase 2	
	US$/t Milled	US$/lb Cu	US$/ t Milled	US$/lb Cu	US$/ t Milled	US$/lb Cu
G&A	0.70	0.10	1.17	0.13	0.64	0.10
Mining	2.62	0.38	6.57	0.77	2.08	0.31
Processing	4.76	0.69	5.54	0.65	4.65	0.70
WSF	0.12	0.02	0.23	0.03	0.11	0.02
Total On-site Costs	8.21	1.19	13.51	1.57	7.49	1.12
Off-site Costs						
Concentrate transport	1.65	0.24	2.06	0.24	1.60	0.24
Smelting and refining	1.51	0.22	1.88	0.22	1.46	0.22
Royalties	0.12	0.02	0.15	0.02	0.11	0.02
Total off-site costs	3.32	0.48	4.10	0.48	3.21	0.48
Credits (gold, silver)	(2.67)	(0.39)	(3.72)	(0.43)	(2.52)	(0.38)
Total Operating Cost	8.86	1.28	13.89	1.62	8.17	1.23

The C1 cost of US$1.28/lb of payable copper consists of mining, processing, site G&A, off-site treatment and refining, transport, and royalties net of by-product credits (gold and silver). The C3 cost of US$1.39/lb of payable copper includes the C1 costs plus sustaining capital, expansion capital, and closure costs.

21.3.2 Basis of Estimate

Common to all operating cost estimates are the following assumptions:

- Majority of the labour requirement is assumed to come from neighbouring municipalities;
- Processing unit operations were benchmarked against similar or comparable processing plants;
- Equipment and materials will be purchased as new;
- Grinding media consumption rates have been estimated based on the material characteristics;
- Reagent consumption rates have been estimated on the metallurgical characteristics;
- The mobile equipment cost provides for fuel and maintenance;
- The fuel price provided for the Project was $1.085/L delivered to the site; and
- An electricity price of $0.065 per kilowatt hour was used.

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 200% 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 21-12.

Table 21-12: Mine Staffing Requirements and Annual Employee Salaries (Year 5)

Position	Employees	Annual Salary (US$/y)
Mine Maintenance		
Maintenance Superintendent	1	227,500
Maintenance General Foreman	1	82,500
Maintenance Shift Foremen	6	63,750
Maintenance Planner/Contract Administration	4	30,000
Clerk	2	15,000
Subtotal	**14**	
Mine Operations		
Mine Operations/Technical Superintendent	1	245,000
Mine General Foreman	1	82,500
Senior Shift Foreman	4	63,750
Junior Shift Foreman	4	41,250
Trainers	1	60,000
Road Crew/Services Foreman	1	60,000
Clerk	2	15,000
Subtotal	**14**	
Mine Engineering		
Chief Engineer	1	82,500
Senior Engineer	1	63,750
Open Pit Planning Engineer	2	45,000
Geotechnical Engineer	1	41,250

Position	Employees	Annual Salary (US$/y)
Blasting Engineer	1	41,250
Blasting/Geotechnical Technician	2	26,250
Dispatch Technician	4	26,250
Surveyor/Mining Technician	2	26,250
Surveyor/Mining Technician Helper	2	22,500
Clerk	1	15,000
Subtotal	**17**	
Geology		
Chief Geologist	1	82,500
Senior Geologist	1	63,750
Grade Control Geologist/Modeller	2	45,000
Sampling/Geology Technician	4	22,500
Clerk	1	15,000
Subtotal	**9**	
Total	**54**	

The mine staff labour remains constant from Year -1 until Year 10 when the trainer positions are eliminated. 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 21-13.

Table 21-13: Hourly Manpower Requirements and Annual Salaries (Year 5)

Position	Employees	Annual Salary (US$/y)
Mine General		
General Equipment Operator	8	22,200
Road/Pump Crew	4	20,300
General Mine Labourer	8	20,300
Light Duty Mechanic	4	33,300
Tire Technician	4	27,700
Lube Truck Driver	4	22,200
Subtotal	32	
Mine Operations		
Driller	16	33,300
Blaster	-	33,300
Blast Helper	-	22,200
Loader Operator	4	33,300
Hydraulic Shovel Operator	12	33,300
Haul Truck Driver	144	22,200
Dozer Operator	12	24,000
Grader Operator	6	24,000
Crusher Loader Operator	4	33,300
Water Truck	8	22,200
Subtotal	206	
Mine Maintenance		
Heavy Duty Mechanics	51	37,000
Welder	24	37,000
Electrician	2	37,000
Apprentice	7	27,700
Subtotal	**84**	
Total Hourly	**322**	

Labour costs are based on an Owner-operated scenario, with Candente 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 one 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 Training Foreman roles are required on site until the end of Year 10, at which time the position is eliminated. The Mine Operations department will have its own Clerk/Secretary.

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 two Grade Control Geologists/Modellers; one will be in short range and 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 until Year 5) 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 16 Drillers in Year 3 and maintains that level until Year 8 and then drops down over time as the drilling hours are diminished.

Shovel and Loader Operators peak at 16 in Year 4 for two years then holds at 12 until Year 19 when the number starts to taper off. Haulage Truck Drivers peak at 148 in Year 3 until and Year 6 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$44,200, resulting in a cost per hour for tires of US$52.99 /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 21-14.

Table 21-14: 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)	54.27	5.43	-	63.00	73.04	195.73
Production drill (270 mm)	135.67	13.57	6.00	95.00	108.40	358.64
Production loader (33 m3)	152.49	15.25	69.31	191.62	15.00	443.67
Electric hydraulic shovel (34 m^3)	123.24	-	204.51	293.14	49.76	670.65
Haulage truck (221 t)	158.46	15.85	52.99	86.50	6.00	319.80
Crusher loader (20 m^3)	179.09	26.86	51.43	143.52	15.00	415.90
Track dozer (455 kw)	78.04	7.80	23.61	51.34	6.56	167.35
Grader (146 kW)	15.95	1.60	4.00	17.48	0.50	39.53

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 21-15.

Table 21-15: 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 21-16.

Table 21-16: 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
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 21-17.

Table 21-17: Design Powder Factors

	Unit	160-mm Drill		270-mm Drill	
		Mill Feed	Waste	Mill Feed	Waste
Powder Factor	kg/m^3	0.70	0.70	0.71	0.71
Powder Factor	kg/t	0.28	0.28	0.28	0.28

The blasting cost was estimated using quotations from a local explosives vendor. The explosives price was US$69.58/100 kg. The explosives vendor will provide an all-in load and shot service for the mine. The total monthly cost was US$40,900 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 21-18 at Year 5. This highlights the focus of the shovels over the loader.

Table 21-18: Loading Parameters – Year 5

	Unit	Hydraulic Shovel	Front End Loader
Bucket capacity	m^3	34	33
Truck capacity loaded	t	221	221
Waste tonnage loaded	%	80	20
Mill feed tonnage loaded	%	75	25
Bucket fill factor	%	87	90
Cycle time	sec	33	45
Trucks present at loading unit	%	80	75
Loading time	min	2.60	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 21-19.

Table 21-19: Haulage Cycle Speeds

	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)	50	40	16	12.1	30	30

	Flat (0%) On Surface	Flat (0%) In pit, Crusher, Dump	Slope Up (8%)	Slope Up (10%)	Slope Down (8%)	Slope Down (10%)
Empty (km/hr)	50	40	35	25	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 21-20.

Table 21-20: 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	8	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 five track dozers, three graders, and one crusher loader. Their tasks will include clean-up of the loader faces, roads, WSF, 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$21.3 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 = 4.25%; and
- Residual = $0.

The initial capital, down payments, and annual leasing costs were included in Section 21.2.2.2.

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$0.19/t to the mine operating cost over the LOM. On a cost per tonne of feed basis, it equates to US$0.31/t mill feed.

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 21-21: and Table 21-22.

Table 21-21: 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$/t mined	0.12	0.08	0.10	0.10
Drilling	US$/t mined	0.11	0.11	0.11	0.11
Blasting	US$/t mined	0.25	0.25	0.25	0.25
Loading	US$/t mined	0.20	0.20	0.20	0.20
Hauling	US$/t mined	0.91	0.64	0.49	0.58
Support	US$/t mined	0.23	0.13	0.15	0.16
Financing costs	US$/t mined	0.63	0.13	0.09	0.19
Dewatering	US$/t mined	0.03	0.04	0.05	0.05
Total	**US$/t mined**	**2.47**	**1.58**	**1.43**	**1.62**

Table 21-22: 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$/t mill feed	0.40	0.30	0.15	0.16
Drilling	US$/t mill feed	0.40	0.43	0.17	0.18
Blasting	US$/t mill feed	0.87	0.92	0.38	0.40
Loading	US$/t mill feed	0.70	0.75	0.30	0.32
Hauling	US$/t mill feed	3.18	2.38	0.74	0.92
Support	US$/t mill feed	0.79	0.49	0.22	0.25
Financing costs	US$/t mill feed	2.17	0.49	0.14	0.31
Dewatering	US$/t mill feed	0.10	0.16	0.08	0.08
Total	**US$/t mill feed**	**8.60**	**5.91**	**2.17**	**2.62**

21.3.4 Process Operating Costs

The operating costs were developed based on the production of copper concentrate at plant feed rates and waste handling strategy as per the production schedule.

Average annual processing cost forecasts for power, consumables, maintenance consumables and labour are summarised in Table 21-23.

