NI 43-101 Technical Report on

Antamina Mining Operation

Peru

Prepared For:

Teck Resources Limited

Prepared By:

Mr. Lucio Canchis, RM SME, Minera Antamina

Mr. Fernando Angeles, P.Eng., Minera Antamina

Mr. Hernando Valdivia, FAusIMM, Minera Antamina

Mr. Carlos Aguirre, FAusIMM, Minera Antamina

Effective Date:

31 December, 2024

CERTIFICATE OF QUALIFIED PERSON

I, Lucio Canchis, RM SME, am employed as the Resource Estimation Superintendent with Compañía Minera Antamina S.A. (Minera Antamina), with office address at El Derby Avenue #055, Cronos Building, Tower 1, 8th Floor, Surco, Lima, Peru.

This certificate applies to the technical report titled "NI 43-101 Technical Report on Antamina Mining Operation, Peru" that has an effective date of 31 December 2024 (the "technical report").

I am a registered member of the Society for Mining, Metallurgical & Exploration (SME); membership number 4172150.

I obtained a Bachelor's Degree in Geological Engineering from the National University of San Marcos, Peru, in 1994.

I have 30 years of experience in my profession since graduating. I have been directly involved in geological modeling, mineral resource estimation, resource confidence classification, quality assurance and quality control programs and reviews, design of drill programs, geometallurgy, and database integrity.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 Standards of Disclosure for Mineral Projects (NI 43–101) for those sections of the technical report that I am responsible for preparing.

I have worked at the Antamina Operations since 2001. This familiarity with operations serves as my personal inspection.

I am responsible for Sections 1.1, 1.2, 1.3, 1.4, 1.5, 1.6, 1.7, 1.8, 1.10, 1.11, 1.22.1 (mineral resources), 1.22.2 (mineral resources), 1.24; Sections 2.1, 2.2, 2.3, 2.4.1, 2.5, 2.6, 2.7; Section 3; Section 4; Section 5; Section 6; Section 7; Section 8; Section 9; Section 10; Section 11; Sections 12.1, 12.2, 12.3.1; Section 14; Sections 25.1, 25.2, 25.3, 25.4, 25.5, 25.6, 25.8, 25.17 (mineral resources), 25.18 (mineral resources); Section 26; and Section 27 of the technical report.

I am not independent of Teck Resources Limited or Minera Antamina as independence is described by Section 1.5 of NI 43–101.

I have been involved with the Antamina Resource Estimation since 2011 as a Minera Antamina employee.

I have read NI 43–101, and the sections of the technical report I am responsible for have been prepared in compliance with that Instrument.

As of the effective date of the technical report, to the best of my knowledge, information, and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

Dated: 19 February 2025

"Signed"

Lucio Canchis RM SME

CERTIFICATE OF QUALIFIED PERSON

I, Fernando Angeles, P.Eng., am employed as the Long-term Planning Superintendent with Compañía Minera Antamina S.A. (Minera Antamina), with an office address at El Derby Avenue #055, Cronos Building, Tower 1, 8th Floor, Surco, Lima, Peru..

This certificate applies to the technical report titled "NI 43-101 Technical Report on Antamina Mining Operation, Peru" that has an effective date of 31 December, 2024 (the "technical report").

I am a Professional Engineer of Engineers & Geoscientist BC (EGBC), membership number 178264.

I graduated from Pontifical Catholic University of Peru with a Bachelor in Mining Engineering and from the University of British Columbia with a Master of Engineering degree.

I have practiced my profession for 20 years. I have been directly involved in the production planning process at the Antamina mine. As head of the life-of-mine planning process, I provide expertise in the main variables of the production chain, such as production constraints, reconciliation of the predictive models, existing and future infrastructure needs, and regulatory permitting processes.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 *Standards of Disclosure for Mineral Projects* (NI 43–101) for those sections of the technical report that I am responsible for preparing.

I worked at the Antamina operation from 2007 to 2011 as a site planning engineer and since 2018 as a Long-Term Planning Superintendent. This familiarity with operations serves as my personal inspection.

I am responsible for Sections 1.1, 1.2, 1.8, 1.12, 1.13, 1.14, 1.16 (excepting TSF, port and water management), 1.17, 1.18, 1.19, 1.20, 1.21, 1.22.1 (excepting mineral resources), 1.22.2 (excepting mineral resources), 1.23, 1.24; Sections 2.1, 2.2, 2.3, 2.4.2, 2.5, 2.6; Section 3; Section 12.3.2; Section 15; Sections 16.1, 16.4, 16.5, 16.6, 16.7, 16.8, 16.9, 16.10; Sections 18.1, 18.2, 18.3, 18.4, 18.10, 18.11; Section 19; Section 20; Section 21; Section 22; Section 24; Sections 25.1, 25.6, 25.9, 25.10, 25.12, 25.13, 25.14, 25.15, 25.16, 25.17, 25.18.1 (excepting mineral resources), 25.18.2 (excepting mineral resources), 25.19; Section 26; and Section 27 of the technical report.

I am not independent of Teck Resources Limited or Minera Antamina as independence is described by Section 1.5 of NI 43–101.

I have been involved with the Antamina Operations for more than 10 years, from 2007 to 2011, and then rejoined the company from 2018 to now.

I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

Dated: 19 February 2025

“Signed and sealed”

Fernando Angeles, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

I, Hernando Valdivia, FAusIMM, am employed as the Superintendent of Metallurgy and Chemical Laboratory with Compañía Minera Antamina S.A. (Minera Antamina), with an office address at El Derby Avenue #055, Cronos Building, Tower 1, 8th Floor, Surco, Lima, Peru.

This certificate applies to the technical report titled "NI 43-101 Technical Report on Antamina Mining Operation, Peru" that has an effective date of 31 December, 2024 (the "technical report").

I am a Fellow of the Australasian Institute of Mining and Metallurgy, membership number 3048291.

I graduated in December 1997 from the National University of San Agustín, Peru, earning a degree in Metallurgical Engineering.

I have been practicing my profession for 25 years, specializing in mineral concentration through grinding and flotation processes, as well as conducting geometallurgical investigations to optimize the recovery of polymetallic minerals.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 *Standards of Disclosure for Mineral Projects* (NI 43–101) for those sections of the technical report that I am responsible for preparing.

I work at the Antamina Operations and have done so since 2001. This familiarity with operations serves as my personal inspection.

I am responsible for Sections 1.1, 1.2, 1.8, 1.9, 1.15, 1.16 (pipeline, port and power only), 1.17 (markets only), 1.22.2 (process only), 1.24; Sections 2.1, 2.2, 2.3, 2.4.3, 2.5; Section 3; Section 12.3.3; Section 13; Section 17; Sections 18.6, 18.7, 18.11; Section 19.1; Sections 25.1, 25.6, 25.7, 25.11, 25.12 (pipeline, port and power only), 25.13 (markets only), 25.18 (process only); Section 26 and Section 27 of the technical report.

I am not independent of Teck Resources Limited or Minera Antamina as independence is described by Section 1.5 of NI 43–101.

I have been involved with the Antamina Operations since 2001 as a Minera Antamina employee.

I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

Dated: 19 February 2025

"Signed"

Hernando Valdivia, FAusIMM

CERTIFICATE OF QUALIFIED PERSON

I, Carlos Aguirre, FAusIMM, am employed as the Geotechnical and Hydrogeological Manager, Business Planning and Development with Compañía Minera Antamina S.A. (Minera Antamina), with an office address at El Derby Avenue #055, Cronos Building, Tower 1, 8th Floor, Surco, Lima, Peru.

This certificate applies to the technical report titled "NI 43-101 Technical Report on Antamina Mining Operation, Peru" that has an effective date of 31 December, 2024 (the "technical report").

I am a Fellow of the Australasian Institute of Mining and Metallurgy, membership number 3047509.

I graduated from Pontificia Universidad Católica del Perú with a degree in Civil Engineering in February, 2005.

I have practiced my profession for 20 years. I have been directly involved in geotechnical and hydrogeological field investigations, explorations, studies and mine facilities geotechnical designs for open pits, waste rock storage facilities, leach pads, water reservoirs and tailings storage facilities, in both the gold and copper mining industries.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 Standards of Disclosure for Mineral Projects (NI 43–101) for those sections of the technical report that I am responsible for preparing.

I work at the Antamina Operations and have done so since 2013. This familiarity with operations serves as my personal inspection.

I am responsible for Sections 1.1, 1.2, 1.8, 1.14 (geotechnical and hydrological only), 1.16 (tailings storage and water management only), 1.24; Sections 2.1, 2.2, 2.3, 2.4.4, 2.6; Section 3; Section 12.3.4; Sections 16.2, 16.3; Sections 18.5, 18.8, 18.9; Section 19; Sections 25.1, 25.6, 25.10 (geotechnical and hydrological only), 25.12 (tailings storage and water management only); Section 26; and Section 27 of the technical report.

I am not independent of Teck Resources Limited or Minera Antamina as independence is described by Section 1.5 of NI 43–101.

I have been involved with the Antamina Operations since 2013 as a Minera Antamina employee.

I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

Dated: 19 February 2025

“Signed”

Carlos Aguirre, FAusIMM

FORWARD LOOKING INFORMATION

This Technical Report contains certain forward-looking information and forward-looking statements as defined in applicable securities laws (collectively referred to as forward-looking statements). These statements relate to future events or our future performance. All statements other than statements of historical fact are forward-looking statements. The use of any of the words “anticipate”, “plan”, “continue”, “estimate”, “expect”, “may”, “will”, “project”, “predict”, “potential”, “should”, “believe” and similar expressions is intended to identify forward-looking statements. These statements involve known and unknown risks, uncertainties and other factors that may cause actual results or events to differ materially from those anticipated in such forward-looking statements. These statements speak only as of the date of this Technical Report.

Forward looking statements in this Technical Report include, but are not limited to, statements regarding mineral resource and mineral reserve estimates, commodity prices, estimated of equipment and infrastructure required for the remainder of the life of mine, operating, capital, closure and other costs for the remainder of the life of mine, timing for submission of regulatory documents and expectations regarding approvals, estimated mine life and strip ratios, life of mine plans, closure activities and permitting renewals or modifications.

Inherent in forward-looking statements are risks and uncertainties beyond our ability to predict or control which may cause actual results to differ materially from those expressed or implied by the forward-looking statements contained in this Technical Report, including: risks that may affect our operating or capital plans; risks generally encountered in the permitting and development of mineral properties such as unusual or unexpected geological formations; risks associated with volatility in financial and commodities markets and global uncertainty; risks associated with fluctuations in the market prices of commodities, which are cyclical and subject to substantial price fluctuations; risks relating to delays associated with permit appeals or other regulatory processes, ground control problems, adverse weather conditions, process upsets, equipment malfunctions or technology failures; risks related to inflation; risks associated with climate change, environmental compliance, changes in environmental legislation and regulation or changes to closure obligations; risks associated with unanticipated metallurgical difficulties; risks associated with labour disturbances and availability of skilled labour; risks associated with changes to the tax and royalty regimes; risks associated with lack of access to markets; risks associated with mineral reserve and resource estimates; risks posed by fluctuations in exchange rates and interest rates, as well as general economic conditions; risks associated with the need to procure goods and services, including risks relating to availability, prices, quality and timely delivery of goods and services; social and political risks associated with operations in Peru; risks related to trade barriers, restrictions or tariffs; risks associated with information technology, including cybersecurity risks and risks associated with the failure of such information technology; risks associated with our ability to obtain or maintain insurance and risks associated with tax reassessments and legal proceedings. Risks and assumptions regarding Mineral Resources are summarized in the footnotes to the Mineral Resources table and outlined in Section 14. Risks and assumptions regarding Mineral Reserves are summarized in the footnotes of the Mineral Reserve table and outlined in Section 15 and Section 16 of the Report. Risks and assumptions regarding life of mine plans, including estimates of required equipment and infrastructure, are outlined throughout the Report, including in Sections 16 and 18. Risks and assumptions regarding operating, capital, closure, and operating costs are outlined throughout the Report, including Section 21.

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1.1 Introduction

Mr. Lucio Canchis, RM SME, Mr. Fernando Angeles, P.Eng., Mr. Hernando Valdivia, FAusIMM and Mr. Carlos Aguirre, FAusIMM, prepared this technical report (the Report) for Teck Resources Limited (Teck) on the Antamina mining operation (the Antamina Operations or the Project), located in in the San Marcos District, Province of Huari, Ancash Department, Peru.

The operations are owned by Compañía Minera Antamina S.A. (Minera Antamina), which is a Peruvian limited liability company (sociedad anonima). There are four shareholders with the following ownership percentages:

- BHP Group Ltd. (BHP): 33.75%;
- Glencore plc. (Glencore): 33.75%;
- Teck (through its subsidiary Teck Base Metals Ltd.): 22.50%;
- Mitsubishi Corp. (Mitsubishi): 10.00%.

Mineral Resources and Mineral Reserves are estimated for the Antamina deposit.

1.2 Terms of Reference

The Report was prepared to support Teck's Annual Information Form for the year-end 31 December 2024.

Mineral Resources and Mineral Reserves are reported using the confidence categories in the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves (May 2014; the 2014 CIM Definition Standards).

Currency is expressed in United States dollars (US$) unless stated otherwise. The Peruvian currency is the Peruvian Nuevo Sol (PEN or S/). Units presented are metric units, such as metric tonnes, unless otherwise noted. The Report uses Canadian English.

1.3 Project Setting

The Antamina operations are situated in the San Marcos District, Province of Huari, Ancash Department, approximately 270 km north of the Peruvian capital, Lima, and 38 km from the closest town, San Marcos. Huaraz, the department capital, can be reached through 200 km of paved road or 156 km of partial dirt road.

Access to the operations is by paved road. The mine road connects with the Peruvian National Highway 3N at Conococha Lake, and this highway then connects to the Pan American Highway via National Highway 16.

An airport with scheduled regular flights is located at Anta, approximately 20 km north of Huaraz.

The workforce is primarily sourced within Peru, including from the local Ancash region.

Goods and supplies are primarily sourced from Lima, and trucked to the site.

The Antamina Operations are located in a high-alpine climate. Operations are conducted year-round. Some short-term interruptions may occur because of localized intense thunderstorms.

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

Mineral tenure consists of 170 mining concessions, covering a total area of approximately 111,757.08 ha. Minera Antamina holds the following additional concessions in support of operations:

- Huincush beneficiation concession: granted in 1999; covers an area of 1,060 ha; annual payment of S/355,298.55 in 2024, which is tied to the production rate;
- Pipeline concession for concentrate transport from operations to the port: granted in 1999; covers an area of 181.2 ha; annual payment of S/46,931.07 in 2024, which is tied to the approximately 304 km pipeline length.

Minera Antamina holds approximately 15,650.96 ha of surface rights under 19 titles. The surface rights are sufficient to support the life-of-mine (LOM) plan.

Minera Antamina holds a licence to use surface water and another licence that allows use of groundwater. The company also holds two discharge authorizations to the Ayash and Antamina rivers. Minera Antamina was granted two authorizations for the reuse of treated wastewater in Huarmey, the coastal town that is the location of the Punta Lobitos port facility.

There are no non-government royalties that must be paid on the concessions that host the Mineral Reserve or Mineral Resource estimates.

The Peruvian Government levies a mining royalty charge that is based on operating income margins with graduated rates ranging from 1–12% of operating profits; the minimum royalty charge is equivalent to 1% of net sales. If the operating income margin is ≤10%, the royalty charge is 1%. For each 5% increment in the operating income margin, the royalty charge rate increases by 0.75%, to a maximum of 12%. A Special Mining Tax is levied by the Peruvian Government, based on operating income. Rates range from 2–8.4%. If the operating income margin is ≤10%, the Special Mining Tax is 2%. For each 5% increment in the operating income margin, the special mining rate increases by 0.4%, to a maximum of 8.4%.

Under a long-term streaming agreement with FN Holdings ULC (FN Holdings), a subsidiary of Franco-Nevada Corporation, Teck has agreed to deliver silver to FN Holdings equivalent to 22.5% of the payable silver sold by Minera Antamina. FN Holdings made a payment of US$610 M on closing of the arrangement in 2015 and pays 5% of the spot price at the time of delivery for each ounce of silver delivered under the agreement. After 86 Moz of silver have been delivered under the agreement, the stream will be reduced by one-third. A total of 29.1 Moz of silver has been delivered under the agreement from the agreement effective date in 2015 to December 31, 2024.

1.5 Geology and Mineralization

The Antamina deposit is considered to be an example of a giant skarn deposit.

The mineralization within the Peruvian Andean metallogenic belt is primarily hosted within the Marañón Fold–Thrust Belt, a structurally and stratigraphically complex deformation zone. In the region of the Antamina operations, the Marañón Fold–Thrust Belt is overlain by Palaeozoic and Mesozoic shelf carbonates and clastic sedimentary rocks, including the Permian Mitu Group, the Triassic–Jurassic Pucara Group, and Chicama Formation; and the Cretaceous Oyón Formation, Goyllarisquizga Group, Pariahuanca Formation, Jumasha Formation, and Celendín Formation. Eocene and Miocene intrusions can locally intrude the sedimentary sequence, and the entire sequence can locally be overlain by Miocene volcanic rocks.

The sedimentary rocks have been intruded by a multi-phase porphyry complex, locally referred to as the Antamina Porphyry Complex. The main mineralized area consists of three northeast–southwest-oriented contiguous porphyry centres. Porphyry emplacement was strongly influenced by the pre-existing structural framework, including a series of northwesterly-trending thrust faults intersected by the northeasterly-trending Valley Fault system.

Skarns formed in tandem with each phase of porphyry emplacement, finally coalescing to form a continuous skarn body. Within the skarns there are systematic zoning patterns defined by garnet colour, garnet:clinopyroxene ratio, and garnet and clinopyroxene compositions. The skarns zone outward from the porphyry centre as follows: brown garnet endoskarn, mixed brown and green garnet indeterminate skarn, mixed brown and green garnet exoskarn, green garnet exoskarn, diopside exoskarn, wollastonite exoskarn, hornfels, marble, and limestone.

A zone of brecciation is located on the southeast side of the deposit. Most individual breccia zones are <30 m wide, and can be traced down-dip and along strike for a maximum of 400 m.

The main core of the Antamina deposit is about 2,000 m long and approximately 1,500 m wide. The mineralization extends from the surface at approximately 4,500 m above mean sea level and has been intersected in core drill holes below 2,500 m above mean sea level. The Usupallares zone mineralization extends from surface at 4,200 m above mean sea level and has been intersected in drill holes to approximately 3,000 m above mean sea level. The Usupallares zone is about 800 m wide, and approximately 1,000 m long.

The majority of mineralisation at Antamina formed during retrograde alteration and occurs in endoskarns, exoskarns, and in veins cutting across porphyries. A minor amount of mineralisation is noted in marble and hornfels.

Copper, molybdenum, and zinc sulphides comprise the principal ore minerals; these include chalcopyrite, bornite, molybdenite, and sphalerite. Pyrite and magnetite are common, but of lesser economic importance. Minor minerals include galena and pyrrhotite. Rare occurrences of chalcocite and wittichenite (Cu_3BiS_3)

have been observed. Sulfides occur disseminated interstitial to garnet, as irregular massive sulfide zones, and as veinlets. The mineralization is separated into a number of different geometallurgical types.

1.6 History

Prior to 2013, the following companies had conducted exploration and related activities in the area, either individually or as part of joint ventures: Northern Perú Copper, Cerro of Pasco Corporation, Empresa Minera Especial, Geomin, Empresa Minera del Centro del Perú S.A. (Centromin), Rio Algom Limited (Rio Algorm), Inmet Mining Corporation (Inmet), Noranda Inc. (Noranda), Teck Corporation (now Teck Resources Limited), Mitsubishi Corporation (Mitsubishi), BHP, (Falconbridge), Xstrata plc (Xstrata), and Glencore plc (Glencore). Work completed included geological mapping, rock chip sampling, underground development, core drilling, mining, and technical studies, and metallurgical testwork.

The ownership interests since 2013 are: 33.75% BHP, 33.75% Glencore, 22.50% Teck, and 10% Mitsubishi. The Antamina mine commenced operations in 2001. Mine expansions were conducted in 2004, 2009 and 2014.

1.7 Drilling and Sampling

Drilling to December 31, 2024 in the Project area totals 4,265 drill holes (1,355,884 m). Of this total, 4,111 holes are core holes (1,334,705 m), 65 are channel samples treated as drill holes, and 89 drill holes are reverse circulation (RC) drill holes. Of the total drilling, 3,860 drill holes (1,233,897 m) are used to support the Mineral Resource estimates. The database used for estimation has a close-out date of August 15, 2023.

Core is logged for geological and geotechnical parameters, and photographed. Core recoveries have typically averaged >95% for most programs since 2007. During Minera Antamina campaigns, drill holes were sited using global positioning system (GPS) equipment. Following drill hole completion, the collars were located by mine survey personnel using GPS instruments. Down hole survey data were collected using various methods including acid tube, Maxibor, Sperry Sun, EZ_Shot, Flexit or gyroscopic tools.

On average, the true width of the mineralization is about 70 m. When using traditional drilling, the mineralization drill intercept is typically greater than the true width of the mineralization. When using directional drilling, the drilled and true widths are approximately the same.

Drill data are considered acceptable to support Mineral Resource and Mineral Reserve estimation, and can be used for mine planning. There are no drilling, sampling, or recovery factors that could materially impact the accuracy and reliability of the results.

Pre-2002 RC holes were sampled every 3 m; RC holes drilled during the 2002–2003 campaign were sampled on 1.5 m intervals. Core sample intervals are set at 3 m length on average; however, the sample length may be adjusted to start and end at major geological (lithology, alteration, mineralization) contacts. The whole length of the drill hole is sampled, whether mineralized or not, except where low drilling recovery prevents the collection of a representative sample.

There are 65,238 density determinations in the Project database, providing adequate coverage of mineralized and waste lithologies.

A number of independent sample preparation and analytical laboratories have been used over time, including SGS, Lima, Peru; Chemex, Vancouver, Canada; XRAL, Toronto, Canada; Cone Geochemical Inc., Colorado; ALS Chemex, Lima, Peru; ALS Chemex, Vancouver, Canada; Acme Analytical; Actlabs Skyline Peru S.A.C., Genalysis Laboratory Services Pty Ltd; Ultra Trace Pty Ltd.; Certimin S.A.; and Inspectorate Services Peru S.A.C. Depending on the date the laboratory was used, accreditations could include ISO9001 and ISO17025. The non-independent Minera Antamina site laboratory is used for blasthole sampling.

Sample preparation methods varied over time, and could include: drying; crushing to 75% passing -10 mesh, 90% passing -10 mesh, better than 90% passing -2 mm, or better than 70% passing -2.0 mm; and pulverizing to ~95% passing -150 mesh, better than 95% passing -0.106 mm, or better than 85% passing -75 µm. Analytical methods also varied over time, and could include inductively coupled plasma (ICP) atomic emission spectroscopy (AES) or mass spectroscopy (MS), and atomic absorption spectroscopy (AA).

Quality assurance and quality control (QA/QC) programs from 2003 onward included submission of blank, standard reference materials (standards) and duplicates (core, coarse-reject, and pulp duplicates). Sieve checks were completed during some campaigns, initially on the -150-mesh fraction, and in later campaigns on the crusher and pulveriser products. Check assays at an umpire laboratory were performed from 2004 onward. Protocols are in place should the QA/QC results show out-of-control results. These can range from acceptance of the sample results to re-assay of partial or full batches. If significant contamination is observed, the laboratory is requested to improve its preparation or assaying procedures. No significant analytical biases have been noted in the analytical data collected from 2003 to the Report effective date.

The Antamina drill hole database is managed using acQuire information management software that is based on Microsoft SQL Server, a relational database system. There are processes and procedures for importing, validating, and exporting drill hole data for use in Mineral Resource estimation.

Sample security has relied upon the fact that the samples are always attended to or locked at 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 to make certain that all samples are received by the laboratory.

The sampling, sample preparation, assaying, and QC procedures used by Minera Antamina are acceptable for Mineral Resource and Mineral Reserve estimation and can be used for mine planning purposes.

1.8 Data Verification

Minera Antamina staff routinely undertake data verification on the data uploaded to the database. Data were manually checked for errors and gaps prior to database upload, and where issues arose, these were corrected.

A number of independent third-party consultants have verified the database over time. No material issues were identified during these programs. Where issues were identified, these were corrected.

The QPs work at the mine site, and this familiarity with the day to day operations serves as their personal inspection. The QPs individually reviewed the information in their areas of expertise, and concluded that the information supported Mineral Resource and Mineral Reserve estimation, and could be used in mine planning and in the economic analysis that supports the Mineral Reserve estimates.

1.9 Metallurgical Testwork

A number of metallurgical testwork facilities and metallurgical experts have been involved with metallurgical testwork and interpretation of that testwork over the Project history, including Lakefield Research Limited, G and T Metallurgical Services Ltd., A.R. MacPherson Consultants Ltd., AMTEL, SGS Canada Inc., Larox, Inc., Jenike and Johanson Ltd., Fuller Company, and Pocock Industrial Inc. Testwork included comminution, mineralogical, pilot and bench scale flotation, dewatering, filtration, and coarse ore and concentrate flow characteristics tests. This testwork was used to support flowsheet design and process plant construction and operation. The plant has been operational since 2001.

During 2015, two metallurgical programs were completed on material at depth within the proposed open pit outline, consisting of 67 copper–molybdenum and 54 copper–zinc samples. Tests completed included comminution and flotation. A variability program was undertaken in 2023, consisting of 34 variability samples and four composites. Samples were subject to comminution, rougher and cleaner flotation testwork.

The correlations between head grade and recovery are well understood by process plant staff. Using predictive recovery equations, annual metal production targets can be calculated from the production plans. The copper and zinc metal production represents 90% of total revenues and these show very good correlation between actual versus predicted recoveries, based on geometallurgical type.

Depending on the geometallurgical type, recovered copper to copper concentrate ranges from 73.72–93.40%, recovered zinc to zinc concentrate ranges from 72.48–83.70%, recovered silver to copper concentrate ranges from 41.06–84.50%, and recovered molybdenum to molybdenum concentrate ranges from 26.12–57.62%.

Samples selected for testing were representative of the various types and styles of mineralization. Samples were selected from a range of depths within the deposit. Sufficient samples were taken so that tests were performed on sufficient sample mass, including individual tests to assess variability.

The major deleterious elements are bismuth and arsenic. These are managed using campaigning.

Geometallurgical types M1 and M4B are campaign-treated to produce high value, high quality copper and zinc concentrates. Geometallurgical types M2 and M2A are campaign-treated to produce separate copper concentrates with higher bismuth and arsenic grades. While there are processes within the industry that can be used to reduce bismuth and arsenic grades in concentrates, installation of these at the Antamina process plant would not be cost effective, due to the relatively small amount of copper concentrates with elevated bismuth and arsenic contents. Geometallurgical types M5 and M6 are always processed separately from the M1 and M4B geometallurgical types to minimize contamination of the higher-grade copper concentrates with the bismuth and arsenic contents associated with the bornite ore.

Concentrates are produced and transported in separate batches to the port, where port storage is designed to keep concentrate batches separate. The concentrate storage shed is divided into 18 different areas; 14 are reserved for copper concentrate, one for bornite concentrate, and three for zinc concentrate. Concentrate blending can be completed at the port by mixing concentrate qualities to meet concentrate specifications as required.

1.10 Mineral Resource Estimation

The database supporting the Mineral Resource estimates was closed as at August 15, 2023. Software packages used in the estimation workflow include Leapfrog and Leapfrog Edge, Isatis Neo, and Supervisor.

For the Mineral Resources potentially amenable to open pit mining methods, grades were initially estimated into 20 x 20 x 15 m blocks, and then reblocked into 5 x 5 x 7.5 m sub-blocks that represent a volume of 187.5 m^3, or 3.125% of the volume of the 20 x 20 x 15 m blocks. There is a total of 32 of these sub-blocks, which are used for improving volumetrics, contained in each 20 x 20 x 15 m block. Grade estimation for all elements is referred to the centroid of the 20 x 20 x 15 m blocks. For the Mineral Resources potentially amenable to underground mining methods, the 20 x 20 x 15 m blocks were re-blocked to 10 m wide x 10 m long x 15 m high.

A total of 21 lithology and breccia wireframes were constructed. Two pre-mineralization faults have been interpreted and modeled. For some elements (zinc, arsenic, and bismuth) additional control is provided through the use of grade shells. For soluble copper and zinc, mineralogical zones were modelled and used to control estimation.

Exploratory data analysis was used to guide the zone definition, domain selection, and definition of interpolation parameters.

Data are composited to 3 m lengths, and were broken at geological contacts. Outliers were managed using a combination of volumetric restriction and grade capping.

Grade estimation was systematically performed using four ordinary kriging (OK) passes followed by a fifth-pass assignment of the average of blocks estimated in pass 4. A one-pass nearest neighbour (NN) estimation was completed for validation purposes.

Grade estimation was controlled by the following criteria:

- Lithological unit (domain estimations; all models);
- Grade shells (Zn, Bi and As);
- Distance to the contact mineralization/waste (density);
- Distance control to contact (Mo);

- Mineralogical zones (CuAC, ZnAC, and CuCN).

Bulk-density estimation was performed using four OK passes followed by a fifth pass assignment of the average of blocks estimated in pass 4. A one-pass NN estimate was completed for validation purposes.

Validation was performed to confirm the modeling and/or estimation adequacy and test for biases, and included: visual validation in plan and section; swath plots; and scatterplots. No significant biases were noted from the validation steps.

Mineral Resource confidence classification was based on a combination of geological groups, the presence of breccia and drill spacing.

Reasonable prospects of eventual economic extraction were assumed based on underground and open pit mining methods. Blocks considered potentially amenable to open pit mining methods were confined within a conceptual pit shell. The property boundaries are used as a hard constraint to define the Mineral-Resource conceptual pit. Blocks considered potentially amenable to underground mining methods were confined within potentially mineable shapes.

Blocks with a positive net value are considered to have reasonable prospects assuming open pit mining methods. Block value is calculated using the following expression:

- Block value per block (BVO) = revenues - operating cost.

Given the variability of the rock hardness, the mill rate is also incorporated into the block value by multiplying the profit of each block ($/t) to the mill throughput (t/h) for the corresponding ore type:

- Mineralization value per hour (OVPHR) = (profit x daily mill throughput) / (24 x 1000).

For the Mineral Resources potentially amenable to open pit mining, Minera Antamina uses a cut-off based on dollar-per-mill hours (OVPHR), which is set at >US$0/h.

Blocks above an NSR of US$53.8/t are considered to have reasonable prospects assuming an underground selective mining method.

1.11 Mineral Resource Statement

Mineral Resources are reported in situ, using the 2014 CIM Definition standards. Mineral Resources are reported exclusive of those Mineral Resources converted to Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Mineral Resources are reported on a 100% basis. There are four shareholders with the following ownership percentages: BHP plc, 33.75%; Glencore plc, 33.75%; Teck, 22.50%; and Mitsubishi Corporation, 10.00%.

The Qualified Person for the estimate is Mr. Lucio Canchis, RM SME, a Minera Antamina employee.

The Mineral Resources considered potentially amenable to underground mining are reported exclusive of the estimated Mineral Resources potentially amenable to open pit mining.

Mineral Resources considered potentially amenable to open pit mining methods are reported in Table 1-1 and have an effective date of 31 December, 2024.

Mineral Resources considered potentially amenable to underground mining methods are reported in Table 1-2 and have an effective date of 31 December, 2024.

Factors that may affect the estimate include changes to metal price and exchange rate assumptions; changes to the assumptions used to generate the copper equivalent grade cut-off grade and the OVPHR; changes in local interpretations of mineralization geometry and continuity of mineralized zones; changes to geological and mineralization shapes, and geological and grade continuity assumptions; density and domain assignments; changes to geotechnical assumptions including pit slope angles; changes to mining and metallurgical recovery assumptions; changes to the input and design parameter assumptions that pertain to the conceptual pit constraining the estimates potentially amenable to open pit mining methods; changes to the input and design parameter assumptions that pertain to the conceptual mineable shapes constraining the estimates potentially amenable to underground mining methods; and assumptions as to the continued ability to access the site, retain mineral and surface rights titles, maintain environment and other regulatory permits, and maintain the social license to operate.

1.12 Mineral Reserve Estimation

Mineral Reserves were estimated from Measured and Indicated Mineral Resources, assuming open pit mining methods. Inferred Mineral Resources within the mine plan were set to waste.

Pit optimization work was conducted using a Lerchs–Grossmann algorithm in Whittle software. The pit shell at a revenue factor of 0.85 was selected as the reference pit shell. There is approximately 193 Mt of mineralization classified as Measured and Indicated Mineral Resources that is contained between the Mineral Reserve pit design and the Mineral Resource pit shell that was not converted to Mineral Reserves.

Pit optimization included consideration of the TSF capacity, process plant campaigning requirements, and metallurgical recoveries.

An operational dilution and ore loss algorithm was applied to the resource block model reducing the copper content by 1.01% and zinc content by 3.57% within the Mineral Reserves pit.

The high-grade stockpiles are classified as Proven Mineral Reserves, and converted from Measured Mineral Resources.

Given the polymetallic nature of the deposit and the variability in ore comminution characteristics between metallurgical types, Minera Antamina uses a ranking that incorporates the block value and time to process ore, such that the mill rate is also incorporated into the block value by multiplying the profit of each block (US$/t) with the effective mill throughput (t/h) and the mill efficiency (%). An OVPHR field in the block model is used for scheduling as it incorporates all of the variables used to generate an optimized mill feed program. The minimum cut-off OVPHR value is US$6,000/h, which ensures that stockpiling and rehandle costs can be met. The value includes provision for extra loading and hauling from stockpiles to the crusher.

Table 1-1: Mineral Resources Potentially Amenable to Open Pit Methods

Confidence Category	Geometallurgical Type	Tonnage (Mt)	Cu (%)	Zn (%)	Ag (g/t)	Mo (%)
Measured	Cu	86	0.66	0.09	6.5	0.014
	Cu–Zn	18	0.46	1.15	25.9	0.008
	Sub-total	*104*	*0.62*	*0.28*	*9.9*	*0.013*
Indicated	Cu	150	0.78	0.13	8.6	0.021
	Cu–Zn	59	0.98	1.66	17.5	0.006
	Sub-total	*209*	*0.84*	*0.56*	*11.1*	*0.017*
Total Measured + Indicated Mineral Resources		***313***	***0.77***	***0.47***	***10.7***	***0.015***
Inferred	Cu	587	0.88	0.14	8.3	0.024
	Cu–Zn	197	1.03	1.62	15.6	0.008
Total Inferred Mineral Resources		***784***	***0.92***	***0.51***	***10.2***	***0.020***

Notes to accompany the estimate of Mineral Resources potentially amenable to open pit mining methods:

1. Mineral Resources are reported in situ, using the 2014 CIM Definition Standards, and have an effective date of 31 December, 2024. The Qualified Person for the estimate is Mr. Lucio Canchis, RM SME, a Minera Antamina employee.
2. Mineral Resources are reported on a 100% basis. There are four shareholders with the following ownership percentages: BHP Group Ltd., 33.75%; Glencore plc, 33.75%; Teck Resources Limited (through its subsidiary Teck Base Metals Ltd.), 22.50%; and Mitsubishi Corporation,10.00%.
3. Mineral Resources are reported exclusive of those Mineral Resources converted to Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
4. Mineral Resources potentially amenable to open pit mining methods are confined within a conceptual pit shell that uses the following input parameters: commodity prices of US$3.50/lb Cu, US$1.25/lb Zn, US$24.63/oz Ag, and US$13.30/lb Mo; variable metallurgical recoveries on a block-by-block basis; the average metallurgical recoveries for the copper geometallurgical mineralization type are 92% Cu, 0% Zn, 79% Ag, and 46% Mo, whereas for the copper–zinc geometallurgical mineralization type, metallurgical recoveries are 83% Cu, 84% Zn, 60% Ag, and 0% Mo; average mining cost of US$3.97/t mined, average processing cost of US$11.11/t processed, and general and administrative cost of US$2.77/t processed; depending on concentrate type: treatment charges that range US$73–186/dmt concentrate, refining charges that range US$0.073–1.10/lb payable metal, and freight costs that range US$34–79/wmt concentrate; pit slope angles that average 38–55°. The project boundaries are used as a hard constraint to define the limits of the conceptual pit shell. Lead and zinc are not recovered from the copper geometallurgical mineralization type, and molybdenum is not recovered from the copper–zinc geometallurgical mineralization type.
5. Mineral Resources are reported above a zero dollar-per-mill hours (OVPHR) cut-off, using the following equations. Mineralization value per block (BVO) = (revenues - operating costs) where mineralization value per block: expected profit on a block in US$/t; revenues or net smelter return (NSR): profits from the payable metals in the concentrate minus the realization cost (penalties, treatment charge (TC), refining charge (RC), and freight costs); operating costs: mining and processing cost (fixed and variable), including sustaining capital. The concentrate transport costs to the port are also included. Given the variability of the rock hardness, the mill rate is also incorporated to the block value by multiplying the profit of each block (US$/t) with the effective mill throughput (t/h) and the mill efficiency (%): mineralization value per hour (OVPHR) = (profit x effective mill throughput x mill efficiency)/(1,000), where mineral value per hour: expected value in US$'000/mill-hour or k$/h; profit: positive BVO; effective mill throughput: long-term average processing rate by geometallurgical type in t/h; mill efficiency: long-term average processing performance in %; conversion factor: division by 1,000 (to convert to thousand dollar units).
6. Numbers have been rounded and may not sum.

Table 1-2: Mineral Resources Potentially Amenable to Underground Methods

Confidence Category	Geometallurgical Type	Tonnage (Mt)	Cu (%)	Zn (%)	Ag (g/t)	Mo (%)
Inferred	Cu	282	1.23	0.20	10.8	0.017
	Cu–Zn	151	1.11	1.50	15.5	0.006
Total Inferred Mineral Resources		***433***	***1.19***	***0.65***	***12.5***	***0.013***

Notes to accompany the estimate of Mineral Resources potentially amenable to underground mining methods

1. Mineral Resources are reported in situ, using the 2014 CIM Definition Standards, and have an effective date of 31 December, 2024. The Qualified Person for the estimate is Mr. Lucio Canchis, RM SME, a Minera Antamina employee.
2. Mineral Resources are reported on a 100% basis. There are four shareholders with the following ownership percentages: BHP Group Ltd., 33.75%; Glencore plc, 33.75%; Teck Resources Limited (through its subsidiary Teck Base Metals Ltd.), 22.50%; and Mitsubishi Corporation,10.00%.
3. Mineral Resources potentially amenable to underground mining methods are confined within conceptual mineable shapes assuming a sublevel-stoping scenario that use the following input parameters: commodity prices of US$3.50/lb Cu, US$1.25/lb Zn, US$24.63/oz Ag, and US$13.30/lb Mo; variable metallurgical recoveries on a block-by-block basis; the average metallurgical recoveries for the copper geometallurgical mineralization type are 92% Cu, 0% Zn, 79% Ag, and 46% Mo, whereas the metallurgical recoveries for the copper–zinc geometallurgical mineralization type are 83% Cu, 84% Zn, 60% Ag, and 0% Mo; assumptions of a 100 m thick crown pillar under the Mineral Resource pit shell; below-crown-pillar cost assumptions of: mining cost of US$30.90/t mined, capital costs of US$8.00/t mined, processing costs of US$10.70/t mined, and general and administrative costs of US$4.20/t mined; within-crown-pillar cost assumptions of: mining costs of US$59.20/t mined, capital costs of US$8.00/t mined, processing costs of US$10.70/t mined, and general and administrative costs of US$4.20/t mined; depending on concentrate type: treatment charges that range US$73–186/dmt concentrate, refining charges that range US$0.073–1.10/lb payable metal, and freight costs that range US$34–79/wmt concentrate; maximum internal dilution of 50% and external dilution of 10%. The property boundaries are used as a hard constraint to define the limits of the conceptual mineable shapes. Zinc is not recovered from the copper geometallurgical mineralization type, and molybdenum is not recovered from the copper–zinc geometallurgical mineralization type.
4. Mineral Resources are reported above a US$53.80/t net smelter return (NSR) cut-off; Mineral Resources within the crown pillar are reported above a US$82.20/t NSR cut-off.
5. Numbers have been rounded and may not sum.

1.13 Mineral Reserves Statement

Mineral Reserves are reported at the point of delivery to the process plant, using the 2014 CIM Definition Standards.

The Qualified Person for the estimate is Mr. Fernando Angeles, P.Eng., a Minera Antamina employee.

Mineral Reserves are reported on a 100% basis. There are four shareholders with the following ownership percentages: BHP plc, 33.75%; Glencore, 33.75%; Teck, 22.50%; and Mitsubishi, 10.00%.

Mineral Reserves are reported in Table 1-3, and have an effective date of 31 December, 2024.

Table 1-3: Mineral Reserves Statement

Category	Geometallurgical Type	Tonnage (Mt)	Cu (%)	Zn (%)	Ag (g/t)	Mo (%)
Proven	Cu	198	0.82	0.12	8.1	0.029
	Cu–Zn	50	1.02	1.88	18.1	0.011
	Sub-total	*247*	*0.86*	*0.47*	*10.1*	*0.025*
Probable	Cu	190	0.91	0.15	9.4	0.030
	Cu–Zn	113	1.07	1.99	19.2	0.008
	Sub-total	*302*	*0.97*	*0.84*	*13.0*	*0.022*
Total Proven + Probable Reserves		***550***	***0.92***	***0.67***	***11.7***	***0.023***

Note to accompany estimate of Mineral Reserves Table

1. Mineral Reserves are reported at the point of delivery to the process plant, using the 2014 CIM Definition Standards, and have an effective date of 31 December, 2024. The Qualified Person for the estimate is Mr. Fernando Angeles, P.Eng., a Minera Antamina employee.
2. Mineral Reserves are reported on a 100% basis. There are four shareholders with the following ownership percentages: BHP Group Ltd., 33.75%; Glencore plc, 33.75%; Teck Resources Limited (through its subsidiary Teck Base Metals Ltd.), 22.50%; and Mitsubishi Corporation,10.00%.
3. Mineral Reserves are confined within an operational pit design that uses the following input parameters: commodity prices of US$3.54/lb Cu, US$1.15/lb Zn, US$21.46/oz Ag, and US$11.10/lb Mo; variable metallurgical recoveries on a block-by-block basis; average metallurgical recoveries for the Cu geometallurgical type of 92% Cu, 79% Ag, and 46% Mo, with zinc not recoverable; average Cu–Zn geometallurgical type recoveries of 83% Cu, 84% Zn, and 60% Ag, with molybdenum not recoverable; average mining cost of US$4.30/t mined, average processing cost of US$9.31/t processed, and general and administrative cost of US$2.95/t processed; depending on concentrate type; treatment charges that range from US$73–186/dmt concentrate, refining charges that range from US$0.073–1.10/lb payable metal, and freight costs that range from US$34–79/wmt concentrate; and pit slope angles that average 33–55°. The remaining tailings capacity is used as a hard constraint to define the limits of the reference pit shell, which is the guide for the operational pit design. Mineral Reserves are reported inclusive of dilution and mining recovery. The dilution and ore loss algorithm applied reduced the copper content by 1.01% and zinc content by 3.57%.
4. Mineral Reserves are reported above a US$6,000 dollar-per-mill hours (OVPHR) cut-off, using the following equations. Ore value per block (BVO) = (revenues - operating costs) where ore value per block: expected profit on a block in US$/t; revenues or net smelter return (NSR): profits from the payable metals in the concentrate minus the realization cost (penalties, treatment charge (TC), refining charge (RC), and freight costs); operating costs: mining and processing cost (fixed and variable), including sustaining capital. The concentrate transport costs to the port are also included. Given the variability of the rock hardness, the mill rate is also incorporated to the block value by multiplying the profit of each block (US$/t) with the effective mill throughput (t/h) and the mill efficiency (%): ore value per hour (OVPHR) = (profit x effective mill throughput x mill efficiency)/(1,000), where mineral value per hour: expected value in US$'000/mill-hour or k$/h; profit: positive BVO; effective mill throughput: long-term average processing rate by geometallurgical type in t/h; mill efficiency: long-term average processing performance in %; conversion factor: division by 1,000 (to convert to thousand dollar units).
5. Numbers have been rounded and may not sum.

Factors that may affect the estimate include: Metal price and exchange rate assumptions; changes to the assumptions used to generate the OVPHR cut-off; changes in local interpretations of mineralization geometry and continuity of mineralized zones; changes to geological and mineralization shapes, and geological and grade continuity assumptions; density and domain assignments; changes to geotechnical assumptions including pit slope angles; changes to hydrological and hydrogeological assumptions; changes to mining and metallurgical recovery assumptions; changes to the input and design parameter assumptions that pertain to the open pit shell constraining the estimates; and assumptions as to the continued ability to access the site, retain mineral and surface rights titles, maintain environment and other regulatory permits, and maintain the social license to operate, and the remaining TSF storage capacity at an elevation of 4,195 masl.

1.14 Mining Methods

The mining operations use conventional open pit mining methods and conventional equipment.

Pit slopes were updated by the engineer of record, Piteau Associates Engineering Ltd., in 2023. The updated study involved assessing the structural fabric data (mapping, photogrammetric, and televiewer), and determining appropriate slope design parameters considering the real bench performance achieved. The geotechnical design is based on the following five models: geological, major structural, structural fabric, rock mass, and hydrogeological. Rock type groups were combined into six rock mass units, to simplify geological and geotechnical interpretation and modelling.

A numerical groundwater flow model is in place, provides data to support geotechnical evaluations, and is used to design future well locations and pumping rates for a given set of operational constraints.

The reference pit shell selected from the pit optimization process was used as a guide to produce the final pit design by integrating operational design characteristics including geotechnical parameters, ramp locations and gradients, overburden storage facilities, mining widths, bench and inter-ramp heights, and other practical mining considerations given the pit geometry. The pit design includes four mining phases:

- Phases 8, 9, and 10 are currently in production;
- Phase 11 waste stripping is planned to start in 2025.

Mining phases are extracted using 15 m high benches. Depending on geotechnical recommendations, single or double benches can be used. Maximum mining and movement rates are approximately 249 and 297 Mt/a, respectively. The maximum processing rate is approximately 53.6 Mt/a.

The remaining mine life is 12 years, from 2025 to 2036, with 2036 being a partial year. About 546 Mt of ore and 1,535 Mt of waste are planned to be mined from the open pit with an average stripping ratio of 2.79. Phase 10 has the highest stripping ratio of the remaining pit phases at 3.1. The total open pit production forecast is approximately 2,080 Mt. The total material movement will be approximately 2,637 Mt, including production and rehandling. On average, chalcopyrite copper-bearing ores are the most common (66.5%) in the LOM plan, followed by chalcopyrite copper–zinc-bearing ores (24.7%). The balance (8.8%) corresponds to bornite-rich copper- and copper–zinc-bearing ores.

Two types of stockpiles are in place: high-grade and low-grade. The high-grade stockpile is used for blending purposes and to provide short-term management for mining of geometallurgical types. Low-grade stockpiling is used to store material that has been mined using variable cut-off grades, and to provide sufficient storage space for mineralization that is not included in the current LOM plan.

Three waste rock storage facilities (WRSFs) are required for the LOM plan; two are in operation and one, an extension to an existing WRSF, is to be constructed. The WRSF permitted capacities are sufficient for the remaining LOM plan. A geotechnical monitoring system is in place.

The equipment fleet is suitable for the LOM plan, and equipment numbers will peak in 2029. The main and ancillary mining equipment is owned and maintained by Minera Antamina. Buffer and trim drilling for wall control and blasting services are contracted out.

1.15 Recovery Methods

The concentrator was initially designed to process an average of 70,000 t/d of ore at an assumed 90% total plant availability. Capital investment in 2008 in the pebble crushing plant, and in 2012 for grinding capacity increases, and optimizations of the unit processes, resulted in a substantial increase in the average processing rate to approximately 145,000 t/d by the end of 2020.

The ore is amenable to traditional differential flotation techniques using standard reagent schemes. The plant feed is separated into distinct geometallurgical types that are campaigned through the plant using different plant configurations to optimize the recovery of payable metals and provide the best opportunity to manage product quality.

Ore is relatively coarse grained. Satisfactory metallurgical results are obtained at a P80 secondary grind size of 220–245 µm for copper-only geometallurgical types (M1, M2 and M2A), at a P80 secondary grind size of 210–240 µm for copper–zinc geometallurgical types (M4B) and at a P80 secondary grind size of 180–220 µm for the bornite geometallurgical types (M5 and M6).

The different geometallurgical types have distinct grinding characteristics, including the total grinding power required and the relative power requirement split between primary semi-autogenous grinding (SAG) and secondary ball milling. Operating flexibility and control are critical for the grinding circuits, and flexibility was increased in 2008 by the addition of a pebble crushing circuit.

For the copper–high bismuth geometallurgical type (M4B), bismuth minerals can be differentially floated from the bulk copper concentrate using standard differential flotation techniques resulting in a lead–bismuth(–silver) concentrate, although this results in some copper and silver recovery losses.