Table 21-23: Processing Costs (US$)

Processing Cost item	LOM		Phase 1		Phase 2	
	US$M/y	US$/t	US$M/y	US$/t	US$M/y	US$/t
Power	46.00	1.83	30.32	2.17	50.28	1.79
Labour	9.63	0.38	8.59	0.61	9.92	0.35
Reagents	35.28	1.41	19.12	1.37	39.69	1.41
Regrind Media and Consumables	19.06	0.76	10.83	0.77	21.30	0.76
Maintenance Materials – Vehicle & Mobile Equipment	9.61	0.38	7.48	0.53	10.19	0.36
Total	**119.58**	**4.76**	**76.33**	**5.45**	**131.38**	**4.67**

21.3.4.1 Power

Power costs were calculated from an estimate of annual power consumption derived from mechanical equipment list and using a unit cost of US$0.065/kWh.

Power consumption was derived from calculated power draw of grinding and regrind mills, plus an allowance for the remainder of the plant, based on typical flotation plants. The average on-line power draw is estimated at 75 MW in phase 1 (Years 1–6) and 115 MW in phase 2 (Years 7–28).

Annual energy consumption is estimated at 604,851 MWh for Years 1–6 and 923,434 MWh for Years 7–28.

21.3.4.2 Labour

Labour costs include all processing and maintenance costs and are shown in Table 21-24.

Table 21-24: Labour Costs

Cost Centre	Phase 1		Phase 2	
	Number	Annual Cost (US$M)	Number	Annual Cost (US$M)
Management	7	0.820	8	0.937
Technical Services	8	0.739	9	0.831
Operations	82	2.521	102	3.136
Maintenance	52	2.302	64	2.833
Laboratory	21	0.996	26	1.233
Contract Labour	29	1.209	22	0.948
Total	**170**	**8.587**	**209**	**9.918**

Costs were estimated from a breakdown of staffing positions, estimated at 170 for Phase 1 and 209 in total for Phase 2, excluding G&A manpower.

Positions were grouped into six broad categories:

- General management;
- Technical services;
- Operations;
- Maintenance;
- Laboratory; and
- Contract services

Costs are average pays inclusive of all loadings applicable to the site.

21.3.4.3 Consumables

Processing reagent and consumable costs were estimated based on the throughput.

Costs are summarised in Table 21-25.

Table 21-25: Processing Reagent and Consumable Costs (US$)

Consumable	Phase 1 (US$M/y)	Phase 2 (US$M/y)
Grinding media	5.59	11.61
Crushing and Grinding Consumables	5.24	9.69
Reagents	19.12	39.69
Total	**29.95**	**60.99**

Costs for liners were estimated based on vendor information and benchmarking similar plants. Grinding media cost were estimated from grinding media consumption rates based on the abrasion index (Ai) and the expected average mill power draws. These costs are summarized in Table 21-26.

Table 21-26: Costs for Media and General Consumables by Area

Area	Consumables	Consumption Set/y	Unit Cost US$/Set	No. of Machines	Annual Cost US$	No. of Machines	Annual Cost US$
Crushing and Conveying					1,117,212		1,117,212
	Mantle Liner - Standard	1.0	152,906	2	305,811	2	305,811
	Mantle Liner - Oversize	1.0	156,764	2	313,529	2	313,529
	Concave Liner	1.0	118,867	2	237,734	2	237,734
	Parts - Spider cap and liner	0.5	149,729	2	149,729	2	149,729
	Parts - Gear cover and liner	0.5	60,409	2	60,409	2	60,409
	General Consumables	----	----	----	50,000	----	50,000
Concentrator					4,933,172		9,866,344
	SAG Mill Liner	1.0	1,808,400	1	1,808,400	2	3,616,800
	Ball Mill Liner	1.0	946,000	1	946,000	2	1,892,000
	Pebble Crusher Mantle Liner	2.0	29,478	1	58,956	2	117,911
	Pebble Crusher Bowl Liner	2.0	29,517	1	59,035	2	118,070
	Regrind Mill Liner	1.0	605,000	2	1,210,000	4	2,420,000
	Filter Cloth	4.0	12,698	1	50,794	2	101,587
	General Consumables	----	----	----	37,500	----	75,000

Reagent costs were based on:

- Consumption rates determined from metallurgical test work; and
- Data base unit costs for the reagents.

Reagent costs are summarised in Table 21-27.

Table 21-27 Reagent Costs

Reagent	Addition Rate (kg/t)	Unit Cost (US$/t)	Phase 1 (US$M/y)	Phase 2 (US$M/y)
Quicklime	1.60	337.90	7.32	15.05
Collector 1	0.02	2,174.55	0.56	1.15
Collector 2	0.03	3,161.00	1.07	2.20
Frother	0.07	2,725.00	2.40	4.93
Flocculant (concentrate)	0.02	3,270.00	0.66	1.37
Flocculant (tailings)	0.02	3,270.00	0.89	1.82
Antiscalant	0.01	2,509.00	0.17	0.35
Biocide	0.00	1,759.00	0.05	0.10
Calcium carbonate	8.00	158.05	5.99	12.32
Total			**19.12**	**39.28**

21.3.4.4 Maintenance Consumables

Annual maintenance spares and consumable costs were factored at 3.5% of total installed costs for mechanical equipment, plate work, support steel and electrics.

This results in an annual maintenance consumables cost estimate of US$7.48 M for phase 1 and US$ 10.19 M for phase 2.

21.3.5 General and Administration

The G&A operating costs have been derived from each area within G&A group. The estimate for each area was built up using budget quotes from suppliers, benchmarked data from comparable projects in similar locations and estimates using industry standards.. The G&A costs were divided into the following areas:

- G&A maintenance, including mobile equipment and road maintenance
- Personnel
- human resources, including training, recruiting, and community relations
- infrastructure power, including power requirements for HVAC and administrative building
- assets operation for non-operational-related vehicle
- health and safety, including personal protective equipment, hospital service cost, and first aid
- environmental, including water sampling

- IT and telecommunications, including hardware and support services
- contract services, including insurance, sanitation and cleaning, and licence fees

The annual G&A cost is estimated at US$17.56 M/yr.

21.3.6 WSF Operating Costs

The operating costs for the WSF include the mechanical spreading and compacting equipment, water truck for dust suppression. The conveyor and stacking equipment, and CWMP pump and pipeline to the plant and water treatment plant are located in the processing operating cost. The mechanical equipment operating costs were estimated based on benchmark data from similar projects in similar locations. The costs include fuel, equipment maintenance, and labor.

The LOM operating costs for the WSF is 0.12 US$/tonne processed and 0.02 US$/lb copper.

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 estimates;
- Assumed commodity prices and exchange rates;
- Proposed mine production plan;
- Projected mining and process recovery rates;
- Assumptions as to mining dilution and estimated future production;
- Sustaining costs and proposed operating costs;
- Assumptions as to closure costs and closure requirements; and
- 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;

- Changes to site access, use of water for mining purposes and to time to obtain environment and other regulatory permits;
- Ability to maintain the social licence to operate;
- Changes to interest rates; and
- Changes to tax rates.

Readers are cautioned that the 2022 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 2022 PEA will be realized.

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

22.2 Methodologies Used

The Project was evaluated using a discounted cash flow (DCF) analysis based on an 8% discount rate. Cash inflows consisted of annual revenue projections. Cash outflows consisted of capital expenditures, including pre-production costs; operating costs; taxes; and royalties. These were subtracted from the inflows to arrive at the annual cash flow projections. Cash flows were 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 metals price, discount rate, head grade, total operating cost, and total capital costs.

The capital and operating cost estimates are presented in Section 21 of this Report in Q4 2021 American dollars. The economic analysis was run on a constant dollar basis with no inflation.

22.3 Financial Model Parameters

22.3.1 Assumptions

The economic analysis was performed assuming a gold price of US$1,650/oz, silver price of US$21.50/oz and copper price of US$3.50/lb; these metal prices were based on consensus analyst estimates and recently published economic studies. 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:

- Construction period of three years;
- Mine life of 28 years (last year is a partial year);
- Results based on 100% ownership with an 0.5% NSR royalty;
- 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;
- Treatment and refining charges as described in section 19.0
- Project revenue is derived from the sale copper concentrate; and
- No contractual arrangements for refining currently exist.

22.3.2 Taxes

The Project was evaluated on a post-tax basis to provide an approximate value of the potential economics. The tax model was compiled by a third-party tax consultant and calculations are based on the tax regime as of the date of the 2022 PEA . Depreciation on capital equipment, development and exploration cost as permitted by Peru tax regulations was applied.

The taxes assumed in the economic analysis include:

- Peruvian corporate income tax of 29.5% (plus 2% during the term of an assumed stability agreement described below);
- Employee profit sharing of 8% of taxable income; and
- Mining taxes for the exploitation of Mineral Resources based on the operating profit (mining royalties on a sliding scale of 1% to 12% with a minimum of 1% of sales and special mining tax on a sliding scale of 2% to 8.4%).

Under the General Mining Law, taxpayers that develop mining activities may enter into a stability regime upon subscription of an agreement with the Peruvian authorities. Under the mining regime, local mining companies may enter into stability agreements of guarantees and investment promotion measures that guarantee the following for 10, 12, or 15 years:

- Stability of the overall tax regime;
- Stability of the overall administrative regime;
- Free disposition of funds (foreign currency) arising from export operations;
- No exchange rate discrimination;
- Free trade of products; and
- Stability of special regimes for tax refunds, temporary importation, etc.

For a stability regime of 15 years, the requirements are either to mine at a rate of at least 15,000–20,000 t/d mineralization, or to invest a minimum amount of US$500 M in a project.

Mining companies that enter into 15-year stability agreements have the added benefit of an accelerated depreciation rate of 20% (without the limit of the financial depreciation) for machinery, industrial equipment, and other assets under criteria set by the Tax Authorities (SUNAT and the Tax Court). These benefits are included in the 2022 PEA.