For the bornite geometallurgical types (M5 and M6) the primary bismuth minerals are witichenite (Cu_3BiS_3) and emplectite ($CuBiS_2$), which are present as very fine inclusions in the bornite crystals. Bismuth cannot be removed by differential flotation and these copper concentrates have elevated bismuth content.

Silver reports to both the final copper concentrate and the lead–bismuth concentrate.

A concentrate pipeline transports the final copper and zinc concentrates from the mine site to the port at Punta Lobitos in the department of Huarmey. A single pipeline is used to transport the concentrates (depending on the geometallurgical type being campaigned) on a batch basis, using a water plug between concentrate geometallurgical type campaigns.

The molybdenum and lead–bismuth concentrates are dewatered at the mine site, packaged in bags, and transported to various ports or clients using highway trucks.

The main reagents used in the concentrator include lime, frother, sodium hydrosulphide, sodium cyanide, zinc sulphate, copper sulphate, fuel oil, flocculant, potassium amyl xanthate (PAX), sodium isopropyl xanthate (SIXP), sodium hydroxide, activated carbon, carbon dioxide, and grinding media.

The main freshwater storage facility is the Nescafé impoundment. Water from the process plant is recirculated from the tailings storage facility (TSF) and fresh water is needed only where clean water is required.

The SAG mills, ball mills and regrind mills are the main power consumers with a combined installed power of around 90,000 kW. The combined installed power for the primary crusher, pebble crushers, flotation cells, and concentrate pumping is approximately 10,200 kW.

1.16 Project Infrastructure

Existing infrastructure includes:

- Open pit;
- Access, shared-use, and haul roads;
- Truck shop, spare-parts warehouse, washing bays, light vehicles shop;
- Diesel fuel storage points;
- Warehouse and lay down yards;
- Explosives storage facility;
- Core storage facility;
- Administration offices, gatehouse;
- Ore crusher, transfer towers, conveyors;
- Process plant and laboratory;
- High-grade and low-grade stockpiles;
- Two WRSFs;
- TSF;
- Concentrate pipeline;

- Port facilities;
- Power line;
- Nescafé impoundment (water storage facility);
- Water storage and settling ponds;
- Water diversion facilities;
- Accommodations camp.

Additional infrastructure that will need to be constructed for the remaining LOM includes:

- Realigned access and shared-use roads;
- Additional haul roads;
- Replacement ore crusher;
- One WRSF extension;
- Two TSF raises.

A new ore crusher is required in 2025 to support the last pit phases, as the current crusher location will be mined out by pit phase 9c. The environmental permitting process for this crusher was completed in 2021. The crusher was in the final construction stage at the Report effective date.

Tailings discharge commenced in 2001 with the completion of the starter dam. Five dam raises have been completed, and at the Report effective date, a sixth was underway. There are two remaining raises to take the TSF to the environmentally-permitted maximum elevation of 4,195 masl. The final design has a permitted capacity of about 1,527 Mt (844 Mm^3), an approximate final footprint of 905 ha, and a maximum height of 310 m. The permitted capacity of the TSF is sufficient for the LOM plan. A geotechnical monitoring system is in place.

Concentrate, at a slurry density of 60–65% solids, is pumped 304 km from the mine to the port at Punta Lobitos. Copper and zinc concentrates are dewatered to final shipping moisture content using large pressure filters. The filtered concentrate is stored in separate piles within the storage shed depending on type. This also allows for blending when required.

The Nescafé impoundment, situated at the southeast end of the Yanacancha WRSF, is the main source of freshwater. The impoundment has an approximate 2.86 Mm^3 storage capacity, which is sufficient for operational and domestic needs. The water demand is estimated at about 2.57 Mm^3 per year, which is within the limits of the currently-approved surface water use license.

The current accommodations camp has a 12,930 bed capacity.

Minera Antamina has an energy supply contract with Engie Energía Perú for 170 MW from the Peruvian Electric System to support mine and port operations. The current average electricity demand is 120 MW, with a maximum demand of 147 MW. Power is supplied is from the National Interconnected Electric System

(SEIN) via two 50 km long 220 kV transmission lines, from the Vizcarra substation to the Antamina substation. Once at the Antamina Operations substation, power is transformed to medium voltage (23 kV) to supply power to the process plant, concentrate pipeline system, shovels and drills, the pit dewatering system, truck and welding shop, water management facilities, and auxiliary facilities. A set of diesel generators onsite provide backup power for the essential and critical load facilities. In the port area, the power supply is sourced from SEIN via a 66 kV transmission line from the 09 of October substation to the PPL substation. The average port power demand is 2.6 MW.

1.17 Markets and Contracts

Four concentrate types are produced: copper concentrate; zinc concentrate; molybdenum concentrate; and lead–bismuth(–silver) concentrate. Minera Antamina sells all products directly to its shareholders; therefore, no market studies have been conducted by Minera Antamina. Copper, zinc, and molybdenum production is allocated to each of the shareholders based on their attributable interest in Minera Antamina.

Short-term and medium- to long-term forecasts from a number of sources were reviewed by Minera Antamina staff to generate consensus commodity pricing for use in the Mineral Resource and Mineral Reserves estimates, and in the economic analysis that supports the Mineral Reserves. Mineral Resource estimates used a single long-term price forecast for each commodity.

The copper, zinc, and molybdenum concentrates were sold under three off-take sales agreements with the Minera Antamina shareholders as at the Report effective date. Terms and conditions under the agreements are equally applicable to each of the Minera Antamina shareholders. The lead–bismuth concentrate is currently directly sold by Minera Antamina either through tender processes, or to local companies in Peru.

1.18 Environmental, Permitting and Social Considerations

1.18.1 ENVIRONMENTAL CONSIDERATIONS

The original environmental impact assessment (EIA) was reviewed and approved by the relevant Peruvian regulatory bodies and was also presented in a public stakeholder forum in 1998. Between 1999 and 2006, Minera Antamina submitted six amendments to the original EIA. In 2007, the original EIA was updated, public forums conducted, and was approved in 2008. Four amendments were approved from 2008–2011, and six minor amendments were approved in 2011.

Baseline and supporting studies have included:

- Soil quality;
- Surface water quality;
- Air quality;
- Biodiversity;
- Project social and economic influence.

Monitoring programs are in place, and focus on waste rock storage, tailings, water, soils, and air quality.

1.18.2 CLOSURE CONSIDERATIONS

Minera Antamina has complied with regulatory requirements for a closure plan for each facility within the mining operation. An update of the regulatory closure plan must be submitted in February 2025 as a result of the approval of the Modification of the Environmental Impact Assessment (MEIA) in 2024.

The progressive, final, and post-closure costs are estimated to be US$831 M at 31 December, 2024.

1.18.3 PERMITTING CONSIDERATIONS

Minera Antamina holds more than 450 licenses and permits in relation to its mining activities. The licenses and permits are managed using a computerized management system with alerts on renewals and expiry dates. All permits were in good standing as at the Report effective date. Minera Antamina holds all of the key permits required to support the LOM plan.

1.18.4 SOCIAL CONSIDERATIONS

Minera Antamina's social management is governed under the multi-stakeholder model. Minera Antamina has implemented several measures to address potential social impacts in the area of social influence generated by mining activities and to strengthen the bonds of trust between the population and the company. These are detailed in a social and community agreement plan established under the Community Relations Plan. The Community Relations Plan consists of several programs that have objectives, guidelines, and activities that focus on the company's social commitments and obligations until mine closure in 2029.

1.19 Capital Cost Estimates

Capital cost forecasts are based on a 12-year mine life, from 2025 to 2036, with the last year of operations being a partial year.

Capital cost estimates are based on pre-feasibility- and feasibility-level studies, and incorporate Minera Antamina's operating experience with in-place vendor agreements and contracts. The capital expenditures reflect budgets to support or increase the production, including funding for new facilities, growth infrastructure, equipment additions or replacements, and in-pit drilling.

Capital cost estimates for the LOM are summarized in Table 1-4 and total US$4,022 M.

Table 1-4: Capital Cost Estimate Summary Table (nominal terms)

Area	Item	Units	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	Total
Mine	Sustaining mine equipment	US$M	254	337	271	114	48	7	20	6	9	3	6	—	***1,073***
	Sustaining truckshop	US$M	3	13	7	—	—	—	—	—	—	—	—	—	***23***
	Mine utilities/technology	US$M	25	3	—	—	—	—	—	—	—	—	—	—	***30***
	WRSF extensions	US$M	40	38	42	40	11	—	—	—	—	—	—	—	***172***
Process	Tailings site	US$M	225	279	243	192	158	147	104	122	30	44	56	11	***1,610***
	Sustaining ball/SAG mill maintenance	US$M	24	21	22	49	49	31	—	30	—	18	—	—	***244***
	Sustaining pipeline/ort projects	US$M	17	9	6	4	7	6	3	1	4	6	2	—	***65***
Other		US$M	238	150	75	64	60	29	28	32	28	37	46	20	***807***
Total estimated capital costs		***US$M***	***825***	***850***	***667***	***464***	***333***	***221***	***155***	***190***	***70***	***108***	***109***	***31***	***4,022***

1.20 Operating Cost Estimates

Operating cost forecasts are based on a 12-year mine life, from 2025 to 2036, with the last year of operations being a partial year.

Operating costs are related to the mine, concentrator, maintenance activities, and support services. Cost estimates are based on actual mine operating costs and budgetary estimates. Consumption rates are based on history performance of the equipment and consumable prices (such as dollar per gallon of diesel or dollar per kg explosive) are used to estimate the variable costs.

Operating cost estimates for the LOM are summarized in Table 1-5 and total US$14,681 M. Over the LOM, mine operating costs are estimated to average US$6.70/t mined and process costs are estimated to average US$7.90/t processed.

1.21 Economic Analysis

Teck is using the provision for producing issuers, whereby producing issuers may exclude the information required under Item 22 for technical reports on properties currently in production and where no material production expansion is planned.

Mineral Reserve declaration is supported by overall site positive cash flows and net present value assessments.

1.22 Risks and Opportunities

1.22.1 RISKS

Minera Antamina maintains a risk register using SIGRA software. This is reviewed and updated by senior management on a quarterly basis. Risks have a systematic control process in place, with periodic audits and reviews of the management for each individual risk.

Key risks identified by Minera Antamina at 31 December, 2024, include:

- Accidents during personnel transport, due to such causes as vehicle mechanical failure, interactions with other, non-Minera Antamina, vehicles using the access route, severe weather conditions, or reckless driving;
- Risk of collision due to intensive truck traffic and congestion in the open pit access ramps affecting haul trucks and other vehicles as a result of driver-related issues, weather conditions, poor road maintenance, or mechanical conditions of the equipment;
- Accidents during the transportation of hazardous material to site, causing spills on the public road;
- Social conflict in the Antamina area of influence, for example conflicts that could be caused by breaches of commitments, political and/or social instability;

Table 1-5: Operating Cost Estimate Summary Table (real terms)

Area	Units	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	Total
Mine	US$M	594	621	597	627	698	675	623	585	465	413	371	257	***6,526***
Maintenance	US$M	385	401	370	359	393	359	325	304	269	277	246	136	***3,822***
Concentrator	US$M	220	212	214	221	232	232	214	209	182	217	189	62	***2,403***
Port	US$M	8	8	8	8	8	9	8	9	8	8	7	6	***97***
Support services	US$M	180	166	162	153	160	161	148	145	142	139	140	137	***1,833***
Total estimated operating costs		***1,387***	***1,409***	***1,351***	***1,367***	***1,492***	***1,435***	***1,318***	***1,252***	***1,066***	***1,053***	***953***	***597***	***14,681***

- Risk of rocks falling from slopes and upper banks in the open pit, for example, caused by blasting effects, loose rocks in walls and slopes, plugged benches, or interaction of works in benches and phases;
- Sub-optimal Mineral Resource estimates caused by, for example, limitations in drilling, failures in the sample capture and processing, assumptions in the modelling process, and assumptions as to forecasts used in assessing reasonable prospects of eventual economic extraction;
- Sub-optimal Mineral Reserves estimation, caused by, for example, uncertainty in the metallurgical behaviour of the ore, operational dilution, deficiencies in the construction of the block model, operational assumptions, and assumptions as to forecasts used in mine planning and the economic analysis that supports the Mineral Reserves.

Risks to the production plan that were assigned a “moderate” classification as at 31 December 2024, include:

- Failure to comply with the performance assumed for loading equipment and trucks fleet, which could be caused by, for example, poor haul road maintenance, inefficiencies in the mining cycle, unexpected traffic or equipment congestion, lack of expertise on the part of the equipment operators, and labour conditions;
- Production of out-of-specification concentrate due to the unexpected higher presence of deleterious metals such as bismuth, arsenic, fluorine, and lead. Engineering studies are underway to support construction of a new filtration plant, which would allow the processing of high-lead ores;
- Failure of the grade control drilling programs because of potential interference between drill rigs and active mining.

1.22.2 OPPORTUNITIES

Opportunities include:

- There is upside potential for the Mineral Resource estimates to be converted to Mineral Reserves once the studies, engineering, economics, and permits are completed for the infrastructure required for mining support;
- There are blocks classified as Inferred within the Mineral Reserve pit that may eventually be able to be upgraded to higher-confidence Mineral Resource categories and converted to Mineral Reserves with additional drilling and studies;
- Technological initiatives are under evaluation to potentially increase productivity and metal recovery, with resulting effects on project economics if successful. These initiatives include ore sorting, coarse-particle flotation, and tailings densification.

1.23 Interpretation and Conclusions

Under the assumptions in this Report, the Mineral Reserve declaration is supported by overall site positive cash flows and net present value assessments, which supports Mineral Reserves. The mine plan is achievable under the set of assumptions and parameters used in this Report.

1.24 Recommendations

As Antamina is an operating mine, the QPs have no material recommendations to make.

2 INTRODUCTION

2.1 Introduction

Mr. Lucio Canchis, RM SME, Mr. Fernando Angeles, P.Eng., Mr. Hernando Valdivia, FAusIMM and Mr. Carlos Aguirre, FAusIMM, prepared this technical report (the Report) for Teck Resources Limited (Teck) on the Antamina mining operation (the Antamina Operations or the Project), located in the San Marcos District, Province of Huari, Ancash Department, Peru (Figure 2-1).

The operations are owned by Compañíа Minera Antamina S.A. (Minera Antamina), which is a Peruvian limited liability company (sociedad anonima). There are four shareholders with the following ownership percentages:

- BHP Group Ltd. (BHP): 33.75%;
- Glencore plc. (Glencore): 33.75%;
- Teck (through its subsidiary Teck Base Metals Ltd.): 22.50%;
- Mitsubishi Corp. (Mitsubishi): 10.00%.

Mineral Resources and Mineral Reserves are estimated for the Antamina deposit.

2.2 Terms of Reference

The Report was prepared to support Teck's Annual Information Form for the year-end 31 December 2024.

Mineral Resources and Mineral Reserves are reported using the confidence categories in the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves (May 2014; the 2014 CIM Definition Standards).

Currency is expressed in United States dollars (US$) unless stated otherwise. The Peruvian currency is the Peruvian Nuevo Sol (PEN or S/).

Units presented are metric units, such as metric tonnes, unless otherwise noted.

The Report uses Canadian English.

Note: Figure prepared by Minera Antamina, 2024.

Figure 2-1: Project Location Plan

2.3 Qualified Persons

This Report has been prepared by the following Qualified Persons (QPs):

- Mr. Lucio Canchis, RM SME, Resource Estimation Superintendent, Minera Antamina;
- Mr. Fernando Angeles, P.Eng., Long-term Planning Superintendent; Minera Antamina;
- Mr. Hernando Valdivia, FAusIMM, Superintendent of Metallurgy and Chemical Laboratory, Minera Antamina;
- Mr. Carlos Aguirre, FAusIMM, Geotechnical and Hydrogeological Manager, Business Planning and Development, Minera Antamina.

2.4 Site Visits and Scope of Personal Inspection

2.4.1 MR. LUCIO CANCHIS

Mr. Canchis has been a Minera Antamina employee since 2001. He works at the mine site and has been involved in resource estimation since 2011. While on site he has visited and inspected the open pit workings, verified regulatory permitting, and visited key infrastructure locations.

2.4.2 MR. FERNANDO ANGELES

Mr. Angeles has been a Minera Antamina employee for more than 10 years, initially from 2007–2011 and currently from 2018 to the Report effective date, and works at the mine site. He performs regular inspections of the open pit, waste rock storage facilities (WRSFs), progress of selected mine projects (e.g. crusher removal) and other facilities (e.g. tailings storage facility (TSF)).

2.4.3 MR. HERNANDO VALDIVIA

Mr. Valdivia has been a Minera Antamina employee since 2001, and works at the mine site. While on site, he has visited and inspected the progress and development of each of the mining phases in the pit, verifying performance of the material mined in each phase when processed through the plant, and fine-tuning the mineral flotation responses in the laboratory to ensure compliance with production budgets.

2.4.4 MR. CARLOS AGUIRRE

Mr. Aguirre has been a Minera Antamina employee since 2013, and works at the mine site. While on site he has visited and inspected the mine main geotechnical components including at the open pit, East and Tucush WRSFs, TSF, and other key infrastructure locations such as the water reservoirs (polishing pond and Nescafé impoundment) and the stockpiles. The purpose of the visits is to verify that the facility construction and operations are in compliance with the design intent.

2.5 Effective Dates

There are a number of effective dates pertinent to the Report, as follows:

- Date of closure of database used for resource estimation: 31 August, 2023;
- Date of last drilling information included in the Report: 31 December, 2024;
- Date of Mineral Resource estimate: 31 December, 2024;
- Date of Mineral Reserve estimate: 31 December, 2024;

The overall Report effective date is 31 December 2024, and is based on the effective date of the Mineral Reserve estimates.

2.6 Information Sources and References

The reports and documents listed in Section 27 of this Report were used to support Report preparation.

Additional information was sought from Teck or Minera Antamina personnel where required.

2.7 Previous Technical Reports

Teck has previously filed the following technical report on the Project:

- Lozada, L., and Espinoza, J., 2011: Technical Report, Mineral Reserves And Resources, Antamina Deposit, Perú 2010: report prepared for Compañía Minera Antamina S.A., owned by Teck Resources Limited, BHP Billiton Group Xstrata PLC, Mitsubishi Corporation; effective date 31 January, 2011.

3 RELIANCE ON OTHER EXPERTS

This section is not relevant to this Report.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Introduction

The Antamina operations are situated in the San Marcos District, Province of Huari, Ancash Department, approximately 270 km north of the Peruvian capital, Lima, and 38 km from the closest town, San Marcos.

The Project is situated at approximately 9º32'17" south latitude and 77º 03'51" west longitude. Mining operations are at an altitude of 4,300 m. The main core of the Antamina deposit is located at 273,454 m E and 8,944,940 m N. The Usupallares zone is located at 272,746 m E and 8,943,770 m N.

4.2 Ownership

The operations are owned by Compañía Minera Antamina S.A., which is a Peruvian limited liability company (sociedad anonima). There are four shareholders with the following ownership percentages:

- BHP Group Ltd. (BHP): 33.75%;
- Glencore plc. (Glencore): 33.75%;
- Teck: 22.50%;
- Mitsubishi Corp. (Mitsubishi): 10.00%.

4.3 Agreements

BHP, Glencore, Teck and Mitsubishi, together with certain subsidiaries, and Minera Antamina are parties to a shareholders agreement governing the relationship between the parties.

4.4 Mineral Tenure

Mineral tenure consists of 170 mining concessions, covering a total area of approximately 111,757.08 ha (Table 4-1). Concession locations are shown in Figure 4-1.

All mining concessions are in the name of Minera Antamina. The mining concessions can be held indefinitely, contingent upon the payment of annual license fees and the provision of minimum annual production from each mining concession. Failure to pay the applicable license fees for two consecutive years will result in the cancellation of the mining concession.

Table 4-1: Mineral Tenure Table

Item	Code	Concession Name	Concession Area (ha)
1	09000429Y01	Chapi	5.99
2	010295203	Cordillera Negra 2 EC	200.00
3	010526706	Anta 11	1,000.00
4	010527106	Anta 16	1,000.00
5	010527606	Anta 22	1,000.00
6	010249903	Cordillera Negra EC	1,000.00
7	010486295	Recuay 12	1,000.00
8	010133398	Robyn 1	800.00
9	010133598	Robyn 3	1,000.00
10	010088201	Alicia Dos Mil Uno	300.00
11	010529706	Anta 43	800.00
12	010528106	Anta 27	1,000.00
13	010525806	Anta 4	700.00
14	010526006	Anta Ocho 2007	800.00
15	010526406	Anta Diez 2006	1,000.00
16	010041501	Andrea Natalia	600.00
17	010528606	Anta 32	1,000.00
18	010529006	Anta 36	1,000.00
19	010529206	Anta 39	900.00
20	010155412	Anta 66	900.00
21	010148313	Anta 74	300.00
22	010122811	Anta 53	1,000.00
23	010208009	Anta 46	1,000.00
24	010124909	Shuyhua	400.00
25	010063212	Anta 57	1000.00
26	010063612	Anta 61	100.00
27	010131515	Anta 78	100.00
28	010131915	Anta 82	200.00
29	010040513	Anta 71	200.00
30	010266015	Anta 85	100.00
31	010266215	Anta 87	900.00

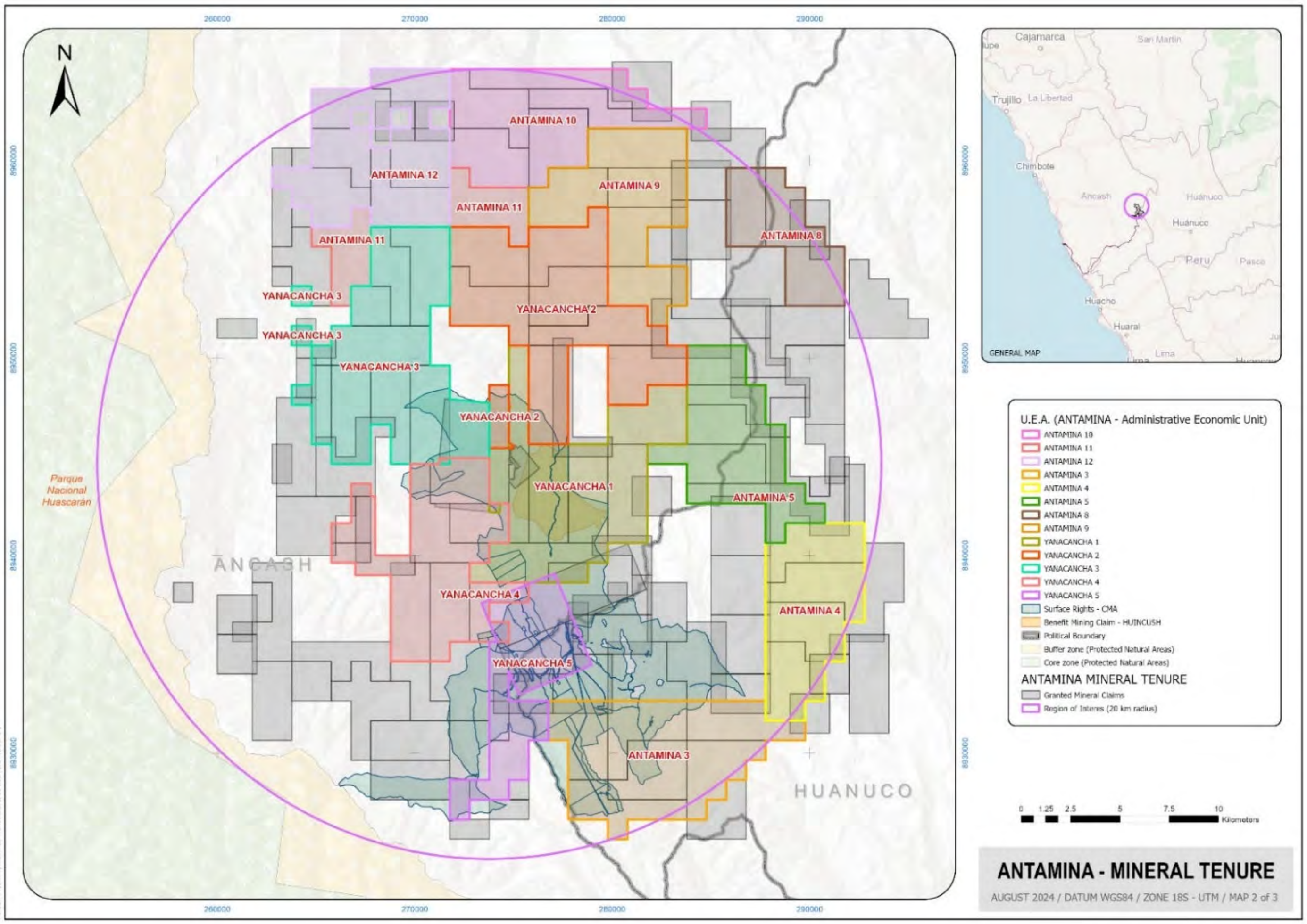
Item	Code	Concession Name	Concession Area (ha)
32	010260817	Anta 101	1,000.00
33	010130716	Anta 91	100.00
34	010260217	Anta 97	600.00
35	010106019	Anta 128	400.00
36	010022819	Anta 126	600.00
37	010107319	Anta 130	300.00
38	010000518LA	Dprincipal Anta 1	325.23
39	010205718	Anta 113	100.00
40	010312018	Anta 117	900.00
41	010081023	Anta 138	100.00
42	010529806	Anta 44	900.00
43	010142804	Estrella 4	1000.00
44	010015402	Eclipse Dosmil Dos	200.00
45	010133498	Robyn 2	1,000.00
46	09014093X01	Don Augusto	1,000.00
47	010523295	Recuay 22	1,000.00
48	010396695	Recuay 13	1,000.00
49	010527206	Anta Dieciocho 2007	1,000.00
50	010527506	Anta 20	1,000.00
51	010527706	Anta 23	1,000.00
52	010525706	Anta 3	900.00
53	010022701	Enrique Dos Mil Uno	200.00
54	010525906	Anta 5	1,000.00
55	010526306	Anta Nueve 2006	1,000.00
56	010526506	Anta 12	1,000.00
57	010528306	Anta 29	1,000.00
58	010526806	Anta 14	1,000.00
59	010528906	Anta 34	1,000.00
60	010528806	Anta 35	900.00
61	010529406	Anta 41	800.00
62	010040413	Anta 73	600.00
63	010040613	Anta 70	200.00
64	010063512	Anta 60	700.00

Item	Code	Concession Name	Concession Area (ha)
65	010203012	Anta 63	100.00
66	010266115	Anta 86	900.00
67	010131815	Anta 81	100.00
68	010170514	Anta 76	100.00
69	010475510	Anta 49	600.00
70	010260017	Anta 95	500.00
71	010260117	Anta 93	1,000.00
72	010260317	Anta 96	600.00
73	010260517	Anta 100	500.00
74	010260717	Anta 102	200.00
75	010130616	Anta 92	100.00
76	010205618	Anta 109	400.00
77	010000518L	Acumulacion Antamina Principal	472.96
78	010000518LC	Dprincipal Anta 3	758.85
79	010220218	Anta 112	500.00
80	010012018	Anta 103	500.00
81	010312118	Anta 116	900.00
82	010312618	Anta 120	400.00
83	010107219	Anta 134	100.00
84	010107419	Anta 131	400.00
85	010158600	Recuay 30-2000	200.00
86	09011850X01	Estrella Del Norte	1,000.00
87	010487195	Recuay 29	1,000.00
88	010522695	Recuay 23	1,000.00
89	010489895	Recuay 20	1,000.00
90	010128695	Recuay 21	600.00
91	010486495	Recuay 11	1,000.00
92	09011851X01	Viejo Bromley	1,000.00
93	010523695	Recuay 24	1,000.00
94	010142704	Estrella 5	1,000.00
95	010526206	Anta Siete 2007	900.00
96	010526606	Anta 13	1,000.00
97	010526906	Anta 15	1,000.00

Item	Code	Concession Name	Concession Area (ha)
98	010527006	Anta Diecisiete 2006	1,000.00
99	010527306	Anta 19	1,000.00
100	010527406	Anta 21	1,000.00
101	010526106	Anta Seis 2007	1,000.00
102	010527806	Anta 24	1,000.00
103	010527906	Anta 26	1,000.00
104	010528006	Anta 25	1,000.00
105	010528706	Anta 33	1,000.00
106	010529606	Anta 42	800.00
107	010529306	Anta 38	600.00
108	010528406	Anta 30	1,000.00
109	010529106	Anta 37	500.00
110	010529506	Anta 40	800.00
111	010529906	Anta 45	700.00
112	010528206	Anta 28	1,000.00
113	010525506	Anta Uno 2007	700.00
114	010525606	Anta 2	1,000.00
115	010528506	Anta 31	1,000.00
116	010217409	Anta 47	700.00
117	010122911	Anta 52	500.00
118	010123011	Anta 51	800.00
119	010122711	Anta 54	400.00
120	010475410	Anta 48	400.00
121	010170212	Anta 65	1,000.00
122	010170112	Anta 69	400.00
123	010202912	Anta 64	200.00
124	010063312	Anta 58	100.00
125	010063012	Anta 62	100.00
126	010063412	Anta 59	100.00
127	010155312	Anta 68	800.00
128	010155512	Anta 67	600.00
129	010063112	Anta 56	400.00
130	010040313	Anta 72	900.00

Item	Code	Concession Name	Concession Area (ha)
131	010257914	Anta 77	100.00
132	010170414	Anta 75	100.00
133	010131715	Anta 80	100.00
134	010132015	Anta 83	400.00
135	010132115	Anta 84	800.00
136	010062912	Anta 55	200.00
137	010131615	Anta 79	100.00
138	010131016	Anta 88	300.00
139	010130816	Anta 90	800.00
140	010130916	Anta 89	800.00
141	010281222	Anta 137	1,000.00
142	010260917	Anta 94	1,000.00
143	010260417	Anta 99	400.00
144	010260617	Anta 98	1,000.00
145	010012118	Anta 104	800.00
146	010205318	Anta 107	600.00
147	010205418	Anta 108	200.00
148	010205218	Anta 106	800.00
149	010312218	Anta 115	900.00
150	010205118	Anta 105	400.00
151	010312418	Anta 119	400.00
152	010312518	Anta 118	600.00
153	010413318	Anta 121	800.00
154	010022919	Anta 125	900.00
155	010106119	Anta 129	900.00
156	010107019	Anta 133	200.00
157	010107119	Anta 127	800.00
158	010000518LB	Dprincipal Anta 2	694.07
159	010205818	Anta 111	400.00
160	010220318	Anta 114	400.00
161	010224520	Anta 135	1,000.00
162	010413018	Anta 123	100.00
163	010286723	Anta 139	400.00

Item	Code	Concession Name	Concession Area (ha)
164	10355195	Huarmey 2	500.00
165	10355395	Huarmey 1	999.99
166	10056600	Huarmey 4	400.00
167	10056700	Huarmey 5	500.00
168	10098000	Huarmey 6	999.99
169	10190200	Huarmey 3-2000	100.00
170	10208798	Vulcano	400.00
Total			***111,757.08***



Note: Figure prepared by Minera Antamina, 2024.

Figure 4-1: Mineral Tenure Location Map

In addition to mining concessions, Minera Antamina holds the following concessions in support of operations:

- Huincush beneficiation concession: granted in 1999; covers an area of 1,060 ha; annual payment of S/355,298.55 in 2024, which is tied to the production rate. Beneficiation concessions follow the same rules as for mining concessions. A fee must be paid that reflects the nominal capacity of the processing plant or production level. Failure to pay the fees or fines for two years would result in the loss of the beneficiation concession;
- Pipeline concession for concentrate transport from operations to the port: granted in 1999; covers an area of 181.2 ha; annual payment of S/46,931.07 in 2024, which is tied to the approximately 304 km pipeline length.

All required annual payments for these two concessions had been made at the Report effective date.

4.5 Surface Rights

Minera Antamina holds approximately 15,650.96 ha of surface rights under 19 titles. These are summarized in Table 4-1. The boundary of the surface rights holdings is shown in Figure 4-1.

The surface rights are sufficient to support the life-of-mine (LOM) plan.

4.6 Water Rights

Minera Antamina holds a licence to use surface water and another licence that allows use of groundwater. The company also holds two discharge authorizations to the Ayash and Antamina rivers.

Minera Antamina was granted two authorizations for the reuse of treated wastewater in Huarmey, the coastal town that is the location of the Punta Lobitos port facility.

To maintain valid water rights, the grantee must make all required payments including water tariffs, and abide by the conditions of the water right in that water is only used for the purpose granted.

4.7 Royalties

In 2011, the Peruvian Congress approved an amendment to the mining royalty charge. This charge is based on operating income margins with graduated rates ranging from 1–12% of operating profits; the minimum royalty charge is equivalent to 1% of net sales. If the operating income margin is ≤10%, the royalty charge is 1%. For each 5% increment in the operating income margin, the royalty charge rate increases by 0.75%, to a maximum of 12%.

The Peruvian Congress has enacted a Special Mining Tax that is also based on operating income. Rates range from 2–8.4%. If the operating income margin is ≤10%, the Special Mining Tax is 2%. For each 5% increment in the operating income margin, the special mining rate increases by 0.4%, to a maximum of 8.4%.

Table 4-2: Surface Rights Table

Number	Surface-Right Parcels	Area (ha)
1	Antamina San Marcos (22 parcels)	6,113.73
2	Tranca	259.50
3	Shahuanga I	92.96
4	Shaguanga II	80.52
5	Shaguanga III	80.52
6	Itcuro Utcush (8 parcels)	1,088.56
7	Itcuro Utcush (22 parcels)	4,070.18
8	Ichic Colla & Ganyas	968.64
9	Charac Lote 12	20.00
10	Charac Lote 13	30.19
11	Charac Lote 14	27.86
12	Charac Lote 09	25.30
13	Tuctu	467.50
14	Ichic Colla II	503.40
15	Ichic Colla I	90.00
16	Parga	654.54
17	Mashra	982.95
18	Tucush III A (3 parcels)	68.70
19	Yanacancha IV	25.91
Total		***15,650.96***

There are no private royalties payable on the claims that host the Mineral Resource or Mineral Reserve estimates.

A 2.5% contractual royalty is payable to Osisko Gold Royalties Ltd (Osisko) on the following concessions: Recuay 11, Recuay 12, Recuay 13, Recuay 20, Recuay 21, Recuay 22, Recuay 23, Recuay 24 and Recuay 29. None of these concessions are currently in production, nor do they host estimated Mineral Resources or Mineral Reserves.

4.8 Streaming Agreements

Teck and Glencore have entered into streaming agreements regarding their share of silver production from the Antamina mine. Minera Antamina, which owns and operates the Antamina Operations, is not a party to those silver streaming agreements and operations are not affected by the agreements.

Under a long-term streaming agreement with FN Holdings ULC (FN Holdings), a subsidiary of Franco-Nevada Corporation, Teck has agreed to deliver silver to FN Holdings equivalent to 22.5% of the payable silver sold by Minera Antamina. FN Holdings made a payment of US$610 M on closing of the arrangement in 2015 and pays 5% of the spot price at the time of delivery for each ounce of silver delivered under the agreement. After 86 Moz of silver have been delivered under the agreement, the stream will be reduced by one-third. A total of 29.1 Moz of silver has been delivered under the agreement from the agreement effective date in 2015 to December 31, 2024.

The streaming agreement restricts distributions from Teck Base Metals Ltd., the subsidiary that holds Teck's 22.5% interest in Minera Antamina, to the extent of unpaid amounts under the agreement if there is an event of default under the streaming agreement or an insolvency of Teck.

4.9 Permitting Considerations

Permitting for the Antamina operations is discussed in Section 20.

4.10 Environmental Considerations

Environmental considerations and monitoring programs for the Antamina operations are discussed in Section 20.

There are environmental liabilities associated with the mining and processing activities. In order to minimize these environmental liabilities, Minera Antamina has secured all required environmental permits and conducts work in compliance with these permits. Additionally, Minera Antamina endeavors to comply with all applicable legal and other obligations.

4.11 Social Considerations

Social considerations for the Antamina operations are discussed in Section 20.

4.12 QP Comment on Item 4 "Property Description and Location"

To the extent known to the QPs, there are no other significant factors and risks that may affect access, title, or the right or ability to perform work on the Project that are not discussed in this Report.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

Access to the operations is by paved road. The mine road connects with the Peruvian National Highway 3N at Conococha Lake, and this highway then connects to the Pan American Highway via National Highway 16 (refer to Figure 2-1).

The closest town to the operations is San Marcos, located approximately 38 km from the mine via a dirt road. Huaraz, the department capital, can be reached through 200 km of paved road or 156 km of partial dirt road.

An airport with scheduled regular flights is located at Anta, approximately 20 km north of Huaraz.

Copper and zinc concentrates are delivered to the Punta Lobitos port at Huarmey through a concentrate pipeline. The port site consists of administration buildings, a concentrate thickener, and filters and a concentrate storage warehouse. The concentrate is loaded to boats at a pier.

The molybdenum and lead–bismuth concentrates are transported via road to the Callao port, adjacent to the national capital of Lima.

5.2 Climate

The Antamina Operations are located in a high-alpine climate. The various site infrastructures are within a number of sub-climates including, from highest altitude to lowest:

- Tropical alpine rainy tundra (4,300–5,000 masl);
- Tropical subalpine humid paramo (4,000–4,300 masl);
- Tropical subalpine very humid paramo (3,900–4,500 masl);
- Tropical subalpine rainy paramo (3,900–4,500 masl);
- Tropical montane steppe (2,800–3,800 masl).

Summers are cool, averaging highs of 11.1ºC and lows of 4ºC. Winters are typically wet and cold, with maxima of about 9ºC.

Rainfall averages about 1,205 mm, primarily falling in the October–March period. Snowfall typically occurs in December–February, averaging about 155 mm.

Operations are conducted year-round. Some short-term interruptions may occur because of localized intense thunderstorms.

5.3 Local Resources and Infrastructure

Infrastructure that supports the current operations is in place (see also discussions in Section 16, Section 17, and Section 18 of this Report). These Report sections also discuss water sources, electricity, personnel, and supplies for the LOM plan.

Process and potable water is sourced from rainfall collected in the Nescafe reservoir, situated on the southern side of the Yanacancha WRSF, close to the accommodations camp.

Minera Antamina has an energy supply contract with Engie Energía Perú.

The workforce is primarily sourced within Peru, including from the local Ancash region.

Goods and supplies are typically sourced from Lima, and trucked to the site.

5.4 Physiography

The Project setting is characterized by short glacial valleys with lakes and deep, steep-sided river canyons and valleys. The dominant ridge-and-valley trend is northwest, reflecting the regional structural and tectonic fabric. Shorter, structurally-controlled, northeast-trending valleys occur within the regional fabric, such as the valley that hosts the Antamina operations. The topography is steep, averaging 4,500–4,800 m in altitude, with the highest point at 5,073 m elevation.

The valley that hosts the Antamina deposit is a 4-km long U-shaped glacial valley with steep sides, a flat floor and cirque lake in an upper valley at the valley head. The lake was separated from the main valley by a rock ridge, known as the Taco ridge. There are two hanging valleys, Usu Pallares and Vallesito.

The original vegetation has been significantly affected by human activities. Grassland areas support livestock grazing, and any areas of woodland were harvested for firewood. Much of the Project area is now covered by highland grasses or “ichu”.

5.5 Sufficiency of Surface Rights

Surface rights (see Section 4.6) are sufficient to support the LOM plan.

6 HISTORY

6.1 Ownership, Exploration, and Development History

The exploration and development history is outlined in Table 6-1.

6.2 Production History

The production history is provided in Table 6-2.

Table 6-1: Ownership, Exploration and Development History

Year	Operator	Work Conducted
Pre-1700s		Indigenous mining activities
1850s	Leopold Pflucker	Constructed a small lead–copper smelter
1903–1914	Vicente Lezameta	Produced copper matte. Unsuccessful attempt to leach copper from the deposit
1925	Northern Perú Copper	Company representative recommended core drilling to assess porphyry copper potential. Eight holes (780 m) were drilled.
1952–1970	Cerro of Pasco Corporation	Constructed adits for underground access. Drifted and crosscut 4,300 m within the eastern zone and drove raises totalling 220 m in the centre of the zone. Completed 32 core holes (3,200 m), 18 from surface and 14 from underground
1970	Government of Peru	Expropriated the deposit area.
1971–1993	Minero Perú, Empresa Minera Especial, Geomin, Empresa Minera del Centro del Perú S.A. (Centromin)	Following expropriation, 2,200 ha of mining rights were transferred to Minero Perú, the mining administration agency of the Peruvian government Formed Empresa Minera Especial in 1974, in partnership with the Government of Romania's mining agency, Geomin. Bench and pilot-plant metallurgical work was completed from 1975–1978 in Romania. Different throughput-rate engineering studies completed in 1978, 1982 and 1989. Evaluated open-pit design, mine-equipment selection, concentrator design, surface infrastructure, local social impacts, geotechnical studies, marketing, and economic analyses. The Antamina property was transferred to Centromin and became part of a privatisation sale package in 1993.
1995	Rio Algom Limited (Rio Algom), Inmet Mining Corporation (Inmet)	Obtained property interest through sales process, and formed a 50:50 joint venture (JV), Compañía Minera Antamina S.A. (Minera Antamina)
1998	Rio Algom, Inmet, Noranda Inc. (Noranda), Teck	In 1998, Inmet sold its JV interest, and Minera Antamina was restructured under an ownership of 37.5% Rio Algom, 37.5% Noranda, and 25% Teck. Completed feasibility study assuming open pit mining methods.
1999	Rio Algom, Inmet, Noranda Inc., Teck, Mitsubishi Corporation (Mitsubishi)	Mitsubishi acquires 10% JV interest, resulting in the ownership percentage being 33.75% Rio Algom, 33.75% Noranda, 22.50% Teck, and 10% Mitsubishi.
2000–2013	BHP, Inmet, Noranda, Falconbridge Limited (Falconbridge), Xstrata plc (Xstrata), Glencore plc (Glencore), Teck, Mitsubishi	BHP acquires Rio Algom in 2000. Falconbridge acquired Noranda in 2005. Falconbridge acquired by Xstrata in 2006, and in 2013, Xstrata acquired by Glencore. Resulting ownership percentage is 33.75% BHP, 33.75% Glencore, 22.50% Teck, and 10% Mitsubishi.

Year	Operator	Work Conducted
2001–2009	Minera Antamina	Trial mining operations began in 2002, with the first full year of operations (72,000 t/d). Production increased in 2004 (85,000 t/d) and 2009 (92,000 t/d).
2011	Minera Antamina	Life extension until 2028 with TSF to level 4165 extends the mine life. Production rate increase to 130,000 t/d following completion of completion of the Antamina expansion project
2014	Minera Antamina	Production increased to 145,000 t/d.

Table 6-2: Production Table

Year	Copper (kt)	Zinc (kt)	Year	Copper (kt)	Zinc (kt)
2001	177	60	2013	440	260
2002	325	230	2014	346	210
2003	261	373	2015	390	235
2004	365	184	2016	432	204
2005	368	185	2017	422	366
2006	379	161	2018	448	413
2007	336	287	2019	453	303
2008	339	355	2020	378	430
2009	317	456	2021	444	463
2010	301	389	2022	452	429
2011	335	234	2023	422	464
2012	448	214	2024	413	249

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

The Peruvian Andean metallogenic belt, which hosts numerous large base and precious metals deposits, is about 1,000 km long, and trends north–northwest–south–southeast. The belt is bounded by two major east–west-oriented shear zones, the Huancabamba deflection, at about 5°S, and the Abancay deflection, at about 14°S.

The mineralization within the Peruvian Andean metallogenic belt is primarily hosted within the Marañón Fold–Thrust Belt, a structurally and stratigraphically complex deformation zone between the Cordillera Occidental, a Cainozoic magmatic arc, and the Cordillera Oriental, a zone of post-Permian uplift that exposed Precambrian crystalline and Palaeozoic sedimentary and igneous rocks.

In the region of the Antamina operations, the Marañón Fold–Thrust Belt consists of Precambrian to Cambrian metamorphic basement rocks (the Marañón Complex) overlain by Palaeozoic and Mesozoic shelf carbonates and clastic sedimentary rocks. The sedimentary sequence is dominated by carbonate-rich units including the Permian Mitu Group, the Triassic–Jurassic Pucara Group, and Chicama Formation; and the Cretaceous Oyón Formation, Goyllarisquizga Group, Pariahuanca Formation, Jumasha Formation, and Celendín Formation (Figure 7-1).

Eocene and Miocene intrusions can locally intrude the sedimentary sequence, and the entire sequence can locally be overlain by Miocene volcanic rocks.

Deformation has included at least five events.

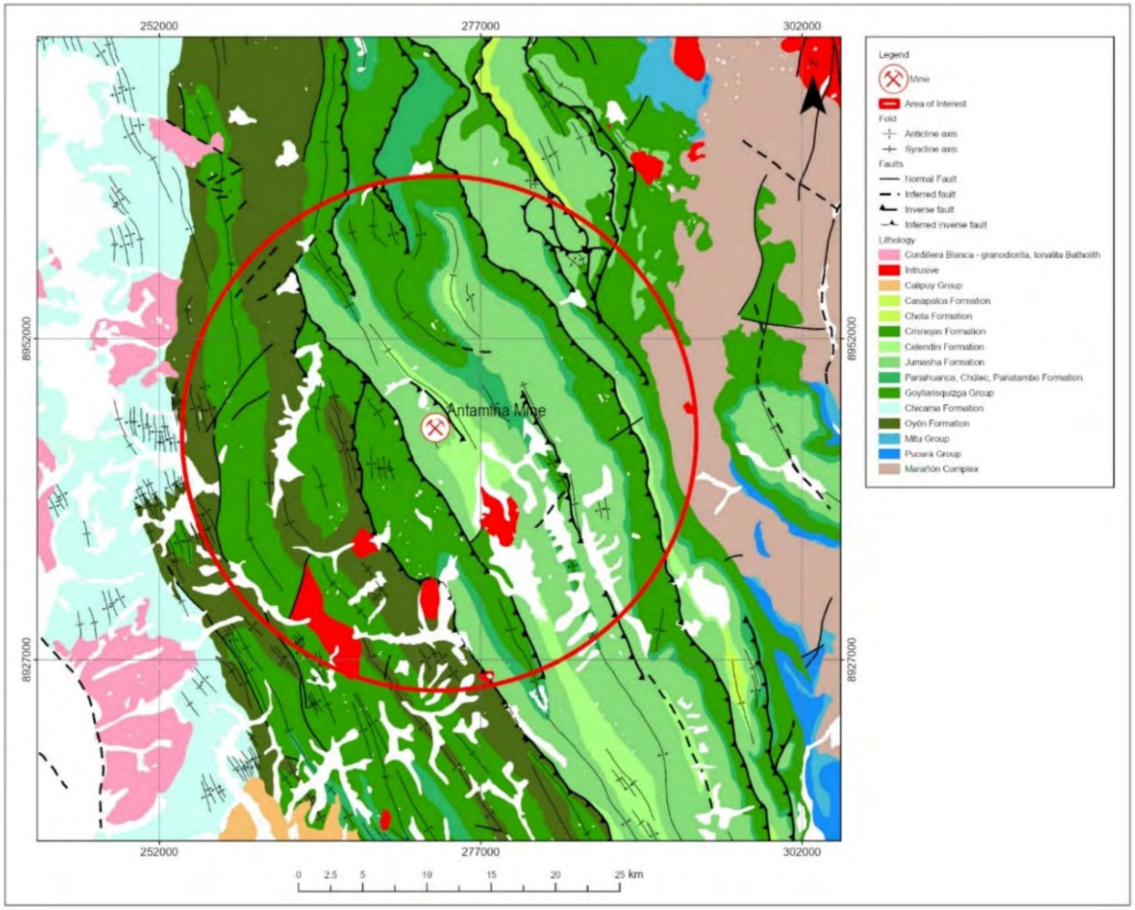
7.2 Project Geology

The formation of the Antamina deposit is attributed to the emplacement of multiple mineralized porphyries (strictly along a northeast-trending fault zone into reactive wall rocks between 10.95 ± 0.20 Ma (oldest uranium–lead zircon age) and 9.68 ± 0.05 Ma (youngest rhenium–osmium molybdenite age).

7.2.1 LITHOLOGIES

The geology of the general Project area includes the sedimentary lithologies summarized in Table 7-1. Thicknesses given in the table are those for the Project area, not necessarily those of the individual formations overall.

The sedimentary rocks have been intruded by a multi-phase porphyry complex, locally referred to as the Antamina Porphyry Complex. The main mineralized area consists of three northeast–southwest-oriented contiguous porphyry centres: the Antamina Main zone, the Usupallares zone, and the Condorcocha zone, which is located about 1 km north of the Antamina Main zone. Technically, the intrusive bodies are trachyandesite porphyries in composition, but are referred to as quartz monzonites at the mine and in many published papers on the deposit.



Note: Figure prepared by Minera Antamina, 2024.

Figure 7-1: Regional Structural and Geology Plan

The Antamina Porphyry Complex hosts five porphyry phases. The Usupallares zone occupies a smaller southwest portion of the Antamina Porphyry Complex, and consists of three porphyry phases. The Condorcocha zone consists of at least three porphyry phases. Identification of the first phase is complicated by the presence of Lake Condorcocha, which obscures the earliest intrusion under water. Two sub-parallel porphyry dykes that intersect the Antamina Main and Condorcocha zones along their northeast margins are referred to as the Oscarina dykes.

A general geology map showing the locations of the intrusions is provided in Figure 7-2.

7.2.2 STRUCTURE

The primary mineralization control is the Valley Fault system, which is a northeast-striking sub-vertical fault zone that defines the trend of the Antamina Valley. Mineralization is best developed a dilation zone along the fault system (Figure 7-3).

The Antamina Main zone coincides with maximum dilation along this structure, whereas Usupallares formed in a structurally-controlled hanging valley perpendicular to the main Valley Fault zone.

Intrusion ages, and the intrusion paragenesis, indicate the overall deposit formed in several short magmatic–hydrothermal pulses rather than during a protracted period of intense magmatic–hydrothermal activity. The southeastwardly progressive younging of relatively short duration magmatic–hydrothermal events was likely controlled by the dilation along the Valley Fault.

7.2.3 METAMORPHISM AND ALTERATION

Skarns formed in tandem with each phase of porphyry emplacement, finally coalescing to form a continuous skarn body.

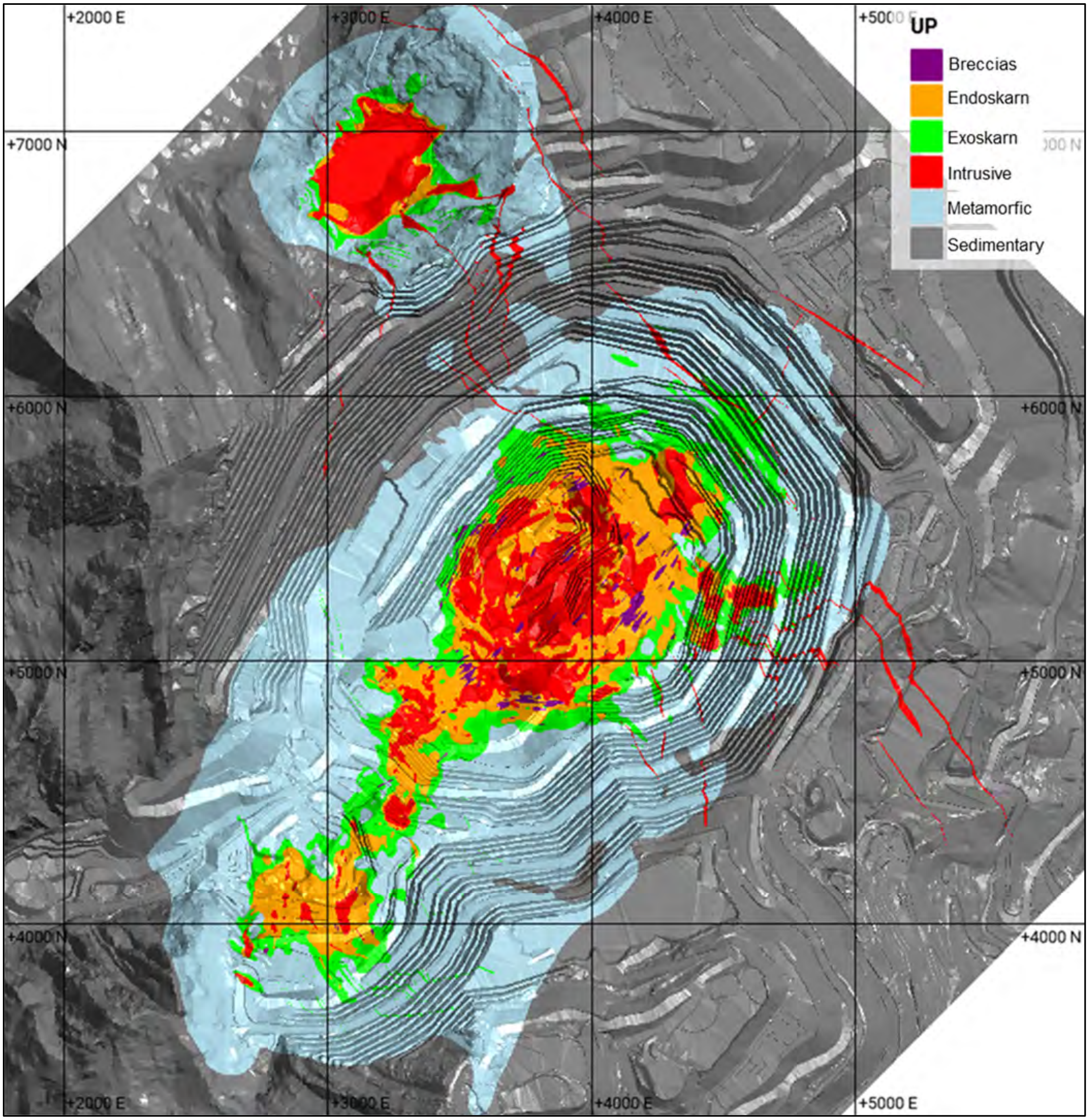
The Antamina Porphyry Complex is partially altered to prograde, pink/brown-garnet endoskarn, which occurs as a 20–100 m wide annulus around the entire complex, as well as in many internal zones, which can reach as much as 75 m wide, occurring to depths well below 3,000 m above mean sea level. Internal endoskarn zones are irregular but can be followed along strike for several tens of metres.

A schematic showing the likely formation over time is included as Figure 7-4.

P1 is the causative skarn-forming intrusion in each porphyry centre. Intrusion of P1 into marble forms early skarns and mineralisation. Endoskarn and exoskarn are genetically related to P1 porphyries. Stockwork quartz veins (≤ 40% volume) and hydrothermal biotite (potassic alteration) are locally abundant in the central porphyry complex. Stockwork veins form in P1, but they do not propagate into the wall rock as porphyry-style alteration; skarns form instead (refer to Figure 7-4).

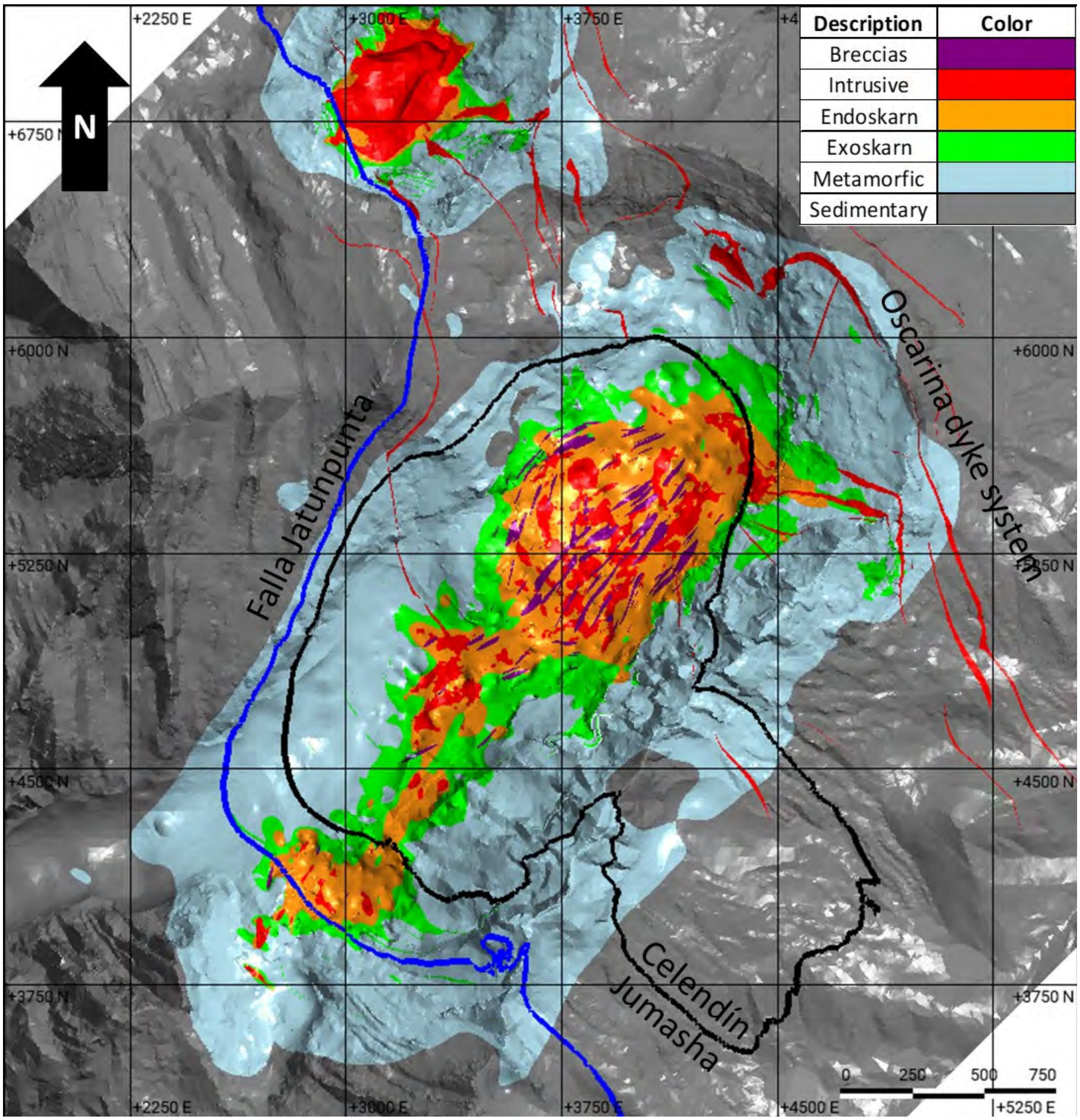
Table 7-1: Lithology Table

Unit	Symbol	Age	Description
Quaternary		Quaternary	
Intrusion	N-rd	Neogene	Rhyodacite
	N-mgr		Antamina Taulli Monzogranite
		Paleogene	Nescafé Monzogranite
Celendín Formation	Ks-ce	Upper Cretaceous	Thin-bedded, predominantly marly limestone, with shale and lesser limestone
Jumasha Formation	Ks-j	Middle Cretaceous	Medium-to-thickly bedded bioclastic limestone with lesser amounts of marl, dolomite, and chert nodules
Pariatambo Formation	Ki-pt	Cretaceous	Dark grey, fossiliferous bituminous marls and limestones with chert intercalations
Chúlec Formation	Ki-chu	Cretaceous	150 m thickness of calcareous mudstone and shale, interbedded with micritic limestone
Pariahuanca Formation	Ki-ph	Cretaceous	120 m thick unit of nodular and micritic limestone, interbedded with calcareous sandstone, calcareous mudstone, and some gypsum layers
Farrat Formation	Ki-f	Cretaceous	80–100 m thickness of quartzite and sandstone
Carhuáz Formation	Ki-ca	Cretaceous	600 m thick unit of interbedded sandstone, shale, and siltstone
Santa Formation	Ki-s	Cretaceous	35–45 m thickness of micritic limestone and calcareous mudstone
Chimú Formation	Ki-chi	Cretaceous	600 m thick unit of quartz and quartzite sandstone, intercalated with coal layers
Oyón Formation	Ki-oy	Cretaceous	>600 m thick unit of light grey quartz sandstone and quartzite, interbedded with mudstone and siltstone
Chicama Formation	Js-ch	Jurassic	>1,000 m thickness. Consists of grey to red conglomerate interbedded with quartzitic sandstone and carbonaceous shales. Minor andesitic sills.



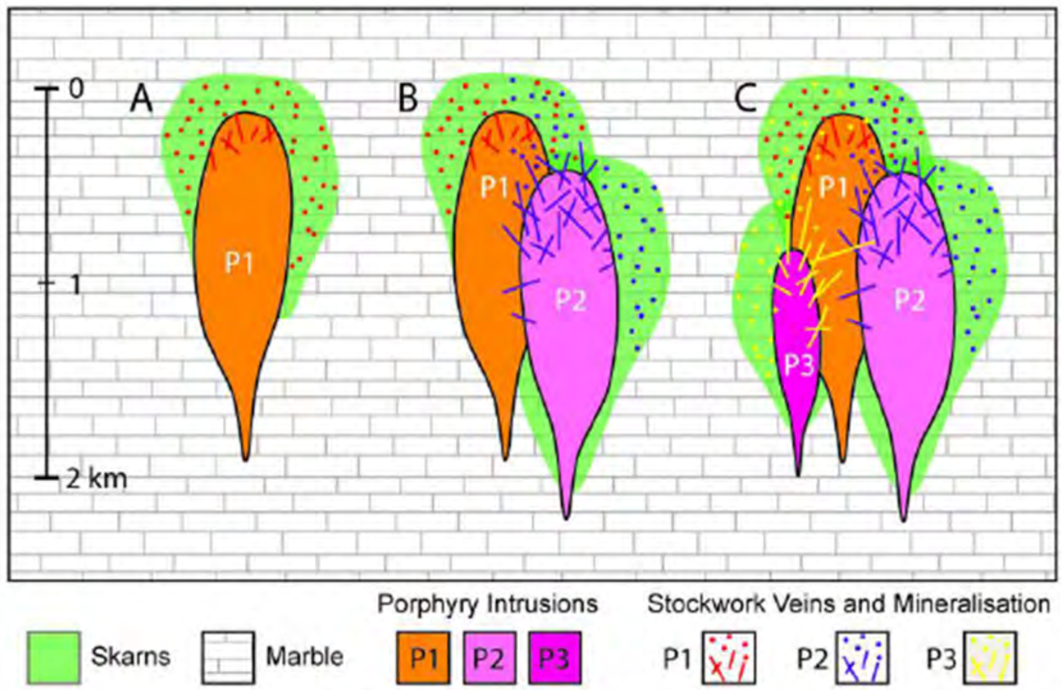
Note: Figure prepared by Minera Antamina, 2024. North is to top of figure.

Figure 7-2: General Geology, Antamina Deposit



Note: Figure prepared by Minera Antamina, 2024. Falla = fault.

Figure 7-3: Structural Setting



Note: Figure from Mrozek, 2018.

Figure 7-4: Skarn Formation Schematic

P2 intrusions cut P1 intrusions, related skarns, and early quartz veins. Locally, P2 can contain xenoliths that can reach several metres in length of P1 exoskarn, P1 endoskarn, and refractory quartz vein fragments. Quartz stockwork veins (± pyrite ± chalcopyrite ± molybdenite) extend outward from P2 porphyries and cross-cut P1 skarns. Where P2 intersects marble, additional skarns form, extending the vertical and lateral extent of skarn alteration already present from P1 intrusive phases.

Intrusion of P3 is accompanied by stockwork veins that cut across P1, P2, and earlier-formed skarns. Where P3 intersects marble, more skarns form and additional skarns also form within the existing P1 and P2 skarns, further extending the vertical and lateral extent of skarn alteration generated during P1 and P2.

Neither P2 nor P3 contain endoskarn alteration, although they do contain veins and disseminations of secondary biotite overprinted by sericite–chlorite alteration.

Molybdenite mineralisation occurs in two stages across the Taco and Usupallares zones. Stage I molybdenite occurs in skarns; and Stage II as molybdenite + quartz veins that cut across P2 and P3 porphyries.

In addition to the skarns, there are units that show marbling and hornfelsing. Marble is usually medium- to coarse-grained, pale grey to white, and varies from banded to massive. It becomes more and more massive closer to the skarn front. There are local instances where skarn is in direct contact with limestone, with no visible metamorphic effects.

Hornfels units are exposed in the walls of the northeastern side of the Antamina valley, dipping gently to the east, and are disturbed by thrust faults. Hornfels units vary in colour from pale grey to green and brown.

7.2.4 MINERALIZATION

Mineralisation and metal zoning shows an outward progression from molybdenum ± copper in the central porphyry, to copper (± silver, bismuth)–zinc–lead from proximal to distal exoskarns.

The ore mineralogy is dominated by molybdenite, chalcopyrite, bornite, and sphalerite, with lesser galena and minor bismuth–silver–sulphur minerals.

A detailed description of the mineralization types is included in Section 7.3.7.

7.3 Deposit Descriptions

7.3.1 DEPOSIT DIMENSIONS

The main core of the Antamina deposit is about 2,000 m long and approximately 1,500 m wide. The mineralization extends from the surface at approximately 4,500 m above mean sea level and has been intersected in core drill holes below 2,500 m above mean sea level.

The Usupallares zone mineralization lies to the south of the Antamina Main zone, extends from surface at 4,200 m above mean sea level, and has been intersected in drill holes to approximately 3,000 m above mean sea level. The Usupallares zone is about 800 m wide, and approximately 1,000 m long.

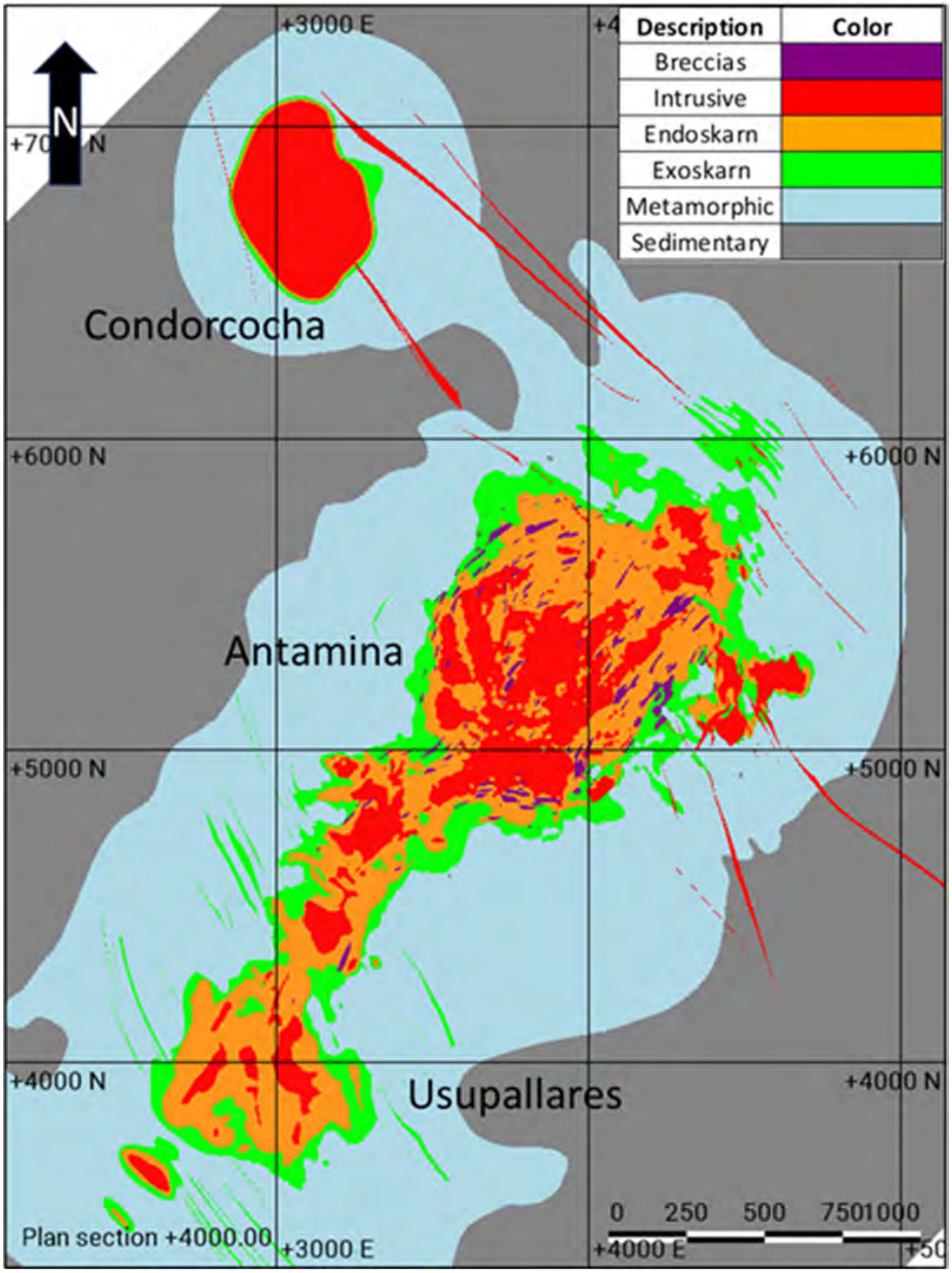
The general boundary between the Antamina Main and Usupallares zones is shown in Figure 7-5.

7.3.2 LITHOLOGIES

The Jumasha Formation (see description in Table 7-1) is considered the main protolith of the Antamina exoskarn. This unit is dominant to the south of the location of the former Lake Antamina, where thrust faulting has caused it to extend partially over the Celendín Formation lithologies.

Intrusive rocks are identified based on composition, texture, and location into four major areas of intrusive activity:

- Antamina Main;
- Usupallares;
- Condorcocha;
- Oscarina dykes.



Note: Figure prepared by Minera Antamina, 2024.

Figure 7-5: Boundary, Antamina Main and Usupallares Zones

The major porphyry phases documented in each intrusive centre are classified as P1 (early), P2 (intermineralisation), and P3 (late inter-mineralisation). Sub-phases are assigned an alphanumeric code. The intrusions and their inter-relationships are summarized in Table 7-2.

A plan view showing the grouped lithologies as used in resource estimation is included as Figure 7-6. A cross-section through that grouped lithological interpretation is provided in Figure 7-7.

7.3.3 SKARNS

Within the skarns there are systematic zoning patterns defined by garnet colour, garnet:clinopyroxene ratio, and garnet and clinopyroxene compositions.

Skarn garnet colour changes from pink to red to brown in endoskarn, and from red to brown and green (from proximal to distal) in exoskarn. Garnet becomes more andradite-rich and grossularite-poor from proximal to distal skarns, and clinopyroxene becomes more hedenbergite-rich and diopside-poor along the same trend.

The skarns zone outward from the porphyry centre as follows: brown garnet endoskarn, mixed brown and green garnet indeterminate skarn, mixed brown and green garnet exoskarn, green garnet exoskarn, diopside exoskarn, wollastonite exoskarn, hornfels, marble, and limestone.

A summary of the major skarn units as used in the geological modelling supporting the Mineral Resource estimate is provided in Table 7-3.

Skarn alteration zones are narrower in the Usupallares zone (approximately 500 m) compared to the Antamina Main zone (approximately 800–1,000 m).

7.3.4 STRUCTURE

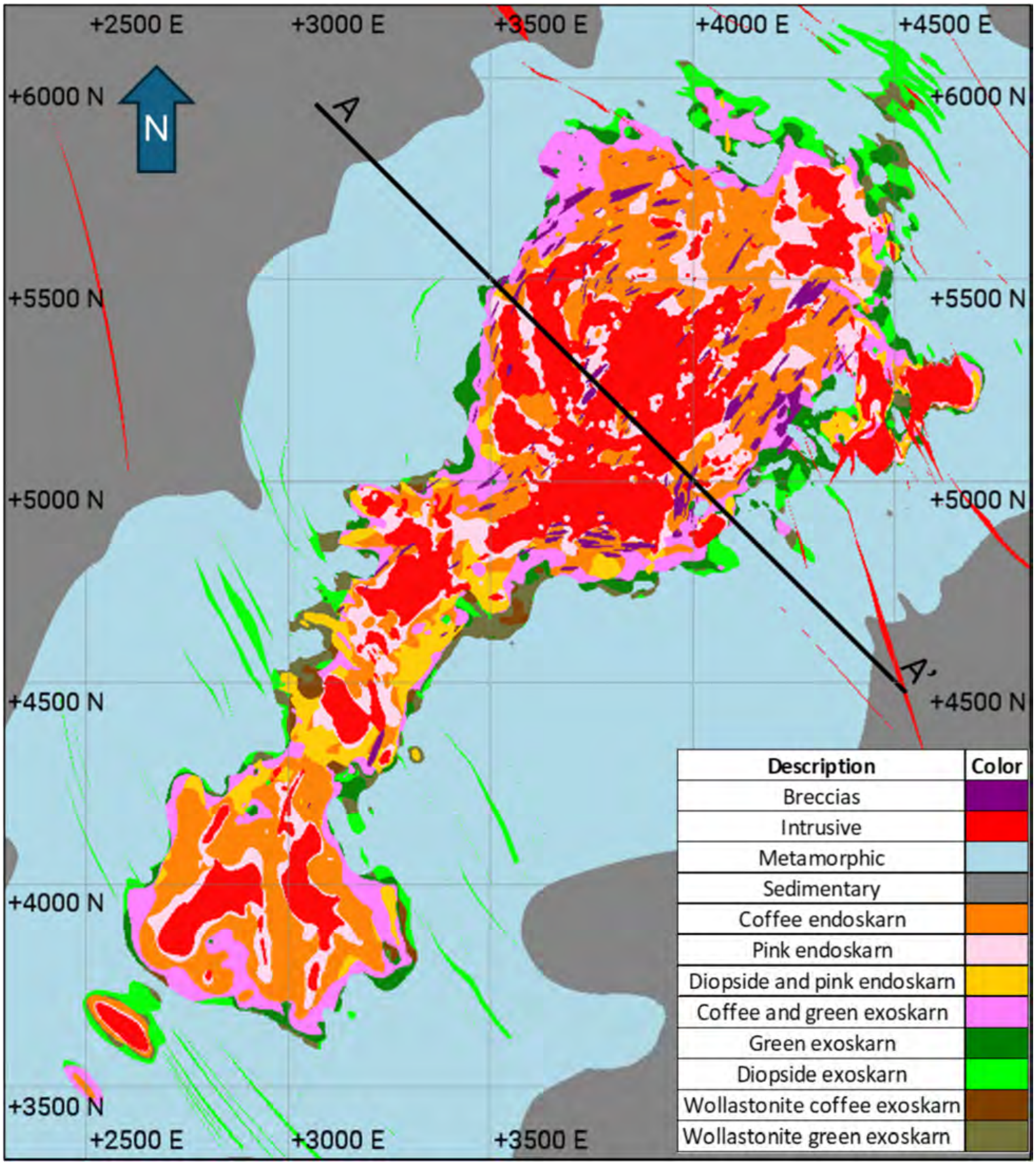
Porphyry emplacement was strongly influenced by the pre-existing structural framework, including a series of northwesterly-trending thrust faults intersected by the northeasterly-trending Valley Fault system (refer to discussion in Section 7.2.2).

The Valley Fault system focused the emplacement of the Antamina Porphyry Complex, and was key to the circulation of hydrothermal fluids that allowed higher-grade mineralization to be deposited in a relatively localised area.

Table 7-2: Porphyry Intrusions

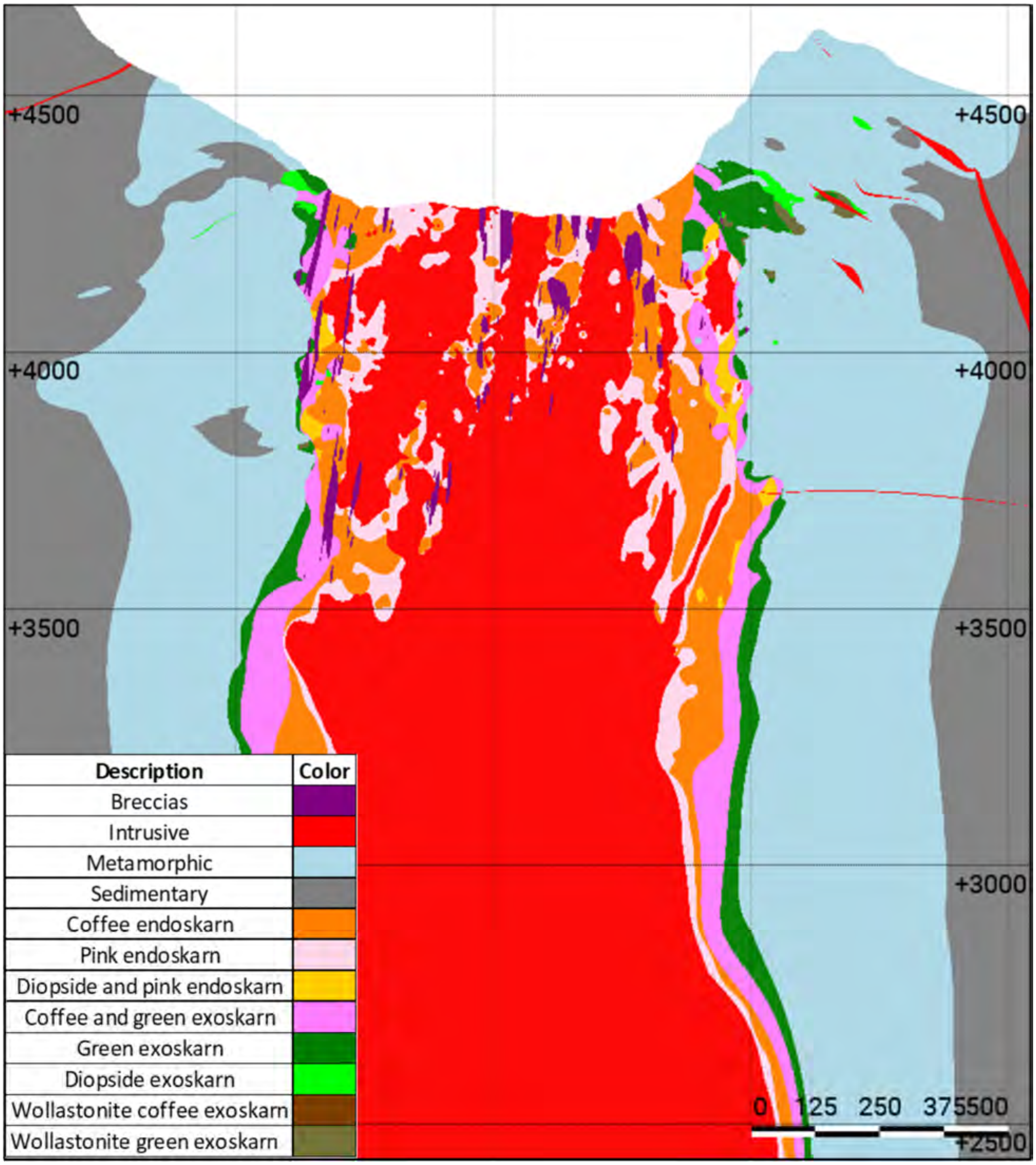
Complex Name	Intrusive Generation	Sub-Generation	Description
Antamina Main	P1		Trachyandesite porphyritic composition. Emplaced as a series of dykes ranging in width from <1 m to tens of meters. Least-altered P1 is porphyritic and contains up to 75% phenocrysts of quartz, plagioclase, and K-feldspar; biotite and amphibole are also present, but have largely been replaced by secondary biotite and endoskarn. Can form xenoliths and sometimes large blocks or dismembered zones in P2 and P3. Contains continuous pervasive and massive endoskarn alteration along the P1-wall rock (exoskarn) contact. P1 and its associated skarns are cut across by P2 and P3 porphyries. Stockwork quartz veins cut across P1 porphyry, endoskarn, and exoskarn.
	P2		Forms broad dykes and a stock that occupies the centre of the Antamina Main zone. Consists of two compositionally similar trachyandesite porphyries, P2a and P2b. Neither P2a nor P2b contain endoskarn alteration, and they cut across endoskarn and exoskarn. Stockwork quartz veins are variable in abundance.
		P2a	Slightly coarser grained compared to P2b and displays a variably megacrystic porphyry texture.
		P2b	Truncates veins in P2a.
	P3		Trachyandesite porphyry that includes plagioclase glomerocrysts and K-feldspar megacrysts. Significantly less veined.
Usupallares	P1		Trachyandesite porphyry. Displays endoskarn alteration along outer margins in contact with exoskarn-altered country rock. Endoskarn-altered dyke margins grade into a core of less altered P1 porphyry with weak secondary biotite alteration. Quartz veins cut across P1 endoskarn and exoskarn.
	P2		Trachyandesite porphyry. Lacks endoskarn alteration and cuts across P1 with sharp contacts. Contains quartz veining.
	P3		Occurs as narrow dykes along P1–P2 contacts, has a dark grey groundmass and abundant skarn xenoliths. Lacks endoskarn alteration, and rarely contains quartz veins. Cuts across P2 and truncates veins in that porphyry phase
Oscarina	P1, P2		Emplaced along steeply dipping (~80°), southeast-striking (~140°) regional thrust faults. Lack endoskarn alteration
		P1	Trachyandesite porphyry with localised patches of calcite + epidote ± prehnite alteration

Complex Name	Intrusive Generation	Sub-Generation	Description
		P2	Trachyte porphyry that contains patchy, weak pervasive calcite + chlorite + illite/muscovite alteration
Condorcocha	P1		Interpreted as an exoskarn
	P2		Trachyandesite porphyry that lacks endoskarn but contains hydrothermal biotite veinlets (potassic alteration) and quartz stockwork veins. P1-exoskarn is cut by P2.
	P3		Andesite porphyry dyke with no quartz stockworks



Note: Figure prepared by Minera Antamina, 2024. Section line shows the location of Figure 7-7.

Figure 7-6: Grouped Lithology Plan View



Note: Figure prepared by Minera Antamina, 2024. Section location is shown in Figure 7-6. Section looks northeast.

Figure 7-7: Grouped Lithology Section View

Table 7-3: Major Skarn Units

Name	Description	Comment
Brown endoskarn	Brown garnet-bearing endoskarn. Can vary from fine-grained (<1 mm) to coarse-grained (>10 mm), occurring as a compact garnet skarn, but often disaggregated in character. Lithology is harder in the Usupallares area.	Most important ore-hosting lithology. Hosts the Cu-rich (chalcopyrite), low-Bi mineralization type.
Pink endoskarn	Coarse-grained, pink garnet-bearing. Consists of milky white, plagioclase-rich matrix enclosing large pink garnets and sparser maroon garnets. Displays a relict porphyritic texture.	Relatively poor Cu mineralisation. A favourable lithology for Mo mineralization.
Pink diopside endoskarn	Fine-grained pink and beige garnet-bearing. Consists of a mixture of milky white plagioclase and greenish diopside-rich matrix enclosing pink garnets and diopside. Does not display a relict porphyritic texture.	Relatively less Cu mineralization than the pink Endoskarn.
Brown–green exoskarn	Mixture of brown and green garnets, in which brown garnet commonly occurs in veinlets cutting green garnet-dominated rock. In places, brown garnet skarn preferentially replaces some layers in green garnet exoskarn at a centimetre scale, producing banded, brown, and green garnet skarn. In the Usupallares area, the occurrence of cafe-green Exoskarn is in the form of a narrow band <30 m wide.	One of the main hosts to Cu–Zn mineralization at Usupallares.
Green exoskarn	Zone of green garnet replacing calcite, commonly found adjacent to zones of marble or hornfels.	Can be an important source of Cu–Zn mineralization.
Diopside exoskarn	Consists of diopside, with minor amounts of garnet, calcite, and other minerals. Confined to higher elevations, primarily at the northern end of the deposit, and generally adjacent to hornfels.	Can host Cu–Zn mineralization.
Green and wollastonite exoskarn	Wollastonite Exoskarn comprises an inner zone (contiguous with the green Exoskarn) of bornite mineralization, and an outer zone (closer to marble) of bornite–sphalerite mineralization. The contact between Wollastonite Exoskarn and green Exoskarn is a broad gradational replacement zone where green garnet replaces wollastonite.	Can host Cu–Zn mineralization.

7.3.5 ALTERATION

The Antamina deposit overall displays systematic alteration patterns. Three major stages of alteration are recognised across the porphyry-skarn transition, as summarized in Table 7-4:

- Stage 1: potassic and prograde alteration;
- Stage 2: retrograde alteration and copper–molybdenum–zinc–lead mineralization;
- Stage 3: late phyllic alteration and molybdenum mineralization.

Outward from the centre of the Antamina Main zone, the alteration consists of potassic alteration (hydrothermal biotite) transitioning into endoskarn, then exoskarn, bleached marble, and fluid escape structures in the most distal deposit areas.

7.3.6 BRECCIATION

A zone of brecciation is located on the southeast side of the open pit (refer to Figure 7-3). The brecciation is oriented northeast, parallel to the deposit's long dimension. Most zones occur as steeply dipping and bifurcating sheets within structural corridors up to 120 m wide and over 600 m long. Most individual breccia zones are <30 m wide, and can be traced down-dip and along strike for a maximum of 400 m.

7.3.7 MINERALIZATION

The majority of mineralisation at Antamina formed during Stage 2 retrograde alteration and occurs in endoskarns, exoskarns, and in veins cutting across porphyries. A minor amount of mineralisation is noted in marble and hornfels.

The deposit was unroofed by glaciation and is exposed in a glacial valley; hence there is no significant oxidation or enrichment.

The Antamina Main and Usupallares zones are horizontally and vertically zoned over a depth of at least 2 km with respect to major metal components:

- Copper is relatively evenly distributed from endoskarn to the limestone contact, and chalcopyrite is distributed throughout all skarn zones;
- Zinc and bismuth tend to occur within 70 m of the contact of green garnet skarn with limestone/marble/hornfels. The appearance of sphalerite approximately coincides with the transition from brown to green garnet;
- Molybdenite occurs in the intrusion and adjacent skarn, as well as being abundant in the wollastonite-diopside skarn;
- Silver, lead, and bismuth values are highest in the outer part of the copper-zinc zone and adjacent marble. Silver can be present in any of the skarn lithologies. Lead is generally hosted by green garnet exoskarn, diopside exoskarn, and hornfels.

Table 7-4: Alteration Types

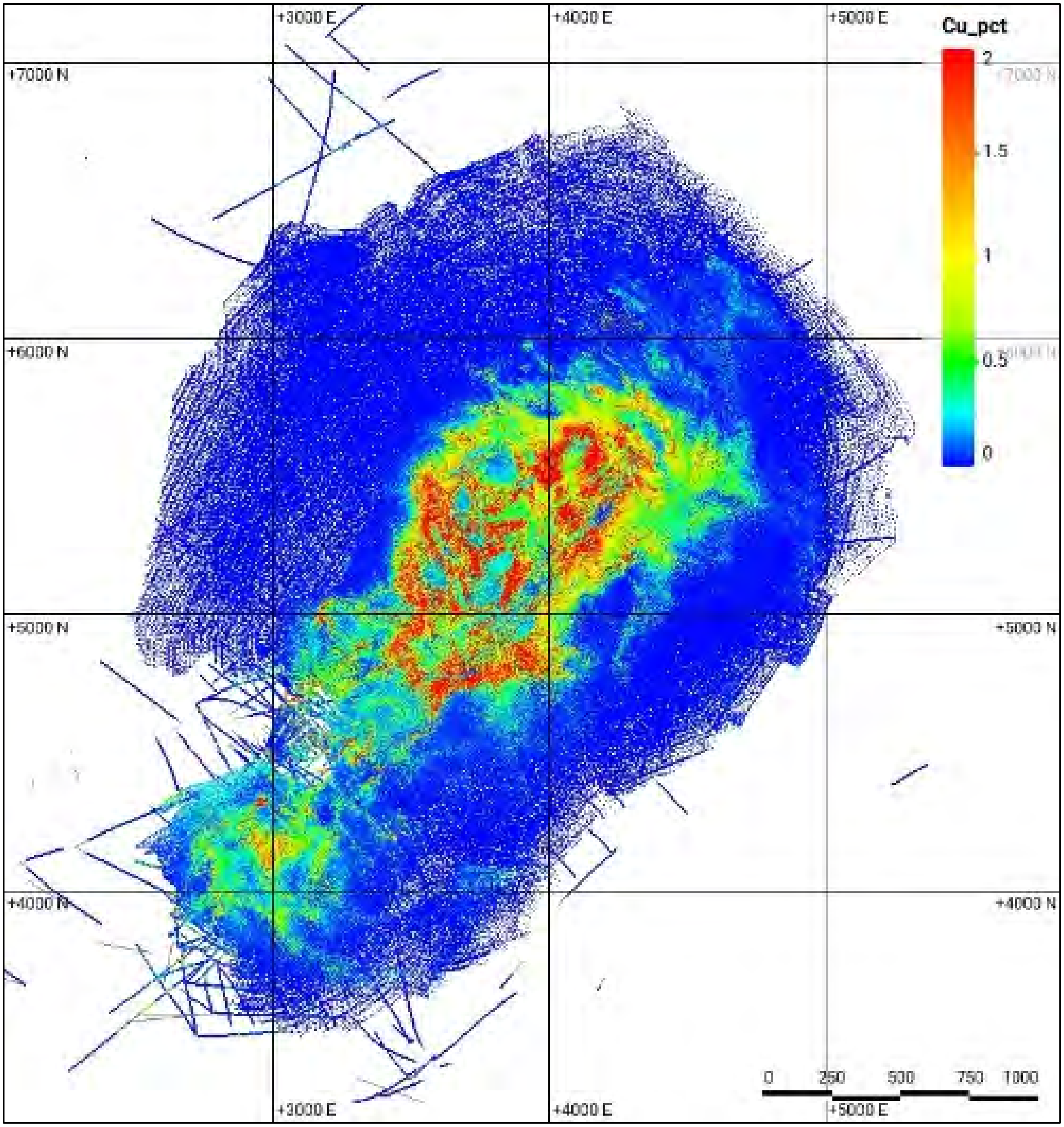
Type	Description
Stage 1	Hydrothermal biotite ± K-feldspar (potassic alteration) in intrusive rocks. Garnet ± clinopyroxene in prograde endoskarn and exoskarn. Endoskarn garnet is pink to red and exoskarn garnet is red, brown, or green. Endoskarn and exoskarn formed early during prograde skarn alteration in igneous rocks and wall rocks, respectively. Both endoskarn and exoskarn occur as massive to partial replacements of their respective host rocks. Prograde skarn mineralogy is dominated by garnet with lesser clinopyroxene ± wollastonite. Endoskarn occurs in P1 porphyries in the Antamina Main, Usupallares, and Condorcocha zones, but not in later porphyry stages. Purple anhydrite + quartz veins in potassically-altered rocks in the central Antamina Main–Bornita zone
Stage 2	Intensity is structurally controlled, with the strongest alteration noted along contacts or veins. The dominant retrograde assemblage consists of quartz + sulphides ± calcite ± epidote ± vesuvianite ± chlorite ± magnetite, with lesser fluorite, phlogopite, and rare ilvaite
Stage 3	Banded molybdenum + quartz veins, pyrite + calcite + quartz + chlorite veins, and trace garnet + scheelite + sericite veins. Cut across Stage 2 alteration and mineralisation

By rock type, the highest concentrations of zinc are hosted in exoskarn, copper in endoskarn and exoskarn, and molybdenum in endoskarn and P2 (Antamina Main and Usupallares). Bismuth displays two prominent zones of high concentration, one in the north–northeast Antamina Main zone and another in the Bornita zone. The Bornita zone is a name used for a region of the deposit where bornite is associated with wollastonite exoskarn. Silver distribution overlaps with copper and zinc.

Zonation of the main ore minerals typically follows the same trend as their constituent metals, with molybdenite concentrated in the Antamina Main and Bornita porphyry centres, chalcopyrite slightly outward of molybdenite in endoskarn and exoskarn, and sphalerite outward from chalcopyrite in exoskarns. The patterns of chalcopyrite and sphalerite are similar in the Usupallares zone; however, the minerals are less abundant. Bornite has a more localized distribution, occurring mostly southwest of the Antamina Main zone in the Bornita zone. Galena occurs in minor amounts in distal exoskarns where its occurrence overlaps with sphalerite. Scheelite was observed in only one occurrence in the northeast Antamina Main and Oscarina zones, although the distribution of tungsten indicates that it may be more common than noted in distal exoskarns.

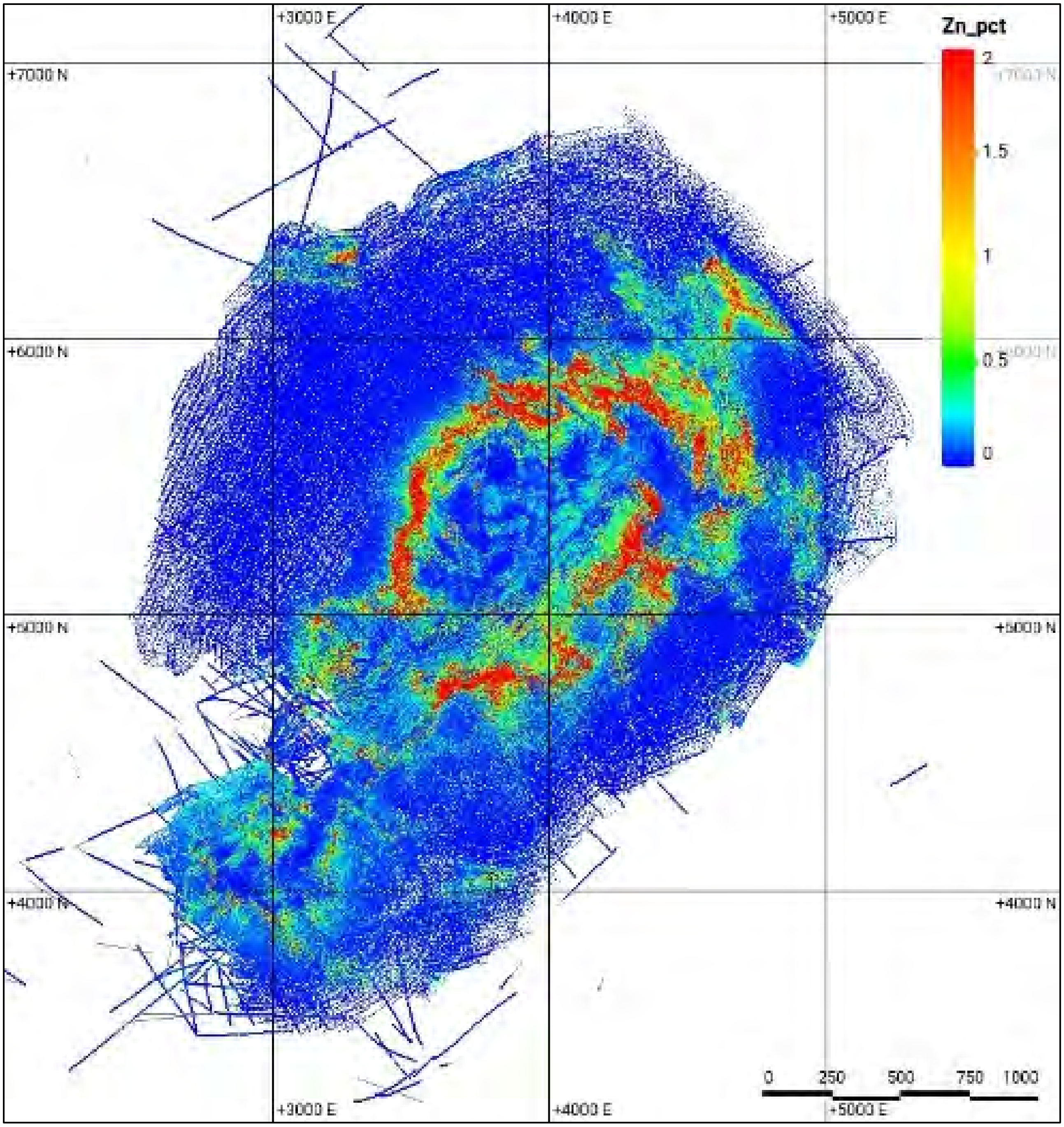
Copper, molybdenum, and zinc sulphides comprise the principal ore minerals; these include chalcopyrite, bornite, molybdenite, and sphalerite. Pyrite and magnetite are common, but of lesser economic importance. Minor minerals include galena and pyrrhotite. Rare occurrences of chalcocite and wittichenite (Cu_3BiS_3) have been observed. Sulphides occur disseminated interstitial to garnet, as irregular massive sulphide zones, and as veinlets. The mineralization distribution for copper, zinc, silver, molybdenum and lead within the open pit is provided in Figure 7-8, Figure 7-9, Figure 7-10, Figure 7-11 and Figure 7-12 respectively. Cross-sections through the mineralization are included in Section 10.

The mineralization has been separated into a number of different geometallurgical types (Table 7-5).