22.3.3 Royalties

Based on the agreements in place, a 0.5% NSR royalty was assumed for the Project, resulting in approximately US$83 M in undiscounted royalty payments over the LOM.

22.4 Economic Analysis

The pre-tax NPV discounted at 8% is $2,022.5 M; the internal rate of return IRR is 21.6%, and payback period is 6.1 years. On a post-tax basis, the NPV discounted at 8% is $1,010.3 M; the internal rate of return IRR is 16.3%, and payback period is 7.1 years.

A summary of Project economics is shown in Table 22-1.

Table 22-1: Economic Analysis Summary

General	LOM Total / Avg.
Copper price (US$/lb)	3.50
Gold price (US$/oz)	1,650
Silver price (US$/oz)	21.50
Mine life (years)	28
Total waste tonnes mined (Mt)	465
Total mill feed tonnes (Mt)	703
Strip ratio	0.66
Production	**LOM Total / Avg.**
Mill head grade - copper (%)	0.38
Mill head grade - gold (g/t)	0.07
Mill head grade - silver (g/t)	1.69
Mill recovery rate - copper (%)	88.12
Mill recovery rate - gold (%)	64.67
Mill recovery rate - silver (%)	57.18
Total mill recovered - copper (mlb)	5,225
Total mill recovered - gold (koz)	960
Total mill recovered - silver (koz)	21,889
Total average annual payable production - copper (mlb)	173
Total average annual payable production - gold (koz)	31
Total average annual payable production - silver (koz)	704

On-site Costs	LOM Total / Avg.	
	US$/t processed	US$/lb Cu
Mining	2.62	0.38
Processing	4.76	0.69
WSF	0.12	0.02
General & administration	0.70	0.10
Total on-site costs	8.21	1.19
Off-site Costs	**LOM Total / Avg.**	
	US$/t concentrate	**US$/lb Cu**
Concentrate transport	$127.47	$0.24
Smelting & refining	$119.10	$0.22
Total off-site costs	$246.57	$0.46
By-product credits (gold, silver)	-	($0.39)
Cash Costs	**LOM Total**	
C1 Cost * (US$/lb Cu)	$1.28	
C3 Cost ** (US$/lb Cu)	$1.39	
Capital Costs	**LOM Total**	
Initial capital (US$M)	$1,043	
Expansion capital (US$M)	$305	
Sustaining capital (US$M)	$119	
Closure costs (US$M)	$104	
Financials	**Pre-Tax**	**Post-Tax**
NPV (8%) (US$M)	$2,023	$1,010
IRR (%)	21.6%	16.3%
Payback (years)	6.1	7.1

Note: * C1 costs consist of mining , processing , site G&A, off-site treatment & refining, transport, and royalties net of by-product credits (gold and silver).
** C3 costs consist of C1 plus sustaining capital, expansion capital, and closure.

The cashflow analysis was completed on an annual cashflow basis; the cashflow output is shown in Table 22-2.

Table 22-2: Project Cash Flow
(Dollar figures in Real 2021 US$M unless otherwise noted)

		Year	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041	2042	2043	2044	2045	2046	2047	2048	2049	2050	2051	2052	2053	2054
Copper Price	US$/lb	3.50	3.50	3.50	3.50	3.50	3.50	3.50	3.50	3.50	3.50	3.50	3.50	3.50	3.50	3.50	3.50	3.50	3.50	3.50	3.50	3.50	3.50	3.50	3.50	3.50	3.50	3.50	3.50	3.50	3.50	3.50	3.50
Gold Price	US$/oz	1,650	1,650	1,650	1,650	1,650	1,650	1,650	1,650	1,650	1,650	1,650	1,650	1,650	1,650	1,650	1,650	1,650	1,650	1,650	1,650	1,650	1,650	1,650	1,650	1,650	1,650	1,650	1,650	1,650	1,650	1,650	1,650
Silver Price	US$/oz	21.50	21.50	21.50	21.50	21.50	21.50	21.50	21.50	21.50	21.50	21.50	21.50	21.50	21.50	21.50	21.50	21.50	21.50	21.50	21.50	21.50	21.50	21.50	21.50	21.50	21.50	21.50	21.50	21.50	21.50	21.50	21.50
Revenue	US$M	18,844	–	–	–	449	526	394	509	524	435	730	601	727	922	810	892	882	783	824	799	626	434	537	658	764	775	742	745	872	805	815	265
Operating Cost	US$M	(5,767)	–	–	–	(173)	(199)	(206)	(194)	(189)	(175)	(235)	(237)	(223)	(221)	(221)	(214)	(213)	(219)	(228)	(233)	(233)	(229)	(221)	(212)	(209)	(207)	(204)	(202)	(204)	(206)	(206)	(54)
Off-Site Costs	US$M	(2,248)	–	–	–	(52)	(61)	(47)	(60)	(61)	(51)	(84)	(73)	(87)	(107)	(97)	(105)	(104)	(92)	(96)	(93)	(74)	(57)	(71)	(84)	(94)	(95)	(89)	(88)	(103)	(95)	(96)	(31)
Royalties	US$M	(83)	–	–	–	(2)	(2)	(2)	(2)	(2)	(2)	(3)	(3)	(3)	(4)	(4)	(4)	(4)	(3)	(4)	(4)	(3)	(2)	(2)	(3)	(3)	(3)	(3)	(3)	(4)	(4)	(4)	(1)
EBITDA	US$M	10,745	–	–	–	223	263	139	253	272	207	407	289	413	590	489	568	560	468	496	470	317	145	243	359	457	470	446	451	562	500	509	178
Initial Capex	US$M	(1,043)	(161)	(369)	(513)	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Sustaining Capex	US$M	(119)	–	–	–	(11)	(17)	(1)	(2)	(10)	(4)	(1)	(2)	(1)	(12)	(1)	(1)	(4)	(9)	(13)	(4)	(1)	(3)	(4)	(6)	(0)	(3)	(1)	(0)	(5)	(3)	(0)	(0)
Closure Capex	US$M	(104)	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	(104)
Pre-Tax Unlevered Free Cash Flow	**US$M**	**9,174**	**(161)**	**(369)**	**(513)**	**211**	**247**	**138**	**251**	**109**	**51**	**406**	**287**	**412**	**577**	**488**	**567**	**556**	**460**	**483**	**466**	**316**	**142**	**239**	**353**	**456**	**467**	**445**	**451**	**556**	**497**	**509**	**74**
Income Tax, Mining Tax & Mining Royalties	US$M	(3,806)	–	–	–	(13)	(53)	(11)	(59)	(66)	(61)	(136)	(85)	(139)	(218)	(184)	(231)	(227)	(186)	(197)	(178)	(114)	(45)	(83)	(132)	(181)	(187)	(177)	(179)	(226)	(200)	(203)	(34)
Post-Tax Unlevered Free Cash Flow	**US$M**	**5,368**	**(161)**	**(369)**	**(513)**	**198**	**194**	**127**	**193**	**43**	**(10)**	**270**	**202**	**273**	**359**	**304**	**337**	**329**	**274**	**286**	**288**	**203**	**98**	**156**	**221**	**275**	**280**	**268**	**272**	**330**	**297**	**305**	**40**
Production																																	
Total Material Mined	Mt	1,168	–	14	21	38	45	55	55	55	55	55	55	43	44	41	41	41	41	41	41	38	38	37	37	37	35	33	32	31	31	30	9
Strip Ratio	w:o	0.66	–	–	–	2.47	2.05	2.74	2.74	2.74	2.74	1.08	0.87	0.48	0.51	0.41	0.41	0.41	0.41	0.41	0.41	0.30	0.31	0.25	0.25	0.25	0.21	0.15	0.10	0.07	0.06	0.04	0.03
Project Life	yrs	28	–	–	–	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	0.4
Mill Feed	Mt	703	–	–	–	11	15	15	15	15	15	26	29	29	29	29	29	29	29	29	29	29	29	29	29	29	29	29	29	29	29	29	8
Mill Head Grade (Cu)	%	0.38	0.00	0.00	0.00	0.55	0.49	0.38	0.48	0.50	0.42	0.39	0.30	0.35	0.44	0.40	0.43	0.43	0.38	0.40	0.39	0.31	0.23	0.28	0.34	0.38	0.38	0.37	0.37	0.42	0.39	0.40	0.45
Mill Recovery (Cu)	%	88.12	–	–	–	90.99	90.38	89.46	90.38	89.52	88.75	87.62	85.83	88.28	88.34	88.70	89.15	88.83	88.07	88.56	88.20	87.66	83.55	85.08	86.91	88.68	88.39	87.54	86.98	88.54	87.57	87.85	88.30
Recovered Copper	mlbs	5,225	–	–	–	121.18	143.90	109.43	140.34	143.34	119.33	198.18	165.14	200.90	252.43	226.23	247.38	244.70	215.95	226.84	219.02	172.32	122.74	153.78	187.67	215.60	218.89	208.61	206.96	241.10	222.41	227.02	73.92
Mill Head Grade (Au)	g/t	0.07	–	–	–	0.11	0.09	0.06	0.08	0.09	0.08	0.08	0.06	0.06	0.09	0.06	0.07	0.07	0.07	0.08	0.08	0.06	0.03	0.03	0.04	0.05	0.05	0.06	0.06	0.07	0.07	0.07	0.07
Mill Recovery (Au)	%	64.67	–	–	–	73.37	71.70	66.98	69.99	70.42	69.27	66.44	60.69	65.82	68.54	63.05	63.55	64.24	63.49	64.79	66.54	66.78	55.23	54.16	56.43	62.72	62.06	60.20	60.98	65.39	64.12	65.39	66.82
Recovered Gold	koz	960	–	–	–	28.26	29.51	18.61	27.59	30.54	24.46	46.35	32.71	37.14	55.19	37.39	44.65	44.58	42.39	45.95	47.10	35.53	16.91	17.68	23.16	31.21	32.03	32.89	36.61	45.39	42.37	40.90	12.53
Mill Head Grade (Ag)	g/t	1.69	–	–	–	2.81	2.42	1.89	1.99	2.09	1.78	1.61	1.54	1.73	1.62	2.01	2.01	1.86	1.68	1.68	1.52	1.16	1.11	1.34	1.63	1.69	1.70	1.62	1.68	1.87	1.77	1.47	1.77
Mill Recovery (Ag)	%	57.18	–	–	–	67.64	62.16	60.05	63.82	64.19	62.36	60.30	54.58	57.97	62.37	55.00	55.85	57.02	56.36	58.10	59.07	59.66	49.72	47.29	48.95	55.59	54.73	54.87	55.28	58.26	56.79	56.85	58.60
Recovered Silver	koz	21,889	–	–	–	672.75	705.17	533.59	595.08	630.90	520.23	818.61	791.47	941.38	951.49	1,037.11	1,053.79	996.05	888.07	915.57	845.47	650.45	516.04	593.73	747.05	882.29	872.67	831.92	870.48	1,022.44	941.92	783.22	280.49
Wet Concentrate Produced	kt	10,020	–	–	–	232	276	210	269	275	229	380	317	385	484	434	474	469	414	435	420	330	235	295	360	413	420	400	397	462	426	435	142
Payable Copper	mlbs	4,848	–	–	–	112	134	102	130	133	111	184	153	186	234	210	230	227	200	210	203	160	114	143	174	200	203	194	192	224	206	211	69
Payable Gold	koz	879	–	–	–	26.00	27.15	17.12	25.38	28.09	22.50	42.64	30.10	34.17	50.77	33.65	41.08	41.02	39.00	42.27	43.33	32.69	15.22	15.92	20.84	28.09	28.83	29.60	33.68	41.76	38.98	37.63	11.53
Payable Silver	koz	19,700	–	–	–	605	635	480	536	568	468	737	712	847	856	933	948	896	799	824	761	585	464	534	672	794	785	749	783	920	848	705	252
Revenue - Copper	US$M	16,970	–	–	–	394	467	355	456	465	388	644	536	652	820	735	803	795	701	737	711	560	399	499	609	700	711	677	672	783	722	737	240
Revenue - Gold	US$M	1,450	–	–	–	43	45	28	42	46	37	70	50	56	84	56	68	68	64	70	71	54	25	26	34	46	48	49	56	69	64	62	19
Revenue - Silver	US$M	424	–	–	–	13	14	10	12	12	10	16	15	18	18	20	20	19	17	18	16	13	10	11	14	17	17	16	17	20	18	15	5
Operating Costs																																	
Total Operating Costs	US$M	5,767	–	–	–	173	199	206	194	189	175	235	237	223	221	221	214	213	219	228	233	233	229	221	212	209	207	204	202	204	206	206	54
Mine Operating Costs	US$M	1,841	–	–	–	95	96	103	91	86	81	76	80	66	63	63	57	56	62	71	76	75	72	64	55	52	50	47	45	47	49	48	14
Mill Processing	US$M	3,345	–	–	–	61	81	81	81	81	81	140	135	135	135	135	135	135	135	135	135	135	135	135	135	135	135	135	135	135	135	135	32
G&A Costs	US$M	494	–	–	–	14	19	19	19	19	9	17	19	19	19	19	19	19	19	19	19	19	19	19	19	19	19	19	19	19	19	19	5
Total Operating Costs	$/t Processed	8.21	–	–	–	15.72	13.62	14.09	13.27	12.95	11.96	8.95	8.12	7.65	7.56	7.55	7.33	7.31	7.51	7.82	7.98	7.96	7.84	7.57	7.26	7.17	7.09	7.00	6.91	6.97	7.07	7.04	6.40
C1 Cost *	US$/lb Cu	1.28	–	–	–	1.52	1.53	2.13	1.55	1.46	1.63	1.29	1.61	1.28	0.98	1.17	1.02	1.03	1.16	1.14	1.19	1.52	2.22	1.80	1.44	1.22	1.18	1.20	1.15	0.99	1.08	1.08	0.90
C3 Cost **	US$/lb Cu	1.39	–	–	–	1.62	1.65	2.14	1.57	2.68	3.04	1.29	1.63	1.29	1.04	1.18	1.03	1.05	1.21	1.21	1.21	1.52	2.25	1.82	1.47	1.22	1.20	1.20	1.15	1.01	1.09	1.08	2.43
Capital Expenditures																																	
Total Initial Capital	US$M	1,043	161	369	513	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Pre-Production Stripping	US$M	119	–	51	68	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Mining Equipment Capital	US$M	26	–	19	6	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–

		Year	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041	2042	2043	2044	2045	2046	2047	2048	2049	2050	2051	2052	2053	2054
Miscellaneous Mine Capital	US$M	14	6	6	2	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Sitewide Water Services	US$M	7	2	3	3	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Sitewide Power & Lighting	US$M	7	2	2	2	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Mining	US$M	31	9	11	11	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Process Plant	US$M	360	--	108	252	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Internal Infrastructure	US$M	129	39	45	45	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
External Infrastructure	US$M	39	12	13	13	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Common Construction Facilities & Services	US$M	62	19	22	22	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
EPCM	US$M	75	22	26	26	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Owners Costs	US$M	162	49	57	57	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Total Expansion Capital	US$M	305	--	--	--	--	--	--	--	152	152	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Sitewide Water Services	US$M	1	--	--	--	--	--	--	--	1	1	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Process Plant	US$M	204	--	--	--	--	--	--	--	102	102	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Site Services And Utilities	US$M	1	--	--	--	--	--	--	--	0	0	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Internal Infrastructure	US$M	1	--	--	--	--	--	--	--	1	1	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Common Construction Facilities And Services	US$M	15	--	--	--	--	--	--	--	7	7	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
EPCM	US$M	28	--	--	--	--	--	--	--	14	14	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Owners Costs	US$M	55	--	--	--	--	--	--	--	27	27	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Total Sustaining Capital	**US$M**	119	--	--	--	11	17	1	2	10	4	1	2	1	12	1	1	4	9	13	4	1	3	4	6	0	3	1	0	5	3	0	0
Mining Equipment Capital	US$M	46	--	--	--	9	1	0	1	0	0	0	2	1	2	1	1	4	7	7	4	1	1	3	1	0	1	1	--	0	1	--	--
Miscellaneous Mine Capital	US$M	18	--	--	--	0	2	0	0	0	2	0	0	0	2	0	0	0	2	0	0	0	2	0	0	0	2	0	0	0	2	0	0
Co-mingle Storage Facility	US$M	49	--	--	--	3	13	--	--	8	--	--	--	--	8	--	--	--	--	6	--	--	--	--	6	--	--	--	--	5	--	--	--
Co-mingle Mobile Equipment (lease payments)	US$M	6	--	--	--	--	1	1	1	1	1	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Closure Cost	**US$M**	104	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	104

Note: * C1 costs consist of mining , processing , site G&A, off-site treatment & refining, transport, and royalties net of by-product credits (gold and silver).
** C3 costs consist of C1 plus sustaining capital, expansion capital, and closure.

22.5 Sensitivity Analysis

A sensitivity analysis was conducted on the base case pre-tax and post-tax NPV (Net Present Value) and IRR (Internal Rate of Return) of the Project, using the following variables: commodity prices, discount rate, total operating cost, total capital cost, head grade, and foreign exchange.

Table 22-3 shows the post-tax sensitivity analysis results; pre-tax sensitivity results are shown in Table 22-4.

As presented in Figure 22-1 and Figure 22-2, the sensitivity analysis showed that the Project is most sensitive to changes in commodity price and head grade. In order of decreasing sensitivity, the project is less sensitive to changes in total capital cost, total operating cost, and foreign exchange rate.