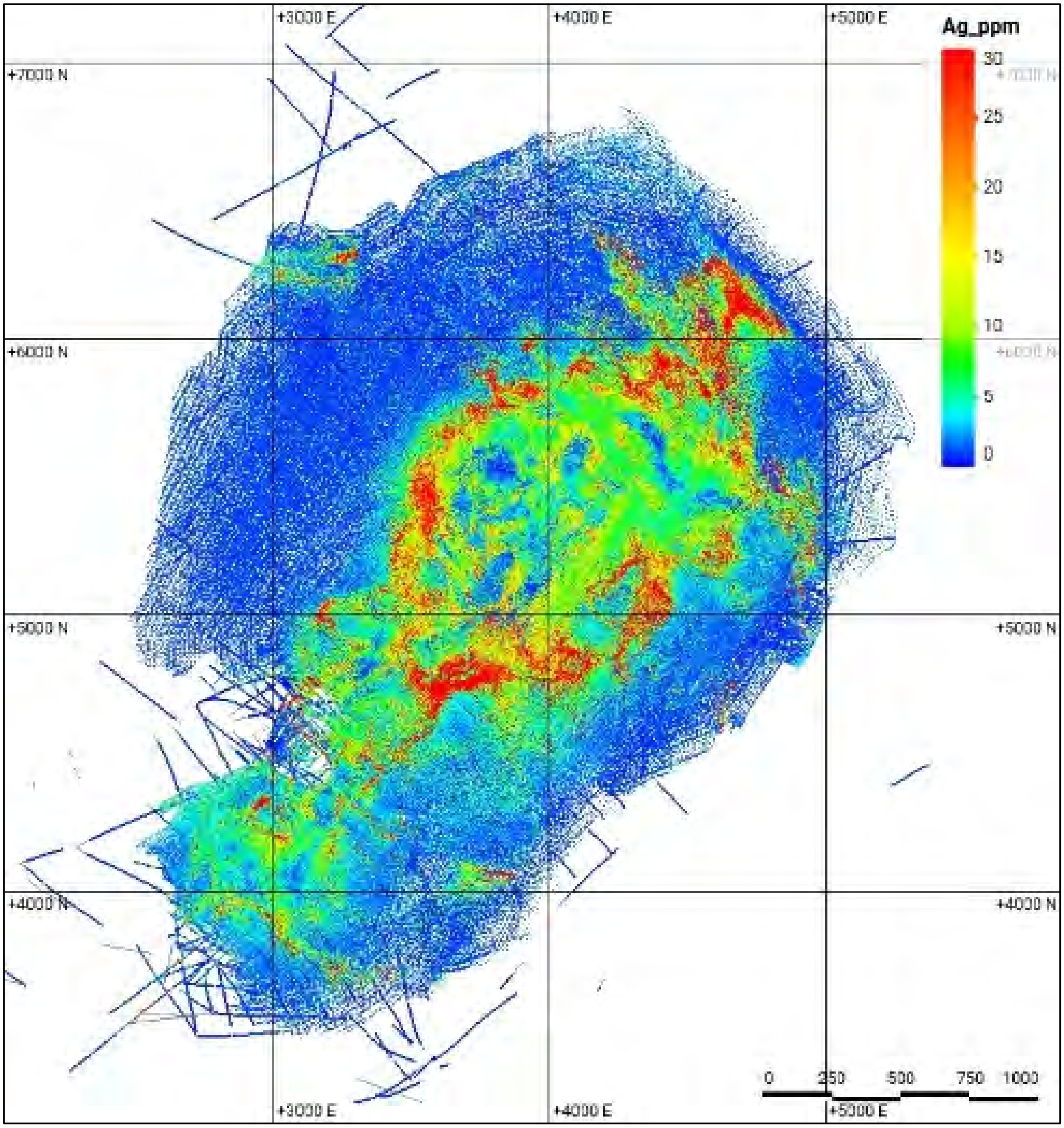
Note: Figure prepared by Minera Antamina, 2024.

Figure 7-8: Mineralization Distribution, Copper



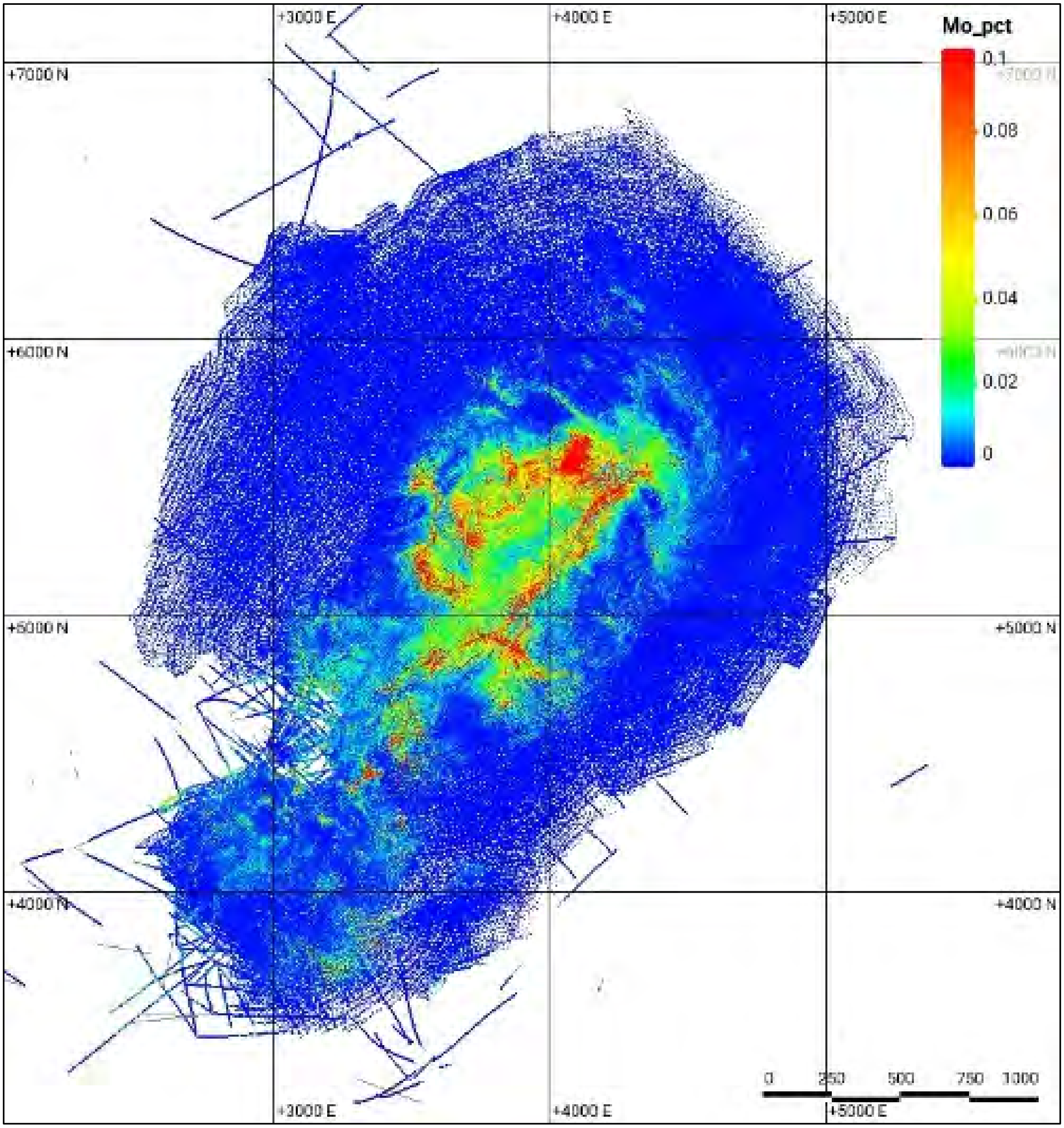
Note: Figure prepared by Minera Antamina, 2024.

Figure 7-9: Mineralization Distribution, Zinc



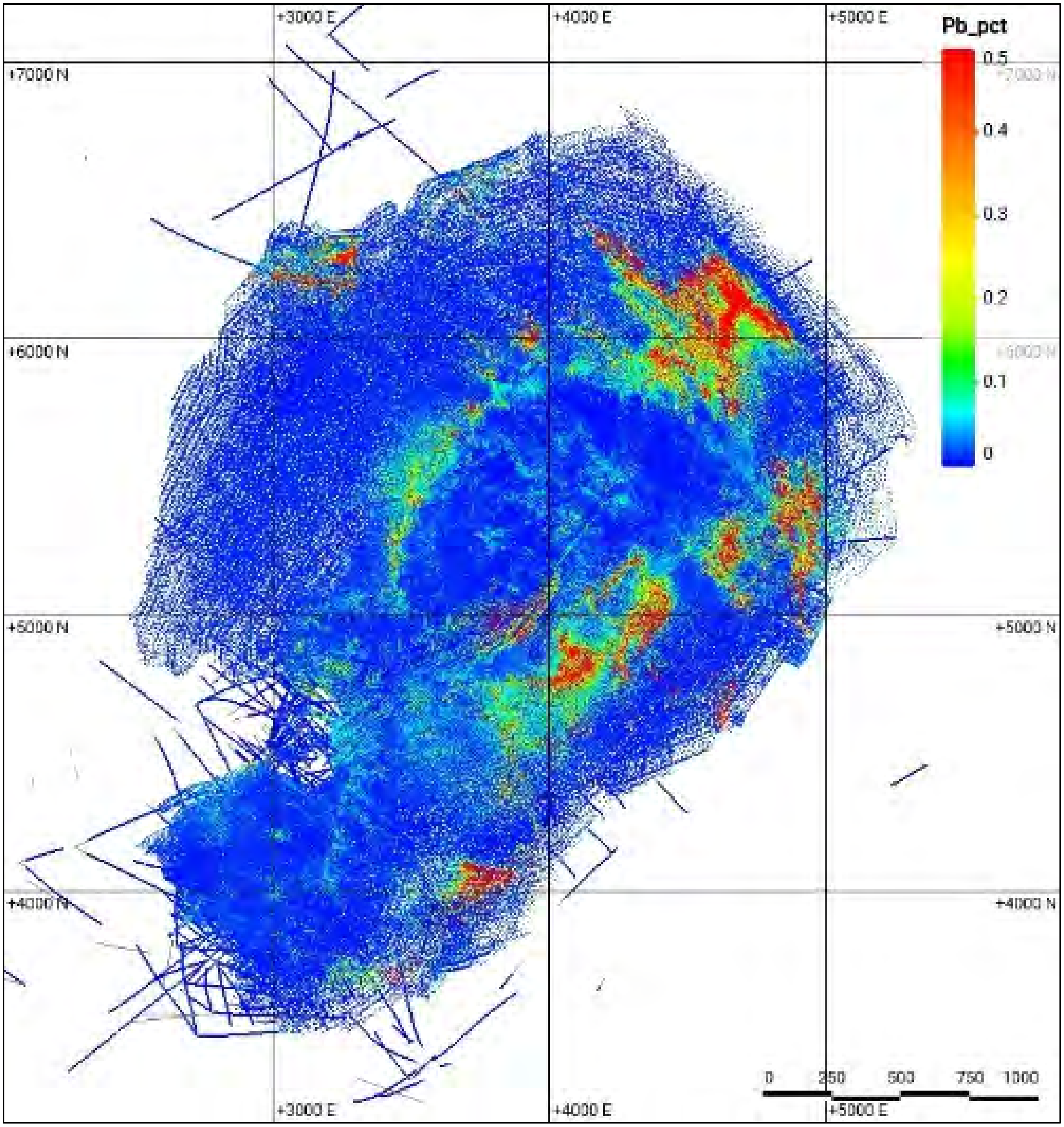
Note: Figure prepared by Minera Antamina, 2024.

Figure 7-10: Mineralization Distribution, Silver



Note: Figure prepared by Minera Antamina, 2024.

Figure 7-11: Mineralization Distribution, Molybdenum



Note: Figure prepared by Minera Antamina, 2024.

Figure 7-12: Mineralization Distribution, Lead

Table 7-5: Geometallurgical Ore Types

Mineral Type	Geometallurgical Code	Geometallurgical Type	Grade Range
Cu	M1	Copper, low bismuth	Bi/Cu <9.4, <0.6% Zn (ratio ppm Bi/%Cu)
Cu	M2	Copper, high bismuth	9.4 ≤Bi/Cu <35.3, <0.6% Zn (ratio ppm Bi/%Cu)
Cu	M2A	Copper, very high bismuth	Bi/Cu ≥35.3, <0.5% Zn (ratio ppm Bi/%Cu)
Cu	M2AT	Copper, transition	CuCN/Cu > 0.1* (% ratio)
Cu–Zn	M4B	Copper–zinc, low bismuth	≥0.5%Zn
Cu–Zn	M4BT	Copper–zinc transition	CuCN/Cu > 0.1*, ≥0.5% Zn (% ratio)
Cu (bornite)	M5	Bornite, low zinc	<0.5% Zn
Cu–Zn (bornite)	M6	Bornite, high zinc	≥0.5% Zn

8 DEPOSIT TYPES

8.1 Introduction

The Antamina deposit is considered to be an example of a giant skarn deposit.

8.2 Deposit Type Description

Skarn-hosted deposits typically form in orogenic belts at convergent plate margins and are related to intrusions associated with the development of oceanic island arcs or back arcs (Ray, 1998; Meinart, 1992; Meinart et al, 2003). Skarn-hosted deposits form during contact or regional metamorphism and metasomatism. The majority of the world's major skarn deposits are interpreted to be related to hydrothermal systems (Einaudi et al., 1981)

Skarns develop in sedimentary carbonate rocks, calcareous clastic rocks, volcaniclastic rocks or (more rarely) volcanic flows in close spatial association with high to intermediate-level stocks, sills and dykes of gabbro, diorite, quartz diorite, or granodiorite composition.

Skarns are classified according to the rock type in which they develop. Endoskarn is skarn developed in intrusions and exoskarn is skarn hosted by sedimentary, volcanic, and metamorphic rocks. Metal deposits hosted by skarns are classified into various types based on metal content (Einaudi and Burt, 1982; Meinart, 1992).

The processes that lead to formation of all types of skarn deposits include

- Stage 1: Iso-chemical contact metamorphism during pluton emplacement;
- Stage 2: Prograde metasomatic skarn formation as the pluton cools and an ore fluid develops;
- Stage 3: Retrograde alteration of earlier-formed mineral assemblages.

Deposition of ore minerals typically accompanies stages 2 and 3. Skarn deposits are commonly zoned mineralogically with respect to pluton contacts, original lithology of host rocks, and (or) fluid pathways. Later petrogenetic stages may partly or completely obliterate earlier stages of skarn development.

Skarn-hosted base and precious metal mineralization frequently displays strong stratigraphic and structural controls. Deposits can form in exoskarn along sill–dike intersections, sill–fault contacts, bedding–fault intersections, fold axes and permeable faults or tension zones.

Deposits range from irregular lenses and veins to tabular or stratiform bodies with lengths ranging up to many hundreds of meters.

Mineral and metal zoning is common in the skarn envelope.

8.3 QP Comment on Item 8 “Deposit Types”

Features of the Antamina deposit that meet the skarn model include:

- A central porphyry intrusion of intruding carbonate stratigraphy. Disseminated and fracture-controlled copper and molybdenum mineralization may be found in this zone;
- Development of an endoskarn zone at the outer margin of the intrusion, with intense alteration of the intrusive protolith and near-total replacement of primary minerals and igneous textures by garnet, diopside, and iron sulfides and oxides. Copper and molybdenum mineralization is strongest in this zone;
- Development of an exoskarn zone in carbonate rocks at the contact between the country rock and the intrusion. Exoskarn is characterized by intense iron and magnesium metasomatism, and near-total replacement of carbonates and sedimentary textures by pyroxene, garnet, and calc-silicate minerals. Base metal and silver mineralization is found in exoskarn. Zinc grades are generally higher than copper grades;
- Local hornfels-grade metamorphism of the limestone and marble near the outer contact of the skarn complex;
- Larger-scale alteration of limestone to marble often along structures and bedding planes. Traces of zinc and lead mineralization can be associated with marble.

The QPs are of the opinion that a skarn deposit type is an appropriate model for exploration and for support of the geological models.

9 EXPLORATION

9.1 Introduction

Limited non-drilling surface exploration work has been conducted. When exploration commenced, prior to deposit definition, surface geochemical and geophysical techniques were used. Currently, all exploration is completed using drill methods, which is consistent with an operating deposit with an adequate level of geological knowledge.

9.2 Grids and Surveys

Topographic data are separated into three time periods.

All pre-1996 work is based on topographic control provided by surveyors. In 1996, a re-survey was completed, and survey control was created. The elevation was increased by +69 m based on the re-survey.

In 1998–1999, when mine construction started, Eagle Mapping undertook an aerial survey and established new survey control. Elevation control was established by survey from a known point on the coast of Peru. The elevation was found to be overstated by 67 m; therefore, a -67 m correction was applied to the earlier +69 m-corrected data. All post-1999 topographic data are assumed to be correct.

The current mine topographic survey is completed using a total-station global positioning system (GPS) instrument with sub-metric precision.

The mine grid is a local grid modified from the UTM PSAD 56 system by subtracting 270,000 m from the UTM easting and 8,940,000 m from the UTM northing, with no rotation.

9.3 Geological Mapping

Geological mapping was completed at two scales:

- 1:25,000-scale district geological mapping;
- 1:2,000-scale geological mapping of specific prospects.

Detailed surface geological mapping was restricted to the following areas:

- The immediate vicinity of the Antamina main deposit;
- Rosita of Oro (east of the Antamina main deposit);
- Usupallares;
- Condorcocha.

A compilation of prospect-scale and district-scale geological mapping by Inmet and Antamina shows references to lithology mapping, bedding strike, and dip and alteration, as well as rock chip sample identification numbers, at multiple sites throughout the district. However, detailed notes and results of rock-chip sampling are generally absent.

The results of the geological mapping programs were incorporated into the geological maps included in Section 7.

9.4 Geochemical Sampling

Existing surface litho-geochemistry data are limited to rock-chip samples collected at the Usupallares and Condorcocha prospects, and a few samples collected from the Robyn claims, the Ayash prospect, and the Don Augusto prospect. The INMET 1:25,000 geology compilation map contains locations and assays for 25 rock-chip samples, mostly in and around the Antamina and Contonga mines.

There is no detailed database for the geochemical data. Geochemical data are not used to support Mineral Resource estimates.

9.5 Geophysics

9.5.1 AIRBORNE SURVEYS

Airborne magnetometer and radiometric surveys were flown over most of the district by World Geoscience in 1996 or earlier (World Geoscience 1998), under contract to Rio Tinto. These are available at the MINE as paper copies of colour-shaded relief maps of total magnetic intensity, potassium radiometry, and thorium radiometry at 1:5,000 and 1:20,000 scales.

Total magnetic intensity maps and potassium and thorium radiometry appear to have limited use, and locally show strong coupling with topographic relief.

Electronic copies of the data files are available.

9.5.2 GROUND SURVEYS

Ground geophysical surveys in the Antamina district consist of ground-magnetometry and induced polarization (IP) surveys that were carried out over the Antamina main deposit.

Ground-magnetometry surveys were carried out over the Rosita of Oro, Condorcocha, Usupallares, and Tucush Valley areas. Colour-shaded image plots are available for each of these areas at scales from 1:2,000 to 1:10,000.

Presentations include total magnetic intensity reduced to the Equator, total magnetic intensity continued upward 30 m (to reduce noise), and the vertical derivative of total magnetic intensity to emphasize anomalies (Figure 9-1).

These surveys were undertaken by José Arce in 1995 and Val D’Or Geofísica in 1996. Electronic copies of the data files have not been located to date.

Ground-magnetometry surveys appear to be very effective in identifying areas of skarn mineralization under cover rocks. Such a survey showed strong magnetic signals above magnetite–garnet skarn mineralization and endoskarn along the northern and southern sides of the central intrusive at Condorcocha, three areas of porphyry at Usupallares, and endoskarn and exoskarn throughout the southwestern and southern sides of the Antamina main deposit.

Several magnetic highs were identified in the Rosita of Oro area, and are interpreted to be associated with porphyry dykes.

Ground-magnetometry surveys of the Tucush Valley did not detect anomalies that warranted additional follow-up.

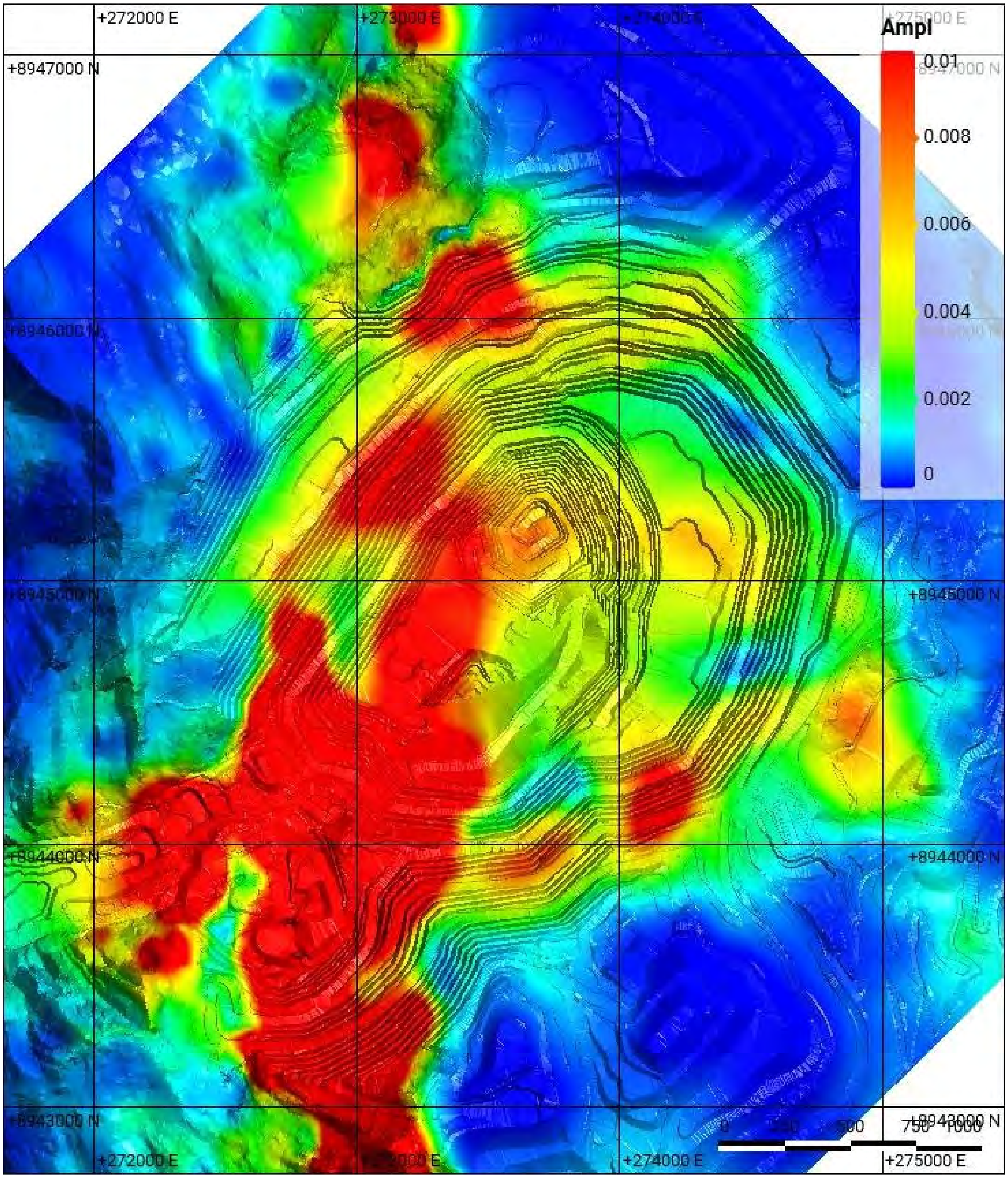
9.6 Exploration Drifts and Tunnels

Cerro of Pasco completed 4,300 m of underground exploration drifts and 220 m of exploration raises in about 1956. Limited documentation is available, and what remains is considered to be of poor quality. The area drifted has since been mined out.

Centromin completed 12,399 m of tunnels between 1956 and 1975. Approximately 6,000 m of underground exploration drifts were channel sampled. The internal Certimin laboratory completed zinc and copper assays. These analytical data are not used in Mineral Resource estimation.

9.7 Petrology, Mineralogy, and Research Studies

Various specialized studies have been completed in support of exploration, development, and metallurgical studies. A number of papers have been published on aspects of the Antamina deposit in scientific and technical journals, and have been presented at mining conferences.



Note: Figure prepared by Minera Antamina, 2024.

Figure 9-1: Magnetic Amplitude Image, Antamina Open Pit

The following research theses have been submitted:

- Aranda Clemente, C., 2010: Assessment of Waste-Rock Weathering Characteristics at the Antamina Mine Based on Field-Cells Experiment: Master of Applied Science thesis, University of British Columbia, Canada;
- Escalante Aramburu, A.D., 2008: Patterns of Distal Alteration Zonation around Antamina Cu-Zn Skarn and Uchucchacua Ag-Base Metal Vein Deposits, Peru: Mineralogical, Chemical, and Isotopic Evidence for Fluid Composition, and Infiltration, and Implications for Mineral Exploration: PhD thesis, University of British Columbia, Canada;
- Mrozek, Stephanie, 2018: The Giant Antamina Deposit, Peru: Uplift History, Intrusive Sequence, and Skarn Mineralisation: PhD thesis, James Cook University, Australia.

9.8 Exploration Potential

Opportunities remain find additional skarn inside the delimited intrusive zone of the deposit, especially in areas where there has been limited drilling. Figure 7-6 shows that there are numerous areas within the intrusions below the 4000 m elevation that have not been investigated for skarn potential, though higher elevations within the intrusive bodies do show skarn development.

10 DRILLING

10.1 Introduction

Drilling to December 31, 2024 in the Project area totals 4,265 drill holes (1,355,884 m). Of this total, 4,111 drill holes are core holes (1,334,705 m), 65 are channel samples that are treated as drill holes, and 89 drill holes are reverse circulation (RC) drill holes. The drilling is summarized in Table 10-1.

Of the total drilling, 3,861 drill holes (1,234,587 m) are used to support the Mineral Resource estimates (refer to Table 10-1). The database used for estimation has a close-out date of 15 August, 2023. Collar locations for those drill holes that are not used in Mineral Resource estimation are shown in Figure 10-1, and the collar locations for the drill holes used for estimation support are shown in Figure 10-2.

Information omitted from estimation support includes 32 core holes completed by Cerro of Pasco (3,200 m), 65 channels completed by Cerro of Pasco (12,399 m), 56 RC holes completed by Minera Antamina in 1997, and 118 holes that had drilling issues or were located outside the area of estimation. Drill holes completed for condemnation and geotechnical purposes are not used in modelling.

10.2 Drill Methods

Drill contractors and drill rig types used in the Cerro of Pasco and Centromin campaigns are unknown. Minera Antamina has used third-party drill contractors Boart Longyear, Bradley-MDH, AK Drilling, and Explomin for drill campaigns from 1996 to the Report effective date.

Core sizes used in the Cerro of Pasco and Centromin campaigns are unknown. Core sizes used by Minera Antamina included HQ (63.5 mm core diameter) and NQ (47.6 mm).

10.3 Logging Procedures

Core is initially subject to a quick log to identify and mark up the major lithological contacts and the main areas of copper and zinc mineralization. The lithology-logging minimum interval is 1 m. Mineralization is assigned a numeric intensity code.

Detailed logging uses a standardized, tablet-based GVMapper format. Lithological units are typically logged on 1 m long intervals, with lithological codes selected on the basis of pre-set abbreviations for the units. Significant structural features (veins, faults, stratification, and contacts) are recorded regardless of the interval length.

Logging also records mineralization type and intensity, geophysical parameters, such as index fracturing (disaggregation index) and the disaggregation ratio, vein bulk density, and vein type.

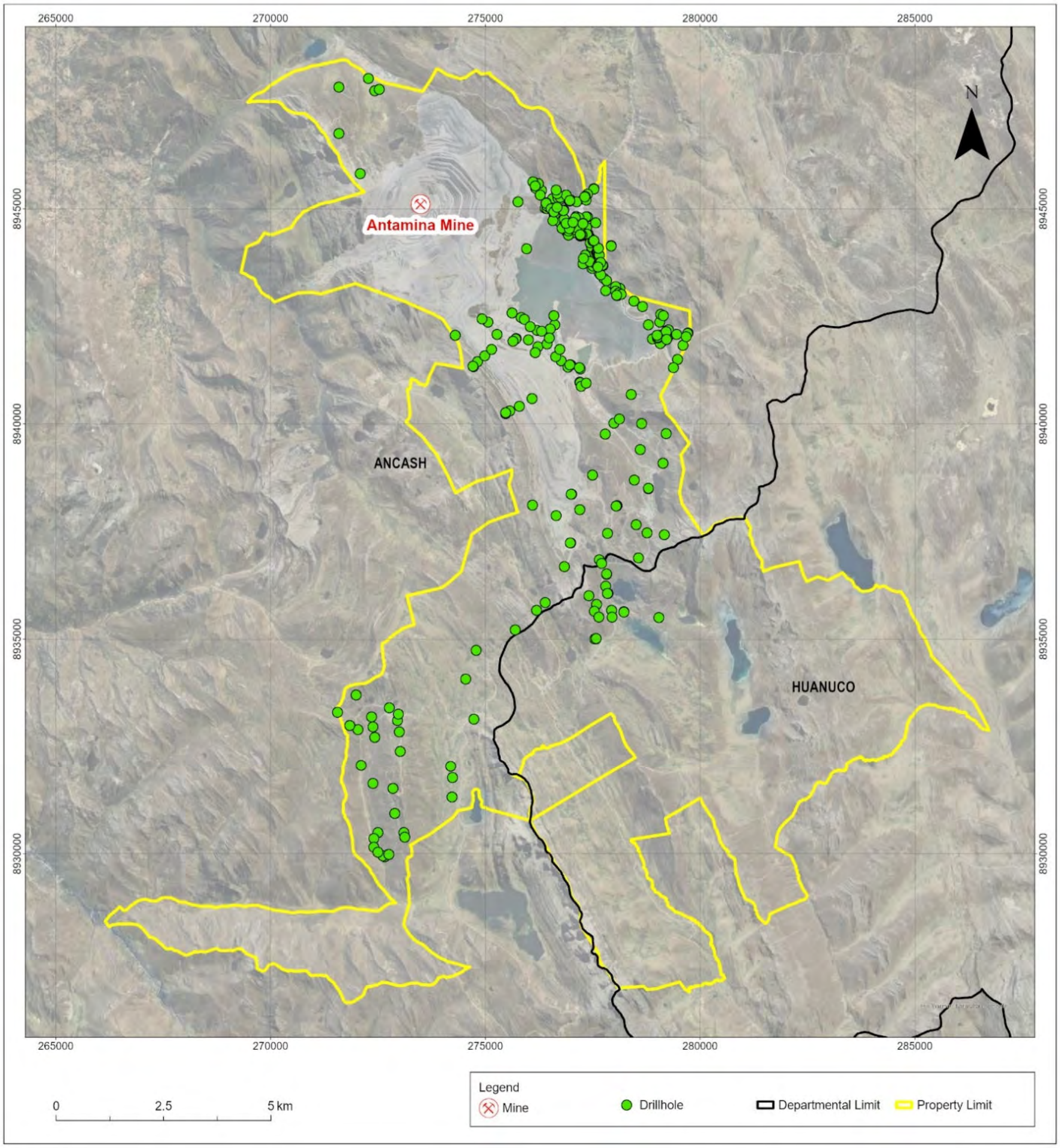
Once logged, core is photographed.

Table 10-1: Drill Summary Table

Company	Year	Drill Type	Number Of Drill Holes	Number Of Assayed Samples	Total Meterage Drilled (m)	Average Core Recovery (%)	Assay Laboratory	Quality Control	Used In Geological Model	Number Of Drill Holes In The Model	Total Meterage In The Model (m)
Cerro of Pasco	1925–1968	Core	32	—	3,200	Unknown	Unknown	Unknown	No	—	—
Centromin	1970–1975	Core	109	6,296	13,486	77	Centromin	Unknown	Yes	102	12,724
Centromin	1970–1975	Channel	65	10,507	12,399	—	Centromin	Unknown	No	—	—
Minera Antamina	1996–1997	Core	296	38,253	106,841	85	SGS and Chemex	Yes (10%)	Yes	275	103,508
Minera Antamina	1997 (assayed 2003)	RC	56	384	2,809	—	ALS Chemex	Yes (10%)	No	—	—
Minera Antamina	1999–2000	Core	98	3,588	12,766	77	SGS And Chemex	Yes (11%)	Yes	87	11,790
Minera Antamina	2002–2003	Core	162	15,000	32,427	91	ALS Chemex	Yes (10%)	Yes	161	32,277
Minera Antamina	2002–2003	RC	31	3,685	5,687	—	ALS Chemex	Yes (10%)	Yes	31	5,687
Minera Antamina	Late 2003	Core	25	2,705	4,686	93	ALS Chemex	Yes (16%)	Yes	25	4,686
Minera Antamina	2004	Core	469	54,961	107,050	91	ALS Chemex	Yes (19%)	Yes	467	106,815
Minera Antamina	2005–2006	Core	200	18,868	42,972	93	ALS Chemex	Yes (21%)	Yes	198	42,812

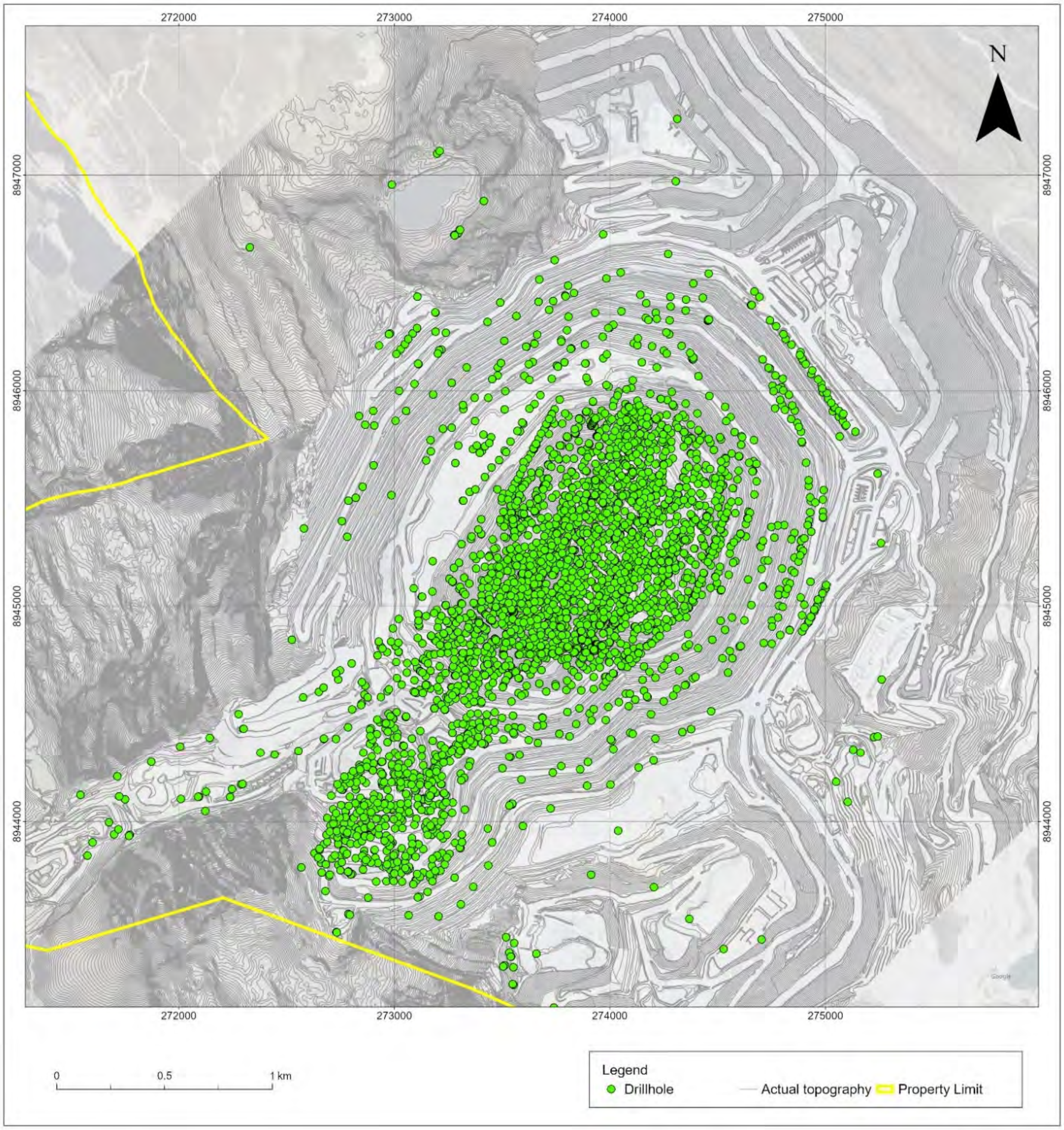
Company	Year	Drill Type	Number Of Drill Holes	Number Of Assayed Samples	Total Meterage Drilled (m)	Average Core Recovery (%)	Assay Laboratory	Quality Control	Used In Geological Model	Number Of Drill Holes In The Model	Total Meterage In The Model (m)
Minera Antamina	2006–2007	Core	261	49,097	105,822	93	ALS Chemex	Yes (20%)	Yes	256	105,057
Minera Antamina	2007–2008	Core	375	77,343	160,237	95	ALS Chemex	Yes (20%)	Yes	372	159,039
Minera Antamina	2008–2011	Core	288	32,228	79,854	95	ALS Chemex	Yes (20%)	Yes	282	77,856
Minera Antamina	2011–2012	Core	394	44,683	104,399	97	ALS Chemex	Yes (24%)	Yes	384	100,311
Minera Antamina	2012–2013	Core	213	32,945	76,449	97	ALS Chemex	Yes (21%)	Yes	195	74,208
Minera Antamina	2013–2014	Core	259	27,002	62,236	97	ALS Chemex	Yes (21%)	Yes	246	61,505
Minera Antamina	2014–2015	Core	182	26,190	60,449	97	ALS Chemex	Yes (15%)	Yes	182	60,496
Minera Antamina	2015–2016	Core	155	24,596	62,982	97	ALS Chemex	Yes (15%)	Yes	150	62,791
Minera Antamina	2016–2017	Core	59	14,338	35,287	98	ALS Chemex	Yes (14%)	Yes	59	35,287
Minera Antamina	2017–2018	Core	69	12,773	36,975	98	ALS Chemex	Yes (15%)	Yes	67	36,294
Minera Antamina	2018–2019	Core	59	9,679	29,793	90	SGS	Yes (16%)	Yes	58	29,393
Minera Antamina	2019–2020	Core	67	6,592	20,753	90	SGS	Yes (16%)	Yes	63	20,106

Company	Year	Drill Type	Number Of Drill Holes	Number Of Assayed Samples	Total Meterage Drilled (m)	Average Core Recovery (%)	Assay Laboratory	Quality Control	Used In Geological Model	Number Of Drill Holes In The Model	Total Meterage In The Model (m)
Minera Antamina	2020–2021	Core	50	7,638	19,410	95	SGS	Yes (14%)	Yes	50	17,914
Minera Antamina	2021–2022	Core	79	9,518	29,407	95	ALS Chemex	Yes (15%)	Yes	78	30,239
Minera Antamina	2022–2023	Core	78	14,815	43,435	95	ALS	Yes (13%)	Yes	72	43,102
Minera Antamina	August 15, 2023 to December 31, 2024	Core	134	28,279	84,075	95	ALS	Yes (15%)	Yes	134	84,075
Total			***4,265***	***571,963***	***1,355,884***					***3,994***	***1,317,972***



Note: Figure prepared by Minera Antamina, 2024.

Figure 10-1: Project Drill Collar Location Plan, Drilling Other than Estimation Support



Note: Figure prepared by Minera Antamina, 2024

Figure 10-2: Location Plan, Drilling Supporting Mineral Resource Estimates

Where oxides are noted during logging, such intervals are identified on the core, so that quality assurance/quality control (QA/QC) samples appropriate to oxide material can be added prior to sample dispatch for assay. The insertion of QA/QC samples is based on a pre-set sampling protocol ratio of the number of QA/QC samples to original core samples.

Blastholes are logged for rock type by the ore control geologists and this information is used as input to the geological modelling process for the mining area.

10.4 Recovery

Recoveries from RC drill holes were not recorded.

Core recoveries were included in Table 10-1, and have typically averaged >95% for most programs since 2007.

10.5 Collar Surveys

There is no collar location information available for the Cerro of Pasco drill program. The Centromin drill holes were marked in the field using welded casing, and where located, were re-surveyed in 1997 by a Peruvian surveyor using GPS instruments.

During the Minera Antamina campaigns, drill holes were sited using GPS equipment. Following drill hole completion, the collars were located by mine survey personnel using GPS instruments.

10.6 Downhole Surveys

There is no down hole survey location information available for the Cerro of Pasco drill program. The Centromin drill holes were not down hole surveyed.

Drill holes were down hole surveyed by the drill contractors, with methods varying over time. Early surveys used acid tube measurements; later surveys were completed using Maxibor, Sperry Sun, EZ_Shot, Flexit or gyroscopic tools. Depending on the campaign, survey intervals down hole could range from 5–25 m.

10.7 Grade Control

Blast hole samples are currently used to construct the ore control model for material classification. Approximately 8–9 kg samples are collected from blast hole cones using a hand-held auger.

Blast holes are drilled for drill-and-blast operations in a 6.0 x 7.0 m pattern for soft ore (M1–M2) and a 5.5 x 6.5 m pattern for hard ore (M4B, M5, and M6). A 9.0 x 10.35 m pattern is used for soft waste, while hard waste follows an 8.0 x 9.2 m pattern.

The blast holes are primarily drilled with Bucyrus, Cat, PIH, and PV235 rigs, using a 311 mm hole diameter.

Collar locations are determined using a rig-based GPS system and are reviewed by the survey department before the ore control models are applied.

The maximum drilling slope is set at 10°. The maximum drill depth in ore is 16.5 m, and 17.5 m in waste.

A separate database for ore control is maintained on-site at the mine.

10.8 Sample Length/True Thickness

On average, the true width of the mineralization for the deposit is about 70 m. When using traditional drilling, the mineralization drilling intersection is greater than the true width of the mineralization. However, when using directional drilling, the drilled and true widths are approximately the same.

Example cross-sections showing the relationship of the copper mineralization to the drilling are provided in Figure 10-3 and Figure 10-4.

10.9 Drilling Completed Since Database Close-out Date

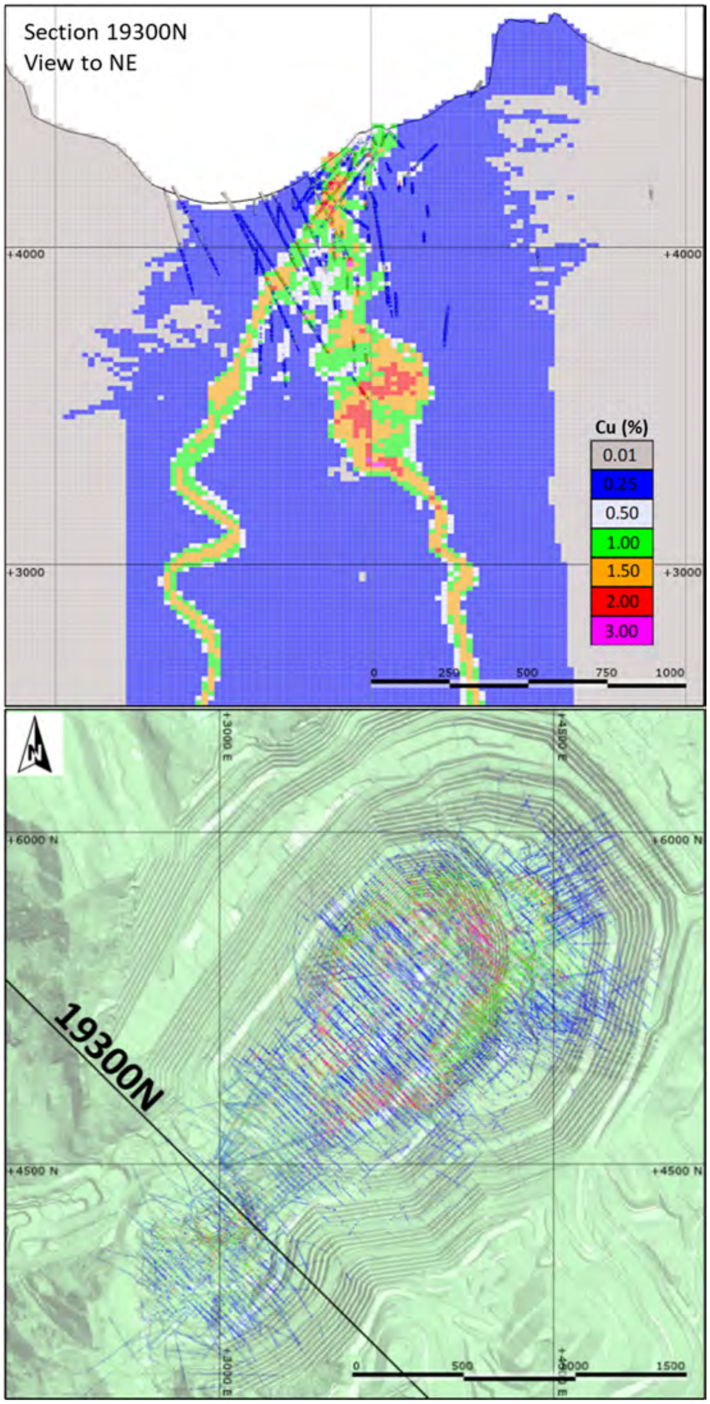
Since the database close-out date, 134 drill holes (84,075 m) have been completed to December 31, 2024.

The Competent Person compared the available drill results to the block model. The drilling results to date confirm approximate mineralization widths as presented in the current Mineral Resource estimate.

10.10 QP Comment on Item 10 "Drilling"

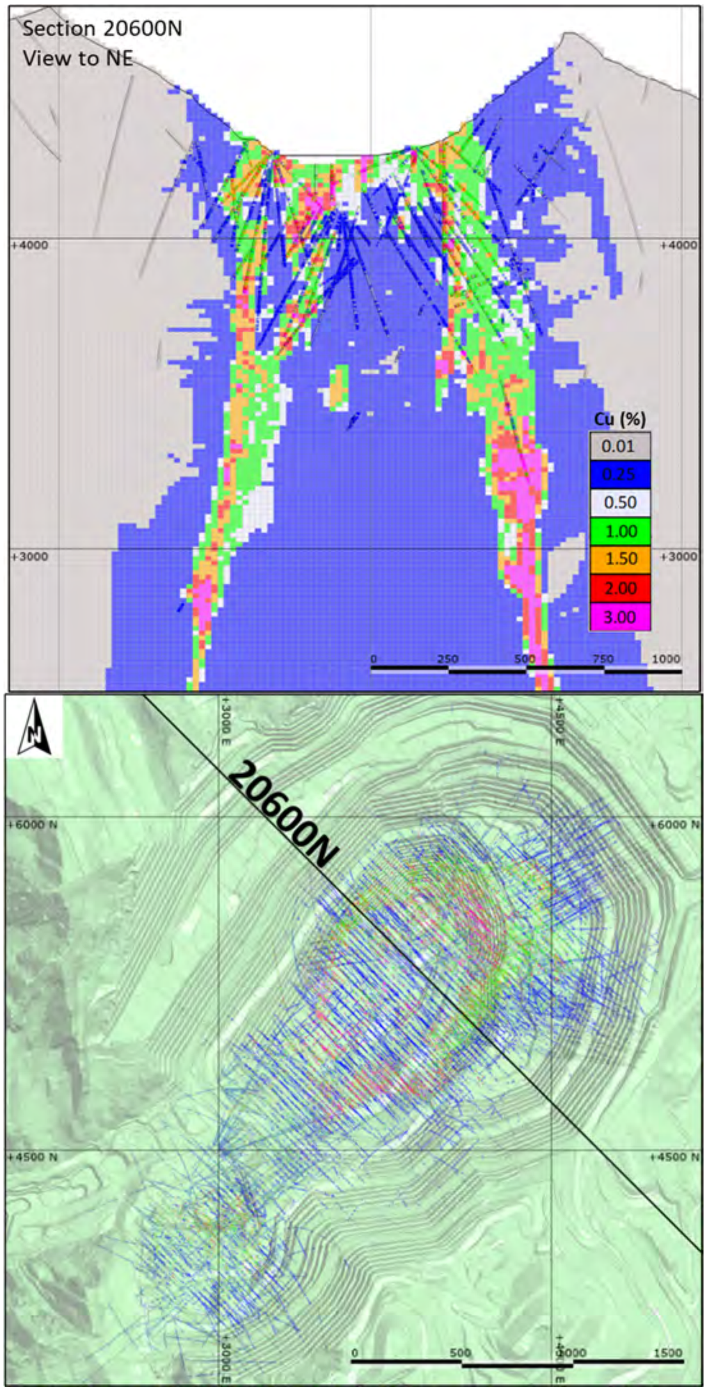
The drilling completed is considered acceptable to support Mineral Resource and Mineral Reserve estimation, and can be used for mine planning.

There are no drilling, sampling, or recovery factors that could materially impact the accuracy and reliability of the results known to the QP other than those discussed in this Report.



Note: Figure prepared by Minera Antamina, 2024.

Figure 10-3: Mineralization Cross-Section, 19300 N



Note: Figure prepared by Minera Antamina, 2024.

Figure 10-4: Mineralization Cross-Section, 20,600N

11 SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1 Sample Methods

11.1.1 RC

Pre-2002 RC holes were sampled every 3 m; RC holes drilled during the 2002–2003 campaign were sampled on 1.5 m intervals.

11.1.2 CORE

Core is split following a crayon line marked by the logging geologist, which serves as a cutting guide to ensure each core half is as representative as possible. The drill core is cut in half using an electrical saw, and sample intervals are set at 3 m length on average; however, the sample length may be adjusted to start and end at major geological (lithology, alteration, mineralization) contacts. All efforts were made to collect samples with a minimum length not less than 1 m, except for bulk density samples, which average approximately 0.15 m in length.

The whole length of the drill hole is sampled, whether mineralized or not, except where low drilling recovery prevents the collection of a representative sample.

11.1.3 GRADE CONTROL

Blast hole samples for submission to the on-site laboratory are collected by the mine geology staff using a hand-held rotary auger.

Cuttings are gathered from a pre-defined pattern within the cone of cuttings produced in the blast holes. In cases with poor recovery, a greater number of insertions is required to achieve a sample weight of 8–9 kg, per sampling protocol.

Blast hole rigs drill a 311 mm diameter hole approximately 17.5 m deep, creating a cone of rock detritus weighing about 2.0–3.5 t around the drill hole. This rock detritus is sampled using an auger bit attached to a battery-powered hand drill. To take an auger sample, the auger is positioned at an angle of 70–80º, roughly three-quarters of the way up the height of the cone. The auger is then slowly inserted into the cone, drilling down to the base. Once the auger reaches the collar, it is carefully pulled out, allowing the detritus material to collect on the spiral. This material is then removed and placed into labelled sampling bags. The sampling protocol requires that 6–10 insertions are taken systematically, and in a representative manner, from the cone. If the cone has adequate recovery, only six insertions are necessary; however, if recovery is poor, up to 10 insertions may be required to achieve the desired 8–9 kg sample weight.

Samples are taken to the geology laboratory, unloaded into two sample trays, dried in the oven, and processed and sent to a site laboratory for analysis.

11.2 Density Determinations

There are 65,238 specific gravity determinations in the Project database, providing adequate coverage of mineralized and waste lithologies. The core data are subdivided into the following major lithological groups: intrusive, endoskarn, exoskarn, breccia, limestone, hornfels, marble, diopsidic marble, and hornfels. The average adjusted specific gravity values for the 21 rock type codes range from 2.45 for intrusive to 3.26 for green exoskarn. In general, exoskarn rock types have higher average specific gravity values than intrusive, endoskarn, and sedimentary rock types.

Most of the data have been collected by Minera Antamina personnel using calliper measurements or wax-coat water immersion. Both methods have been applied to approximately 15 cm long whole-core samples, although some half-core samples were also used. Where both results were available for a single sample, the wax-coat, water-immersion method was considered superior and was selected for use, based on a priority code.

In addition to the routine specific gravity determinations on 15 cm core intervals, specific gravity was also routinely measured on 3 m intervals every 15 m in mineralization and 40 m in unmineralized intervals. These specific gravity measurements were referred to as large-scale bulk-density samples, and were initiated in 2004 as a proxy for in-situ bulk density. These measurements were determined by drying and weighing the core interval, measuring the length and diameter of the core using callipers to get a volume, and dividing the weight by the volume. One or more 15 cm-long wax immersion specific gravity determinations were also performed within the large-scale bulk-density sample intervals. The specific gravity values from the two methods were used to determine correction factors for each rock type to account for the fracturing and disaggregation of the core over longer intervals. All adjustments were negative (resulted in a lower specific gravity) and ranged from <1.0–5.0%. Endoskarn and exoskarn rock types had the largest adjustments.

A 2003 study concluded that the wax immersion and calliper datasets were equivalent.

11.3 Analytical and Test Laboratories

A number of analytical and test laboratories have been used over time. Where known, they are summarized in Table 11-1.

11.4 Sample Preparation

Sample preparation methods varied by laboratory, and are summarized in Table 11-2.

11.5 Analysis

Analytical methods varied by laboratory, and are summarized in Table 11-3.

Table 11-1: Analytical and Test Laboratories

Laboratory	Years Used	Purpose	Accreditation	Independent	Analyses Used in Estimation
SGS, Lima, Peru	1996–2000; April 2018 until July 2021	Primary laboratory; sample preparation, analysis (Cu, Zn, Mo, Ag)	ISO9001:2000	Yes	Yes
	2000–2012	Check laboratory			
Chemex, Vancouver, Canada	1996–1997	Check laboratory	Unknown	Yes	
XRAL, Toronto, Canada	1996–2000	Primary laboratory; sample preparation, analysis (ICP)	Unknown	Yes	
Cone Geochemical Inc., Colorado	1996–1997	Check laboratory	Unknown	Yes	
ALS Chemex, Lima, Peru	2002–2018 August 2021–2024	Primary laboratory; sample preparation, analysis	ISO 9001 (2000) ANSI/ASQ Q 9001:2000	Yes	Yes
Minera Antamina	Not used in Mineral Resource estimation	Blast hole; sample preparation, analysis	Not accredited	No	
Acme Analytical	Not recorded in database	Check laboratory	BS EN ISO 9001:2000	Yes	
Actlabs Skyline Peru S.A.C.	Not recorded in database	Check laboratory	ISO 9001:2000	Yes	
Genalysis Laboratory Services Pty Ltd	Not recorded in database	Check laboratory	ISO/IEC 17025 (1999), NATA Accreditation No 3244	Yes	
Ultra Trace Pty Ltd	Not recorded in database	Check laboratory	ISO/IEC 17025 (2001) NATA Accreditation No 14492	Yes	
Certimin S.A	2012–2017	Check laboratory	ISO9001:2008; ISO/IEC 17025:2006	Yes	Yes
Inspectorate Services Peru S.A.C	2016–2024	Check laboratory	ISO/IEC 17025:2006	Yes	Yes

Note: ICP = inductively coupled plasma

Table 11-2: Sample Preparation Methods

Laboratory	Years Used	Sample Preparation Method
SGS and Chemex, Lima	1996–1998	Crush half-core sample to ~75% passing -10 mesh. Split -10 mesh fraction to ~250 g. Pulverize to ~95% passing -150 mesh.
SGS and Chemex, Lima	1999–2001	Crush half-core sample to ~75% passing -10 mesh. Split -10 mesh fraction to ~5 kg. Pulverize to ~95% passing -150 mesh.
ALS Chemex, Lima	2002–2006	Crush half-core sample to ~90% passing -10 mesh. Split -10 mesh fraction to ~2.5 kg. Pulverize to ~95% passing -150 mesh.
ALS Chemex, Lima	2007–2015	Crush to better than 90% passing -2 mm. Split of 1 kg using a Jones splitter. Pulverize to better than 95% passing -0.106 mm.
ALS Chemex, Lima	2016–2017	Crush to better than 90% passing -2.0 mm. Split of 0.5 kg using a rotary splitter. Pulverize to better than 85% passing -75 µm.
SGS, Lima	2018–2021	Crush to better than 70% passing -2.0 mm. Split of 0.5 kg using a rotary splitter. Pulverize to better than 95% passing -0.106 mm.
ALS Global, Lima	2022–2024	Crush to better than 70% passing -2.0 mm. Split of 0.5 kg using a rotary splitter. Pulverize to better than 95% passing -0.106 mm.

Table 11-3: Analytical Methods

Laboratory	Years Used	Analytical Methods and Elements Assayed
SGS, Lima	1996–2000	Aqua regia digestion, Cu, Zn, Mo, and Ag by AA (ICP: Ag, Al, As, Ba, B, Be, Bi, Ca, Cd, Co, Cr, Cu, Fe, Ga, Hg, K, La, Mg, Mn, Mo, Na, Ni, P, Pb, S, Sb, Sc, Sr, Ti, Tl, U, V, W, Zn)
XRAL	1996–2000	ICP-AES (ICP: Ag, Al, As, Ba, Bi, Ca, Cd, Co, Cr, Cu, Fe, K, Mg, Mn, Mo, Na, Ni, P, Pb, Sb, Sn, Sp, Sr, Ti, V, W, Y, Zn, Zr)
ALS Chemex, Lima	2002–2003, 2004, 2005–2006, 2010–2018	Aqua regia digestion, Cu, Zn, Mo, and Ag by AA (ICP: Ag, Al, As, Ba, B, Be, Bi, Ca, Cd, Co, Cr, Cu, Fe, Ga, Hg, K, La, Mg, Mn, Mo, Na, Ni, P, Pb, S, Sb, Sc, Sr, Ti, Tl, U, V, W, Zn)
ALS Chemex, Vancouver	2002–2003, 2004, 2005–2006, 2010–2018	ICP-AES; Bi by ICP-MS (ICP: Ag, Al, As, Ba, B, Be, Bi, Ca, Cd, Co, Cr, Cu, Fe, Ga, Hg, K, La, Mg, Mn, Mo, Na, Ni, P, Pb, S, Sb, Sc, Sr, Th, Ti, Tl, U, V, W, Zn)
SGS, Lima	2018–2021	ICP-AES; Bi by ICP-MS (ICP: Ag, Al, As, Ba, Be, Bi, Ca, Cd, Co, Cr, Cu, Fe, Ga, Hg, K, La, Li, Mg, Mn, Mo, Na, Nb, Ni, P, Pb, S, Sb, Sc, Se, Sn, Sr, Te, Ti, Tl, V, W, Zn)
ALS Chemex, Vancouver	2022–2024	ICP-AES; Bi by ICP-MS (ICP: Ag, Al, As, B, Be, Bi, Ca, Cd, Co, Cr, Cu, Fe, Ga, Hg, K, La, Li, Mg, Mn, Mo, Na, Ni, P, Pb, S, Sb, Sc, Sr, Th, Ti, Tl, U, V, W, Zn)

Note: AA = atomic absorption; ICP = inductively coupled plasma, AES = atomic emission spectroscopy, MS = mass spectrometry

Current analytical methods are:

- Sulphide: assayed by ICP atomic emission spectroscopy (AES) with aqua-regia digestion for a suite of 34 elements (including arsenic). Copper, lead, zinc, silver, molybdenum, and lead are analysed using atomic absorption (AA) when overlimit results are returned. Bismuth is assayed using HF–HNO_3–$HClO_4$ digestion, followed by HCl leaching and an ICP mass spectrometer (MS) reading;
- Oxide: sodium cyanide-soluble copper (CuCN), ammonium acetate-soluble copper (CuAC), and zinc (ZnAC). All determinations were completed using an AA finish.

Over time, lead was analysed by either ICP-AES or AA methods, depending on the lead grade.

Prior to the 2002–2003 drilling campaign, five methods were used for bismuth analyses; however, some of these methods were not accurate for bismuth grades <70 ppm Bi. An additional concern was that some analytical methods returned slightly elevated bismuth values with higher-grade copper assays. Correction equations were developed, and a priority scheme was created to select the bismuth assays to be used from the pre-2002 drill campaigns.

Starting with the 2002 drilling campaign, samples were assayed for partially-soluble copper and zinc, using acetate-soluble copper and zinc, and cyanide-soluble copper methods. The soluble copper–zinc datasets are less than one-tenth of their respective total metals' datasets. Acetate-soluble copper and zinc are indicators of copper and zinc oxidized by weathering. Cyanide-soluble copper is an indicator of cyanide consumption during ore processing at the mill. The partial-soluble methods are important in the Mineral Resource and Mineral Reserve estimation process since they help forecast metallurgical performance for the copper and zinc concentrates.

11.6 Quality Assurance and Quality Control

Quality assurance and quality control (QA/QC) programs from 2003 onward included submission of blank, standard reference materials (standards) and duplicates (core, coarse-reject, and pulp duplicates). Sieve checks were completed during some campaigns, initially on the -150-mesh fraction, and in later campaigns on the crusher and pulveriser products. Check assays at an umpire laboratory were performed from 2004 onward. The current control sample insertion frequencies are summarized in Table 11-4.

- Currently, all QA/QC data are reviewed by a QA/QC geological specialist. Review criteria include:
- Duplicates: failure rate should not exceed 10% of all pairs;
- Standards: good: between -5% and +5%; questionable, with care: from -5% to -10% or from +5 to +10%; unacceptable: below -10% or above 10%;
- Blanks: significant contamination assumed if blank value exceeds a safe limit of three to five times the practical detection limit. The contamination rate is calculated by determining the proportion of possibly contaminated blanks, and significant contamination is assumed if the contamination rate exceeds 5%;
- Between-laboratory checks: based on linear regression comparisons.

Protocols are in place should the QA/QC results show out-of-control results. These can range from acceptance of the sample results to re-assay of partial or full batches. If significant contamination is observed, the laboratory is requested to improve its preparation or assaying procedures.

No significant analytical biases have been noted in the analytical data collected from 2003 to the Report effective date.

11.7 Databases

The Antamina drill hole database is managed using acQuire information management software that is based on Microsoft SQL Server, a relational database system. There are processes and procedures for importing, validating, and exporting drill hole data for use in resource estimation.

11.8 Sample Security

Sample security has relied upon the fact that the samples are always attended to or locked at 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 to make certain that all samples are received by the laboratory.

Table 11-4: Control Sample Insertion Rates

Sample Type	Insertion Frequency (%)
Twin samples*	2
Coarse duplicates*	2
Coarse blanks*	2
Pulp duplicates*	2
Standards	5
Fine blanks*	2
Check samples**	5

* Assayed by the primary laboratory: ** Assayed by the secondary laboratory; included additional control samples (pulp duplicates, standards, and fine blanks)

11.9 Sample Storage

Samples are retained onsite at the operations, in a dedicated 3 ha facility. The majority of the drill core from exploration to the Report effective date is stored within the facility. Core and reject discard campaigns are completed from time to time, and may result in the discard of samples that are from areas that are mined-out or are known to be non-mineralized.

Sample retention is summarized in Table 11-5.

11.10 QP Comment on Item 11 "Sample Preparation, Analyses and Security"

In the opinion of the QPs, the sampling, sample preparation, assaying, and QC procedures are acceptable for Mineral Resource and Mineral Reserve estimation and can be used for mine planning purposes, based on the following:

- Core handling and sample collection are undertaken in accordance with industry-standard practices, with procedures in place to limit potential sample losses and sampling biases;
- Sample preparation is consistent with industry standards, and appropriate for the mineralization style;
- Analytical methods are industry standard, and appropriate for the mineralization style;
- QA/QC measures have been in place since 2003. Insertion rates are in line with industry norms. No significant biases have been identified during reviews of the QA/QC data;
- Bulk-density determination procedures are consistent with industry standards. There are sufficient acceptable determinations to support the bulk-density values used in waste, and oxide- and sulfide-mineralization tonnage estimates;

Table 11-5: Sample Storage and Retention

Sample Type	Note
Core	Half core is kept inside the same box where it is placed when extracted from the drilled hole. For maximum storage order, boxes are assembled and packed on wooden pallets, which allows them to be moved and placed inside storage racks or shelves, which in turn are covered with a roof for better conservation
Coarse rejects	Once the analysis is completed, the material left over from the process is returned to Minera Antamina. This rock fraction has a particle size of approximately 2 mm (ASTM 10 mesh) and weighs approximately 15 kg. These bags and sacks are coded according to the origin of the drilling well and are placed in 5–6 level racks inside closed warehouses and/or on shelves inside hermetic containers within the storage facility.
Fine rejects	Pulverized rock material corresponding to each sample chemical analysis, arranged in coded paper envelopes, and containing 150–300 g in each envelope. The batches of envelopes with samples are, in turn, arranged in cardboard boxes duly labeled and referenced with dimensions of 45 x 15 x 10 cm and an approximate weight of 10 kg. These boxes are also located in 5-level racks within the closed warehouses within the storage facility.

- Collected data are subject to validation, using in-built program triggers that automatically check data on upload to the database. The checks are appropriate and consistent with industry standards;
- Sample security measures are considered acceptable;
- Current sample-storage procedures and storage areas are consistent with industry standards.

12 DATA VERIFICATION

12.1 Internal Data Verification

Minera Antamina staff routinely undertake the following:

- Internal checks are completed as the lithology, grade, down hole survey, and core recovery data are entered into the geological database to ensure compliance with recorded values;
- QA/QC data are reviewed prior to acceptance of each analytical program in the database;
- Modeling inputs, procedures, parameters, and results for the geological and domain models are reviewed prior to estimation;
- Mineable shapes are reviewed against cross-sections through the shapes and the block model coded for resource definition criteria for domain and cut-off value.

12.2 External Data Verification

Numerous external audits or data collection supervision have been undertaken since 2004, as summarized in Table 12-1.

12.3 Verification by Qualified Persons

12.3.1 MR. LUCIO CANCHIS

Mr. Canchis works at the Antamina Operations as described in Section 2.4.1.

Mr. Canchis completed reviews of the following data and data collection procedures:

- Exploration drill hole data collection methodologies;
- Data upload to database;
- Results of historical and current QA/QC programs;
- Modeling inputs, procedures, parameters, and results for the geological and domain models;
- Development of the constraints and assumptions that were used in assessing reasonable prospects of eventual economic extraction;
- Reconciliation of production to models.

He also reviewed the findings of independent audits and reviews as part of the data verification process.

In the QP’s opinion, the completed verification supports the use of the data in Mineral Resource estimation.

Table 12-1: External Data Verification

Year	Company	Work Undertaken
2004	AMEC	Design and supervision of QC program
2005–2008	AMEC	Design and supervision of QC program
	CRM SA	Audit of QC program
2010–2011	AMEC	Design and supervision of QC program
	CRM SA	Audit of QC program
2011–2012	AMEC	Design and supervision of QC program
	Golder Associates	Audit of QC program
2012–2016	Amec Foster Wheeler	Design and supervision of QC program
	Geosystem International Inc.	Audit of QC program
2017–2021	SRK	Audit of QC program
2020	CSA Global Pty. Ltd.	Audited mine plan and Mineral Reserves

12.3.2 MR. FERNANDO ANGELES

Mr. Angeles works at the Antamina Operations as described in Section 2.4.2.

Mr. Angeles completed reviews of the following aspects of the Mineral Reserve estimates:

- Information provided by geotechnical experts on recommended pit slope angles and geotechnical considerations;
- Block model and selective mining unit sizes;
- Cut-off criteria;
- Pit optimization inputs and selection of optimal pit shell;
- Reconciliation data;
- Mine designs;
- Mine scheduling and planning;
- Infrastructure requirements and capacities;
- Permits;
- Productivity and economic calculations.

In the QP’s opinion, the completed verification supports the use of the data in Mineral Reserve estimation, the life-of-mine (LOM) plan, and in the economic analysis that supports the Mineral Reserves.

12.3.3 MR. HERNANDO VALDIVIA

Mr. Valdivia works at the Antamina Operations as described in Section 2.4.3.

He has participated in:

- Developing methods to optimize the processing of porphyry copper–molybdenum ores;
- Developing interpretations of the lithological influence in the quality control of the ore feed to the concentrator;
- Developing geometallurgical domains and evaluating the influence of those domains on plant performance;
- Optimizing semi-autogenous grind (SAG) grinding in inverse closed circuit with ball mills;
- Optimizing copper–zinc flotation and lead–bismuth and molybdenum separation.

Mr. Valdivia also manages and is responsible for implementation of new technologies in mineral processing, an internal auditor for the ISO 14001 environmental management system, and the OHSAS 18001 Health and Safety Area Manager.

In the QP's opinion, the completed verification supports the use of the metallurgical and process data and supporting assumptions in Mineral Resource and Mineral Reserve estimation and in the economic analysis that supports the Mineral Reserves.

12.3.4 MR. CARLOS AGUIRRE

Mr. Aguirre works at the Antamina Operations as described in Section 2.4.4.

He participates in and reviews specialized studies and design criteria for mine components, such as the open pit and mine WRSFs. He engages with the Engineer of Record by reviewing long-term pit designs, verifying the actual pit conforms to the design criteria, and provide quality assurance on the geotechnical logging from drill hole databases.

Mr. Aguirre also manages independent reviews to ensure mine components are performing as designed, periodically conducting site visits for this purpose. He provides operational expertise in relation to the geotechnical, hydrogeological, and related aspects of the Antamina operations, including WRSFs and TSF.

In the QP's opinion, the completed verification supports the use of the geotechnical and hydrogeological data and supporting assumptions in Mineral Resource and Mineral Reserve estimation, LOM plan, and in the economic analysis that supports the Mineral Reserves.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

A number of metallurgical testwork facilities and metallurgical experts have been involved with metallurgical testwork and interpretation of that testwork over the Project history, including Lakefield Research Limited, G and T Metallurgical Services Ltd., A.R. MacPherson Consultants Ltd., AMTEL, SGS Canada Inc., Larox, Inc., Jenike and Johanson Ltd., Fuller Company, and Pocock Industrial Inc.

13.2 Metallurgical Testwork

13.2.1 HISTORICAL TESTWORK

Testwork included comminution, mineralogy, pilot and bench scale flotation, dewatering, filtration, and coarse ore and concentrate flow characteristics tests. This testwork was used to support flowsheet design and process plant construction and operation.

The plant has been operational since 2001.

13.2.2 2015 PROGRAM

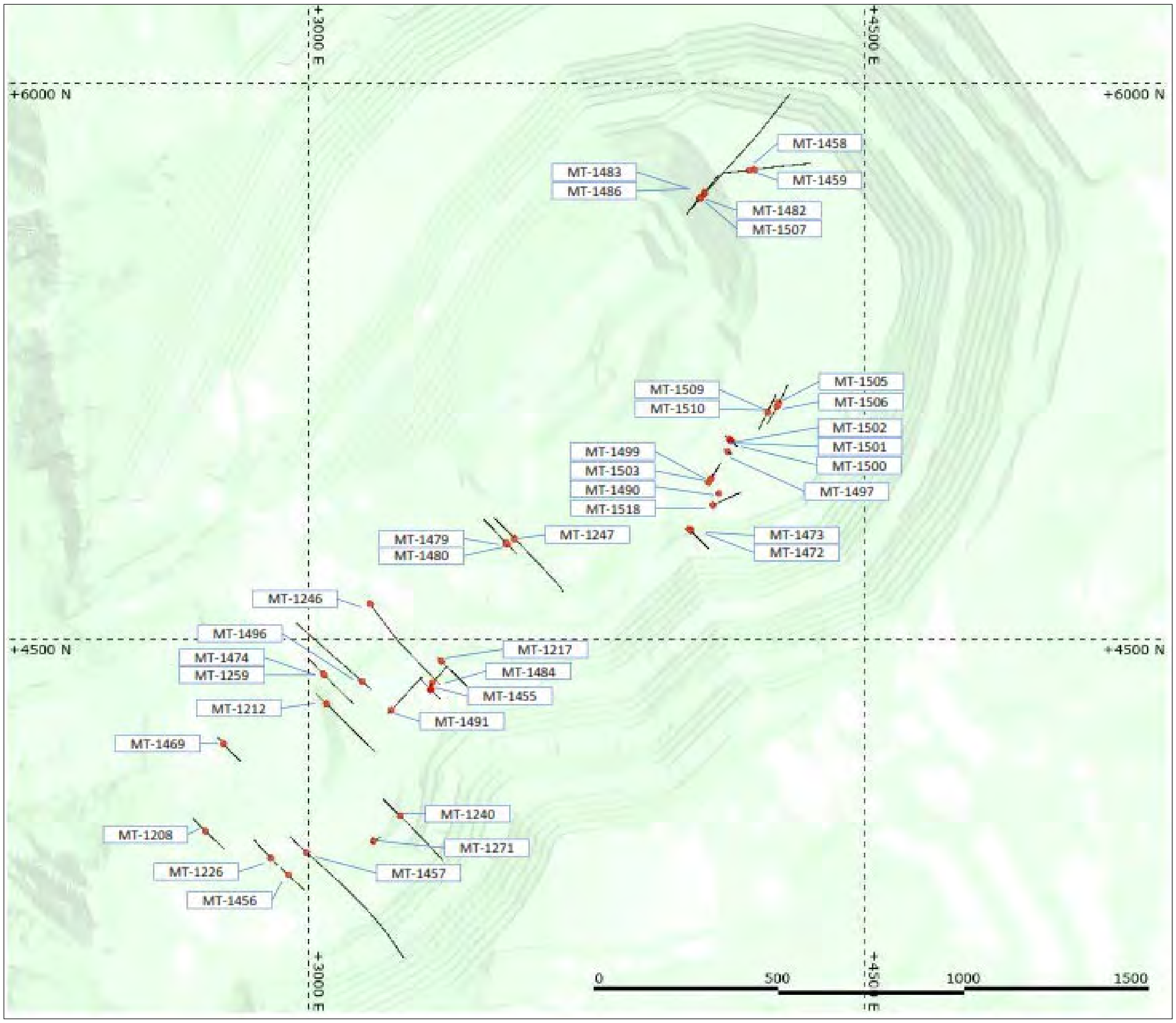
During 2015, two metallurgical programs were completed on material at depth within the proposed open pit outline, consisting of 67 copper–molybdenum and 54 copper–zinc samples (Figure 13-1).

Samples were classified into five geometallurgical types, M1, M2, M2A, M2AT and M4B (see Table 7-5 for an explanation of the geometallurgical types).

From these 121 samples, 39 samples represented mineralization that would be mined and processed during 2023–2028. Tests completed included comminution and flotation (Table 13-1).

The copper–zinc samples displayed a wide range of head grades, and there was an apparent trend of higher recovery with increasing head grade. Such a wide range would not be expected during plant operations. Copper rougher concentrate grades were very high for rougher flotation with copper grades ranging from 14–27%. Harder pulling of the froth, and longer residence times and/or adjustments in reagent doses in the laboratory tests would likely have significantly increased the recoveries. The zinc rougher concentrates showed a similar trend, with zinc grades ranging from 13–50%.

The copper–molybdenum samples showed less variation in head grades, and there was no clear head grade versus recovery trend. Each sample set had one poor result; however, no re-testing was carried out in the laboratory to determine the cause. As with the copper–zinc samples, the rougher concentrate copper grades were high, in the range of 7–27%. This indicated that harder pulling of the froth, longer residence times, and/or adjustments to reagent doses would have resulted in higher recoveries.



Note: Figure prepared by Minera Antamina, 2023.

Figure 13-1: Metallurgical Sample Location Plan, 2015 Testwork

Table 13-1: 2015 Testwork

Geometallurgical Type/Number of Samples Tested	Testwork Type	Note
M1, M2, M2A, M2AT, M4B	Mineralogy	The main copper mineral is chalcopyrite. The main sulphide gangue mineral is pyrite, which occurs in a range of concentrations, but is higher in mineral type M4B than in previous years. This can be controlled with reagents, but some blending may be required. Silicates are present in varying concentrations and include andradite, granites, diopsides, quartz, fluor-vesuvianite, orthoclase, plagioclase, chlorite, micas, and clays. Wollastonite is present in high concentrations in geometallurgical types M5 and M6. Blending may be required to avoid processing problems. Low carbonate concentrations are present as calcite. Low concentrations of oxide minerals are present as hematite, magnetite, and lime. Fluorite is present in low concentrations except for geometallurgical type M5; this will require monitoring to avoid smelter penalties
49	Bond ball mill work index (BWi) A*b Drop weight index (DWi)	Mineralization classified as soft to very soft for SAG milling. Mineralization classified as medium hardness for ball milling. The grinding circuit is extremely flexible and has sufficient grinding power to cope with the ore hardness variations expected.
39	Flotation	M4B, M4BT Variable Zn and Cu head grades. Some samples showed elevated cyanide-soluble Cu. Elevated Zn reported to the Cu concentrate. In plant practice much of the Zn in the Cu rougher concentrate would be rejected in the Cu cleaner stage and returned to the copper stage tailings for recovery in the Zn circuit.
		Geometallurgical types M1, M2A, M2AT, M5. No obvious relationship between Cu head grade and recovery. Generally acceptable Cu recoveries.

Twenty years of operational and metallurgical data have been collected on the current process plant operation. The historical copper recovery is 87% for all geometallurgical types. The highest recoveries are with geometallurgical type M1 (average 93.1%).

The 2023–2028 sample set results were compared to the historical performance (Table 13-2). Some of the 2015 geometallurgical types are not included in Table 13-2 due to insufficient test samples to provide meaningful results.

13.2.3 2023 PROGRAM

A variability program was undertaken in 2023. A total of 34 variability samples and four composites were collected in 2022 (see locations in Figure 13-2), and sent to SGS for metallurgical testing. Samples were representative of the main geometallurgical types proposed to be mined from 2023–2036, focusing on the 2023–2028 period, and consisted of 24 copper–molybdenum and 10 copper–zinc samples. Samples were classified as geometallurgical types M1, M2, M2A and M4B (see Table 7-5). Samples were subject to comminution, rougher and cleaner flotation testwork. The tests completed are summarized in Table 13-3. Cleaner and rougher recovery results are provided in Table 13-4. The 2023–2036 sample set results were compared to the historical performance (Table 13-5).

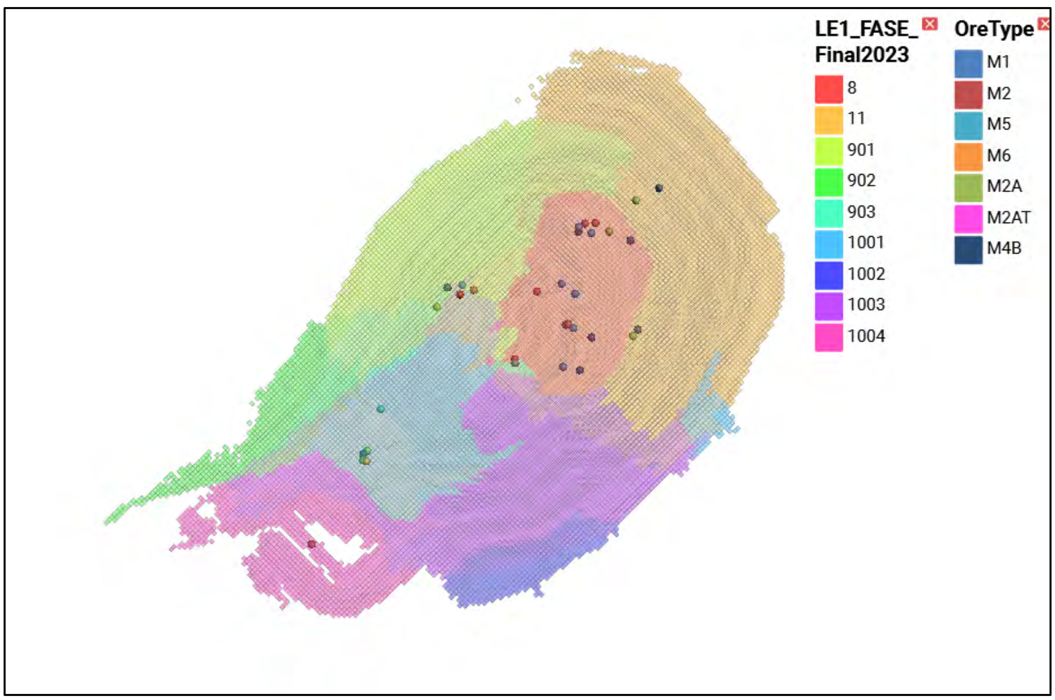
Model recoveries were plotted against those obtained from the 2023 testwork. The results indicate that the copper recoveries from the LOM2024 model are somehow overestimated when compared with copper recoveries from the variability and composite samples. A poor recovery correlation was noted between the zinc LOM2024 Model and the zinc global variability test results. This poor correlation may be a result of the equations used for the LOM2024 model, which represent plant production from the blending of three or four ore sources; such that the equations are the product of a stable process, while the variability samples represent pure unmixed ore types. Composite results showed a better correlation with the model than the variability samples.

A QEMSCAN mineralogical analysis was completed, which identified the following:

- The main copper mineral is chalcopyrite;
- The main sulphide gangue mineral is pyrite, which occurs in a range of concentrations but is higher in geometallurgical type M4B than in previous years. This can be controlled with reagents, but some blending may be required;
- Silicates are present in varying concentrations and include andradite, granites, diopsides, quartz, fluor-vesuvianite, orthoclase, plagioclase, chlorite, micas, and clays. Wollastonite, a calcium–silica–oxide mineral, is present in high concentrations in geometallurgical types M5 and M6 in the future. Blending may be required to avoid processing problems;
- Low carbonate concentrations are present as calcite;
- Low concentrations of oxide minerals are present as hematite, magnetite, and lime;
- Fluorite is present in low concentrations except for geometallurgical type M5; this will require monitoring to avoid smelter penalties.

Table 13-2: 2023–2028 Sample Set Results Compared to Historical Performance

Geometallurgical Type	Note
M1	The average test work Cu head grade is within the range experienced in the last five years and within the predicted grade; the average test work rougher Cu recovery to Cu concentrate is slightly lower than the recoveries experienced and predicted.
M2A	The average test work Cu head grade is higher than the range experienced in the last five years and the predicted grade; the average test work rougher Cu recovery to Cu concentrate is lower than the recoveries experienced and predicted
M4B	The average test work Cu head grade is higher than that experienced in the last five years but within the predicted grade; the average test work rougher Cu recovery to Cu concentrate is within the recoveries experienced and predicted. The average test work Zn head grade is higher than that experienced in the last five years and the predicted grade; the average test work Zn rougher recovery to Zn concentrate is lower than the recoveries experienced and predicted (for Zn the test work circuit does not match the plant circuit)
M5	The average test work Cu head grade is within the range experienced in the last five years and within the predicted grade; the average test work rougher Cu recovery to Cu concentrate is lower than the recoveries experienced and within the predicted range of recoveries. For this ore type there were only two years of previous performance



Note: Figure prepared by Minera Antamina, 2023. Fase = phase.

Figure 13-2: 2023 Variability Program Sample Locations

Table 13-3: 2023 Testwork

Number of Samples Tested	Tests Conducted	Discussion
34	Bond ball mill work index (BWi) A*b Drop weight index (DWi)	Standard procedures for the Antamina flowsheet were used. Most samples have low DWi, so the samples are in the moderately soft to soft range. M4B was the softest material tested, M2 was the hardest.
34	Rougher flotation	M1: Cu head grades ranged from 0.72–1.50%. No apparent relationship between head grade and recovery. 10 samples tested; one had a poor result at 58% Cu recovery; the remaining nine returned recoveries ranging from 84–95%. M2A: Cu head grades ranged from 0.72–1.50%. Four samples tested; one had a poor result at 17% Cu recovery; the remaining three returned recoveries ranging from 88–96%. M2AT: one sample tested. Cu head grade of 0.60%; copper recovery of 86%. M5: Cu head grades ranged from 0.66–0.86%. Recoveries ranged from 74–96%. Rougher flotation test results for Cu–Zn ore types indicate that Zn head grades and Zn recoveries for Zn concentrate are highly variable. An average of 17% of the Zn was reported into the Cu concentrate and 74% to the Zn concentrate. In the process plant workflow, much of the Zn in the Cu rougher concentrate would be rejected in the Cu cleaner stage and returned to the copper stage tailings for recovery in the Zn circuit. The Cu–Zn ore type M4BT showed a similar degree of variability and, on average, contained six times more CuCN soluble Cu than M4B. However, the average Cu recovery was similar at 82%. The main difference between M4BT and M4B was that the average Zn recovery to the Cu concentrate was 46% in M4BT compared to 17% in M4B. Rejection of Zn in the Cu cleaner circuit also applies to M4BT.

Table 13-4: Cleaner and Rougher Recovery Results, 2023 Testwork

Sample Type	Area	Geometallurgical Domain	Recovery (%)
Variability	Cu rougher	M1, M2	>94
		M2A	80
		M4	87
	Cu cleaner	M1, M2	97
		M2A, M4	95
	Mo rougher	M1, M2	75–90
	Mo cleaner	M1, M2	9
	Zn losses to Cu concentrate		33–40
Composites	Cu rougher	M1, M2	>93
		M2A, M4B	88
	Cu cleaner	M1, M2	>97
		M2A, M4B	95
	Mo rougher	M1, M2	75–90
	Mo cleaner	M1, M2	95
	Zn losses to Cu concentrate		33–40

Table 13-5: 2023–2036 Sample Set Results Compared to Historical Performance

Geometallurgical Type	Note
M1	For the variability samples, three points show model overestimation by +5% or more, one on the line and one at -5%; the composite is +2%; this shows a low correlation for the variability samples with a tendency for the model to overestimate recoveries, lower recoveries are related to higher pyrite contents and lower Cu:Zn ratios. The composite result is within a reasonable limit.
M2	For the variability samples, four points within ± 1% of the equilibrium line, one at -5%, one at +2%, and one at +3%; the composite result at +1% agrees with the model. Overall, the correlation is reasonable for M2.
M2A	For the variability samples, three points are at +4%, one at +3%, and one at -5%; the composite result is at +4%, and overall, the correlation is moderate.
M4B	For the variability samples, four points between 1% and 4% above the equilibrium line, one at +5%, and 3 between 2% and 3% below the line; the composite result is at +4%. Overall, the correlation is moderate.

13.3 Recovery Estimates

The correlations between head grade and recovery are well understood by process plant staff. Using the predictive recovery equations, annual metal production targets can be calculated from the production plans.

The copper and zinc metal production represents 90% of total revenues and these show very good correlation between actual versus predicted recoveries, based on ore type.

Table 13-6 shows the average actual recoveries for 2020–2023. Table 13-7 shows the average metallurgical recoveries by geometallurgical type used for the Mineral Resource and Mineral Reserve estimates from 2025–2036. These recoveries are calculated using the predictive equations to correlate head grades and metallurgical recoveries.

The metallurgical equations results used for Mineral Reserve estimates were compared to the actual metallurgical recoveries over the last three years, and the ratios were found to be reasonable.

13.4 Metallurgical Variability

Samples selected for testing were representative of the various types and styles of mineralization. Samples were selected from a range of depths within the deposit.

Sufficient samples were taken so that tests were performed on sufficient sample mass, including individual tests to assess variability.

13.5 Deleterious Elements

The major deleterious elements are bismuth and arsenic. These are managed using campaigning. Geometallurgical types M1 and M4B are campaign-treated to produce high value, high quality copper and zinc concentrates.

Geometallurgical types M2 and M2A are campaign-treated to produce separate copper concentrates with higher bismuth and arsenic grades. While there are processes within the mining industry that can be used to reduce bismuth and arsenic grades in concentrates, installation of these at the Antamina process plant would not be cost effective, due to the relatively small amount of copper concentrates with elevated bismuth and arsenic contents produced.

Geometallurgical types M5 and M6 are always processed separately from the M1 and M4B geometallurgical types to minimize contamination of the higher-grade copper concentrates with the bismuth and arsenic contents associated with the bornite ore.

Concentrates are produced and transported as separate batches to the port, where port storage is designed to keep the concentrate batches separate. The concentrate storage shed is divided into 18 different areas; 14 are reserved for copper concentrate, one for bornite concentrate, and three for zinc concentrate. Concentrate blending can be completed at the port by mixing concentrate qualities to meet concentrate specifications as required.

Table 13-6: Plant Metallurgical Recoveries, 2022–2024

Geometallurgical Type	Recovered Copper to Copper Concentrate (%)	Recovered Zinc to Zinc Concentrate (%)	Recovered Silver to Copper Concentrate (%)	Recovered Molybdenum to Molybdenum Concentrate (%)
M1, M2	92.56	—	85.36	44.98
M2A	84.99	—	74.62	—
M5	—	—	—	—
M6	84.62	75.38	77.10	—
M4B	83.86	85.80	63.77	—
Total	***88.81***	***85.80***	***73.12***	***42.70***

Table 13-7: Forecast Metallurgical Recoveries, 2025–2036

Geometallurgical Type	Recovered Copper to Copper Concentrate (%)	Recovered Zinc to Zinc Concentrate (%)	Recovered Silver to Copper Concentrate (%)	Recovered Molybdenum to Molybdenum Concentrate (%)
M1–M2	93.40	—	84.50	53.64
M2A	89.80	—	73.13	57.62
M2AT	90.17	—	63.66	26.12
M5	79.95		64.29	—
M6	73.72	81.25	67.37	
M4B	84.21	83.70	55.17	
M4BT	78.99	72.48	41.03	
Total	***89.72***	***70.27***	***70.14***	***45.59***

14 MINERAL RESOURCE ESTIMATES

14.1 Introduction

The database supporting the Mineral Resource estimates was closed as at August 15, 2023.

Software packages used in the estimation workflow include Leapfrog and Leapfrog Edge, Isatis Neo, and Supervisor.

For the Mineral Resources potentially amenable to open pit mining methods, grades were initially estimated into 20 x 20 x 15 m blocks, and then reblocked into 5 x 5 x 7.5 m sub-blocks that represent a volume of 187.5 m^3, or 3.125% of the volume of the 20 x 20 x 15 m blocks. There is a total of 32 of these sub-blocks, which are used for improving volumetrics, contained in each 20 x 20 x 15 m block. Grade estimation for all elements is referred to the centroid of the 20 x 20 x 15 m blocks.

For the Mineral Resources potentially amenable to underground mining methods, the 20 x 20 x 15 m blocks were re-blocked to 10 m wide x 10 m long x 15 m high.

14.2 Modelling Approach

14.2.1 GRADE TRENDS

There are broad, deposit-wide, grade trends throughout the Antamina deposit. Most noticeable of these trends are the increase in copper grades and decrease in zinc grades when moving from the skarn–marble contact toward the centre of the deposit. In addition to these deposit-wide trends, more localized trends are associated with the northwest-trending Oscarina feature and the Antamina Main fold axis.

To reduce the variability associated with strong grade trends, the skarn units are divided first by host lithology (endoskarn vs exoskarn) and then by alteration minerals (generally the type of garnet observed). This modelled lithological control defines pseudo-stationary estimation domains; however, both strong grade trends and, in some cases, zones of strongly-elevated grade remain for some elements in some modelled lithological units.

A conditional simulation was completed to assess the variability of the geological contacts between skarn–marble, endoskarn versus exoskarn and porphyry versus endoskarn.

14.2.2 DETERMINISTIC MODELS

Deterministic models were used for lithological and estimation units using both drill hole and blast hole data. Deterministic grade shells were defined for bismuth, arsenic, and zinc (high/low zoning). The grade thresholds used to define the grade shells were 25 ppm, 70 ppm and 2% respectively for bismuth, arsenic, and zinc. The grade shell models did not try to capture the complete detail seen in the grade contours. There was mixing of material above and below cut-off near the contours. This was particularly true for arsenic.

Deterministic models were also developed for the bornite and bornite transition zones. Models of elevated ammonia acetate soluble copper and zinc were prepared to control the estimates of acetate soluble copper and zinc as well as cyanide soluble copper.

14.2.3 GEOLOGICAL MODELS

The geological model is based on geological logging of core drill holes with local support from previous interpretations and includes blast hole information that is used to help identifying large lithological trends.

A total of 21 lithology and breccia wireframes were constructed.

The final wireframes were validated by visual inspection to check spatial continuity, misclassification (interpreted) wireframes vs. logging information, volumetric comparisons for previous models, nearest-neighbour models to check for volumetric biases, and geology reconciliation in mined areas using previous geological models.

Two pre-mineralization faults were interpreted and modelled.

14.3 Models Constructed

The overall model contains a total of 21 separate lithological units. Table 14-1 summarizes the individual models that were constructed.

For some elements (arsenic and bismuth) additional control is provided through the use of grade shells. The grade shell models created to supplement the lithological models were:

- Bismuth: 25 ppm;
- Arsenic: 70 ppm;
- Zinc: 2%.

For soluble copper and zinc, mineralogical zones were modelled and used to control estimation.

14.4 Exploratory Data Analysis

Exploratory data analysis consisted of the construction of tables of statistics by lithological units and estimation domains, swath plots, contact plots, high-grade local capping, and experimental variograms of composite grades for total copper, total zinc, ammonium acetate-soluble copper (CuAC), sodium cyanide-soluble copper (CuCN), ammonium acetate-soluble zinc (ZnAC), molybdenum, silver, arsenic, bismuth, iron, lead, sulphur, calcium, mercury, cadmium, antimony and bulk density in each estimation domain.

The results were used to guide the zone definition, domain selection, and definition of interpolation parameters.

Table 14-1: Models

Model	Note
Copper	Major contributor to net smelter return (NSR) equation. Copper grades vary by lithological unit and, therefore, lithology is a major control on grade.
Zinc	Major contributor to NSR. Higher zinc grades are found in the exoskarn units.
Bismuth	Used to classify the different geometallurgical types. When present in sufficient quantities, bismuth reporting to the copper concentrate results in penalties that reduce NSR and ore value per hour (OVPHR).
Silver	Silver is generally a by-product metal, but, depending on commodity prices, can be a major contributor to the NSR and OVPHR cut-offs. Silver grades are generally controlled by lithology.
Molybdenum	Considered to be a by-product metal.
Arsenic	Arsenic is a deleterious element in copper and zinc smelting. It occurs at high grades in the wollastonite exoskarns and in areas where it coincides with high economic metal grades.
Lead	Lead grades are controlled by lithology.
Ammonia acetate-soluble copper and zinc	Soluble copper and zinc are not recoverable by the flotation circuit and the values of these metals must be discounted during economic evaluations.
Iron	Iron grades are controlled by lithology.
Cyanide soluble copper and bornite	The presence of cyanide soluble copper is used as a bornite indicator.
Density	A density downgrade factor is calculated and applied to the composite data prior to estimation. The downgrade factor varies by lithology and degree of fracturing (fracture frequency index). Within the waste lithologies, density values decrease with distance from the contact with the mineralized units. This trend in density values is modelled using distance class estimation groups. A similar trend is observed in the endoskarn units approaching the exoskarn. This trend was modelled using distance classes.
Waste	When waste rock is reactive or potentially reactive it must be separated and cannot be used for TSF construction. Waste rock types A and B are defined from the union of the following solid models and grades. • Blocks within 80 m of the skarn; • Blocks within 30 m of modelled dikes; • Presence of unit 14 or 19 (primary or secondary logging); • Grades of >400 ppm arsenic; • Grades of >0.07% zinc; • Zones modelled as high-solubility copper or zinc (oxide zones). The remaining waste rock material is considered as potential type C waste, which should have an unconfined compressive strength of >40 MPa and abrasion <60%. The final waste model is thus the result of the intersection between various deterministic models and unconfined compressive strength and abrasion models.

Model	Note
Abrasion and unconfined compressive strength	Used as additional criteria to classify waste as type C
Sulphur	Used to estimate low grades with high confidence in the intrusive lithological unit.
Calcium	A primary component of the sedimentary calcareous lithological units.
RQD	Rock quality model based on RQD data.
Clay	Grouped lithological units are used as estimation domains and indicator kriging was used to estimate this model.
Epidote	Grouped lithological units are used as estimation domains and indicator kriging was used to estimate this model. Located mainly in the endoskarn lithological units. Characteristic pink garnet content.
Pyrite	Grouped lithological units are used as estimation domains and indicator kriging was used to estimate this model. Typically found in the endoskarn lithological units in the north of the deposit.
Magnetite	Grouped lithological units are used as estimation domains and indicator kriging was used to estimate this model. Typically associated with the enriched and brecciated zones lithological units and located in the central deposit area.
Fluorite	Grouped lithological units are used as estimation domains and indicator kriging was used to estimate this model. Associated with the Oscarina area dikes. Can affect concentrate quality.
Silicification	Grouped lithological units are used as estimation domains and indicator kriging was used to estimate this model.
Mercury	Grouped lithological units are used as estimation domains and indicator kriging was used to estimate this model. Higher values localized in the intrusive lithological unit followed by the endoskarn lithological units. Can affect concentrate quality.
Antimony	Grouped lithological units are used as estimation domains and indicator kriging was used to estimate this model. Can affect concentrate quality
Cadmium	Grouped lithological units are used as estimation domains and indicator kriging was used to estimate this model.
Aluminium	Grouped lithological units are used as estimation domains and indicator kriging was used to estimate this model.
Cobalt	Grouped lithological units are used as estimation domains and indicator kriging was used to estimate this model.
Gold	Grouped lithological units are used as estimation domains and indicator kriging was used to estimate this model.

Sharing of samples across model boundaries was limited to elements, such as bismuth and arsenic, which are grade-bound due to the use of a deterministic grade shell, but not in other elements where an increase in grade within the waste units was not desired; and, across distance-based estimation group limits (where failure to share samples would introduce a conditional bias). The zinc model, however, shared samples within 10–50 m distance in some domains based on variography but only where those domains based on variography were within the same lithological domain.

Sample sharing across lithology contacts was generally not permitted.

14.5 Density Assignment

Bulk density estimation was performed using four ordinary kriging (OK) passes and a fifth pass assignment of the average of blocks estimated in pass 4. A one-pass nearest neighbour (NN) estimate was completed for validation purposes.

14.6 Composites

Data were composited to 3 m lengths, and were broken at geological contacts. The minimum composite length was 1.5 m. The maximum composite length was 4.5 m.

14.7 Grade Capping/Outlier Restrictions

Outliers were managed using a combination of volumetric restriction and grade capping. Identified outlier composites were first restricted. The uncapped outlier composites were allowed to participate in the estimation of an element when the block was sufficiently close to the outlier and could be estimated in the first or second estimation passes. For all remaining estimation situations, the outlier grade of the composite was capped.