Table 22-3: Post-Tax Sensitivity Summary

Post-Tax NPV Sensitivity To Discount Rate — Commodity Price (%)

Discount Rate	(20%)	(10%)	0%	10%	20%
3.0%	$1,557	$2,237	$2,889	$3,534	$4,177
5.0%	$921	$1,432	$1,916	$2,396	$2,874
8.0%	$335	$684	$1,010	$1,333	$1,654
10.0%	$93	$372	$630	$884	$1,137
12.0%	($76)	$151	$359	$564	$767

Post-Tax IRR Sensitivity To Discount Rate — Commodity Price (%)

Discount Rate	(20%)	(10%)	0%	10%	20%
3.0%	11.0%	13.9%	16.3%	18.6%	20.7%
5.0%	11.0%	13.9%	16.3%	18.6%	20.7%
8.0%	11.0%	13.9%	16.3%	18.6%	20.7%
10.0%	11.0%	13.9%	16.3%	18.6%	20.7%
12.0%	11.0%	13.9%	16.3%	18.6%	20.7%

Post-Tax NPV Sensitivity To Opex — Commodity Price (%)

Opex	(20%)	(10%)	0%	10%	20%
(20.0%)	$568	$893	$1,216	$1,535	$1,853
(10.0%)	$458	$790	$1,114	$1,435	$1,754
--	$335	$684	$1,010	$1,333	$1,654
10.0%	$211	$576	$905	$1,230	$1,553
20.0%	$86	$457	$799	$1,126	$1,450

Post-Tax IRR Sensitivity To Opex — Commodity Price (%)

Opex	(20%)	(10%)	0%	10%	20%
(20.0%)	13.1%	15.6%	17.9%	20.1%	22.1%
(10.0%)	12.1%	14.8%	17.1%	19.4%	21.4%
--	11.0%	13.9%	16.3%	18.6%	20.7%
10.0%	9.9%	13.0%	15.5%	17.8%	20.0%
20.0%	8.8%	12.0%	14.7%	17.1%	19.3%

Post-Tax NPV Sensitivity To Capex — Commodity Price (%)

Capex	(20%)	(10%)	0%	10%	20%
(20.0%)	$554	$902	$1,227	$1,549	$1,869
(10.0%)	$445	$793	$1,119	$1,441	$1,761
--	$335	$684	$1,010	$1,333	$1,654
10.0%	$226	$575	$902	$1,225	$1,546
20.0%	$116	$466	$793	$1,117	$1,438

Post-Tax IRR Sensitivity To Capex — Commodity Price (%)

Capex	(20%)	(10%)	0%	10%	20%
(20.0%)	13.9%	17.2%	20.1%	22.7%	25.1%
(10.0%)	12.3%	15.4%	18.0%	20.5%	22.7%
--	11.0%	13.9%	16.3%	18.6%	20.7%
10.0%	9.9%	12.6%	14.9%	17.1%	19.1%
20.0%	8.9%	11.5%	13.7%	15.7%	17.6%

Post-Tax NPV Sensitivity To Mill Head Grade — Commodity Price (%)

Mill Head Grade	(20%)	(10%)	0%	10%	20%
(20.0%)	($293)	$31	$323	$605	$865
(10.0%)	$38	$369	$680	$972	$1,262
--	$335	$684	$1,010	$1,333	$1,654
10.0%	$619	$979	$1,336	$1,690	$2,042
20.0%	$881	$1,272	$1,660	$2,045	$2,429

Post-Tax IRR Sensitivity To Mill Head Grade — Commodity Price (%)

Mill Head Grade	(20%)	(10%)	0%	10%	20%
(20.0%)	5.1%	8.3%	10.9%	13.3%	15.3%
(10.0%)	8.4%	11.3%	13.9%	16.1%	18.1%
--	11.0%	13.9%	16.3%	18.6%	20.7%
10.0%	13.4%	16.1%	18.6%	21.0%	23.2%
20.0%	15.4%	18.2%	20.8%	23.2%	25.5%

Post-Tax NPV Sensitivity To Foreign Exchange — Commodity Price (%)

Foreign Exchange	(20%)	(10%)	0%	10%	20%
(20.0%)	$94	$447	$774	$1,098	$1,420
(10.0%)	$228	$579	$905	$1,229	$1,550
--	$335	$684	$1,010	$1,333	$1,654
10.0%	$423	$771	$1,096	$1,419	$1,739
20.0%	$496	$843	$1,168	$1,490	$1,810

Post-Tax IRR Sensitivity To Foreign Exchange — Commodity Price (%)

Foreign Exchange	(20%)	(10%)	0%	10%	20%
(20.0%)	8.7%	11.3%	13.5%	15.5%	17.4%
(10.0%)	9.9%	12.7%	15.0%	17.1%	19.1%
--	11.0%	13.9%	16.3%	18.6%	20.7%
10.0%	12.0%	15.1%	17.6%	20.0%	22.3%
20.0%	13.0%	16.2%	18.9%	21.3%	23.7%

Table 22-4: Pre-Tax Sensitivity Analysis

Pre-Tax NPV Sensitivity To Discount Rate (Commodity Price (%))

Discount Rate	(20%)	(10%)	0%	10%	20%
3.0%	$2,819	$3,961	$5,103	$6,245	$7,387
5.0%	$1,809	$2,659	$3,508	$4,357	$5,207
8.0%	$881	$1,452	$2,023	$2,593	$3,164
10.0%	$497	$947	$1,397	$1,848	$2,298
12.0%	$228	$590	$952	$1,315	$1,677

Pre-Tax IRR Sensitivity To Discount Rate (Commodity Price (%))

Discount Rate	(20%)	(10%)	0%	10%	20%
3.0%	14.5%	18.1%	21.6%	24.8%	27.9%
5.0%	14.5%	18.1%	21.6%	24.8%	27.9%
8.0%	14.5%	18.1%	21.6%	24.8%	27.9%
10.0%	14.5%	18.1%	21.6%	24.8%	27.9%
12.0%	14.5%	18.1%	21.6%	24.8%	27.9%

Pre-Tax NPV Sensitivity To Opex (Commodity Price (%))

Opex	(20%)	(10%)	0%	10%	20%
(20.0%)	$1,256	$1,827	$2,398	$2,969	$3,540
(10.0%)	$1,069	$1,640	$2,210	$2,781	$3,352
--	$881	$1,452	$2,023	$2,593	$3,164
10.0%	$693	$1,264	$1,835	$2,406	$2,976
20.0%	$505	$1,076	$1,647	$2,218	$2,789

Pre-Tax IRR Sensitivity To Opex (Commodity Price (%))

Opex	(20%)	(10%)	0%	10%	20%
(20.0%)	17.1%	20.6%	23.9%	27.1%	30.2%
(10.0%)	15.8%	19.4%	22.7%	26.0%	29.0%
--	14.5%	18.1%	21.6%	24.8%	27.9%
10.0%	13.2%	16.9%	20.3%	23.6%	26.8%
20.0%	11.8%	15.6%	19.1%	22.5%	25.6%

Pre-Tax NPV Sensitivity To Capex (Commodity Price (%))

Capex	(20%)	(10%)	0%	10%	20%
(20.0%)	$1,105	$1,676	$2,247	$2,818	$3,389
(10.0%)	$993	$1,564	$2,135	$2,706	$3,277
--	$881	$1,452	$2,023	$2,593	$3,164
10.0%	$769	$1,339	$1,910	$2,481	$3,052
20.0%	$656	$1,227	$1,798	$2,369	$2,940

Pre-Tax IRR Sensitivity To Capex (Commodity Price (%))

Capex	(20%)	(10%)	0%	10%	20%
(20.0%)	17.5%	21.7%	25.7%	29.5%	33.1%
(10.0%)	15.9%	19.8%	23.4%	26.9%	30.3%
--	14.5%	18.1%	21.6%	24.8%	27.9%
10.0%	13.3%	16.7%	19.9%	23.0%	25.9%
20.0%	12.2%	15.5%	18.6%	21.4%	24.2%

Pre-Tax NPV Sensitivity To Mill Head Grade (Commodity Price (%))

Mill Head Grade	(20%)	(10%)	0%	10%	20%
(20.0%)	($41)	$411	$863	$1,315	$1,767
(10.0%)	$420	$932	$1,443	$1,955	$2,467
--	$881	$1,452	$2,023	$2,593	$3,164
10.0%	$1,339	$1,969	$2,599	$3,229	$3,859
20.0%	$1,796	$2,486	$3,175	$3,865	$4,554

Pre-Tax NPV Sensitivity To Mill Head Grade (Commodity Price (%))

Mill Head Grade	(20%)	(10%)	0%	10%	20%
(20.0%)	7.7%	11.2%	14.4%	17.3%	20.0%
(10.0%)	11.3%	14.8%	18.1%	21.2%	24.1%
--	14.5%	18.1%	21.6%	24.8%	27.9%
10.0%	17.4%	21.2%	24.8%	28.2%	31.5%
20.0%	20.2%	24.2%	27.9%	31.5%	35.0%

Pre-Tax NPV Sensitivity To Foreign Exchange (Commodity Price (%))

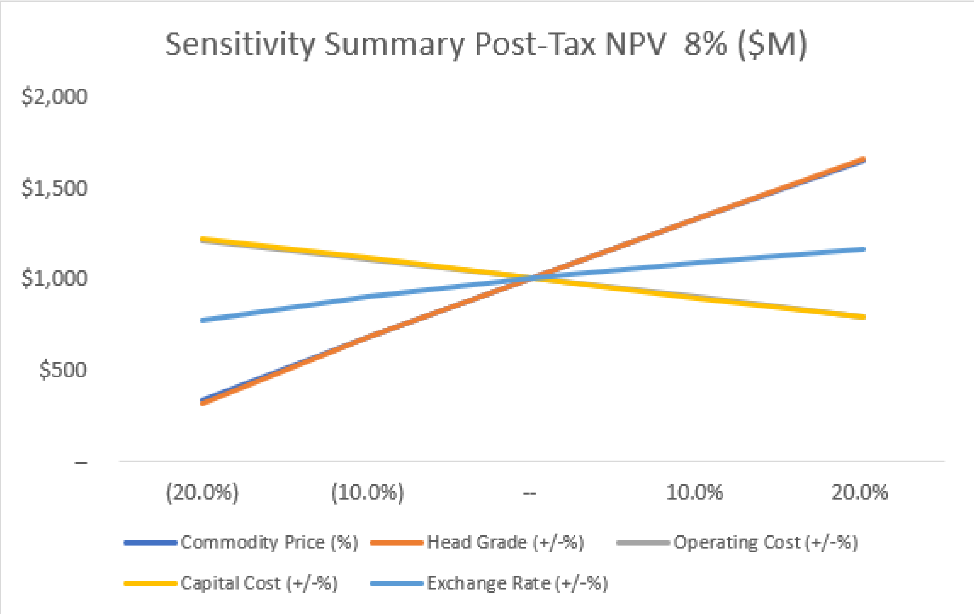
Foreign Exchange	(20%)	(10%)	0%	10%	20%
(20.0%)	$629	$1,200	$1,771	$2,342	$2,913
(10.0%)	$769	$1,339	$1,910	$2,481	$3,052
--	$881	$1,452	$2,023	$2,593	$3,164
10.0%	$973	$1,544	$2,114	$2,685	$3,256
20.0%	$1,049	$1,620	$2,191	$2,762	$3,333