Capping was performed at various risk thresholds, and the threshold selected for capping was determined and based on the number of capped data. Commonly, the risk level that identified 1–2% of the data as outliers per estimation unit was selected.

The selected risk threshold varies widely across the different rock types due to differences in variability and average grades. Typically, outliers are stronger in low-grade units, where a slightly elevated grade can cause an important change in the local average grade.

After capping, the number of capped composites and the theoretical quantity of metal removed were examined. The capped and uncapped grades were also examined using log-probability plots.

14.8 Variography

A localized geostatistical search (LGS) technique was employed, which uses variogram zones that interact with localized rotations (block to block). Due to geological complexities, modelling global anisotropies were generally not possible during the estimation process. Localized anisotropies were calculated for each geological unit directly from the geological model meshes using a tool implemented in the estimation software.

A localized anisotropy algorithm converted the distance to the surfaces into a 3D property in the block model, which was later used in the estimation as the rotation of the search ellipsoid. The distances were calculated from the cell centroid to the nearest point in the surface file. The nearest point is not necessarily a corner of the triangle; it could be any point on the facet.

The variogram modelling search strategy was based on the localized geostatistical search kriging approach for copper, zinc, molybdenum, silver, lead, and iron.

14.9 Estimation/Interpolation Methods

Grade estimation was systematically performed on the 3 m composites using five passes.

Blocks that were sufficiently close to an outlier value were identified and estimated using uncapped composites within restricted search neighbourhoods.

Other blocks were estimated using capped composites and four passes of OK estimation. A fifth estimation pass corresponded to a moving average estimate. A one pass NN estimate was implemented for validation purposes.

A basic search distance of 75 m was selected based on the typical 50 m spacing between drill lines.

Outlier restriction was used in most models to reduce the estimation risk associated with extreme data values. Outlier values were identified locally, inspecting the impact on the neighbourhood mean.

Localized geostatistical search ordinary kriging using Isatis Neo software was used to estimate the block grades. The Isatis Neo software read in a localized search ellipse orientation (localized search neighbourhood) and a localized variogram for each block to be estimated. Orientations of the localized variogram and the search ellipse did not necessarily need to be the same. Search ellipse dimensions were based upon the shape of the zonal variograms and were oriented using the interpolated rotation angles.

Grade estimation was systematically performed in four OK passes and a fifth-pass assignment of the average of blocks estimated in pass 4. A NN estimate was completed for validation purposes.

Grade estimation was controlled by the following criteria:

- Lithological unit (domain estimations; all models);
- Grade shells (Zn, Bi, and As);
- Distance to the contact between mineralization and waste lithologies (density);
- Distance control to contact (Mo);
- Mineralogical zones (CuAC, ZnAC, and CuCN).

Estimation also considered the following:

- Zinc: the high variability forced the adoption of a deterministic grade shell model to control the estimation. A single grade shell was used to distinguish high- and low-grade zones;
- Copper, silver, lead, molybdenum, and iron: lithological controls on grade were sufficient to define estimation domains;
- Soluble copper and zinc: deterministic models of higher solubility zones, the extent of bornite mineralization, and the bornite transition zone, were used to control the estimation
- Bismuth: the steep grade profiles and thin high-grade zones forced the adoption of a deterministic grade shell to control the estimation. The threshold for high and low bismuth grades was 25 ppm;
- Arsenic: trends in mineralization do not appear to have strong lithological unit or structural controls. At the current scale of drilling, definition of these trends is difficult. A single grade shell was used to distinguish high- and low-grade arsenic zones;
- Density: distance-based estimation classes or groups were used to reproduce observed spatial grade trends.

In each pass, the estimation was performed per estimation unit (lithology + grade shell or distance class where appropriate). Minimal sample sharing between estimation units was allowed. The orientation of the search ellipsoid and the anisotropy ratio were defined by the variogram model for search passes 1 and 2. Estimation pass criteria are summarized in Table 14-2.

Blocks not estimated in passes 1 to 4 were estimated by a moving average methodology using the blocks estimated in pass 4 (by estimation unit and zone).

Aspects of the modelling and estimation process are summarized in Table 14-3.

Specific aspects of block grade estimation included the following:

- Zinc: grades were estimated using deterministic high-grade shell to control the estimation and lithological control;
- Bismuth: steep grade profiles and thin high-grade zones forced the adoption of a deterministic grade shell to control the estimation;
- Molybdenum and bulk density: distance-based estimation classes or groups were used to reproduce observed spatial grade trends. Grade–trend modelling was performed in the intrusive unit and endoskarn units;
- Arsenic: trends in mineralization do not appear to have strong rock-type or structural control. At the current drilling scale, the definition of these trends is difficult. A single-grade shell is used to distinguish high- and low-grade arsenic zones;

Table 14-2: Estimation Passes

Pass Number	Estimation	Major Axis Direction	Distance Along Major Axis (m)	Directions And Distances for the Other Two Directions	Octant	Minimum Number of Composites	Maximum Number of Composites	Maximum Number of Consecutive Empty Octants	Maximum Number of Drill Holes
1	OK	Defined by variogram model	75	Defined by variogram model	Four angular sectors divided by 2	14–18	16–24	1	3
2	OK	Defined by variogram model	150	Defined by variogram model	Four angular sectors divided by 2	14–18	16–24	2	3
3	OK	Fixed search: 250 x 250 x 250 m			Four angular sectors divided by 2	14–18	16–24	2	2
4	OK	Fixed search: 300 x 300 x 300 m			None	14–18	16–24	1	2

Note: OK = ordinary kriging.

Table 14-3: Estimation Approach

Element	Grade Distribution	Search Strategy/Capping	Notes
Copper	Copper grades are usually high within the exoskarns and increase with distance from the waste contact. The brown endoskarn is economically the most important unit for copper due to the high grades (>1%) and large volumes. Low grades are found in the contact between the intrusive and other skarn lithologies. The narrow, enriched brecciation-zone units typically contain strongly elevated copper grades, and material adjacent to the enriched brecciation zones also tends to have elevated copper grades.	Based on a review of grade statistics and trend–direction observations per selected spatial zones of the deposit. The final estimation–search directions were consistent with both the observed grade trend and the variogram model.	Model validation was performed separately for each of the deposit zones. The validations demonstrate that there is a strong match between the average grades and the model, both locally and globally. Visual comparisons of the estimated model against the drill hole data indicated a strong similarity in the two grade sets and indicated that the model was spatially well behaved.
Zinc	Zinc grades in the endoskarn are elevated when the endoskarn is located near the exoskarn. Similarly, the grades of most waste rock units have an elevated zinc content along the exoskarn contact.	Limited sharing of exoskarn data in the estimation of the waste-unit blocks was allowed using distance groups. The limitation on sharing considered the distance of the exoskarn data and the waste block from the contact, the capped grade of the exoskarn composite, and the deposit sector.	Various model runs followed by blast-hole reconciliations were conducted to optimize model parameters. A trend model was constructed to define the final estimated zinc grade.
Bismuth	Two bismuth estimation zones based on a 25 ppm grade shell. The interpreted low-grade and high-grade bismuth domains have good continuity over approximately 850 m. Narrow zones associated with high-grade zinc occur in the west margin of the deposit. In the north area of the main deposit, the high-grade bismuth domain is wider than in the south and central areas.	Bismuth interpolation was constrained by grade domains and rock types.	Outlier restriction was applied in the low-grade domain. Spatial zones with elevated Bi were treated as separate estimation domains.
Silver	Silver tends to increase in grade at the outer edge of the skarn system. Zones of elevated		Drill hole and blast hole data indicate that zones of potentially high-grade vein-type

Element	Grade Distribution	Search Strategy/Capping	Notes
	silver grade are also observed in northeast-oriented bodies near the southeastern exoskarn–endoskarn contact and in the extreme northeastern portion of the deposit.		mineralization may be present. To avoid excessive capping of these grades, defined outliers could be used (using the uncapped grade) in the estimation of three blocks. These model blocks were aligned parallel to the apparent orientation of the high-grade mineralization.
Molybdenum	Molybdenum is found in relatively high concentrations in some parts of the deposit. Molybdenum concentrations are controlled by structures and rock types. Economic concentrations (>0.02% Mo) are found in the intrusive, endoskarn (particularly the pink endoskarn), and brown–wollastonite exoskarn lithology types. Strongly elevated grades (>0.1% Mo) are restricted to the main deposit area. As with most other elements, the limestone, marble, and hornfels units are essentially non-mineralized (<0.001% Mo).	Limits of the barren core were identified and modelled. the low-grade core model does not use a hard boundary separating the Intrusive into high- and low-grade volumes. Instead, the distance to this surface is used to define estimation sub-zones. Between these distance-based sub-zones, sample sharing is allowed during estimation.	
Arsenic	High-grade arsenic zones often cut across geological units.	Arsenic was estimated by lithological unit. Given the penalty element property of arsenic, no grade capping was performed in the high-arsenic zone.	Validations indicate that the modelling approach is reasonable, considering the very high-nugget nature of arsenic, whose maxima can be more than a hundred times the average grades. At depth, the drill hole bulk density becomes noticeably less and the confidence in the model estimates decreases. However, data indicate that arsenic grades decrease with depth. As a result, the modelled volume of the high-grade arsenic zone also decreases, reducing the risk of understating arsenic grade estimates.

Element	Grade Distribution	Search Strategy/Capping	Notes
Lead	In the eastern and western areas of the deposit, high lead grades are localized near the contact between skarns and limestone/marble-hornfels. In the north area, lead mineralization is associated with the Oscarina Dike.	Lead grades show little spatial continuity (high nugget effect and short-range variograms).	Reconciliations against blast holes and model validation show that the estimates are globally reasonable but locally imprecise.
Ammonium acetate-soluble copper	Three separate CuAC zones were modelled	Higher CuAC values in Usupallares and along the southeastern edge of the main pit. Bornite zone was used to control the CuAC estimation	The number of data available for soluble metal estimates is well less than is available to estimate total-metal grades. Grades of soluble and total metal were checked before and after grade estimation. In cases where the soluble grade exceeded the total grade, the soluble metal grade was reduced to match the total metal grade.
Ammonium acetate-soluble zinc	Estimated within and outside of a higher-solubility zone		
Iron	Higher iron grades are generally found in the enriched brecciated zones, endoskarn and exoskarn units, except for wollastonite zones and diopside exoskarn units, which have lower iron grades.	No capping of iron grades	Model validation showed strong correspondence between the model and the NN model average grades, both over large blocks and over slices (swaths) through the deposit.
Sodium cyanide-soluble copper	The presence of CuCN is used as an indicator of the presence of bornite. Bornite is most common in the southern portion of the main pit, within the brown wollastonite exoskarn and green wollastonite exoskarn units. Bornite is often found in association with bismuth, arsenic, and molybdenum. When present in the ore feed, a separate bornite concentrate is produced.		Estimated for groups of lithological units within mineralogical zones. The mineral indicator zones, defined by deterministic models, are the bornite indicator zone, the oxide indicator zone, and the chalcopyrite indicator zone.
Bornite		The bornite model was constructed according to intervals of bornite intensity >0.2 (normalized copper mineralogy proportions from soluble assays have	A deterministic model of the bornite zone was interpreted.

Element	Grade Distribution	Search Strategy/Capping	Notes
		generated three indicators: chalcopyrite, bornite, and oxide indicator zones).	
Bulk density			Modelled using distance-class estimation groups (selection of composites and blocks close to certain contacts). A similar trend is observed in the endoskarn and calcareous units approaching the exoskarn.

Note: CuAC = ammonium acetate-soluble copper; CuCN = sodium cyanide-soluble copper.

- Copper, silver, lead, and iron: lithological controls on grade were sufficient to define estimation domains. A localized geostatistical search was applied for these elements;
- Soluble copper and zinc: deterministic models of higher-solubility zones, the extent of bornite mineralization, and the bornite transition zone were used to control the estimation.

14.10 Validation

Validation was performed to confirm the modelling and/or estimation adequacy and biases, and included:

- Visual validation in plan and section;
- Comparison, for each element, of the tonnage estimated, and the average model and declustered data grades;
- Swath plots;
- Scatterplots;
- Change of support checks.

No significant biases were noted from the validation steps.

A review of available reconciliation data against the block model indicated that the estimates were also unbiased with respect to production data through the comparison of the long-range block-model grades with the blast-block model estimated from the production blast holes.

14.11 Classification of Mineral Resources

14.11.1 INITIAL CLASSIFICATION

Mineral Resource confidence classification was based on a combination of the presence of breccia and the drill spacing (Table 14-4).

An example cross-section showing the confidence classifications is included in Figure 14-1.

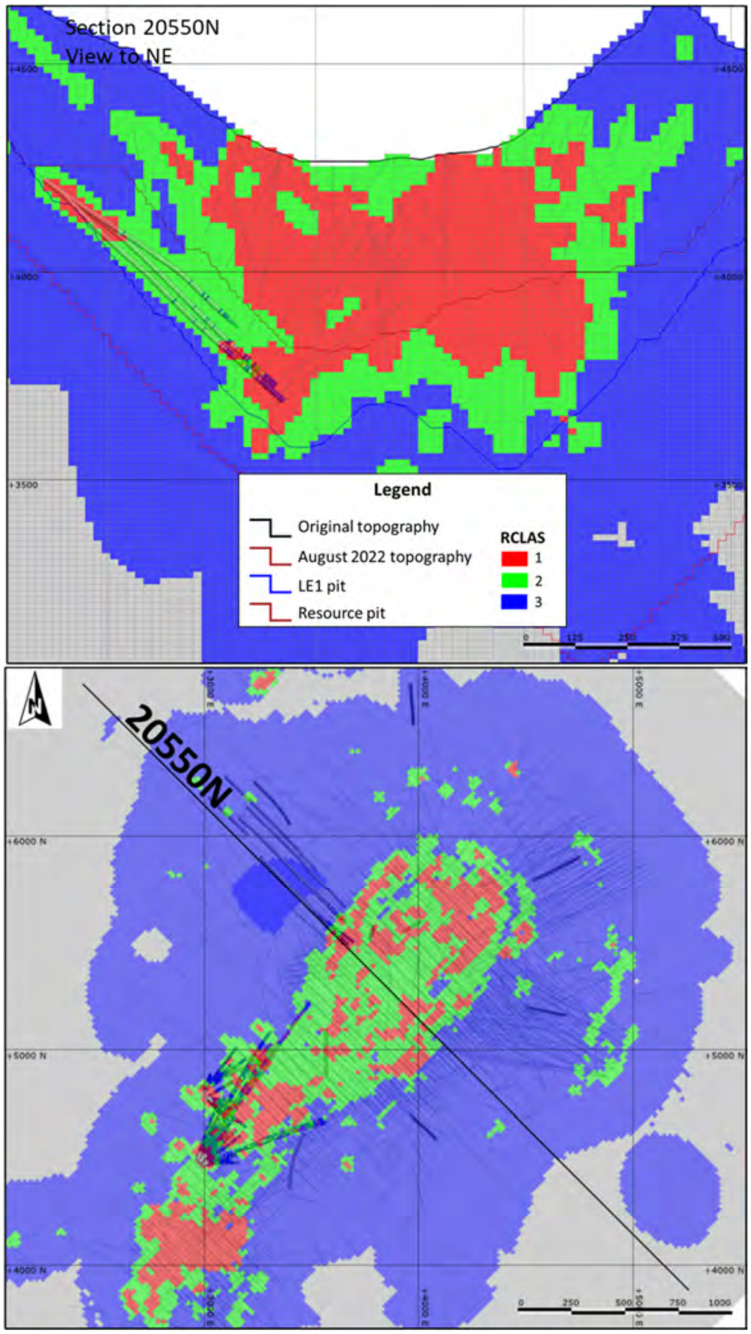
The grid spacings selected for the classification are supported by a detailed reconciliation study. The classification was smoothed to remove isolated blocks of one confidence classification from within zones of another classification.

14.11.2 TREATMENT OF BRECCIA UNITS

The breccia units are geologically complex, and individual breccia bodies are discontinuous. Grades observed in other units are typically elevated in zones of brecciation. Non-breccia samples located closer to the breccia generally have higher grades, and grades are also elevated over the entire volume where breccia is present.

Table 14-4: Confidence Categories

Confidence Classification	Unit	Criteria
Measured (Class 1)	Endoskarn that contains >5% probability of breccia.	Three drill holes within 35 m and at least one drill hole within 30 m.
	Endoskarn that contains <5% probability of breccia.	Three drill holes within 55 m and at least one drill hole within 35 m.
	Intrusive that contains >5% probability of breccia.	Three drill holes within 40 m and at least one drill hole within 30 m.
	Intrusive that contains <5% probability of breccia.	Three drill holes within 45 m and at least one drill hole within 35 m.
	Exoskarn and waste.	Three drill holes within 30 m and at least one drill hole within 25 m.
Indicated (Class 2)	Endoskarn that contains >5% probability of breccia.	Three drill holes within 55 m and at least one drill hole within 45 m.
	Endoskarn that contains <5% probability of breccia.	Three drill holes within 70 m and at least one drill hole within 45 m.
	Intrusive that contains >5% probability of breccia.	Three drill holes within 60 m and at least one drill hole within 45 m.
	Intrusive that contains <5% probability of breccia.	Three drill holes within 80 m and at least one drill hole within 55 m.
	Exoskarn and waste.	Three drill holes within 50 m and at least one drill hole within 40 m.
Inferred (interpolated) Class 3		Two drill holes containing three composites within a 150 m search, or one drill hole containing three composites within 75 m. In the latter case, this applies where there are drill holes at >150 m spacing, and there are blocks classified as Indicated between the drill holes but not adjacent to the single drill hole.
Inferred (extrapolated) Class 4		Blocks within the four-pass estimated copper grades that do not meet the criteria for other classes.



Note: Figure prepared by Minera Antamina, 2024.

Figure 14-1: Example Cross-Section, Confidence Classification

A breccia proportion model was created and used to define volumes where breccia is potentially present, such that those zones were treated differently from non-breccia zones. To model the local breccia proportions, the deposit was divided into four groups based on lithology: waste lithologies (marble, hornfels, and limestones), endoskarns, intrusive rocks, and exoskarns. For these groups, indicator variograms were modelled and the breccia proportions were estimated.

Superimposing the breccia proportion indicator model over the Mineral Resource model indicated qualitatively that copper grades were higher and more variable over zones where breccia was present. Visual observations also indicated that only a small amount of breccia was necessary to locally increase grade variability.

As a result, Mineral Resource classification includes modifications to the required drill spacing based on the presence or absence of breccia.

14.11.3 GEOLOGICAL COMPLEXITY CONSIDERATIONS

Other risks incorporated into the confidence classification included:

- The exoskarn and breccia units carry a significant proportion of the zinc and copper grades. The location and continuity of these units are typically low, causing high uncertainty in the presence/absence of the units and local grades;
- The brown endoskarn unit continuity is a key consideration for the copper estimate. The volume/proportion of this unit changes by northing and easting so that the required drill spacing used for confidence classification must address both the uncertainty in the unit volume and the grades within that unit;
- In the northern portion of the deposit, zinc grades in the brown endoskarn increase when approaching the brown–green exoskarn and green exoskarn. The zinc tonnage estimate is dependent on the rate of the decrease of zinc grades and the data density must be sufficient to accurately model this grade trend;
- Within the exoskarns, the zinc mineralization is not uniform. There are portions of the exoskarn where zinc mineralization is absent. Drilling has to be sufficiently closely spaced to identify and separate such volumes of copper–zinc and copper-only mineralization.

To develop the geological complexity treatment model, the 5 x 5 x 7.5 m sub-blocks were coded using the geological and Mineral Resource models. A Python program was used to convert the blocks within the geological complexity treatment model to waste, or to change the classification from copper–zinc to copper-only. The model was then re-blocked to the 20 x 20 x 15 m resource model block size.

Seven types of blocks were evaluated (Table 14-5).

Table 14-5: Block Types Assessed in Geological Complexity Evaluation

Block Type	Note
Mineralization versus waste	Grades of all potentially economic elements were set to detection limit values for all 5 m blocks coded as waste lithologies
Copper-only and copper–zinc mineralization	If the copper–zinc zones were insufficiently large to create a separate mineable polygon, the material was included within a copper mineralization polygon. In effect, the copper–zinc mineralization was converted to copper mineralization. This change of mineralization type resulted in a loss of zinc metal since zinc is not recovered from copper-only mineralization. To partially reproduce this change in mineralization type, narrow zones of copper–zinc mineralization surrounded by copper mineralization were identified and the copper–zinc mineralization classification was converted to copper mineralization. When this change in mineralization type was made, the zinc grade was set to the analytical detection limit. To track the changes in zinc grade associated with this assignment, the operational dilution model contained two zinc grades: • ZNR: grade without considering the conversion of copper–zinc to copper mineralization; • ZNR2: zinc grade when zinc grades are removed from converted copper–zinc mineralization. Changes were made when the following conditions were met: • When copper and copper–zinc geometallurgical types are in contact, the mineralization type is changed. Typically, the change is from copper–zinc to copper; • If the geometallurgical type is 6 (bornite, high Zn), the code is changed to 5 (bornite, low Zn); • Geometallurgical type 7 (copper–zinc) is changed to 1 (Cu–low Bi if Bi/Cu<9.4), 2 (Cu–high Bi if 9.4<Bi/Cu<35.3) or 3 (Cu–very high Bi, if Bi/Cu>35.3); • Geometallurgical type 8 (zinc, bornite transition) is changed to 4 (copper, bornite transition). When mineralization types are switched, an indicator is set to one and the average of the indicator over the final 20 m block defines the proportion of mineralization type switches. When copper mineralization is converted to copper–zinc mineralization, the indicator flag is set but no changes to the grades are made. That is, the Mineral Resource and Mineral Reserve grades are the same. When copper–zinc mineralization is converted to copper, the zinc grade is set to the zinc detection limit.
Blocks located within waste lithologies	Converted to waste by assigning all economic grades to the detection limit. The penalty element grades are not modified.
Blocks defined as mineralization within waste	The algorithm operates in three dimensions examining the nature of the contacts on the six block faces. The geometric conditions required to change ore to waste were: • Waste on two opposing faces (in X, Y or Z); • Waste on three faces where two of the faces are opposing. Initially, waste on any three faces was considered for diluting the block but this resulted in excessive flipping of codes. • Waste on any four or five sides or waste on six sides. When these conditions were met, the mineralized block was converted to waste and the copper, zinc, silver, molybdenum, and lead grades of the block were set to

Block Type	Note
	detection limit values. These revised grades were considered when the grades of the 20 x 20 x 15 m Mineral Reserve model blocks were assigned.
Blocks defined as waste but surrounded by mineralization	Must be recovered as mineralization and will dilute grade. Uses the same algorithm as used to assess the mineralization within waste. This change in mineralization or waste category is completed to simplify the mineralization and waste contact boundaries that could potentially reduce the amount of mineralization to waste conversion.
Contact dilution	Applies to blocks that would be subject to contact dilution if mining were undertaken on 5 m blocks. This dilution was largely accounted for during the creation of 20 x 20 x 15 m blocks, so no additional dilution was included in the final operational dilution model.

Blocks located in volumes where the geometry of the deposit was not complex were coded as either mineralization or waste. For all other areas, five block types of were assessed. The final mineralization, waste, or mineralization-type code was assigned after five passes through the model. After five passes, no additional code alterations were permitted.

Uncertainties regarding sampling and drilling methods, data processing and handling, geological modelling, and estimation were incorporated into the classification assigned. The areas with the most uncertainty was assigned to the Inferred Mineral Resources category, and the areas with fewest uncertainties were classified as Measured Mineral Resources.

Review of reconciliation data supports that the confidence criteria are reflective of actual mine performance.

14.12 Reasonable Prospects of Eventual Economic Extraction

Reasonable prospects of eventual economic extraction were evaluated based on underground and open pit mining methods.

14.12.1 OPEN PIT

Blocks considered potentially amenable to open pit mining methods were confined within a conceptual pit shell that used the parameters summarized in Table 14-6.

Geological complexity criteria were also incorporated into the assessment of reasonable prospects:

- Narrow mineralized zones near the exoskarn/waste contact and the endoskarn/intrusion contact are not recoverable given the scale of mining. A geometric treatment algorithm was used to identify and dilute this material;
- Mineralization within estimation units that are typically waste (marble, hornfels, limestone, etc.) is not recovered. This material was converted to waste in the resource model;

Table 14-6: Input Parameters, Conceptual Constraining Open Pit Shell

Parameter	Unit	Value
Copper price	US$/lb	3.50
Zinc price	US$/lb	1.25
Silver price	US$/oz	24.63
Molybdenum price	US$/lb	13.30
Exchange rate	USD/PEN	3.71
Optimization algorithm		A Lerchs-Grossmann algorithm was used with Whittle software.
Metallurgical recoveries	%	Variable metallurgical recoveries on a block-by-block basis were used, assigned by a script based on a set of equations derived from historical plant performance.
Mining cost	US$/t ore mined	4.90
	US$/t waste mined	3.72
	US$/t total mined	3.97
Processing cost	Variable US$/t processed	4.16
	Sustaining capital US$/t processed	3.49
	Fixed US$/t processed	3.46
	Total US$/t processed	11.11
General and administrative cost	US$/t processed	2.77
Treatment charge	US$/dmt Cu concentrate	73.08
	US$/dmt Zn concentrate	186.0
	US$/dmt Mo concentrate	N/A
Refining charge	US$/lb payable Cu in Cu concentrate	0.073
	US$/lb payable Zn in Zn concentrate	N/A
	US$/lb payable Mo in Mo concentrate	1.10
Freight cost	US$/wmt Cu concentrate	34.0
	US$/wmt Zn concentrate	34.0
	US$/wmt Mo concentrate	78.8
Pit slope angles	Degrees	38–55

Note: N/A = not applicable.

- Mixtures of copper and copper–zinc mineralization were identified, and the mineralization type was modified to define a potentially mineable volume of copper–zinc or copper mineralization. Copper–zinc mineralization surrounded by copper mineralization was converted to copper mineralization and the zinc value of the block was set to the detection limit since zinc is not recovered in the copper mill circuit;
- The bornite transition zone was modelled and applied to the resource model to identify the transitional mineralization type.

The property boundaries were used as a hard constraint to define the conceptual pit shell.

14.12.2 UNDERGROUND

Blocks considered potentially amenable to underground mining methods were confined within conceptual mineable shapes, assuming a selective mining method such as sub-level stoping, using the parameters summarized in Table 14-7. The property boundaries were used as a hard constraint to define the estimates.

14.13 Cut-off Criteria

Block value is calculated using the following expression:

- Mineralization value per block (BVO) = (revenues – operating costs), where:
  - Mineral value per block: expected profit on a block in US$/t;
  - Revenues or net smelter return (NSR): profits from the payable metals in the concentrate minus the realization cost (penalties, treatment charge (TC), refining charge (RC), and freight costs);
  - Operating costs: mining and processing cost (fixed and variable), including sustaining capital. The cost of transport of concentrate to the port is also included.

Given the variability of the rock hardness, the mill rate is also incorporated to the block value by multiplying the profit of each block (US$/t) with the effective mill throughput (t/h) and the mill efficiency (%):

- Mineralization value per hour (OVPHR) = (profit x effective mill throughput x mill efficiency)/(1,000), where:
  - Mineral value per hour: expected value in US$‘000/mill-hour or k$/h;
  - Profit: positive mineralization value per block (BVO);
  - Effective mill throughput: long-term average processing rate by geometallurgical type in t/h;
  - Mill efficiency: long-term average processing performance in %;
  - Conversion factor: division by 1,000 (to convert to thousand dollar units).

Table 14-7: Input Parameters, Conceptual Underground Potentially Mineable Shapes

Parameter	Unit	Value	
Copper price	US$/lb	3.50	
Zinc price	US$/lb	1.25	
Silver price	US$/oz	24.63	
Molybdenum price	US$/lb	13.30	
Exchange rate	USD/PEN	3.71	
Optimization algorithm		The Alford algorithm was used with Deswik software.	
Metallurgical recoveries	%	Variable metallurgical recoveries on a block-by-block basis have been used, assigned by a script based on a set of equations derived from historical plant performance.	
Conceptual mining method		Sublevel stoping; assuming 35,000 t/d (12.0–13.5 Mt/a).	
Underground limit		Conceptual mine depth limited to 2,903 masl.	
Crown pillar		A crown pillar of 100 m has been considered under the Mineral Resource pit shell.	
Costs		Below crown pillar	Within crown pillar
Mining cost	US$/t mined	30.9	59.2
Capital cost	US$/t mined	8.0	8.0
Processing cost	US$/t mined	10.7	10.7
General and administrative cost	US$/t mined	4.2	4.2
Total underground cost	US$/t mined	53.8	82.2
Treatment charge	US$/dmt Cu concentrate	73.08	
	US$/dmt Zn concentrate	186.0	
	US$/dmt Mo concentrate	N/A	
Refining charge	US$/lb payable Cu in Cu concentrate	0.073	
	US$/lb payable Zn in Zn concentrate	N/A	
	US$/lb payable Mo in Mo concentrate	1.10	
Freight cost	US$/wmt Cu concentrate	34.0	
	US$/wmt Zn concentrate	34.0	
	US$/wmt Mo concentrate	78.77	
Dilution	% assumed maximum internal	50	
	% assumed external	10	

Note: N/A = not applicable.

Blocks confined within a conceptual pit shell with an OVPHR above the minimum cut-off value of US$0/h are considered to have reasonable prospects of eventual economic extraction, assuming open pit mining methods. This minimum cut-off value can be expressed in an approximate reference copper equivalent cut-off grade as presented in Table 14-8.

Blocks below the crown pillar confined within conceptual mineable shapes with an NSR above the minimum cut-off value of US$53.8/t are considered to have reasonable prospects of eventual economic extraction assuming an underground selective mining method. This minimum cut-off value can be expressed in an approximate copper equivalent cut-off grade as presented in Table 14-9.

Blocks within the crown pillar confined within conceptual mineable shapes with an NSR above the minimum cut-off value of US$82.2/t are considered to have reasonable prospects of eventual economic extraction assuming an underground selective mining method. This minimum cut-off value can be expressed in an approximate copper equivalent cut-off grade as presented in Table 14-10.

14.14 Mineral Resource Statement

Mineral Resources are reported in situ, using the 2014 CIM Definition Standards.

Mineral Resources are reported exclusive of those Mineral Resources converted to Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

The Mineral Resources considered potentially amenable to underground mining are reported exclusive of the estimated Mineral Resources potentially amenable to open pit mining.

Mineral Resources are reported on a 100% basis. There are four shareholders with the following ownership percentages: BHP 33.75%; Glencore 33.75%; Teck, 22.50%; and Mitsubishi,10.00%.

The Qualified Person for the estimates is Mr. Lucio Canchis, RM SME, a Minera Antamina employee.

Mineral Resources considered potentially amenable to open pit mining methods are reported in Table 14-11 and have an effective date of 31 December, 2024.

Mineral Resources considered potentially amenable to underground mining methods are reported in Table 14-12 and have an effective date of 31 December, 2024.

Table 14-8: OVPHR and Reference Copper Equivalency (Open Pit)

Geometallurgical Type	Cut-off Value OVPHR (US$/h)	CuEq (%)	Cu (%)	Zn (%)	Ag (g/t)	Mo (%)
Cu	0.0	0.22	0.17	N/A	2.0	0.014
Cu–Zn	0.0	0.38	0.08	0.75	4.1	N/A

Notes: Cu geometallurgical type: CuEq (%) = Cu (%) + (Ag (g/t) x 0.008814) + (Mo (%) x 1.887466). Cu–Zn geometallurgical type: CuEq (%) = Cu (%) + (Ag (g/t) x 0.007438) + (Zn (%) x 0.362614).

Table 14-9: NSR and Reference Copper Equivalency (Underground and Below Crown Pillar)

Geometallurgical Type	Cut-off Value NSR (US$/t)	CuEq (%)	Cu (%)	Zn (%)	Ag (g/t)	Mo (%)
Cu	53.8	0.88	0.78	N/A	7.1	0.018
Cu–Zn	53.8	1.02	0.64	0.86	8.8	N/A

Notes: Cu geometallurgical type: CuEq (%) = Cu (%) + (Ag (g/t) x 0.008814) + (Mo (%) x 1.887466). Cu–Zn geometallurgical type: CuEq (%) = Cu (%) + (Ag (g/t) x 0.007438) + (Zn (%) x 0.362614).

Table 14-10: NSR and Reference Copper Equivalency (Underground and Within Crown Pillar)

Geometallurgical Type	Cut-off Value NSR (US$/t)	CuEq (%)	Cu (%)	Zn (%)	Ag (g/t)	Mo (%)
Cu	82.2	1.30	1.17	N/A	10.7	0.020
Cu–Zn	82.2	1.54	1.04	1.12	13.8	N/A

Notes: Cu geometallurgical type: CuEq (%) = Cu (%) + (Ag (g/t) x 0.008814) + (Mo (%) x 1.887466). Cu–Zn geometallurgical type: CuEq (%) = Cu (%) + (Ag (g/t) x 0.007438) + (Zn (%) x 0.362614).

Table 14-11: Mineral Resources Potentially Amenable to Open Pit Methods

Confidence Category	Geometallurgical Type	Tonnage (Mt)	Cu (%)	Zn (%)	Ag (g/t)	Mo (%)
Measured	Cu	86	0.66	0.09	6.5	0.014
	Cu–Zn	18	0.46	1.15	25.9	0.008
	Sub-total	*104*	*0.62*	*0.28*	*9.9*	*0.013*
Indicated	Cu	150	0.78	0.13	8.6	0.021
	Cu–Zn	59	0.98	1.66	17.5	0.006
	Sub-total	*209*	*0.84*	*0.56*	*11.1*	*0.017*
Total Measured + Indicated Mineral Resources		***313***	***0.77***	***0.47***	***10.7***	***0.015***
Inferred	Cu	587	0.88	0.14	8.3	0.024
	Cu–Zn	197	1.03	1.62	15.6	0.008
	Sub-total	*784*	*0.92*	*0.51*	*10.2*	*0.020*

Notes to accompany the estimate of Mineral Resources potentially amenable to open pit mining methods:

1. Mineral Resources are reported in situ, using the 2014 CIM Definition Standards, and have an effective date of 31 December, 2024. The Qualified Person for the estimate is Mr. Lucio Canchis, RM SME, a Minera Antamina employee.
2. Mineral Resources are reported on a 100% basis. There are four shareholders with the following ownership percentages: BHP Group Ltd., 33.75%; Glencore plc, 33.75%; Teck Resources Limited (through its subsidiary Teck Base Metals Ltd.), 22.50%; and Mitsubishi Corporation,10.00%.
3. Mineral Resources are reported exclusive of those Mineral Resources converted to Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
4. Mineral Resources potentially amenable to open pit mining methods are confined within a conceptual pit shell that uses the following input parameters: commodity prices of US$3.50/lb Cu, US$1.25/lb Zn, US$24.63/oz Ag, and US$13.30/lb Mo; variable metallurgical recoveries on a block-by-block basis; the average metallurgical recoveries for the copper geometallurgical mineralization type are 92% Cu, 0% Zn, 79% Ag, and 46% Mo, whereas for the copper–zinc geometallurgical mineralization type, metallurgical recoveries are 83% Cu, 84% Zn, 60% Ag, and 0% Mo; average mining cost of US$3.97/t mined, average processing cost of US$11.11/t processed, and general and administrative cost of US$2.77/t processed; depending on concentrate type: treatment charges that range US$73–186/dmt concentrate, refining charges that range US$0.073–1.10/lb payable metal, and freight costs that range US$34–79/wmt concentrate; pit slope angles that average 38–55°. The project boundaries are used as a hard constraint to define the limits of the conceptual pit shell. Lead and zinc are not recovered from the copper geometallurgical mineralization type, and molybdenum is not recovered from the copper–zinc geometallurgical mineralization type.
5. Mineral Resources are reported above a zero dollar-per-mill hours (OVPHR) cut-off, using the following equations. Mineralization value per block (BVO) = (revenues - operating costs) where mineralization value per block: expected profit on a block in US$/t; revenues or net smelter return (NSR): profits from the payable metals in the concentrate minus the realization cost (penalties, treatment charge (TC), refining charge (RC), and freight costs); operating costs: mining and processing cost (fixed and variable), including sustaining capital. The concentrate transport costs to the port are also included. Given the variability of the rock hardness, the mill rate is also incorporated to the block value by multiplying the profit of each block (US$/t) with the effective mill throughput (t/h) and the mill efficiency (%): mineralization value per hour (OVPHR) = (profit x effective mill throughput x mill efficiency)/(1,000), where mineral value per hour: expected value in US$'000/mill-hour or k$/h; profit: positive BVO; effective mill throughput: long-term average processing rate by geometallurgical type in t/h; mill efficiency: long-term average processing performance in %; conversion factor: division by 1,000 (to convert to thousand dollar units).
6. Numbers have been rounded and may not sum.

Table 14-12: Mineral Resources Potentially Amenable to Underground Methods

Confidence Category	Geometallurgical Type	Tonnage (Mt)	Cu (%)	Zn (%)	Ag (g/t)	Mo (%)
Inferred	Cu	282	1.23	0.20	10.8	0.017
	Cu–Zn	151	1.11	1.50	15.5	0.006
Total Inferred Mineral Resources		***433***	***1.19***	***0.65***	***12.5***	***0.013***

Notes to accompany the estimate of Mineral Resources potentially amenable to underground mining methods

1. Mineral Resources are reported in situ, using the 2014 CIM Definition Standards, and have an effective date of 31 December, 2024. The Qualified Person for the estimate is Mr. Lucio Canchis, RM SME, a Minera Antamina employee.
2. Mineral Resources are reported on a 100% basis. There are four shareholders with the following ownership percentages: BHP Group Ltd., 33.75%; Glencore plc, 33.75%; Teck Resources Limited (through its subsidiary Teck Base Metals Ltd.), 22.50%; and Mitsubishi Corporation,10.00%.
3. Mineral Resources potentially amenable to underground mining methods are confined within conceptual mineable shapes assuming a sublevel-stoping scenario that use the following input parameters: commodity prices of US$3.50/lb Cu, US$1.25/lb Zn, US$24.63/oz Ag, and US$13.30/lb Mo; variable metallurgical recoveries on a block-by-block basis; variable metallurgical recoveries on a block-by-block basis; the average metallurgical recoveries for the copper geometallurgical mineralization type are 92% Cu, 0% Zn, 79% Ag, and 46% Mo, whereas the metallurgical recoveries for the copper–zinc geometallurgical mineralization type are 83% Cu, 84% Zn, 60% Ag, and 0% Mo; assumptions of a 100 m thick crown pillar under the Mineral Resource pit shell; below-crown-pillar cost assumptions of: mining cost of US$30.90/t mined, capital costs of US$8.00/t mined, processing costs of US$10.70/t mined, and general and administrative costs of US$4.20/t mined; within-crown-pillar cost assumptions of: mining costs of US$59.20/t mined, capital costs of US$8.00/t mined, processing costs of US$10.70/t mined, and general and administrative costs of US$4.20/t mined; depending on concentrate type: treatment charges that range US$73–186/dmt concentrate, refining charges that range US$0.073–1.10/lb payable metal, and freight costs that range US$34–79/wmt concentrate; maximum internal dilution of 50% and external dilution of 10%. The property boundaries are used as a hard constraint to define the limits of the conceptual mineable shapes. Zinc is not recovered from the copper geometallurgical mineralization type, and molybdenum is not recovered from the copper–zinc geometallurgical mineralization type.
4. Mineral Resources are reported above a US$53.80/t net smelter return (NSR) cut-off; Mineral Resources within the crown pillar are reported above a US$82.20/t NSR cut-off.
5. Numbers have been rounded and may not sum.

14.15 Factors That May Affect the Mineral Resource Estimate

Factors that may affect the Mineral Resource estimate include:

- Commodity price and exchange rate assumptions;
- Changes to the assumptions used to generate the NSR and OVPHR cut-offs;
- Changes in local interpretations of mineralization geometry and continuity of mineralized zones;
- Changes to geological and mineralization shapes, and geological and grade continuity assumptions;

- Density and domain assignments;
- Changes to geotechnical assumptions;
- Changes to mining and metallurgical recovery assumptions;
- Changes to the input and design parameter assumptions that pertain to the conceptual pit shell constraining the estimates potentially amenable to open pit mining methods;
- Changes to the input and design parameter assumptions that pertain to the conceptual mineable shapes constraining the estimates potentially amenable to underground mining methods;
- Assumptions as to the continued ability to access the site, retain mineral and surface rights titles, maintain environment and other regulatory permits, and maintain the social license to operate.

14.16 QP Comment on Item 14 "Mineral Resource Estimates"

Mineral resources are reported in situ, using the 2014 CIM Definition Standards.

There is upside potential for the Mineral Resource estimate if those Mineral Resources classified as Inferred can be upgraded to higher confidence categories with additional drilling and supporting studies.

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

15.1 Introduction

Mineral Reserves were estimated from Measured and Indicated Mineral Resources, assuming only open pit mining methods. Inferred Mineral Resources within the mine plan were set to waste.

Mineral Resources in low-grade stockpiles were not converted to Mineral Reserves.

The current TSF dam elevation at 4,195 masl was used for the pit limit selection during Mineral Reserve estimation.

15.2 Pit Optimization

The pit optimization work was conducted using Whittle software based on Measured and Indicated Mineral Resource blocks only. The selection of the reference pit shell corresponds to the closest match between pit shell and the target ore tonnes constrained by remaining tailings capacity. The pit shell at a revenue factor of 0.85 was selected as the reference pit shell.

There is approximately 193 Mt of mineralization classified as Measured and Indicated that is contained between the Mineral Reserve pit design and the Mineral Resource pit shell that was not converted to Mineral Reserves.

15.3 Optimization Inputs

Pit optimization included the following considerations:

- TSF capacity:
  - The Mineral Reserves are primarily constrained by the TSF capacity rather than an economic pit limit. The maximum permitted TSF elevation is 4,195 masl, and is used as a hard constraint when converting Mineral Resources to Mineral Reserves. The reference final pit limit is designed to meet the available TSF capacity;
- Process plant campaigning:
  - The concentrator processes eight different geometallurgical types (refer to Table 7-5 for a description of these geometallurgical types) on a campaign basis. These campaigns range from a number of days to an entire month, depending upon ore development and concentrate marketing requirements;
  - To optimize recovery and product quality, both the grinding and flotation circuits are calibrated for each geometallurgical type campaign;
  - The processing rate is 145,000 t/d on average. Mill throughput varies among the geometallurgical types;

  - The chalcopyrite copper geometallurgical types, located in the centre of the main skarn orebody (endoskarn) have the highest mill rates due to the significant alteration of the host rocks;
  - Bornite geometallurgical types, on the south side of the pit, have lower throughput rates but are less common inside the pit;
- Metallurgical recoveries:
  - These are defined on a block-by-block basis using a script that assigns a recovery value based on a set of equations derived from historical plant performance;
  - Metallurgical equations are specific for each of the geometallurgical types that are processed and depend on a combination of inputs, including geometallurgical type, head grades, Cu-to-Zn ratio, Zn-to-Cu ratio, Fe-to-Cu ratio, Ag-to-Cu ratio, Bi-to-Cu ratio, As-to-Cu ratio, and other ratios;
  - The average plant recoveries assumed during Mineral Reserve estimation are 89% for Cu (all geometallurgical types) and 84% for Zn (Cu–Zn geometallurgical types);
  - The average metallurgical recoveries for copper geometallurgical types are 92% Cu, 79% Ag, and 46% Mo. Zinc is not recovered from the copper geometallurgical types;
  - The average metallurgical recoveries for copper–zinc geometallurgical type are 83% Cu, 84% Zn, and 60% Ag. Molybdenum is not recovered from the copper–zinc geometallurgical types.

Inputs used during pit optimization are summarized in Table 15-1.

15.4 Cut-off Criteria

Given the polymetallic nature of the deposit and the variability of the rock hardness, Minera Antamina uses a ranking that incorporates the block value and time to process it so the mill rate is also incorporated into the block value by multiplying the profit of each block (US$/t) with the effective mill throughput (t/h) and the mill efficiency (%).

The block value is calculated using the following expression:

- Ore value per block (BVO) = (revenues – operating costs), where:
  - Ore value per block: expected profit on a block in US$/t;
  - Revenues or net smelter return (NSR): profits from the payable metals in the concentrate minus the realization cost (penalties, treatment charge (TC), refining charge (RC), and freight costs);
  - Operating costs: mining and processing cost (fixed and variable), including sustaining capital. The concentrate transport costs to the port are also included.

Table 15-1: Input Parameters

Parameter	Unit	Value
Copper price	US$/lb	3.54
Zinc price	US$/lb	1.15
Silver price	US$/oz	21.46
Molybdenum price	US$/lb	11.10
Exchange rate	USD/PEN	3.71
Fuel cost	US$/L	1.07
Electricity cost	US$/kWh	0.083
Metallurgical recoveries	%	Variable metallurgical recoveries on a block-by-block basis. Assigned using a script based on a set of equations derived from historical plant performance.
Mining cost	US$/t ore mined	4.20
	US$/t waste mined	4.54
	US$/t total mined	4.30
Processing cost	Variable, US$/t processed	4.29
	Sustaining capital, US$/t processed	2.36
	Fixed US$/t processed	2.67
	Total US$/t processed	9.31
General and administrative cost	US$/t processed	2.95
Treatment charge	US$/dmt Cu concentrate	73.08
	US$/dmt Zn concentrate	186.0
	US$/dmt Mo concentrate	N/A
Refining charge	US$/lb payable Cu in Cu concentrate	0.073
	US$/lb payable Zn in Zn concentrate	N/A
	US$/lb payable Mo in Mo concentrate	1.10
Freight cost	US$/wmt Cu concentrate	34.0
	US$/wmt Zn concentrate	34.0
	US$/wmt Mo concentrate	78.8
Pit slope angles	Degrees	33–55

Subsequently, the ore value per hour (OVPHR) is calculated for cut-off and ore ranking in the mine schedule as follows:

- OVPHR = (profit x effective mill throughput x mill efficiency) / (1,000), where:
  - Ore value per hour: expected value in US$'000/mill-hour or k$/h;
  - Profit: positive ore value per block (BVO);
  - Effective mill throughput: long-term average processing rate by ore type in t/h;
  - Mill efficiency: long-term average processing performance in %;
  - Conversion factor: division by 1,000 (to convert to thousand dollar units).

The OVPHR field on the block model is used for scheduling as it incorporates all of the variables used to generate an optimized ore feed program. The minimum cut-off value is at OVPHR equal to US$6,000/h. This minimum cut-off value ensures that the costs of stockpiling and rehandle can be met. It includes provision for extra loading and hauling from the stockpile to the crusher.

However, the LOM production plan does not include processing, or stacking, of any additional low-grade stockpiles; it is based on direct mill feed only.

Blocks confined within an operational pit design with an OVPHR above the minimum cut-off value of US$6,000/h are considered Mineral Reserves assuming open pit mining methods. This minimum cut-off value can be expressed as an approximate copper equivalent cut-off grade as presented in Table 15-2.

15.5 Dilution and Ore Loss

The reserve block model is a derivation of the resource block model that considers dilution and ore losses that commonly occur during mining.

Dilution, which is related to grade variability and geological contacts, is introduced through the application of a selective mining unit of 20 m wide x 20 m long x 15 m high.

Ore loss, related to non-mineable narrow mineralized structures along the boundary between exoskarn and waste rock, is applied through a dilution and ore loss algorithm (CRMSA, 2017), based on a 5 m wide x 5 m long x 7.5 m high resource block model. As a result, the operational dilution and ore loss algorithm that was applied to the resource block model, reduced the copper content by 1.01% and zinc content by 3.57% within the Mineral Reserves pit.

15.6 Stockpiles

The high-grade stockpiles are classified as Proven Mineral Reserves, and converted from Measured Mineral Resources.

Stockpile tonnages and grades are estimated by adding the daily incoming records and subtracting the outgoing records. These data are under the control of the Ore Control and Mine Planning areas.

Table 15-2: OVPHR and Copper Equivalency (Open Pit)

Geometallurgical Type	Cut-off Value OVPHR (US$/h)	CuEq (%)	Cu (%)	Zn (%)	Ag (g/t)	Mo (%)
Cu	6,000	0.20	0.16	—	1.6	0.017
Cu–Zn	6,000	0.42	0.10	0.87	4.5	—

Notes: Cu geometallurgical type: CuEq (%) = Cu (%) + (Ag (g/t) x 0.007591) + (Mo (%) x 1.557949). Cu–Zn geometallurgical type: CuEq (%) = Cu (%) + (Ag (g/t) x 0.006406 + (Zn (%) x 0.328226).

15.7 Mineral Reserves Statement

Mineral Reserves are reported at the point of delivery to the process plant, using the 2014 CIM Definition Standards.

The Qualified Person for the estimate is Mr. Fernando Angeles, P.Eng., a Minera Antamina employee.

Mineral Reserves are reported on a 100% basis. There are four shareholders with the following ownership percentages: BHP plc, 33.75%; Glencore, 33.75%; Teck, 22.50%; and Mitsubishi, 10.00%.

Mineral Reserves are reported in Table 15-3, and have an effective date of 31 December, 2024.