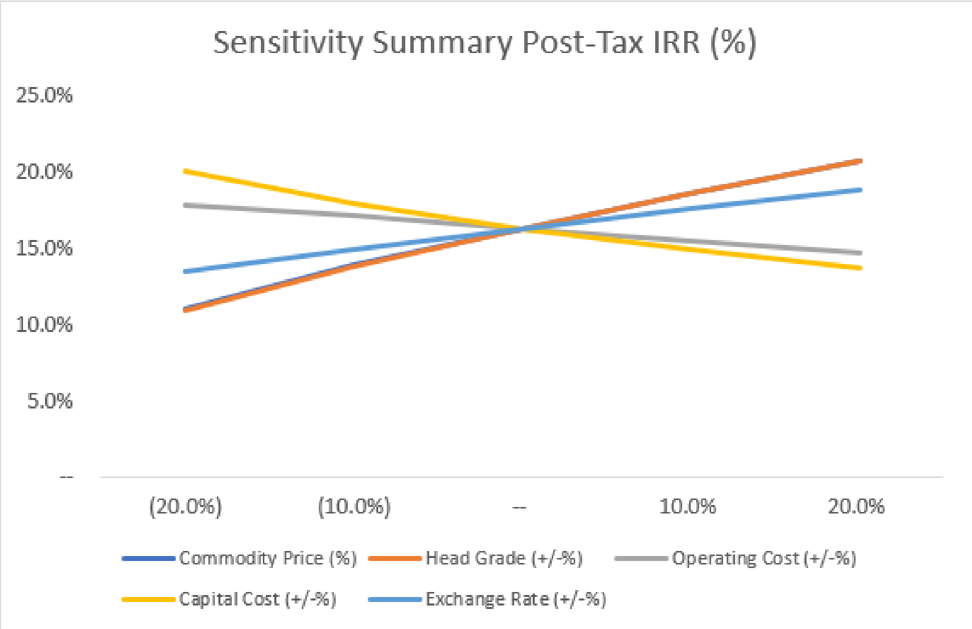
Pre-Tax NPV Sensitivity To Foreign Exchange (Commodity Price (%))

Foreign Exchange	(20%)	(10%)	0%	10%	20%
(20.0%)	12.1%	15.3%	18.3%	21.2%	23.9%
(10.0%)	13.3%	16.8%	20.0%	23.1%	26.0%
--	14.5%	18.1%	21.6%	24.8%	27.9%
10.0%	15.6%	19.4%	23.0%	26.4%	29.7%
20.0%	16.6%	20.6%	24.4%	28.0%	31.4%

Note: prepared by Ausenco, 2022.

Figure 22-1: Post-Tax NPV and IRR Sensitivity Results



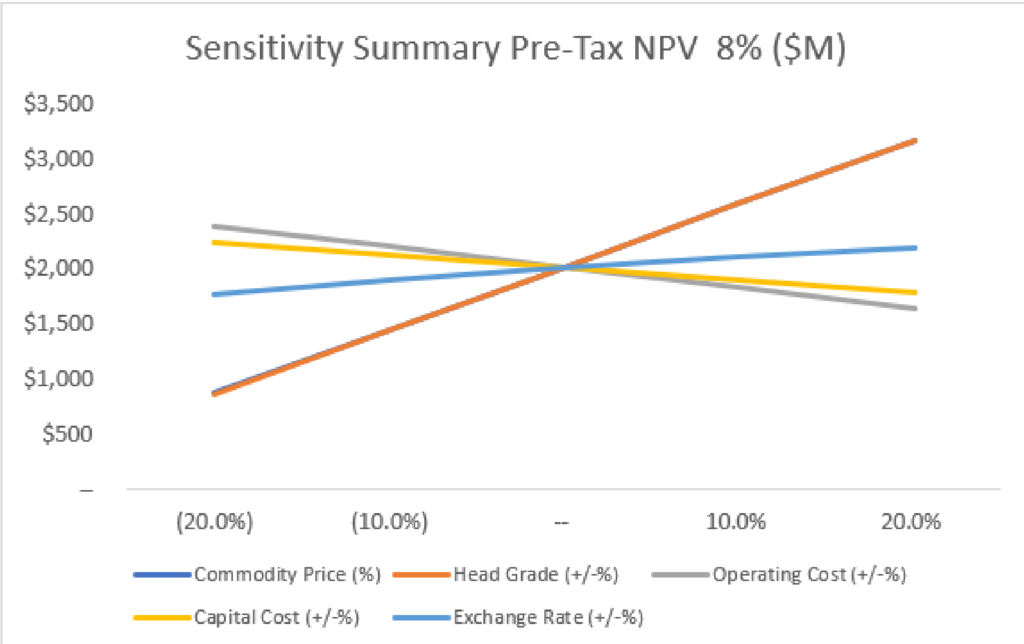


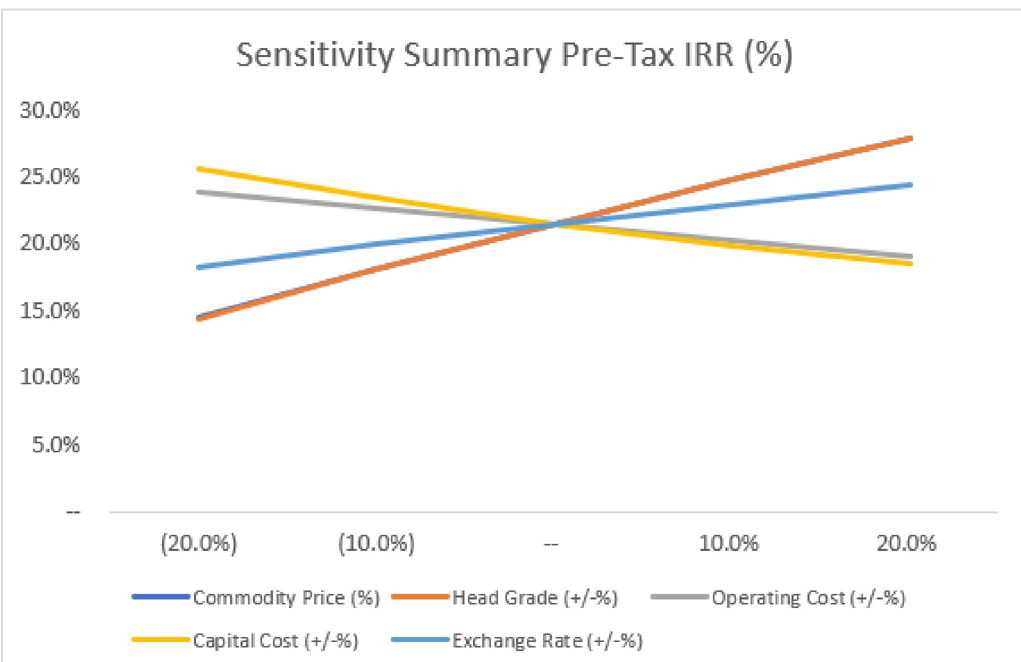
Notes: Figure prepared by Ausenco, 2022.
For the chart titled "Sensitivity Summary Post-Tax NPV 8% ($M)", chart lines overlap for:

- commodity price and head grade, and
- operating cost and capital cost.

For the chart titled "Sensitivity Summary Post-Tax IRR (%)", chart lines overlap for commodity price and head grade.

Figure 22-2: Pre-Tax NPV and IRR Sensitivity Results





Notes: Figure prepared by Ausenco, 2022.
For the chart titled "Sensitivity Summary Pre-Tax NPV 8% ($M)", chart lines overlap for commodity price and head grade.
For the chart titled "Sensitivity Summary Pre-Tax IRR (%)", chart lines overlap for commodity price and head grade.

23 ADJACENT PROPERTIES

This section is not relevant to this Report.

24 OTHER RELEVANT DATA AND INFORMATION

This section is not relevant to this 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

Information obtained from Candente Copper experts and legal experts retained by Candente Copper supports that the support that the mineral tenure held is valid, and is sufficient to support a declaration of Mineral Resources at Cañariaco Norte and Cañariaco Sur.

The Project consists of 15 mining concessions totalling approximately 10,190 ha. Ausenco was advised by Candente 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.

Mineral concession rights do not 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. No surface rights have been acquired in support of any future mine operations.

Candente Copper will need to obtain an authorization from the 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 certain of the Cañariaco concessions. Profit based taxes are due to the Government of Peru.

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, and can support Mineral Resource estimation; however, the deposit is not included in the 2022 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 Candente 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.

Candente Copper acquired 100% ownership of the Project in February 2002. Since that date, Candente 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 Candente Copper drill core was halved and sampled on 2 m intervals.

A total of 9,424 bulk density readings were taken by Candente 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 Candente 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 Candente 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 33-element 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 Candente 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 were verified by Candente Copper personnel.

Drill data collected from the INGEMMET, Placer Dome, and Billiton campaigns were re-logged by Candente Copper personnel, and nine of the drill holes have been re-assayed. Based on the good correlation between the historical grades and the Candente Copper re-assay grades, all of the historical data have been inserted into the final database.