15.8 Factors that May Affect the Mineral Reserves

Factors that may affect the Mineral Reserve estimates include:

- Metal price and exchange rate assumptions;
- Changes to the assumptions used to generate the OVPHR cut-off;
- Changes in local interpretations of mineralization geometry and continuity of mineralized zones;
- Changes to geological and mineralization shapes, and geological and grade continuity assumptions;
- Density and domain assignments;
- Changes to geotechnical assumptions including pit slope angles;
- Changes to hydrological and hydrogeological assumptions;
- Changes to mining and metallurgical recovery assumptions;

Table 15-3: Mineral Reserves Statement

Category	Geometallurgical Type	Tonnage (Mt)	Cu (%)	Zn (%)	Ag (g/t)	Mo (%)
Proven	Cu	198	0.82	0.12	8.1	0.029
	Cu–Zn	50	1.02	1.88	18.1	0.011
	Sub-total	*247*	*0.86*	*0.47*	*10.1*	*0.025*
Probable	Cu	190	0.91	0.15	9.4	0.030
	Cu–Zn	113	1.07	1.99	19.2	0.008
	Sub-total	*302*	*0.97*	*0.84*	*13.0*	*0.022*
Total Proven + Probable Reserves		***550***	***0.92***	***0.67***	***11.7***	***0.023***

Notes to accompany the Mineral Reserves estimate

1. Mineral Reserves are reported at the point of delivery to the process plant, using the 2014 CIM Definition Standards, and have an effective date of 31 December, 2024. The Qualified Person for the estimate is Mr. Fernando Angeles, P.Eng., a Minera Antamina employee.

2. Mineral Reserves are reported on a 100% basis. There are four shareholders with the following ownership percentages: BHP Group Ltd., 33.75%; Glencore plc, 33.75%; Teck Resources Limited (through its subsidiary Teck Base Metals Ltd.), 22.50%; and Mitsubishi Corporation,10.00%.

3. Mineral Reserves are confined within an operational pit design that uses the following input parameters: commodity prices of US$3.50/lb Cu, US$1.15/lb Zn, US$21.46/oz Ag, and US$11.10/lb Mo; variable metallurgical recoveries on a block-by-block basis; average metallurgical recoveries for the Cu geometallurgical type of 92% Cu, 79% Ag, and 46% Mo, with zinc not recoverable; average Cu–Zn geometallurgical type recoveries of 83% Cu, 84% Zn, and 60% Ag, with molybdenum not recoverable; average mining cost of US$4.30/t mined, average processing cost of US$9.31/t processed, and general and administrative cost of US$2.95/t processed; depending on concentrate type; treatment charges that range from US$73–186/dmt concentrate, refining charges that range from US$0.073–1.10/lb payable metal, and freight costs that range from US$34–79/wmt concentrate; and pit slope angles that average 33–55°. The remaining tailings capacity is used as a hard constraint to define the limits of the reference pit shell, which is the guide for the operational pit design. Mineral Reserves are reported inclusive of dilution and mining recovery. The dilution and ore loss algorithm applied reduced the copper content by 1.01% and zinc content by 3.57%.

4. Mineral Reserves are reported above a US$6,000 dollar-per-mill hours (OVPHR) cut-off, using the following equations. Ore value per block (BVO) = (revenues - operating costs) where ore value per block: expected profit on a block in US$/t; revenues or net smelter return (NSR): profits from the payable metals in the concentrate minus the realization cost (penalties, treatment charge (TC), refining charge (RC), and freight costs); operating costs: mining and processing cost (fixed and variable), including sustaining capital. The concentrate transport costs to the port are also included. Given the variability of the rock hardness, the mill rate is also incorporated to the block value by multiplying the profit of each block (US$/t) with the effective mill throughput (t/h) and the mill efficiency (%): ore value per hour (OVPHR) = (profit x effective mill throughput x mill efficiency)/(1,000), where mineral value per hour: expected value in US$'000/mill-hour or k$/h; profit: positive BVO; effective mill throughput: long-term average processing rate by geometallurgical type in t/h; mill efficiency: long-term average processing performance in %; conversion factor: division by 1,000 (to convert to thousand dollar units)..

5. Numbers have been rounded and may not sum.

- Changes to the input and design parameter assumptions that pertain to the open pit shell constraining the estimates;
- Assumptions as to the continued ability to access the site, retain mineral and surface rights titles, maintain environment and other regulatory permits, and maintain the social license to operate;
- The remaining TSF storage capacity at an elevation of 4,195 masl.

15.9 QP Comment on Item 15 "Mineral Reserve Estimates"

Mineral Reserves are reported using 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 Reserves that are not discussed in this Report.

16 MINING METHODS

16.1 Overview

The mining operations use conventional open pit mining methods and conventional equipment.

16.2 Geotechnical Considerations

Pit slopes were updated by the engineer of record, Piteau Associates Engineering Ltd. (Piteau), in 2023. The updated study involved assessing the structural fabric data (mapping, photogrammetric, and televiewer), and determining appropriate slope design parameters considering the real bench performance achieved.

The geotechnical design is based on the following five models: a geological, major structural, structural fabric, rock mass, and hydrogeological.

Rock type groups were combined into six rock mass units (RMU), to simplify geological and geotechnical interpretation and modelling. Intact rock strengths range from strong to moderately strong and the rock mass quality from Fair to Good. The average unconfined compressive strengths vary from 27–87 MPa, and the rock mass rating from 47–75.

The resulting geotechnical domains are shown in Figure 16-1.

16.3 Hydrogeological Considerations

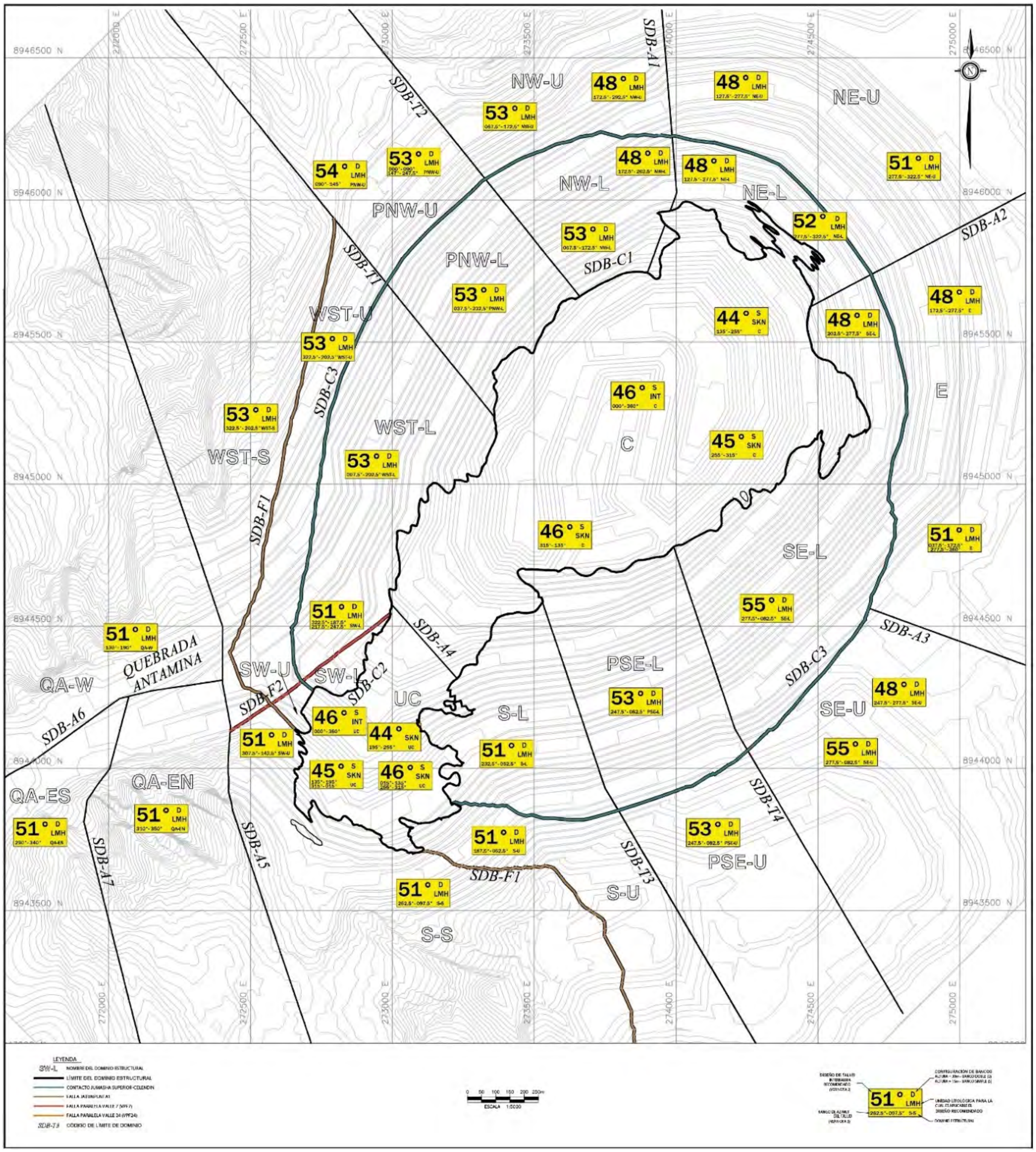
A numerical groundwater flow model is in place, provides data to support geotechnical evaluations, and is used to design future well locations and pumping rates for a given set of operational constraints.

The results of the model predict that the pore pressures in the pit slopes for future mining phases will be reasonable and consistent with the observed historical trends. Predictive results are considered conservative, as pore pressures appear to be overestimated.

The model predicts that most depressurization will occur in the central intrusive unit, with less depressurization in the less permeable exoskarns, hornfels, and marbles. The endoskarn and intrusive units are expected to behave similarly.

Drainage flows until the end of operations are forecast to be between 100–150 L/s in the dry season and 300–400 L/s in the wet season. Much of this flow will come from drainage wells located in the intrusive units (at least 70%), with a smaller contribution from pit wall seepage (<30%).

Sensitivity analyses indicate that the model is sensitive to small changes in hydraulic conductivity, recharge, and specific storage. The total amount of estimated water pumped for the high-drainage and low-drainage scenarios is 210 Mm^3 (in both scenarios), which is reasonably consistent with the base case estimate of 190 Mm^3.



Note: Figure prepared by Piteau, 2023.

Figure 16-1: Geotechnical Domains

Approximately half of the water pumped from the open pit comes from local recharge and the other half from storage within fractured intrusions and disturbed zones. Less than 10% of the water is expected to come from adjacent limestone units. The intact rock in the pit area has a low permeability. Groundwater movement is influenced by structural trends and particularly by the degree of secondary fracturing and faulting of individual lithological units.

The drainage and groundwater monitoring system has been operating since 2004. The objective of the drainage program is to maintain the water table in the intrusive and skarn units at least 20 m below the lowest level of the operating pit floor. The drainage and depressurization program consists of:

- Installation of vertical drainage wells with a drilling diameter of up to 18 inches (450 mm), and a casing diameter of 12 inches (300 mm);
- Installation of piezometers within the drainable hydrogeological units (intrusive and skarn units) to establish water levels in and around the pit floor;
- Installation of piezometers inside limestone, hornfels and marble units to measure the pore pressure in the pit slopes.

The average current pumping flow is 250 L/s. The pit drainage flows are slightly higher in the northern and central sectors of the open pit, as compared to the southern sector.

16.4 Mine Design

The reference pit shell selected from the pit optimization process was used as a guide to produce the final pit design by integrating operational design characteristics including geotechnical parameters, ramp locations and gradients, overburden storage facilities, mining widths, bench and inter-ramp heights, and other practical mining considerations given the pit geometry.

The Mineral Reserve pit was determined using blocks that were classified as Measured, Indicated, and Inferred. However, all the Inferred blocks (46 Mt) were treated as waste rock for mine scheduling and tabulation of the Mineral Reserve estimate.

The pit design includes four mining phases (Figure 16-2):

- Phases 8, 9, and 10 are currently in production;
- Phase 11 waste stripping is planned for 2025.

Both the final pit and mining phases were designed using QPit software, using the following design criteria:

- Bench height of 15 m in intrusive–skarn lithologies;
- Bench height of 30 m in limestone–marble–hornfels lithologies;
- Minimum mining width of 60 m;
- Haul road design width of 40 m;
- Maximum road gradient of 10%;

Note: Figure prepared by Minera Antamina, 2023. Image is an oblique view so a scale is not applicable.

Figure 16-2: Pit Phase Location

- Maximum inter-ramp height of 150 m in intrusive–skarn lithologies
- Maximum inter-ramp height of 300 m in limestone–marble–hornfels lithologies;
- Geotechnical berm of 30 m;
- Variable bench face angles, inter-ramp angles, and catch berm widths, depending on geotechnical domain.

16.5 Operational Considerations

Minera Antamina operates 24 hours a day, 365 days a year. Mining activities are carried out in two 12-hour daily shifts on a seven-day-per-week basis.

Mining phases are extracted using 15 m high benches. Depending on geotechnical recommendations, single or double benches can be used.

Maximum mining and movement rates are approximately 249 and 297 Mt/a, respectively, and are constrained by:

- Mining restrictions;
- Maximum sinking rate of eight benches per year;
- Mine equipment performance, availability, and utilization;
- The area within the mining bench.

The maximum processing rate is approximately 53.6 Mt/a, and is constrained by:

- Variable processing rate for each geometallurgical type;
- Overall mill availability and utilization (this includes the primary crusher);
- Minimum feed to plant for two days by geometallurgical type campaign.

To ensure there is sufficient ore to feed the plant, a minimum of two mining phases (if available) with exposed ore are required.

Based on the current phase designs, backfill material will need to be used to create access roads for selected mining phases.

16.6 Mining Infrastructure

16.6.1 ORE CRUSHER

A new ore primary crusher is being commissioned in 2025 to replace the existing crusher in the Quebrada Antamina area. This will allow pit phase 9 to continue expanding to the south. The new crusher has a nominal capacity of 13,000 t/h, which exceeds the mill capacity.

16.6.2 STOCKPILES

Two types of stockpiles are used: high-grade and low-grade.

High-grade stockpiling is used for blending purposes and to provide short-term management for mining of geometallurgical types. The high-grade stockpiles have an approximate 5 Mt (2.3 Mm^3) capacity.

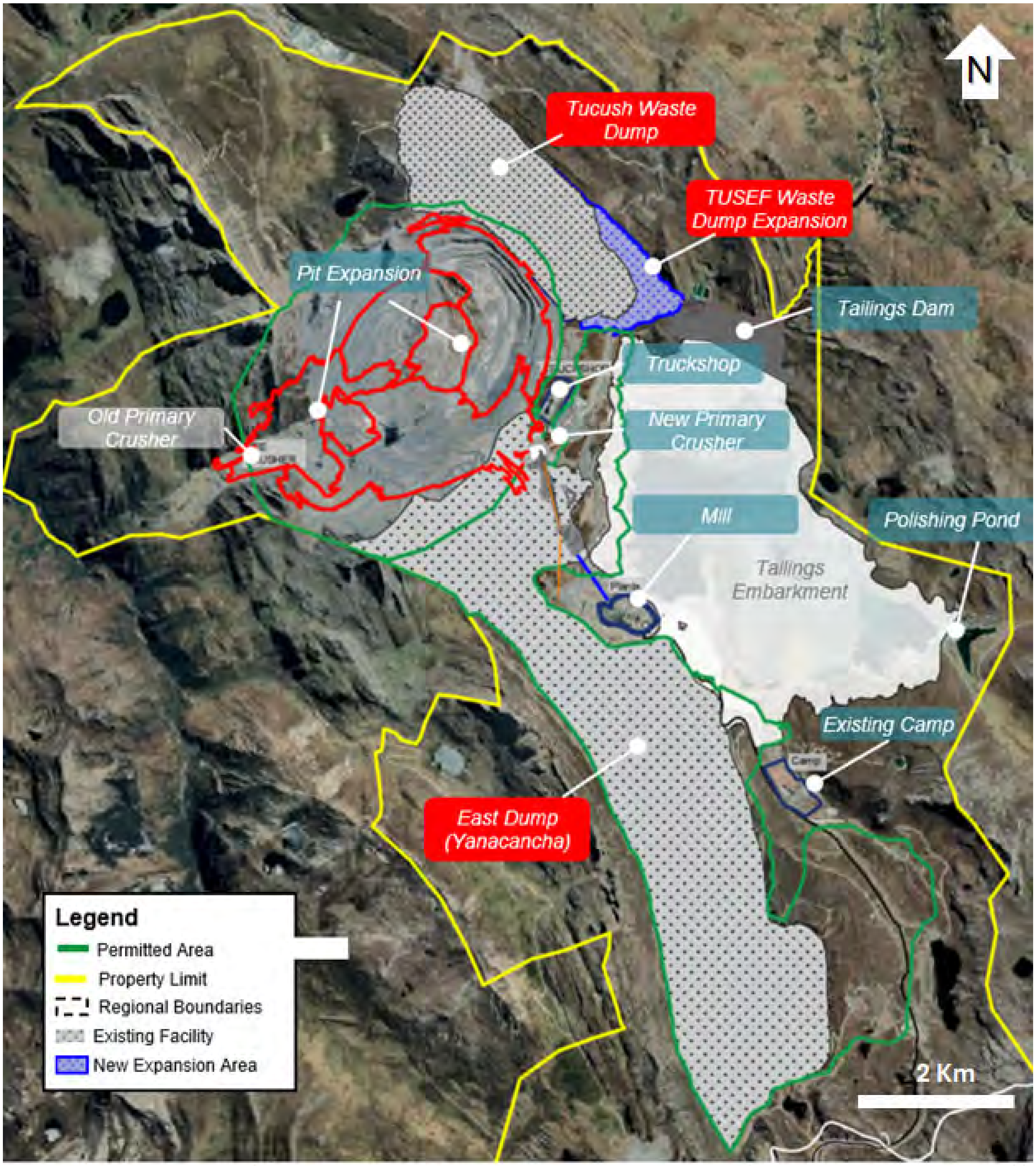
Low-grade stockpiling is used to segregate material at various cut-off grades during mining, and to provide sufficient storage space for mineralization that is not in the current LOM plan. The low-grade stockpile has an approximate 50 Mt (22.8 Mm^3) capacity and is located on the north side of the East WRSF.

16.6.3 WASTE ROCK STORAGE FACILITIES

The WRSF locations are provided in Figure 16-3, and are designed to store the waste types summarized in Table 7-5.

Key features include:

- East (Yanacancha) WRSF: located at the southeast end of the open pit, near the process plant. Designed to contain reactive and non-reactive waste (A, Aox, B, and C waste rock types). Permitted capacity of about 3,052 Mt (1,526 Mm^3), approximate final footprint of 1,101 ha, maximum height of 714 m, and maximum elevation of 4,868 masl;



Note: Figure prepared by Minera Antamina, 2024. Note "Dump" = WRSF.

Figure 16-3: WRSF Location Plan

Table 16-1: Waste Rock Types

Waste Rock Type	CuAc/Cu Ratio	Zn (%)	As (ppm)	Lithology	Description
A	<0.4	≥0.15	≥400	—	Reactive
Aox	≥0.4	≥0.15	≥400	—	Reactive oxide
B	—	≥0.07 to <0.15	<400	Limestone, marble, hornfels	Potentially reactive
C	—	<0.07	<400	Limestone, marble, hornfels	Non-reactive

- Tucush: located at the northeast of the open pit and to the northwest of the TSF. Designed to contain only potentially reactive and non-reactive waste (B and C waste rock types). Permitted capacity of about 971 Mt (486 Mm^3), approximate final footprint of 321 ha, maximum height of 555 m, and maximum elevation of 4,643 masl;
- TUSEF (extension to the Tucush WRSF): located to the northeast of the open pit and southeast of the Tucush WRSF. Designed to contain potentially reactive and non-reactive waste (B and C waste rock classes). Permitted capacity of about 309 Mt (155 Mm^3), approximate final footprint of 26 ha, maximum height of 503 m, and maximum elevation of 4,523 masl.

WRSF designs are based on 60 m lift heights, 50 m berm widths, and 37º face angles. Designs incorporate a minimum static factor of safety of 1.5 and a minimum pseudostatic factor of safety of 1.0.

The WRSF permitted capacities are sufficient for the remaining LOM plan.

Operational facilities have geotechnical monitoring in place.

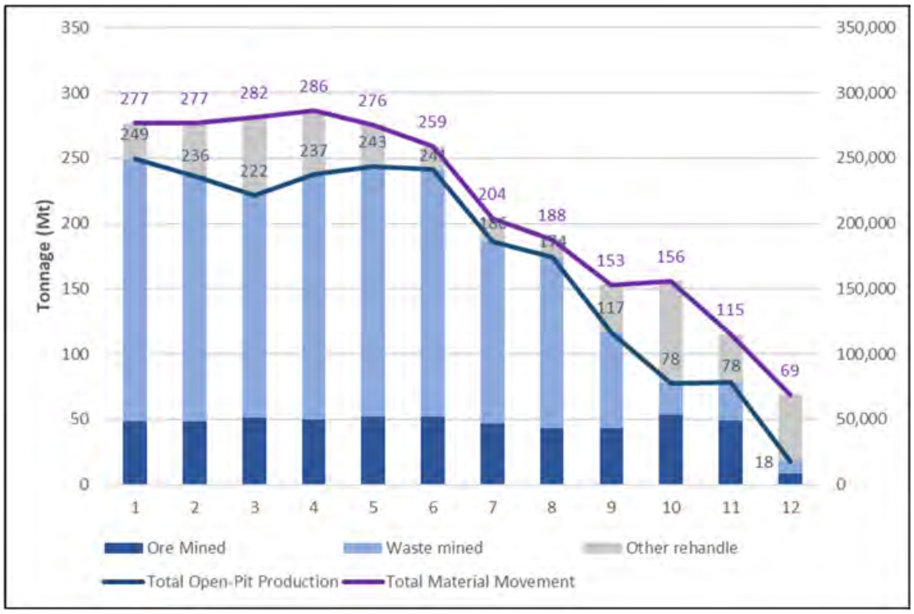
16.7 Life-Of-Mine Plan

The remaining mine life is 12 years, from 2025 to 2036, with 2036 being a partial year.

About 546 Mt of ore and 1,535 Mt of waste are planned to be mined from the open pit with an average stripping ratio of 2.79. Phase 10 has the highest stripping ratio of the remaining pit phases at 3.1.

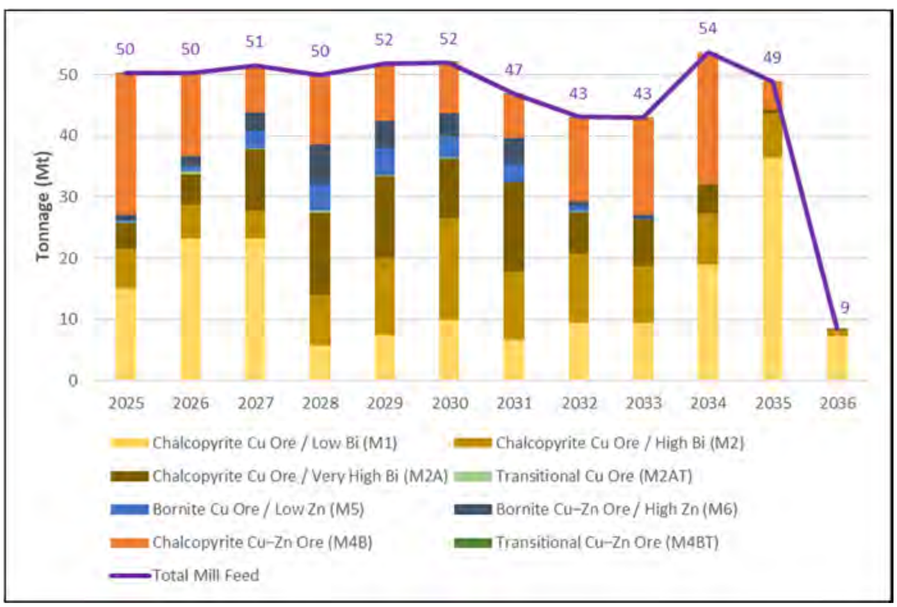
Figure 16-4 shows the total material movement by destination. The total open-pit production is approximately 2,080 Mt, while the total material movement is approximately 2,637 Mt, which considers production, rehandling, and other material movements.

Figure 16-5 shows the total mill feed by geometallurgical campaign. On average, chalcopyrite copper-bearing ores are the most common (66.5%), followed by chalcopyrite copper–zinc-bearing ores (24.7%), and the balance (8.8%) corresponds to bornite-rich copper- and copper–zinc-bearing ores.



Note: Figure prepared by Minera Antamina, 2024. HG = high-grade. Year 1 = 2025.

Figure 16-4: LOM Production Plan by Destination



Note: Figure prepared by Minera Antamina, 2024.

Figure 16-5: LOM Mill Feed Production Plan by Geometallurgical Type

16.8 Blasting and Explosives

Blasting consumables include bulk explosives, electronic detonators, and delays.

Large diameter (311 mm) production drilling is performed by Minera Antamina staff. The blasting contractor undertakes buffer and trim blasting, using 251 and 127 mm diameter blast holes.

The bulk explosive used in dry conditions is ANFO (ammonium nitrate and fuel oil). In wet conditions, heavy ANFO (ANFO and emulsion) is used.

16.9 Grade Control

Grade control sampling includes preparing a sampling bag with blasthole identification information, collecting samples from six different points around the drill cutting pile generated by the blasthole drilling, placing the material inside the sampling bag, and verifying that the bag has the required weight (8–9 kg) for the laboratory.

The spacing between production blastholes used for ore and waste drill patterns are approximately 6.3 and 9.2 m, respectively.

16.10 Equipment

The main and ancillary mining equipment is owned and maintained by Minera Antamina. Buffer drilling and blasting services are outsourced.

The drilling fleet consists of electric and diesel rotary drills. Large diameter drills are used for production drilling, while medium and small diameter drills are used for trim drilling.

The loading fleet consists of electric-rope shovels, hydraulic shovels, and front-end loaders. Electric-rope shovels are used for mine production, while hydraulic shovels and front-end loaders are used for rehandling and secondary works.

To 2025, the hauling fleet consists of 92 Komatsu-930E trucks, and 46 ultra class trucks. Mine equipment is constantly renewed according to the equipment life and accounting for to the increased depths in the open pit and greater distances to the WRSFs.

Appropriate ancillary equipment and light-duty vehicles provide support for the production operations. Dust control is conducted by water-trucks and automated sprinkler systems on the main haul roads.

The main equipment list forecast is presented in Table 16-2. Equipment numbers are expected to peak in 2029.

Table 16-2: Equipment List Forecast

Equipment Type	Capacity	Equipment Number Forecast											
		2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036
Production drill	311 mm	9	8	8	8	8	8	7	6	6	5	4	1
Buffer drill	251 mm	3	3	3	3	3	3	2	2	2	1	1	1
Rope shovel	67 m^3	—	1	2	2	2	2	2	2	2	2	2	2
Rope shovel	56 m^3	7	6	5	5	5	5	3	2	2	1	1	—
Hydraulic shovel	29 m^3	4	4	4	4	4	4	4	4	4	4	1	1
Haul trucks	300 t	92	92	59	31	26	12	—	—	—	—	—	—
Haul trucks	370 t	46	52	88	116	124	124	122	111	86	77	67	45

17 RECOVERY METHODS

17.1 Introduction

The process plant was designed and constructed based on the metallurgical testwork programs summarized in Section 13 of this Report, and has been operating since 2001.

The concentrator was initially designed to process an average of 70,000 t/d of ore at an assumed 90% total plant availability. Capital investment in 2008 in the pebble crushing plant, in 2012 for grinding capacity increases, and optimizations of the unit processes have resulted in a substantial increase in the average processing rate to approximately 145,000 t/d by the end of 2020.

The ore is amenable to traditional differential flotation techniques using standard reagent schemes. The plant feed is separated into distinct geometallurgical types that are campaigned through the plant using different plant configurations to optimize the recovery of payable metals and provide the best opportunity to manage product quality.

Ore is relatively coarse grained. Satisfactory metallurgical results are obtained at a P80 secondary grind size of 220–245 µm for copper-only geometallurgical types (M1, M2 and M2A), at a P80 secondary grind size of 210–240 µm for copper–zinc geometallurgical types (M4B) and at a P80 secondary grind size of 180–220 µm for the bornite geometallurgical types (M5 and M6).

The different geometallurgical types have very different grinding characteristics, including the total grinding power required and the relative power requirement split between primary semi-autogenous grinding (SAG) and secondary ball milling. Operating flexibility and control are critical for the grinding circuits, and flexibility was increased in 2008 by the addition of a pebble crushing circuit.

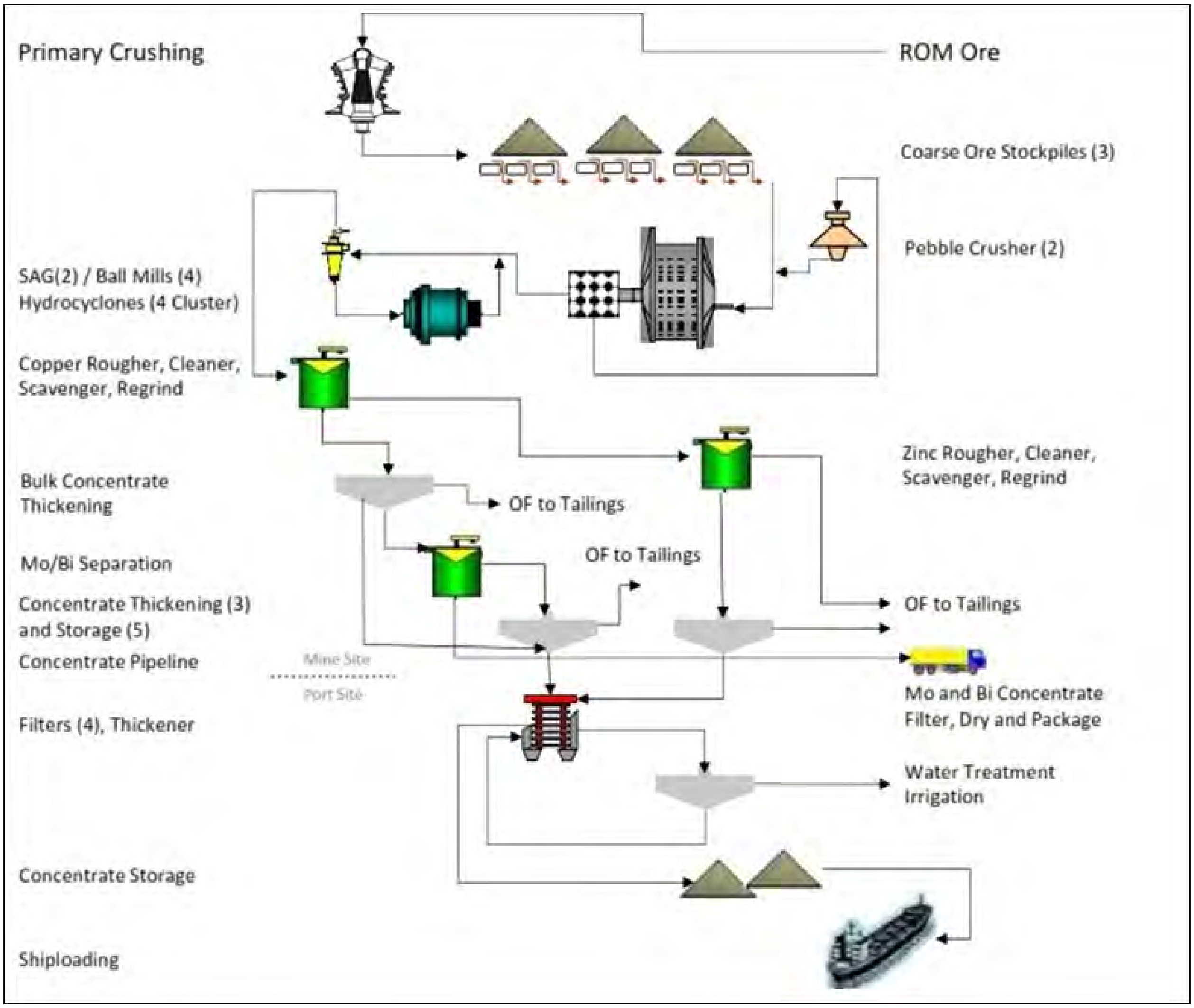
For the high bismuth–chalcopyrite geometallurgical type (M4B), bismuth minerals can be differentially floated from the bulk copper concentrate using standard differential flotation techniques resulting in a lead–bismuth(–silver) concentrate, although this results in some copper and silver recovery losses.

For the bornite geometallurgical types (M5 and M6) the primary bismuth minerals are witichenite (Cu_3BiS_3) and emplectite ($CuBiS_2$), which are present as very fine inclusions in the bornite crystals. Bismuth cannot be removed by differential flotation and these copper concentrates have elevated bismuth content.

Silver reports to both the final copper concentrate and the lead–bismuth concentrate.

17.2 Process Flow Sheet

A schematic process flowsheet is shown in Figure 17-1.



Note: Figure prepared by Minera Antamina, 2023.

Figure 17-1: Process Flow Sheet

17.3 Plant Design

Run-of-mine (ROM) ore is delivered to the 1.524 m x 2.261 m primary gyratory crusher by the mine haul trucks. Ore is reduced in size from 1.5 m to a nominal -178 mm product. The primary crushed product is transported via a 3 km conveyor through a 2.7 km long tunnel that runs southeast from the crusher to the Yanacancha Valley where the concentrator facilities are located.

The Antamina flotation flowsheet is designed to campaign-treat eight different geometallurgical types (refer to Table 7-5). Consequently, the plant is operated differently for each of the different geometallurgical types.

The coarse ore conveyor passes the material onto a radial stacker that deposits the crushed ore onto one of three 50,000 t capacity (live) coarse ore stockpiles. Coarse ore is reclaimed by apron feeders located underneath the stockpiles onto the SAG mill feed conveyors that run in a reclaim tunnel longitudinally to the stockpiles.

The particle size is reduced in stages in primary and secondary grinding and in regrinding to separate the ore minerals from the gangue minerals. The oversize discharge from the SAG mill is fed to the pebble crushers and the undersize is fed to secondary grinding. Steel balls are used as the grinding media.

The SAG mill feed conveyors discharge into a SAG mill circuit (two 11.6 m diameter x 6.4 m effective grinding length (EGL) mills).

The SAG mill discharge is distributed to four ball mills (each 7.3 m diameter x 10.8 m long). The ball mills are operated in closed circuit with hydrocyclones and further reduce the particle size prior to flotation. The hydrocyclones separate the coarser particles (returned to the ball mills) from the finer particles that are flotation feed. The overflow from the hydrocyclones is the flotation feed and the underflow from the hydrocyclones is returned to the ball mills.

The flotation process separates finely-ground sulphide minerals from gangue. In general, sulphide minerals attach to the bubbles that are forced through the slurry and float as the concentrate and non-sulphide minerals (gangue) remain in the slurry.

The initial flotation step is the bulk copper rougher circuit. The flotation product from the bulk copper roughers is fed to the bulk copper cleaner–regrind circuit. The regrind mill is part of the cleaner circuit and further reduces the particle size of the rougher concentrate to allow better separation of the sulphide minerals by further liberation from gangue. The flotation product from the bulk copper cleaner–regrind circuit is fed to the bulk copper concentrate thickener.

Depending on the geometallurgical type the “tailings” are sent to the zinc circuit or are final tailings.

From the bulk copper concentrate thickener, the bulk copper concentrate (depending on the geometallurgical type) is fed to the molybdenum–bismuth circuit, based on the geometallurgical type being campaigned. The circuit is used to either separate a saleable molybdenum concentrate from the bulk copper concentrate, or clean the bulk copper concentrate of bismuth (a penalty element) by producing a lead–bismuth concentrate.

The separation is achieved using differential flotation by conditioning the feed pulp with reagents to depress the copper minerals. For the molybdenum concentrate, sodium hydrosulphide is used, and for the bismuth concentrate sodium cyanide is used. Other flotation conditions such as pH and oxidation reduction potential (ORP) are adjusted to provide the correct metallurgical conditions. There are two products from this differential flotation circuit, a molybdenum concentrate or a lead–bismuth concentrate, depending on the feed, and the "tailings" (copper concentrate). Bismuth is a contaminant in the copper concentrate. The molybdenum concentrate and the lead–bismuth concentrate are sold as separate products.

The flotation product (either the molybdenum concentrate or the lead–bismuth concentrate) is fed to small dewatering and product bagging plant. The tailings from the molybdenum–bismuth circuit constitute a final copper concentrate that is fed to the copper concentrate thickener.

Depending on the geometallurgical type, the tailings from the bulk copper rougher and cleaner circuits are fed to the zinc rougher circuit or become the final tailings. The zinc rougher flotation product passes to the zinc cleaner–regrind circuit. The product is the final zinc concentrate that is fed to the zinc concentrate thickener. Tailings from the zinc circuit are final tailings, which are sent to the TSF. The TSF design and operation is provided in Section 18.5 of this Report.

A list of the key process equipment is provided in Table 17-1.

17.4 Product/Materials Handling

A concentrate pipeline transports the final copper and zinc concentrates from the mine site to the port at Punta Lobitos in the department of Huarmey (Figure 17-2).

A single pipeline is used to transport the concentrates (depending on the geometallurgical type being campaigned) on a batch basis, using a water plug between concentrate campaigns. The pipeline location is included with the offsite infrastructure plan in Section 18.1.

The molybdenum and lead–bismuth concentrates are dewatered at the mine site, packaged in bags, and transported to various ports or clients using highway trucks.

17.5 Energy, Water, and Process Materials Requirements

17.5.1 REAGENTS AND CONSUMABLES

The main reagents used in the concentrator include:

- Lime: used as a pH modifier, consumption depends on pH required and the geometallurgical type;
- Frother F-582 and M-144: used to form stable bubbles;
- Sodium hydrosulphide (NaHS): used to depress copper and iron sulphides to float molybdenum sulphides;
- Sodium cyanide (NaCN): used to depress pyrite, pyrrhotite, marcasite, arsenopyrite, sphalerite and chalcopyrite; in the copper–lead–bismuth circuit it is used as a surface activator to improve flotation of lead minerals;

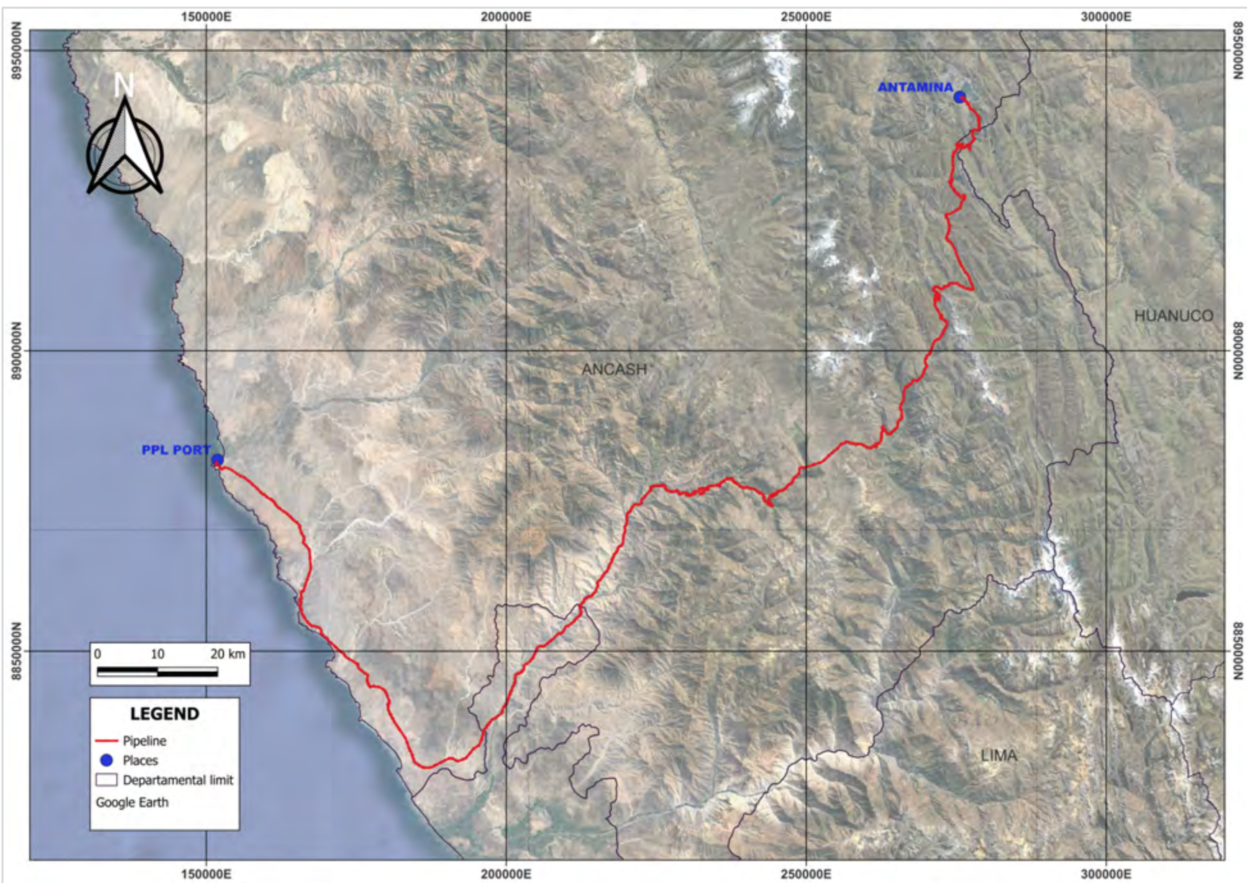
Table 17-1: Process Equipment List

Major Area	Equipment	Unit	Value
SAG mills	No. of mills		2
	Installed power (each)	kW	20,100
	Size (diameter x effective length)	ft	38 x 19
	Feed p80 passing size	mm	150
	Fresh feed per mill	t/h	2,800 to 3,500
	% ball charge (139.7 mm)	%	13 to 16
	Ball consumption	kg/t	0.28
Ball mills	No. of mills		4
	Installed power (each)	kW	11,200
	Size (diameter x effective length)	ft	24 x 35.5
	Feed p80 passing size	um	1,500
	Feed per mill	t/h	3,500 to 4,300
	% solids	%	74 to 76
	% ball charge (63.5 mm)	%	33 to 35
	Ball consumption	kg/t	0.32
Hydrocyclones	No. of cyclone batteries		4
	Cyclones per battery		10 operating/3 stand-by
	Diameter	in	26
	Overflow mass flow	t/h	1,500 to 1,800
	% solids in cyclone overflow	%	48 to 53
	Overflow 80% passing size	um	180 to 250
Pebble crushers	No. of crushers		2
	Type		Cone
	Installed power	HP	800
	Mass flow (each)	t/h	250 to 380
	Closed size setting	mm	12.7
Cu rougher flotation	Volume of each cell	m^3	130
	No. of lines		4
	No. of cells per line		8
	Residence time	min	35 to 40
	Cu recovery	%	Varies with geometallurgical type and head grade 92 to 94 (M1)

Major Area	Equipment	Unit	Value
			80 to 86 (M4B)
	Feed mass flow per line	t/h	1,400 to 1,750
	Concentrate mass flow per line	t/h	56 to 90
	Tailings mass flow per line	t/h	1,344 to 1,660
Cu regrind hydrocyclones	No. of cyclone batteries		1
	Cyclones per battery		17
	Diameter	in	15
	Feed per battery	m^3/h	1,500 to 1,600
	Overflow mass flow	t/h	300 to 350
	Overflow 80% passing size	um	40 to 60
Cu regrind mills	No. of mills		2
	Feed per mill	t/h	150 to 175
	Installed power (each)	kW	746
Cu first cleaner flotation	Volume of each cell	m^3	174
	Diameter	m	4.3
	Height	m	14
	No. of columns		7
	Cu recovery	%	75 to 80
	Feed mass flow	t/h	45 to 50
	Concentrate mass flow	t/h	26 to 30
	Tailings mass flow	t/h	19 to 20
Cu second cleaner flotation	Volume of each cell	m^3	174
	Diameter	m	4.3
	Height	m	14
	No. of columns		3
	Cu recovery	%	59 to 62
	Feed mass flow	t/h	72 to 76
	Concentrate mass flow	t/h	16 to 20
	Tailings mass flow	t/h	56 to 58
Cu scavenger flotation	Volume of each cell	m^3	130
	No. of lines		1
	No. of cells per line		5
	Residence time	min	25 to 30
	Cu recovery	%	88 to 90
	Feed mass flow	t/h	165 to 170

Major Area	Equipment	Unit	Value
	Concentrate mass flow	t/h	88 to 95
	Tailings mass flow	t/h	75 to 77
Zn rougher flotation	Volume of each cell	m^3	130
	No. of lines		4
	No. of cells per line		8
	Residence time	min	20 to 25
	Zn recovery	%	88 to 91
	Feed mass flow	t/h	1,344 to 1,410
	Concentrate mass flow	t/h	54 to 70
	Tailings mass flow	t/h	1,290 to 1,340
Zn regrind hydrocyclones	No. of batteries		1
	Cyclones per battery		16
	Size	in	15
	Feed per battery	m^3/h	1,200 to 1,400
	Overflow mass flow	t/h	240 to 260
	Underflow mass flow	t/h	240 to 260
Zn regrind mills	No. of mills		2
	Installed power (each)	kW	746
	Feed per mill	t/h	120 to 145
	P80 product	um	40 to 50
Zn first cleaner flotation	Volume of each cell	m^3	174
	Diameter	m	4.3
	Height	m	14
	No. of columns		8
	Zn recovery	%	75 to 80
	Feed mass flow	t/h	40 to 45
	Concentrate mass flow	t/h	21 to 25
	Tailings mass flow	t/h	19 to 20
Zn second cleaner flotation	Volume of each cell	m^3	174
	Diameter	m	4.3
	Height	m	14
	No. of columns		2
	Zn recovery	%	73 to 76
	Feed mass flow	t/h	45 to 50
	Concentrate mass flow	t/h	17 to 21
	Tailings mass flow	t/h	26 to 28

Major Area	Equipment	Unit	Value
Zn scavenger flotation	Volume of each cell	m^3	130
	Diameter		1
	Height		5
	Residence time	min	25 to 30
	Zn recovery	%	93 to 96
	Feed mass flow	t/h	130 to 140
	Concentrate mass flow	t/h	80 to 90
	Tailings mass flow	t/h	45 to 50



Note: Figure prepared by Minera Antamina, 2024.

Figure 17-2: Pipeline Corridor

- Zinc sulphate: used alone or in combination with NaCN to depress zinc minerals;
- Copper sulphate: used to activate iron and zinc sulphides;
- Fuel oil: used as a collector for molybdenum minerals in flotation;
- Flocculant 351: used to improve settling in the thickeners;
- Potassium amyl xanthate (PAX): used as a strong and non-selective collector in copper flotation;
- Sodium isopropyl xanthate (SIXP): used as a selective collector for sulphides in zinc flotation;
- Sodium hydroxide: used as a pH modifier for NaCN solution;
- Activated carbon: used to remove reagents remaining in the slurry feed to the copper–lead–bismuth separation circuit;
- Carbon dioxide: used as a pH modifier in the separation circuit;
- Grinding media: 5.5, 2.5, and 1 inch balls.

17.5.2 WATER

The main freshwater storage facility is the Nescafé impoundment. The dam supplies water for treatment to be used as potable water in the camp and fresh water for special uses in the concentrator.

In 2024 a total of 1,254,021 m^3 of fresh water was used in the concentrator. This was 1.15% of the total concentrator water consumption (fresh + recovered/reclaim water).

Water from the process plant is recirculated from the TSF and fresh water is needed only where clean water is required (for the cooling tower, reagent preparation and pump gland seal water).

The water balance is positive due to rainfall catchment in the TSF.

At the port site at Punta Lobitos, water is recovered from the concentrate by filtration, treated and used for irrigation of plants and trees in a reforestation project.

17.5.3 POWER

The SAG mills, ball mills and regrind mills are the main power consumers with a combined installed power of around 90,000 kW. The installed power for the primary crusher, pebble crushers, flotation cells and concentrate pumping combined is approximately 10,200 kW.

18 PROJECT INFRASTRUCTURE

18.1 Introduction

Existing infrastructure includes:

- Open pit;
- Access, shared-use, and haul roads;
- Truck shop, spare-parts warehouse, washing bays, light vehicles shop;
- Diesel fuel storage points;
- Warehouse and lay down yards;
- Explosives storage facility;
- Core storage facility;
- Administration offices, gatehouse;
- Ore crusher, transfer towers, conveyors;
- Process plant and laboratory;
- Two stockpiles (one high-grade, one low-grade);
- Two WRSFs;
- TSF;
- Concentrate pipeline;
- Port facilities;
- Power line;
- Nescafé water storage facility;
- Water storage and settling ponds;
- Water diversion facilities;
- Accommodations camp.

Additional infrastructure that will need to be constructed for the remaining LOM includes:

- Realigned access and shared-use roads;
- Additional haul roads;
- Replacement ore crusher;
- One WRSF extension;

- Two TSF raises.