Three pairs of twinned holes were drilled by Candente 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 Candente 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

Three major phases of testwork were conducted. The first consisted of process development to define the type of processing most applicable to the mineralization. This was followed by more detailed work to optimize process conditions. 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 mill feed variability testing campaign conducted 2011 and 2012 was designed to improve the geometallurgical understanding of the deposit and robustness in the proposed process flowsheet

Testwork included: mineralogy, QEMSCAN examination, comminution and variability comminution tests, tests on the effects of grind sizes, collectors and pH, sulphidization, 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 processing routes, and were performed using samples that are typical of the mineralization styles.

Samples selected for testing from Cañariaco Norte were representative of the various types and styles of mineralization. Samples were selected from a range of depths within the Cañariaco Norte e deposit. Sufficient samples were taken so that tests were performed on sufficient sample mass.

Recovery factors estimated for Cañariaco Norte are based on appropriate metallurgical testwork. Copper recoveries are expected to vary with feed grade ranging from 91% at 0.6% Cu and 58% at 0.1% Cu. 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, gold and silver recoveries are projected to be 64.7% and 57.2% respectively.

As part of the final conclusions of the testing campaigned carried out in 2008 it was noted that the presents of arsenic and antimony 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. This resulted in a final flowsheet generating a low-grade copper concentrate to the smelter with arsenic content controlled to a level where no impact on concentrate marketability is anticipated over the LOM.

No testwork has been performed on mineralization from Cañariaco Sur, and recovery assumptions and metallurgical behaviour of the mineralization are based on Cañariaco Norte as an analogue.

25.6 Mineral Resource Estimates

Mineral Resources are reported using the 2014 CIM Definition Standards and assume 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; 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 title, obtain surface rights to allow mine construction and operations, obtain environment and other regulatory permits, and obtain the social license to operate.

25.7 Mine Plan

The 2022 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.

Based on earlier internal desktop trade-off studies, the mine plan considers an initial 40 kt/d processing rate which is later doubled to 80 kt/d in production Year 7. The peak mining capacity is 54.6 Mt/a. Four phase designs were developed for the planned single open pit. The mine life will run for approximately 28 years, with the last year being a partial year.

Mine planning was performed based on marginal cut-offs applied to the NSR grade item. During the initial Project phase when the concentrator is operating at 40 kt/d, the marginal cut-off is the sum of the mill feed based operating costs, (processing, G&A and tailings management), which is US$7.91/t. When the process rate increases to 80 kt/d, the marginal cut-off decreases to US$6.52/t. No long-term stockpiling of low-grade material has been considered. Within the ultimate pit, at the US$6.52/t NSR cut-off the classification breakdown of the mill feed material is 54% Measured, 38% Indicated and 8% 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.

Three years of pre-production mining activities are required. Mill feed delivery to the crusher in the first production year is forecast at 11.1 Mt, which is inclusive of the pre-production stockpiled mill feed material reclaim. In production Year 2 through Year 6, the full 14.6 Mt (40,000 t/d) will be delivered to the crusher area. In Year 7, the production rate will increase to 26.3 Mt. During Year 8 through Year 27, the full 29.2 Mt (80,000 t/d) will be delivered. The last year of production, Year 28, will be a partial year with 8.2 Mt processed.

25.8 Recovery Plan

The processing plant will be built in two phases. Phase 1 will have a nominal throughput of 14.6 Mt/a during the first six production years. Phase 2 will double the annual production by adding a parallel line, from the mill feed stockpile to concentrate and tailing filtration and will have the same equipment and layout as the Phase 1 process line. The plant is designed to operate 24 hours per day, 365 days per year with an overall plant availability of 92%.

The process plant selected is a conventional copper concentrator and the process design is typical of a concentrator treating copper sulfide mill feed. The process plant feed will be supplied from the open pit mine with a LOM average feed grade of 0.39% Cu and will produce a copper concentrate containing 26% Cu.

25.9 Infrastructure

The mine site facilities are divided into four general areas: the mine, the crusher (that include buildings and structures for repair and maintenance of mine and plant equipment), the plant site, and the camp area, (which includes facilities for personnel accommodations, administration, and security). Other support facilities and services include site access, power supply and distribution, water supply, explosive storage and handling, communication systems and waste storage facility at the site.

The site was selected on a natural elevated area on 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.

A high-level waste storage siting and disposal method was performed for waste rock and tailings. Tailings storage options included slurry tailings storage behind a conventional embankment and filtered tailings in a dry stack facility. Due to the potential ARD issues with the waste rock and tailings it was decided to develop a single waste storage facility (co-deposal) on the west side of the project in the head waters of the Yerma Creek basin to better manage contact water. The waste rock is crushed, the tailings are filtered and both materials are transported to the WSF on conveyor systems either together or separately, depending on the stacking plan. In addition, calcium carbonate will be added to the two streams at the start of the two WSF conveyors to mitigate acid generation of the materials. The facility has been designed in accordance with international standards for storage of tailings and waste rock.

The existing and planned infrastructure, availability of staff, the existing power, water, and communications facilities, the methods whereby goods are transported to the mine, and any planned modifications or supporting studies are well-established, or the requirements to establish such, are well understood by Candente Copper, and can support the declaration of Mineral Resources and the 2022 PEA economic analysis.

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 will range from 625 m^3/hr to a maximum of 2,228 m^3/hr over the life of the Project. A water treatment plant with a capacity to treat up to 144 m^3/hr will be required to treat excess effluent. The raw water, process water and contact water management ponds were sized to meet forecast Project water supply needs. No off-site water supply is required.

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 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.

Completed baseline studies evaluated air quality, noise, hydrography, hydrology and hydrogeology, soils, water quality, ecosystems, flora, and fauna.

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 WSF to supply the freshwater replenishment requirements of the process plant.

The economic analysis in the 2022 PEA assumes a closure cost of US$104 M.

A number of permits will be required in support of construction and operations. For the purposes of the 2022 PEA, Candente 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 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, May 2011 and June 2014, archaeological evaluations were carried out. 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 Peruvian regulatory requirements.

25.11 Markets and Contracts

No market studies or product valuations were completed as part of the 2022 PEA. Market price assumptions were based on a review of public information, industry consensus, standard practiced and specific information from comparable operations in the region.

Project economics were estimated based on long-term metal prices of US$3.50/lb Cu, US$1,650/oz Au and US$21.50/oz Ag, which was established by the Company in conjunction with consensus forecasts from various financial institutions.

No contracts for transportation or off-take of the concentrates are currently in place. However, if and when they are negotiated, they are expected to be within the industry norms. Similarly, there are no contracts currently in place for supply of reagents, utilities, or other bulk commodities required to construct and operate the Project.

25.12 Capital Cost Estimates

Capital costs are provided in Q4 2021 US$. The estimate uses AACE International guidelines and is reported as an AACE Class 5 Order of Magnitude/Conceptual Study estimate with a -30% to +50% accuracy. Exchange rates used include US$1.00 = PENS/4.00, and US$1.00 = C$/1.29.

The overall capital cost estimate was developed by Ausenco with contributions from AGP for the mining cost estimates. Costs were separated into initial and sustaining capital costs, and further subdivided out into what will be needed for the first project phase (40 kt/d) and expansion phase (80 kt/d).

The total direct cost for Phase 1 is US$744.0 and the indirect cost for Phase 1 is US$299.2 M, with an overall cost of US$1043.1 M. The total direct cost for Phase 2 is US$207.5 and the indirect cost for Phase 2 is US$97.4 M, with an overall cost of US$304.9 M. The total sustaining capital cost is estimated at US$119.0 M.

25.13 Operating Cost Estimates

Operating costs are provided in Q4 2021 US$. The estimate uses AACE International guidelines and is reported as an AACE Class 5 Order of Magnitude/Conceptual Study estimate with a -30% to +50% accuracy. Exchange rates used include US$1.00 = PENS/4.00, and US$1.00 = C$/1.29.

Mine 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 are fuel and labour.

Process operating cost were developed based on the production of copper concentrate at plant feed rates and waste handling strategy as per the production schedule. The estimate includes average annual processing costs for power, consumables, maintenance consumables and labour.

The G&A operating costs were estimated based on benchmarked data from similar projects in similar locations. Costs include camp operations, G&A personnel, off-site offices, contracts, and vehicle maintenance, as well as miscellaneous project costs.

The WSF operating cost included the mechanical spreading and compacting equipment, water truck for dust suppression. The conveyor and stacking equipment, and CWMP pump and pipeline to the plant and water treatment plant are located in the processing operating cost. The mechanical equipment operating costs were estimated based on benchmark data from similar projects in similar locations. The costs include fuel, equipment maintenance, and labor.

The onsite operating cost estimate is US$1.19/lb Cu (US$8.21/t milled), the offsite costs are estimated at US$0.46/lb Cu (US$3.16t/milled), a royalty of approximately US$0.02/lb (US$0.12/t milled), and a credit for gold and silver production is estimated at US$0.39/lb Cu (US$2.67/t milled). The overall operating cost estimate for the LOM is US$1.28/lb Cu (US$8.86/t milled).

25.14 Economic Analysis

The 2022 PEA is preliminary in nature and is partly based on 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 2020 PEA based on these Mineral Resources will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

The economic analysis was performed assuming an 8% discount rate. The pre-tax NPV discounted at 8% is $2,022.5 M; the IRR is 21.6%, and payback period is 6.1 years. On a post-tax basis, the NPV discounted at 8% is $1,010.3 M; the IRR is 16.3%, and the payback period is 7.1 years.

The Project is most sensitive to changes in commodity price and head grade. In order of decreasing sensitivity, the project is less sensitive to changes in total capital cost, total operating cost, and foreign exchange rate.

25.15 Risks and Opportunities

25.15.1 Risks

Risks that may affect the Mineral Resource estimate specifically were provided in Section 14.6. Risks that may affect the 2022 PEA that is based on that estimate include:

- Geotechnical design of +900 m pit highwall: the ultimate pit highwall is over 900 m in vertical height, making the proposed Cañariaco Norte pit one of the world's deepest. The current study design basis for the overall pit and mine bench slope angles is pending completion of a geotechnical drilling program that was halted in 2013. The program is required to be completed to provide additional information on the rock quality in the pit wall areas. The current design is based on assessment of scoping level information, visual inspection of available core, and limited physical testing of available material. If future geotechnical investigations result in shallower overall pit slope angles, then the stripping ratio will increase and have a negative impact on the project mining costs;

- Main access road routing: the conceptual design of the main access road is based on low-quality contour data. This may lead to a less-than-optimum design with a risk that the road may not be constructible in some sections. Aerial or land reconnaissance followed by more detailed geotechnical investigations along the proposed road routing is recommended to decrease the risk in the design of this road;

- Landslides: 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 valley slopes will be required to assess the severity of this risk and what mitigation can be undertaken to reduce such risk;

- Coarse mill feed material stockpile and dusting: the open-air design of the coarse ore stockpile is typical of many operating mines; however, it may result in non-compliant dust loading during dry periods. Dust suppression or a stockpile cover may be required, which will incur additional capital/operating costs for mitigation;

- There is insufficient data too provide a proper 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 any ARD risk ;

- There is no geotechnical information of the WSF foundation. There may be unknow 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 WSF will be required to assess any potential risk to the facilities and develop any potential mitigation measures;

- Inability to meet filtered tailings moisture content that could create trafficability issues for the conveyor system. Need additional filtering test program to validate that filter design moisture content can be achieved;

- 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.

- Arsenic in concentrate: Arsenic content in the copper concentrate will attract penalties. Further marketing studies are required to confirm what penalties may be imposed on the low-grade copper concentrate;

- Delivery of long-lead capital equipment: market conditions may increase delivery schedules for long-lead capital equipment items. Equipment may need to be secured and purchased earlier than anticipated. This may result in increases to the capital costs as envisaged in the 2022 PEA;

- Port availability: current assumptions are that the marine facilities to be developed by Lumina will be available for Project use. This should be confirmed in negotiations with Lumina.

25.15.2 Opportunities

Opportunities include:

- Mineral resource estimation: additional drilling should be undertaken in the vicinity of Cañariaco Norte with the objective of providing sufficient data to support potential upgrading of mineralization that is currently classified as Inferred to higher confidence categories. The potential exists to include the mineralization at the Cañariaco Sur deposit, which is also currently classified as Inferred, with further drilling, in more detailed studies. The Quebrada Verde prospect retains exploration potential;

- Mine plan: there is potential to optimize the mine plan in future more detailed studies. This may potentially result in a decrease in the forecast Project payback period, and an increase in the IRR;

- Flowsheet: there is potential to optimize the process flowsheet plan in future more detailed studies. This may potentially result in a decrease in capital cost, forecasted Project payback period, and an increase in the IRR;

- Geo-metallurgy/metallurgy: increased testwork and database has the potential to optimize the process parameters in future more detailed studies. This may potentially result in an increase payables recoveries and/ or concentrate quality and/ or decreased deleterious element recoveries. This may potentially result in increased revenue, NPV and IRR;

- WSF there is a potential to optimize the deposition program in future studies. This may result in decrease capital, sustaining capital, and operating costs.

25.16 Conclusions

Under the assumptions and parameters presented in this Report, the 2022 PEA shows a positive economic outcome. Additional mining and technical studies are warranted.

26 RECOMMENDATIONS

26.1 Introduction

A two-phase work program is recommended.

Recommendations Phase 1 will consist of a review of available geotechnical data to determine if any additional pit geotechnical drilling is required, and an exploration program to better delineate the mineralization at Cañariaco Sur and provide core for an initial metallurgical testwork program. The recommended budget for this program is approximately US$6 M.

A second phase of work is also recommended. A portion of the second recommendations phase is dependent on the drill campaign for provision of fresh drill core from Cañariaco Sur; however, the majority of the studies can be conducted concurrently with the Cañariaco Sur drill program. The second phase includes updated Mineral Resource estimates for Cañariaco Norte and Cañariaco Sur. Metallurgical testwork consisting of variability tests for Cañariaco Norte using core in cold storage from the 2013 metallurgical drill program, and initial metallurgical testwork on Cañariaco Sur is proposed. Other recommendations include mining and technical studies for support of process plant, infrastructure, and WSF and water management designs. A CIRA must be obtained for all areas that are planned to or could host infrastructure. The recommended budget for this program is about US$2.7 M.

26.2 Recommendations Phase 1

The first phase consists of geotechnical data review and analysis, and Mineral Resource estimation support drilling at Cañariaco Sur.

26.2.1 Geotechnical

The ultimate pit highwall envisaged in the 2022 PEA is over 900 m in vertical height, making the proposed Cañariaco Norte pit one of the world's deepest. A detailed review and analysis of the existing drill core and geotechnical data is required to support geotechnical assumptions for open pit slope design that would be used to support more detailed mine designs, and to determine if any additional specific geotechnical drill holes will be required to support FS level pit slope designs.

This program is estimated at US$0.05 M.

26.2.2 Exploration Drilling - Cañariaco Sur

A 40-hole drill program is recommended for Cañariaco Sur. Locations of the proposed drill holes were shown on Figure 7-9 and Figure 7-10. These drill collar locations are conceptual, and as drill results come to hand during program execution, the locations may be modified to reflect those results. Drill holes are assumed to be a nominal 500 m on average.

Assuming an all-in core drilling cost of US$300/m, this program is estimated at US$6 M.

26.3 Recommendations Phase 2

The second recommendations phase consists of various studies and testwork that should be completed in support of more detailed Project evaluation.

26.3.1 Mineral Resource Estimation, Cañariaco Norte

As part of an updated Mineral Resource estimate for the Cañariaco Norte area, the following should be completed:

- A mineral zonation model should 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;
- A mineralization hardness model should be constructed to provide support for the metallurgical throughput assumptions;
- Density values were assigned to blocks during Mineral Resource estimation based on the lithological codes. Additional bulk density determinations should be collected from core samples to provide a statistical database of density determinations for each major mineralization and waste lithology.

This program is estimated to require a budget of approximately US$0.05 M.

26.3.2 Mineral Resource Estimation, Cañariaco Sur

Once results are to hand from the exploration drilling at Cañariaco Sur, an updated Mineral Resource estimate should be completed. This work is estimated at US$0.15 M.

26.3.3 Metallurgical Testwork

26.3.3.1 Cañariaco Norte

Variability samples are required to understand the responses of the various mineralized zones to flotation kinetics and contaminant correlations. This work does not require the completion of any metallurgical drill program as the drill core from the 2013 metallurgical drill program has been maintained in cold storage.

Additional comminution tests (e.g. crushing work index, rod work index, SMC and abrasion index)) are recommended on material representative of the first 3–5 years of the planned operation, to provide more confidence in equipment selection, and to ensure that there is sufficient comminution information that is spatially representative of the variability within the various mineralized zones.

Flotation tests, including optimisation, locked cycle, QEMSCAN and contaminant removal tests, should be undertaken on variability samples. Performance optimization, contaminant liberation and/or confirmation should be tested on individual variability samples as per zone-optimized conditions determined from flotation kinetic tests.

A tailings dewatering test and a concentrate dewatering test are recommended. Rheology of the tailings slurry stream should be conducted on the combined composite sample, without any prior treatment that would affect the rheology the testwork program is expected to cost approximately US$0.37 M.

26.3.3.2 Cañariaco Sur

Initial metallurgical testwork is required for the Cañariaco Sur deposit. This should include comminution, flotation, thickening and filtrations tests to ensure Cañariaco Sur mineralization reacts similar to Cañariaco Norte as expected. This work requires the completion of the proposed drill program outlined in Section 26.2.4 so that fresh core is available for testing.

The budget estimate for this work is US$0.27 M.

26.3.4 Tailings and Waste Facilities Engineering

Additional geotechnical drilling and sampling is recommended in the area of 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 should be completed 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 WSF 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 WSF 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 is to perform this work is US$1.5 M.

26.3.5 Water Management

The geotechnical drill holes recommended in Recommendations Phase 1 should be used to provide additional hydrological data and act, where appropriate as monitoring wells. Piezometers and monitoring wells should be installed/tested to assess groundwater gradients.

Packer testing should be conducted on the geotechnical drill holes that intersect the planned highwall, as most K values were measured in the mineralized zone and are not necessarily representative of K on the pit slopes. Selected drill holes should be packer-tested to provide data on the conditions at depth within the proposed pit area, because the K measurements were predominantly in the top 200 m and do not necessarily represent K at greater depth.

Pumping tests should be conducted in high conductivity locations (e.g. fault structures) for assessing storage and boundary conditions.

A three-dimensional numerical model should be constructed to simulate the groundwater system and assess pit depressurization requirements.

Hydrometric station(s) should be installed and monitored in the Rio Cañariaco to assess the interaction of the river with groundwater.

These studies will require a budget of approximately US$0.3 M to complete.

26.3.6 Environmental, Permitting and Social

To date, Candente Copper has carried out the Archaeological Evaluation Project for the planned open pit area. The results showed that this area and the surrounding wooded areas have no archaeological sites or remains. In order to start construction activities, Candente Copper must obtain CIRA(s) for the other planned Project facilities area.

This study will require a budget of approximately US$0.05 M to complete.

27 REFERENCES

27.0 Document References

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