An onsite infrastructure layout plan is included in Figure 18-1. Offsite infrastructure locations in relation to the mine site are shown in Figure 18-2, and include the port location and the concentrator pipeline route.

A new ore crusher is required in 2025 to support the last pit phases, as the current crusher location will be mined out by pit phase 9. The environmental permitting process for this crusher was completed in 2021.

18.2 Road and Logistics

The Project access is discussed in Section 5.1.

18.3 Stockpiles

Stockpiles are discussed in Section 16.6.2.

18.4 Waste Rock Storage Facilities

WRSFs are discussed in Section 16.6.3.

18.5 Tailings Storage Facility

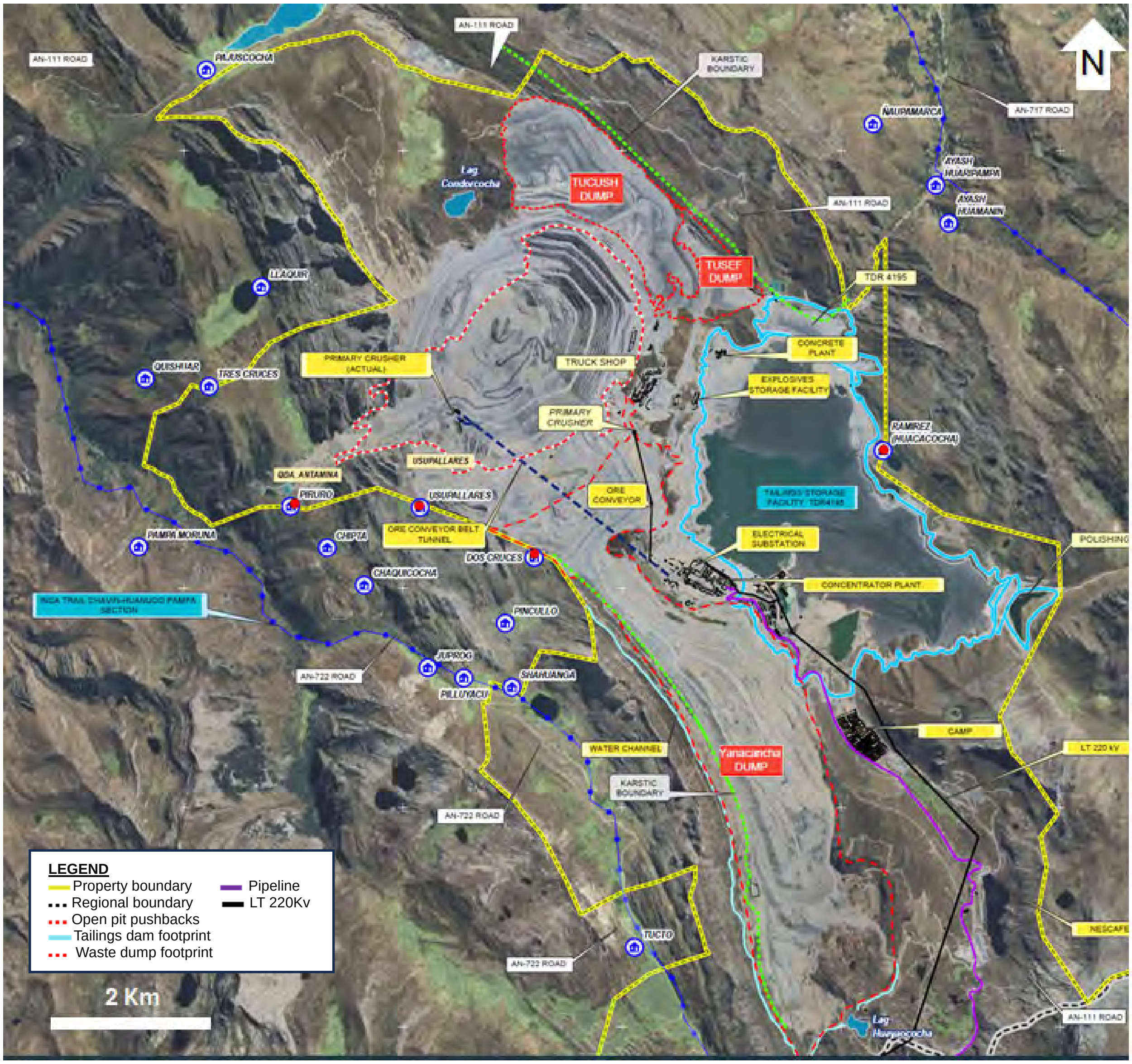
The TSF is situated in the Huincush valley, about 3 km from the process plant.

The main components of the TSF are summarized in Table 18-1. The TSF layout is shown in Figure 18-3.

Tailings discharge commenced in 2001 with the completion of the starter dam. Five dam raises have been completed, and at the overall date of the Report, a sixth was underway. There are two remaining raises to take the TSF to the environmentally permitted maximum elevation of 4,195 masl. The final design has a permitted capacity of about 1,527 Mt (844 Mm^3), approximate final footprint of 905 ha, and maximum height of 310 m. The TSF design incorporates a minimum static factor of safety of 1.5, a minimum pseudostatic factor of safety of 1.0, and a minimum post-earthquake factor of safety of 1.2.

The permitted capacity of the TSF is sufficient for the LOM plan. A geotechnical monitoring system is in place.

The TSF is a rockfill structure founded on a rocky basement and built with the downstream method up to an elevation of 4,113 masl. From that elevation to an elevation of 4,135 masl, the center line method has been used. Finally, from that last elevation, the facility has been and will be constructed with the downstream method up to an elevation of 4,195 masl.



Note: Figure prepared by Minera Antamina, 2024.

Figure 18-1: Onsite Project Infrastructure Layout Plan

Note: Figure prepared by Minera Antamina, 2024.

Figure 18-2: Offsite Project Infrastructure Layout Plan

Table 18-1: Key Components, TSF

Item	Note
TSF main dam	
Corridor auxiliary dam	Designed to retain tailings upstream of the TSF to prevent the tailings from flooding the concentrator area
Huacaccocha 1 and Huacaccocha 2 auxiliary dams	Designed to retain water and tailings upstream of the TSF to prevent flooding of the Huacaccocha lagoon
Control pond	Designed to retain upstream tailings to protect a downstream organic material stockpile located in the eastern sector of the TSF, between the Huacaccocha lagoon and the polishing pond
Polishing pond	Designed to contain natural water on the downstream side of the TSF and retain tailings on the upstream side. There is a design constraint, in that the level of the pond must be maintained within the property limit, so it cannot be higher 4,138 masl
Tailings transportation system	Four pumping lines with two pumps in series
Water recirculation system	
Water management system	Diversion system, around the tailings dam basin, to divert runoff flows from valleys A to D around the TSF, and maintain a minimum flow of 150 L/s in the Ayash River, downstream of the dam. Consists of the upper catchment and diversion canal, the CD Canal, the polishing pond, and a settlement system

Note: Figure prepared by Minera Antamina, 2024. Image is an oblique view so a scale is not applicable. Dump = WRSF.

Figure 18-3: TSF Layout

TSF construction has been primarily completed using mine pit-run rockfill, with upstream filter sections. Concrete and geomembrane are used to line faces for seepage control, and to prevent tailings migration into the rockfill. The facility is designed for the maximum credible earthquake, and can contain extreme rainfall events.

Geotechnical instrumentation provides permanent monitoring of settlement, deformation, changes in water levels, and changes in infiltration. Large scale geotechnical tests were performed on rockfill, and a deformation model was constructed to evaluate liquefaction potential and overall stability.

The TSF is monitored by international experts and by Peruvian government organizations (the Environmental Evaluation and Oversight Agency and the Supervisory Agency for Investment in Energy and Mining). The TSF undergoes an inspection by a qualified engineer, and an independent dam safety review every year. A comprehensive dam safety review is carried out every three years.

Tailings consist of approximately 45% solids as discharged from the plant, with an approximate in situ dry density of 1.81 t/m^3. Tailings are currently discharged using pumping. There are two main distribution pipelines and one secondary distribution pipeline.

Downstream seepage is collected in a collection pond at the dam toe, and is either pumped back to the dam or released to the Ayash River. Non-contact catchment water is diverted around the TSF to the Ayash River beyond the toe of the rockfill tailings dam.

Minera Antamina recovers and recycles 98% of the water used in the process, minimizing freshwater requirements. Four floating barges with a total of 16 vertical turbine pumps the supernatant water to the recirculating water tank back up to the concentrator. The maximum design flow is 16,000 m^3/h and the current demand is approximately 10,000 m^3/h. The system has instrumentation such as level sensors and flow meters to monitor the line.

18.6 Concentrate Pipeline

Concentrates from the final thickeners are stored in eight agitated storage tanks (18 m diameter by 18 m high) ahead of the concentrate pipeline to manage the geometallurgical type batching process.

Concentrate, at a slurry density of 60–65% solids, is pumped 304 km from the mine to the port at Punta Lobitos (refer to pipeline location in Figure 17-2). The pipeline has one pumping station and three choke stations to decrease the pressure during the ascent/descent from the mine to the coast. Concentrate is run through the pipeline continuously during the geometallurgical type campaigns, with water batches between each geometallurgical type campaign.

At the mine pump station there are four centrifugal charging pumps (two for copper concentrate and two for zinc concentrate) and four main positive displacement pumps (2,200 L/min each). A sump collects any spillage and returns it to the storage tanks. There are four intermediate valve stations, four pressure monitoring stations, and one terminal station. Rupture discs are installed at the valve stations and terminal station. There is an emergency pond at each valve station to collect spillage.

A fibre-optic line runs along the pipeline to feed signals from the instruments to the system for data collection and distribution (SCADA) system that is monitored at the control rooms at the mine and port.

18.7 Port

The port site is remote but is linked to the Pan-American Highway by a 6 km long paved road. There is an accommodation camp area and an industrial area including a power generating plant. The port has six mooring buoys (three bow and three stern) and navigation lights.

At the port site the concentrate is received at the terminal station and stored in three large, agitated storage tanks that provide surge capacity between the pipeline and the dewatering plant. Copper and zinc concentrates are dewatered to final shipping moisture content (8.5–9.5% moisture) using large pressure filters (four Larox 144 m^2 vertical filters (three operating and one standby) before being deposited in the dry concentrate storage shed that has a nominal capacity of 160,000 wmt.

The filtered concentrate is stored in separate piles within the storage shed depending on concentrate type. This also allows for blending when required.

Concentrates are loaded onto ocean-going vessels by front end loaders and conveyors (enclosed to prevent concentrate loss) feeding a ship loading facility. There is a dust collection system serving the storage shed and concentrate loading system.

The water treatment plant removes reagents left from the process plant from the liquid effluent using three treatment tanks and a clarifier followed by sand filters to reduce the solids to <10 ppm. The site has zero water discharge, all treated effluents are used to irrigate a forest area or are evaporated.

18.8 Water Supply

The Nescafé impoundment, situated at the southeast end of the Yanacancha WRSF is the main source of freshwater (refer to location in Figure 18-1) for the accommodations camp and operations. It captures rainfall, and drainage entering into the Project area from the south.

The impoundment has an approximate 2.86 Mm^3 storage capacity, which is sufficient for operational and domestic needs. The remaining demand is estimated at about 2.57 Mm^3 per year, which is within the limits of the currently-approved surface water use license.

18.9 Water Management

There are two main water basins that require management:

- Huincush and Tucush sub-basins, discharging at Ayash basin; these basin areas include the TSF and WRSFs;
- Antamina Creek discharging at the Pampa Moruna basin; this basin area includes the open pit.

Water from the open pit is drained along the valley to the southwest. Storage and settling ponds are located at the south end of the pit.

Water treatment includes:

- Surface-water management system, mainly for treating copper, zinc, and iron. The system includes physical treatment (sedimentation ponds and diversion channels) and chemical treatment (coagulation, flocculation, and precipitation by using sodium hydroxide);
- The Tucush WRSF settlement ponds system to control sediment levels.

Additional information is provided in Section 20.1.

At the Punta Lobitos port, water is recovered from concentrate by filtration and treated and used for irrigation in a reforestation project.

18.10 Camps and Accommodation

The current camp has a 12,930 bed capacity. The original camp was decommissioned in 2023.

18.11 Power and Electrical

Minera Antamina has an energy supply contract with Engie Energía Perú for 170 MW from the Peruvian Electric System to support mine and port operations. The current average electricity demand is 120 MW with a maximum demand of 147 MW.

Power is supplied is from the National Interconnected Electric System (SEIN) via two 50 km long 220 kV transmission lines, from the Vizcarra substation to the Antamina substation. Once at the Antamina Operations substation, power is transformed to medium voltage (23 kV) to supply power to the process plant, concentrate pipeline system, shovels and drills, the pit dewatering system, truck and welding shop, water management facilities, and auxiliary facilities.

A set of diesel generators onsite provide backup power for the essential and critical load facilities.

In the port area, power is obtained from SEIN via a 66 kV transmission line from the 09 of Octubre substation to the PPL substation. The average port power demand is 2.6 MW.

19 MARKET STUDIES AND CONTRACTS

19.1 Markets

Four concentrate types are produced:

- Copper concentrate;
- Zinc concentrate;
- Molybdenum concentrate;
- Lead–bismuth concentrate.

Minera Antamina sells the copper, zinc, and molybdenum concentrates directly to its shareholders; therefore, no market studies have been conducted by Minera Antamina. Copper, zinc, and molybdenum production are distributed to the shareholders, according to their share participation in the company, which is: BHP, 33.75%, Glencore, 33.75%, Teck, 22.50%, and Mitsubishi,10.00%.

The sale of lead–bismuth concentrate has been directly managed by Minera Antamina's Commercial Department since the start of production in 2001, and includes sales by spot tenders to third parties as local sales.

19.2 Commodity Price Projections

Minera Antamina uses two information sources for commodity price forecasts:

- Forward-prices on a month-to-month basis provided by the London Metal Exchange (LME);
- Forecast commodity prices from major banks and financial institutions, compiled by Morgan Stanley and the Canadian Imperial Bank of Commerce (CIBC) who consolidate and deliver market observations from various brokers. For copper and zinc prices forecasts, additional information is sourced from Wood Mackenzie's metal prices outlook report.

Minera Antamina's market analysis team reviewed and integrated these reports to create a set of commodity "consensus prices". The forecast includes annual prices for the first five years and a long-term price projection for the remainder of the period. The team required a minimum of 10 different predictions for use in forecasting; typically, the reports include 30–50 predictions.

Following this review, Minera Antamina provides commodity price forecasts on an annual basis for the first five years of the cashflow model and for the Mineral Reserve estimate, then price forecasts revert to a long-term price for each commodity for the remainder of the cashflow (Table 19-1).

Table 19-1: Commodity Price and Exchange Rate Forecast, Mineral Reserves and Cashflow

Metal	Units	2025	2026	2027	2028	2029	2030 and Long-Term Price
Copper	US $/lb	3.51	3.64	3.68	3.68	3.66	3.54
Zinc	US $/lb	1.25	1.20	1.15	1.11	1.10	1.15
Silver	US $/oz	21.32	21.15	20.36	20.40	21.46	21.46
Molybdenum	US $/lb	12.91	12.25	11.46	11.08	11.10	11.10
Exchange rate	US$/PEN	3.95	3.78	3.75	3.72	3.71	3.71

The following approach is used:

- Years 1–3: an average of views on the consensus prices. For copper and zinc, this average includes consensus prices and Wood Mackenzie's metal prices outlook report;
- Year 4: an interpolation of consensus prices is used to establish a smooth transition to the long-term price. For copper and zinc, the interpolated price is averaged with Wood Mackenzie's outlook;
- Year 5: the average of the percentiles P5/P95 from the long-term consensus price views is used and maintained as long term price for subsequent periods, with outliers excluded to reduce variability. For copper and zinc, an interpolated consensus price is averaged with Wood Mackenzie's outlook;
- Year 6 forward: for copper and zinc, the forecast price is calculated as the average of the percentiles P5/P95 from the long-term consensus price views, excluding outliers to reduce variability. This average is then averaged with the price from Wood Mackenzie's metal price outlook report.

Mineral Resources use the price forecasts in Table 19-2.

19.3 Contracts

The copper, zinc and molybdenum concentrates are sold under three off-take sales agreements with the Minera Antamina shareholders as at the overall date of the Report. The agreements covers all of the concentrate production. Terms and conditions under the agreements are applicable equally to all of the Minera Antamina shareholders.

The copper and zinc off-take agreements have a five-year term, which renews automatically for successive five-year terms unless terminated in accordance with the terms of the off-take agreement. The current agreements automatically renewed on January 1, 2025. However, in accordance with the terms of the off-take, the parties are currently reviewing the commercial terms to consider amendments that may be required to reflect the predominant market trends and conditions.

Table 19-2: Commodity Price and Exchange Rate Forecast, Mineral Resources

Metal	Units	Value
Copper	US$/lb	3.50
Zinc	US$/lb	1.25
Silver	US$/oz	24.63
Molybdenum	US$/lb	13.30
Exchange rate	US$/PEN	3.71

The molybdenum off-take agreement has a three-year term. The current agreement will expire in December 2025, but will be automatically renewed for successive three-year terms unless terminated in accordance with the terms of off-take agreement.

For each off-take agreement, the shareholders are obliged to receive and purchase all concentrates. Each shipment is currently delivered under cost, insurance, and freight (CIF, Incoterms 2020 rules) port of discharge free out terms.

Minera Antamina has entered into spot, medium- and long-term contracts with international shipping companies to deliver concentrates sold under the off-take sales agreements with shareholders.

Copper and zinc concentrates are loaded in bulk through the Punta Lobitos port. Molybdenum concentrates are loaded in containers in big bags through the Callao port.

Minera Antamina enters into lead–bismuth spot contracts directly with third parties; this is not managed by the respective shareholders in Minera Antamina. Lead–bismuth concentrates are delivered ex-works (EXW, Incoterms 2020 rules) in big bags to a third-party warehouse in Callao.

Minera Antamina has contracts in place with third-parties that provide services, supplies, and construction service to the site. Contracts are managed by Minera Antamina's logistics department.

19.4 QP Comment on Item 19 "Market Studies and Contracts"

All concentrate sales are to shareholders under off-take agreements, and no market studies were performed. Terms and conditions under the agreements are applicable equally to all of the Minera Antamina shareholders. In the QP's opinion, the terms of the off-take agreements are within the reasonable range of market value treatments in similar arm's length third party transactions. Antamina does not hedge its production prices.

Metal price assumptions have a reasonable basis, and can be used in Mineral Resource and Mineral Reserve estimation and in the cashflow analysis that supports the Mineral Reserves.

Contracts other than concentrate sales contracts are typical of such third-party contracts in Peru that the QP is familiar with.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Environmental Considerations

20.1.1 BASELINE STUDIES

Baseline and supporting studies have included:

- Soil quality: soil quality and its characteristics are consistent with Andes-type soil, which is influenced by the underlying geology;
- Surface water quality: initial baseline samples for the Ayash and Carash basins recorded alkaline water, reflective of the surrounding geology. Some elements, such as arsenic, were slightly above the Environmental Quality Standard (EQS) — Category 3, which sets the requirements for water suitable for animal consumption and crop irrigation requirements;
- Air quality: all results fell within the ranges stipulated in the applicable standards;
- Biodiversity: in general, the flora and fauna present in the Project area have changed since mining operations began. There is a trend toward an increasing number of species between monitoring episodes. Biodiversity descriptions focus on the main biological indicators of principal land and water taxonomies;
- Project social and economic influence: the mine site social area of influence covers the San Marcos district and part of the San Pedro de Chana district.

20.1.2 MONITORING

20.1.2.1 Tailings

Tailings are classified according to geometallurgical type, and include copper, copper–zinc, and bornite ore types. Static and kinetic tests have found that all tailings, except for those originating from the bornite–copper and bornite–copper–zinc ore types, can be classified as potentially acid generating. These types are stored in the TSF.

20.1.2.2 Waste Rock

Waste rock is stored based on its potential to generate acid rock drainage. Potentially acid generating waste is stored in the East and Yanacancha WRSFs, and non-potentially acid generating waste is stored in the Tucush WRSF. Additional information on the WRSFs is included in Section 16.6.3.

20.1.2.3 Water

The water management system for the Ayash basin includes managing the Tucush WRSF, TSF dam seepage, and upstream diverted water flows. The water management in the Pampa Moruna basin includes managing water from the open pit.

Key water management features are shown in Figure 20-1. Typical contact and non-contact water management activities are summarized in Table 20-1.

20.1.2.4 Soils

Minera Antamina has instituted the following measures for soil management:

- Erosion and sediment control;
- Management of contaminated soil;
- Topsoil management;
- Rejected material handling.

20.1.2.5 Air Quality

Steps to address air quality include dust suppression, covering stockpiles, planting of windbreaks and vegetative cover, and regularly carrying out air quality monitoring to verify that environmental air quality measures are working as anticipated.

20.2 Closure Considerations

The Ministry of Energy and Mines (MINEM) must be provided with a regulatory closure plan that includes each facility within the mining operation (Table 20-2).

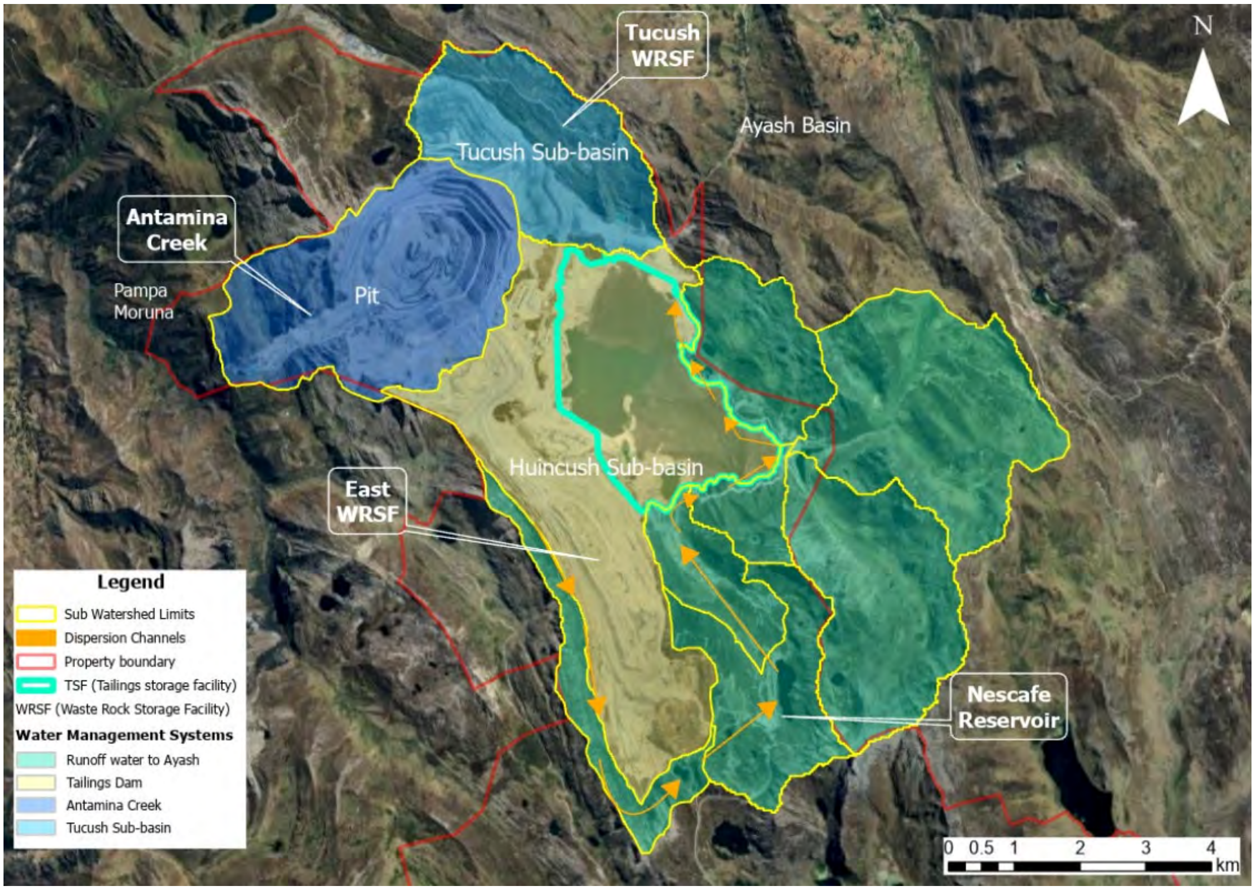
The plan has to cover the mine's short-, medium-, and long-term impacts and document proposed mitigation strategies to protect the environment.

The approved closure plan must be updated three years after its original submission, and on a five-yearly basis thereafter. If an amendment is made to an existing EIA, the closure plan must be updated within a year to reflect the EIA changes.

For the Antamina operations, the update of the regulatory closure plan must be submitted in February 2025, as a result of the approval of the MEIA in 2024. The document is expected to be approved during 2025, and new financial guarantees will be posted in 2026.

Closure plans must be accompanied by payment of a financial guarantee.

Closure cost estimates incorporate costs related to progressive closure, final closure, and post closure.



Note: Figure prepared by Minera Antamina, 2024.

Figure 20-1: Water Management

Table 20-1: Water Management

Basin/Sub-Basin	Facility	Contact Water	Non-Contact Water
Ayash	Yanacancha WRSF	Surface water and seepage is collected at various points at the WRSF toe and directed to the TSF	Two diversion channels on the west and east side of the WRSF, which flow into the Huayaoccocha lake and then to the Nescafé reservoir
	Tucush WRSF	Seepage collection system located at the toe of the WRSF that captures and directs the seepage and runoff from the dump towards the sedimentation ponds. Sedimentation ponds discharge by gravity into the Ayash basin. A pump-back system in the sedimentation ponds is activated when the water does not meet the minimum water quality standard and the water is then diverted to the TSF.	No diversion channels; the karstic nature of the north side of the Tucush Valley limits the amount of surface runoff
	TSF	Supernatant water is reclaimed for ore processing. Seepage is captured at the seepage collection system and, depending on the water quality, is pumped-back to the TSF or discharged to the Ayash basin	Diversion channels are located east and south to the tailings facility to divert non-contact water to the Ayash basin
Pampa Moruna	Open pit	Run-off is collected through a system of channels and ponds located within the pit limits. Sediment control and physical-chemical treatment is conducted before the water is discharged to a control point	

Table 20-2: Closure Planning

Closure Period	Duration	Major Activities To Be Completed
Progressive closure	2035–2036	Re-sloping of WRSFs, ground leveling, starting of placement of organic soil, revegetation, and implementation of water management structures; start the construction of a water treatment plant for seepage from the East WRSF; construction of the emergency spillway on the tailings dam right abutment; construction of a safety berm around the open pit where necessary; as well as the bulkhead construction at the overflow outlet of the open pit in Antamina creek.
Final closure	2037–2041	Dismantling, demolition, reclamation, and disposal of the concentrator plant facilities, pipeline and associated infrastructure, power transmission lines, tailings transportation and distribution systems, and associated infrastructure. Finishing of placement of covers, organic soil, revegetation of WRSFs and TSF, and implementation of water management structures; end construction of a water treatment plant to treat seepage from the East WRSF; and beginning of pit lake formation.
Post-closure	2042–2071 or until closure objectives are met	The water treatment plant, the sedimentation dam facility (polishing pond) and the tailings dam seepage pumping system, will remain for the post-closure period. The main activity will be post-closure monitoring and maintenance activities.

The total estimated closure cost has the following components:

- Direct costs;
- Indirect costs are calculated at a percentage of the direct costs (engineering and inspections at 5%, overhead at 15%, utility at 10% and contractor mobilization/demobilization at 3%);
- 25% of the sum of direct and indirect costs was included as contingency.

Inflation and discount rates used for the annual financial guarantees were assumed to accrue progressively until 2036.

The estimation base for the cash flow uses the unitary costs of the approved 2023 regulatory closure plan developed by Klohn Crippen Berger (KCB, 2023), adjusted to the current designs of life of mine components up to 2036. This amount was scaled for inflation to 2024, resulting in a total of US $831 M, which includes WRSF re-sloping.

20.3 Permitting Considerations

20.3.1 PERMIT HISTORY

The original EIA was reviewed and approved by the relevant Peruvian regulatory bodies and was also presented in a public stakeholder forum in 1998. Between 1999 and 2006, Minera Antamina submitted six amendments to the original EIA which were approved by MINEM. These amendments included modifications to the mining site, concentrate pipeline, ancillary buildings, and changes to the environmental management and monitoring plan.

In 2007, the original EIA was updated to reflect increased mine and plant production, and an expansion in ore reserves from the original project definition. This process also involved public stakeholder forums leading up to the approval of the modified EIAs by MINEM, in April 2008.

Since 2008, Minera Antamina has submitted four amendments to the EIA, three relating to the installation of an auxiliary electricity line, and the other relating to an expansion in ore reserves and mine plan optimization. The fourth amendment was approved by MINEM in 2011.

In addition, Minera Antamina has submitted six minor amendments to the EIA which were approved in 2011. The modifications related to new expansions and modifications of the ancillary infrastructure; these modifications did not generate new or significant negative environmental impacts under the regulatory code.

On 15 February, 2024, after a detailed evaluation process, the National Environmental Certification Authority approved Minera Antamina's MEIA.

The MEIA approval in 2024 allowed Minera Antamina to extend the LOM from 2028 to 2036. The MEIA addressed extensions to, and optimizations of, the open pit, WRSFs and TSF. These projects were designed with strict engineering protocols that prioritized safety and integrated environmental and social sustainability considerations.

20.3.2 CURRENT PERMITS

Minera Antamina holds more than 450 licenses and permits in relation to its mining activities. The licenses and permits are managed using a computerized management system with alerts on renewals and expiry dates.

Key permits cover:

- Land tenure and access permits;
- Construction and operations licenses;
- Government permits;
- Environmental licenses;
- Closure plan;
- Water-management licenses and authorizations.

All permits were in good standing as at the Report effective date. Minera Antamina holds all of the key permits to support operations to 2036; however, some of the permits will require renewal or modifications during the LOM. Based on previous permit approval processes for the mine, obtaining the renewals or modifications is reasonably well understood, and within Minera Antamina's envisaged expected scope and timing in the LOM plan.

20.4 Social Considerations

Social management for the Antamina Operations consists of three major plans:

- Community Relations Plan;
- Social Concertation Plan;
- Community Development Plan.

The social management plans are summarized in Table 20-3.

These support social commitments and obligations in the plans and activities included in the LOM. Plans will be supported by the approved Modification of the Environmental Impact Assessment (MEIA).

Table 20-3: Social Management Plans

Plan	Programs	Activities
Community Relations Plan	Internal and external communications program	Dissemination of prioritised social management programs and/or projects, through newsletters.
		Social responsibility activities that promote values and respect for local culture and the environment.
		Guided tours for the populace within the area of direct and indirect influence of the mine and port.
		Distribution of information material about social management programs, projects, and initiatives in the direct area of influence of the mine and port.
		Radial spots in the direct area of influence of the mine and port.
		Continuity of operation of the permanent information office.
		Panels in high traffic spaces in the direct area of influence of the mine and port.
		Meetings with interest groups (dialogue, development, or environmental spaces) and periodic field visits in the direct area of influence of the mine and port.
		Dissemination of results of environmental monitoring in the direct area of influence of the mine.
		Dissemination of environmental monitoring procedures carried out in the direct area of influence of the mine.
		Informative signage on shared use path detours and sections of shared use roads owned by Minera Antamina.
		Dissemination of blasting procedures in areas located <500 m from blasting occurring within the current pit limits and part of the regular mining activities, particularly in Jatun Piruro and Ushupallares.
		Dissemination of claims procedure.
	Social management system	
Social Concertation Plan	Social contingencies program	Application of the claims management procedure in the direct area of influence of the mine and port and the indirect area of influence of the mine and port (pipelines and power transmission line).
		Implementation of participatory environmental procedures in the direct area of influence of the mine and port.
		Activities for monitoring social commitments and communicating progress of such commitments (e.g. round tables, multi-actor spaces, traditional media,

Plan	Programs	Activities
		social networks) in the direct area of influence of the mine.
		Implementation of the hiring procedure for permanent local labour in the in the direct area of influence of the mine and port, and the indirect area of influence of the mine and port (pipelines and power transmission line).
		Identification and monitoring of community perceptions every three years in the direct area of influence of the mine and port.
Community Development	Plan local/productive development	Program activities to promote annual employability in the direct area of influence of the mine and port.
		Financing or co-financing of the preparation of studies for biennial water management in the direct area of influence of the mine.
		Support with materials to improve irrigation systems. Annual in the direct area of influence of the mine.
		Activities to improve annual livestock production in the direct area of influence of the mine.
	Local/social development	Program support with materials, supplies and/or equipment to improve the offer of annual health facility care in the direct area of influence of the mine.
		Co-financing of health care activities in locations in the direct area of influence of the mine.
		Support in financing medical care in cases of emergency, urgency, or annual complex illness in in the direct area of influence of the mine.
		Financing of activities to prevent malnutrition, anaemia. Guide to healthy and safe eating. Semester in the direct area of influence of the mine.
		Financing of activities to improve permanent educational quality in the direct area of influence of the mine.
		Accessibility. Transfer of community members between sentry boxes. Permanent.
		Accessibility. Barge trailer. Permanent.
	Local development/cultural tourism program.	Co-financing of tourism development activities in San Marcos annually within the direct area of influence of the mine and port.
		Support of cultural activities in the Ayash and San Marcos Basin, quarterly, in the direct area of influence of the mine.
		Support of activities that contribute to promoting the use of the Quechua language. Annually in the direct area of influence of the mine.

Plan	Programs	Activities
	Local development program/acquisition of local products or services	Acquisition of products and services in the direct area of influence of the mine, on annual basis.
		Acquisition of products and services from the Ancash region, annual in the direct area of influence of the mine and port.
		Annual training for local suppliers in the direct area of influence of the mine.
	Local capacity strengthening program/local management	Training and technical assistance activities for local authorities or grassroots organizations on public management issues, annual in the direct area of influence of the mine.
	Local capacity strengthening program/technical and productive training	Activities to promote annual employability in the direct area of influence of the mine and port.
		Financing training activities and/or annual productive technical assistance in the direct area of influence of the mine.
		Financing training and/or business technical assistance activities aimed at entrepreneurs and productive organizations annually in the direct area of influence of the mine.
	Local capacity strengthening program / technical assistance	Annual training activities and/or technical assistance to the local Environmental Committees in the direct area of influence of the mine and port.

21 CAPITAL AND OPERATING COSTS

21.1 Capital Cost Estimates

Capital cost forecasts are based on a 12-year mine life, from 2025 to 2036, with the last year of operations being a partial year.

Capital cost estimates for infrastructure projects are primarily based on pre-feasibility (cost estimate accuracy of -15% to +25% and contingency of 15–25%) and feasibility (cost estimate accuracy of -10% to 15% and contingency of 10–15%) studies based on Minera Antamina's Standard Investment Management System (SIGIA), and includes Minera Antamina's operating experience with incorporating in-place vendor agreements and contracts.

The estimated capital expenditures reflect funds to support operations, including funding for new facilities, growth infrastructure, equipment additions or replacements, and in-pit drilling.

Key elements include:

- TSF raises, including the 4,165 and 4,195 masl dam raises, and associated projects related to auxiliary dam construction, water management, tailings pumping systems, and infrastructure relocation;
- Mine equipment, including additional and replacement equipment for mine fleet;
- Tucush full WRSF extension: TUSEF and associated projects such as road construction, infrastructure relocation, foundation preparation, and water management;
- New primary crusher: ex-W1 ore mechanized system, including execution, and sustaining;
- Capitalized components for ball/SAG mill maintenance, pipeline sustaining capital costs, and Punta Lobitos port sustaining capital costs;
- In-pit drilling.

Capital cost estimates for the LOM are summarized in Table 21-1 and total US$4,022 M.

Table 21-1: Capital Cost Estimate Summary Table (nominal terms)

Area	Item	Units	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	Total
Mine	Sustaining mine equipment	US$M	254	337	271	114	48	7	20	6	9	3	6	—	***1,073***
	Sustaining truckshop	US$M	3	13	7	—	—	—	—	—	—	—	—	—	***23***
	Mine utilities/technology	US$M	25	3	—	—	—	—	—	—	—	—	—	—	***30***
	WRSF extensions	US$M	40	38	42	40	11	—	—	—	—	—	—	—	***172***
Process	Tailings site	US$M	225	279	243	192	158	147	104	122	30	44	56	11	***1,610***
	Sustaining ball/SAG mill maintenance	US$M	24	21	22	49	49	31	—	30	—	18	—	—	***244***
	Sustaining pipeline/ort projects	US$M	17	9	6	4	7	6	3	1	4	6	2	—	***65***
Other		US$M	238	150	75	64	60	29	28	32	28	37	46	20	***807***
Total estimated capital costs		***US$M***	***825***	***850***	***667***	***464***	***333***	***221***	***155***	***190***	***70***	***108***	***109***	***31***	***4,022***

21.2 Operating Cost Estimates

Operating cost forecasts are based on a 12-year mine life, from 2025 to 2036, with the last year of operations being a partial year.

Operating costs are related to the mine, concentrator, maintenance activities, and support services.

Cost estimates are based on actual mine operating costs and budgetary estimates.

Consumption rates are based on history performance of the equipment and consumable prices (such as dollar per gallon of diesel or dollar per kg explosive) are used to estimate the variable costs.

Key cost inputs include:

- Mine operation includes drilling, blasting, loading, and hauling (including labor and consumables), mine supervision and technical services (geotechnical, ore control, planning, and control);
- Mine and mill equipment maintenance expenditures, including fixed and variable components;
- Mill operation includes SAG, ball and pebble mills power, labour, and consumables (including steel and liners), maintenance, and contractors.
- Flotation circuit: reagents for the various concentrates, electricity, labour, and maintenance;
- Water system: electricity, labour, maintenance, and contractors;
- Pipeline and concentrate transport: including electricity, labour, maintenance, and contractors;
- Port operating expenditures: including electricity, labour, maintenance, and other annual fixed and variable costs;
- Support services: mine-site support services for finance, administration, human resources, environment, health and safety, operations management, engineering and project management, community relations, and business development.

Operating cost estimates for the LOM are summarized in Table 21-2 and total US$14,681 M. Over the LOM, mine operating costs are estimated to average US$6.70/t mined and process costs are estimated to average US$7.90/t processed.

Table 21-2: Operating Cost Estimate Summary Table (real terms)

Area	Units	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	Total
Mine	US$M	594	621	597	627	698	675	623	585	465	413	371	257	***6,526***
Maintenance	US$M	385	401	370	359	393	359	325	304	269	277	246	136	***3,822***
Concentrator	US$M	220	212	214	221	232	232	214	209	182	217	189	62	***2,403***
Port	US$M	8	8	8	8	8	9	8	9	8	8	7	6	***97***
Support services	US$M	180	166	162	153	160	161	148	145	142	139	140	137	***1,833***
Total estimated operating costs		***1,387***	***1,409***	***1,351***	***1,367***	***1,492***	***1,435***	***1,318***	***1,252***	***1,066***	***1,053***	***953***	***597***	***14,681***

22 ECONOMIC ANALYSIS

Teck is using the provision for producing issuers, whereby producing issuers may exclude the information required under Item 22 for technical reports on properties currently in production and where no material production expansion is planned.

Mineral Reserve declaration is supported by overall site positive cash flows and net present value assessments.

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, based on the review of data and information available for this Report.

25.2 Ownership

There are four shareholders with the following ownership percentages: BHP plc, 33.75%; Glencore plc, 33.75%; Teck, 22.50%; and Mitsubishi Corporation,10.00%.

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

Information from Minera Antamina in-house experts support that the mining tenure held is valid and sufficient to support a declaration of Mineral Resources and Mineral Reserves.

Minera Antamina holds sufficient surface rights to allow mining activities.

Royalties are payable to the Peruvian Government. This royalty is calculated on the quarterly operating income derived from the sale of concentrates at market value, discounting the cost of those sales and any operating expenses (including sales and administrative expenses).

Under a long-term streaming agreement with FN Holdings ULC (FNH), a subsidiary of Franco-Nevada Corporation, Teck has agreed to deliver silver to FNH equivalent to 22.5% of the payable silver sold by Minera Antamina. The streaming agreement restricts distributions from Teck Base Metals Ltd., the subsidiary that holds Teck's 22.5% interest in Minera Antamina, to the extent of unpaid amounts under the agreement if there is an event of default under the streaming agreement or an insolvency of Teck. Minera Antamina, which owns and operates Antamina, is not a party to the agreement and operations are not affected by it.

25.4 Geology and Mineralization

The Antamina deposit is considered to be an example of a giant skarn deposit.

The understanding of the Antamina deposit settings, lithologies, mineralization, and the geological, structural, and alteration controls on mineralization is sufficient to support estimation of Mineral Resources and Mineral Reserves.

Opportunities remain find additional skarn inside the delimited intrusive zone of the deposit, especially in areas where there has been limited drilling. There are intrusions below the 4000 m elevation that have not been investigated for skarn potential, though higher elevations within the intrusive bodies do show skarn development.

25.5 Drilling, and Sampling

The exploration programs completed to date are appropriate for the deposit style.

Sampling methods are acceptable for Mineral Resource and Mineral Reserve estimation.

Sample preparation, analysis and security are generally performed in accordance with exploration best practices and industry standards at the time the information was collected.

The quantity and quality of the logged geological data, collar, and downhole survey data collected in the exploration and infill drill programs are sufficient to support Mineral Resource estimation.

No material factors were identified with the data collection from the drill programs that could significantly affect Mineral Resource or Mineral Reserve estimation.

The sample preparation, analysis, and security practices are acceptable and meet industry-standard practices at the time that they were undertaken and are sufficient to support Mineral Resource and Mineral Reserve estimation.

Minera Antamina initiated QA/QC measures in 2003. With progressive years, the QA/QC protocol became more comprehensive and detailed. QA/QC submission rates meet industry-accepted standards at the time of the completion of the various campaigns.

The QA/QC programs did not detect any material sample biases in the data reviewed that supports Mineral Resource and Mineral Reserve estimation.

25.6 Data Verification

The data verification programs concluded 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 and Mineral Reserve estimation.

25.7 Metallurgical Testwork

Metallurgical testwork and associated analytical procedures were appropriate to the mineralization type, appropriate to establish the optimal processing routes, and were performed using samples that are typical of the mineralization styles found within the various mineralized zones.

Samples selected for testing were representative of the various types and styles of mineralization. Samples were selected from a range of depths within the deposit. Sufficient samples were taken so that tests were performed on sufficient sample mass, including individual tests to assess variability.

The correlations between head grade and recovery are well understood by process plant staff. Using the predictive recovery equations, annual metal production targets can be calculated from the production plans. The copper and zinc metal production represents 90% of total revenues and these show very good correlation between actual versus predicted recoveries, based on ore type.

The major deleterious elements are bismuth and arsenic. These are managed using campaigning. Geometallurgical types M1 and M4B are campaign treated to produce high value, high quality copper and zinc concentrates.

Geometallurgical types M2 and M2A are campaign treated to produce separate copper concentrates with higher bismuth and arsenic grades. While there are processes within the industry that can be used to bismuth and arsenic grades in concentrates, installation of these at Antamina would not be cost effective, due to the relatively small amount of higher bismuth and arsenic grade copper concentrates.

Geometallurgical types M5 and M6 are always processed separately from the M1 and M4B geometallurgical types to minimize contamination of the higher-grade copper concentrates with the bismuth and arsenic contents associated with the bornite ore.

The concentrates are produced and transported in separate batches to the port, where port storage is designed to keep concentrate batches separate. Concentrate blending can be completed at the port by mixing concentrate qualities to meet concentrate specifications as required.

25.8 Mineral Resource Estimates

The Mineral Resource estimation for the Project conforms to industry-accepted practices and is reported using the 2014 CIM Definition Standards.

Factors that may affect the estimate include changes to metal price and exchange rate assumptions; changes to the assumptions used to generate the copper equivalent grade cut-off grade and the OVPHR; changes in local interpretations of mineralization geometry and continuity of mineralized zones; changes to geological and mineralization shapes, and geological and grade continuity assumptions; density and domain assignments; changes to geotechnical assumptions including pit slope angles; changes to mining and metallurgical recovery assumptions; changes to the input and design parameter assumptions that pertain to the conceptual pit constraining the estimates potentially amenable to open pit mining methods; changes to the input and design parameter assumptions that pertain to the conceptual mineable shapes constraining the estimates potentially amenable to underground mining methods; and assumptions as to the continued ability to access the site, retain mineral and surface rights titles, maintain environment and other regulatory permits, and maintain the social license to operate.

25.9 Mineral Reserve Estimates

The Mineral Reserve estimation for the Project conforms to industry-accepted practices and is reported using the 2014 CIM Definition Standards.

There is approximately 193 Mt of mineralization classified as Measured and Indicated that is contained between the Mineral Reserve pit design and the Mineral Resource pit shell that was not converted to Mineral Reserves. Low-grade stockpiles are not included in the LOM plan.

Factors that may affect the estimate include: metal price and exchange rate assumptions; changes to the assumptions used to generate the OVPHR cut-off; changes in local interpretations of mineralization geometry and continuity of mineralized zones; changes to geological and mineralization shapes, and geological and grade continuity assumptions; density and domain assignments; changes to geotechnical assumptions including pit slope angles; changes to hydrological and hydrogeological assumptions; changes to mining and metallurgical recovery assumptions; changes to the input and design parameter assumptions that pertain to the open pit shell constraining the estimates; and assumptions as to the continued ability to access the site, retain mineral and surface rights titles, maintain environment and other regulatory permits, and maintain the social license to operate, and the remaining TSF storage capacity at elevation 4,195 masl.

25.10 Mining Methods

The mining operations use conventional open pit mining methods and conventional equipment. All of the mining infrastructure (including the WRSFs) are located within property boundaries.

The pit design includes four mining phases. Phases 8, 9, and 10 are currently in production. Phase 11 is planned for 2025.

Maximum mining and movement rates are approximately 249 and 297 Mt/a, respectively, and are constrained by mining restrictions; maximum sinking rate of eight benches per year; mine equipment performance, availability, and utilization; and the area within the mining bench.

The maximum processing rate is approximately 53.6 Mt/a, and is constrained by variable processing rate for each geometallurgical type, overall mill availability and utilization (this includes the primary crusher), and minimum feed to plant for two days by geometallurgical type campaign.

The remaining mine life is 12 years, from 2025 to 2036, with 2036 being a partial year. About 546 Mt of ore and 1,535 Mt of waste are planned to be mined from the open pit with an average stripping ratio of 2.79. Phase 10 has the highest stripping ratio of the remaining pit phases at 3.1.

On average, chalcopyrite copper-bearing ores are the most common (66.5%), followed by chalcopyrite copper–zinc-bearing ores (24.7%), and the balance (8.8%) corresponds to bornite-rich copper- and copper–zinc-bearing ores.

Equipment numbers are expected to peak in 2029.

25.11 Recovery Methods

The process plant design was based on metallurgical testwork and industry-standard practices. The design is conventional to the mining industry and has no novel parameters. The plant has been in operation since 2001.

Plant facilities consist of crushing, grinding and flotation circuits. The Antamina flotation flowsheet is designed to provide the flexibility to treat the eight different ore types as they are campaigned through the plant. Consequently, the plant is operated differently for different ore types. The ore is amenable to traditional differential flotation techniques using standard reagent schemes. The distinct ore types are campaigned through the plant using different plant configurations to optimize the recovery of payable metals and provide the best opportunity to manage product quality.

A concentrate pipeline transports the final copper and zinc concentrates from the mine site to the port at Punta Lobitos in the department of Huarmey. A single pipeline is used to transport both types of concentrate (depending on the ore type being campaigned) on a batch basis, using a water plug between campaigns.

The lead–bismuth–silver and molybdenum concentrates are dewatered at the mine site, packaged in bags, and transported to various ports or clients using highway trucks.

Reagents and consumables are conventional for the flowsheet used. Power is obtained from a number of sources through the Chilean grid. Fresh water is obtained from an on-site storage reservoir, and process water primarily from TSF reclaim.

25.12 Infrastructure

Most of the key infrastructure required for the remaining LOM is in place and is operational. Additional infrastructure that will need to be constructed for the remaining LOM includes realigned access and shared-use roads; additional haul roads; replacement ore crusher; one WRSF extension; and two TSF raises.

There are two remaining raises to take the TSF to the environmentally permitted maximum elevation of 4,195 masl. The final design has a permitted capacity of about 1,527 Mt (844 Mm^3), approximate final footprint of 905 ha, and maximum height of 310 m. The mine plan is constrained by the permitted TSF capacity. The TSF has sufficient capacity for the LOM plan.

Site water management includes management of contact water by way of storage ponds and collection of seepage water. Non-contact water is managed using water diversion structures.

Power is supplied by SEIN.

25.13 Market Studies and Contracts

Minera Antamina sells the copper, zinc, and molybdenum concentrates directly to its shareholders; therefore, no market studies have been conducted by Minera Antamina. The Minera Antamina shareholders are obligated to take all of these concentrates, based on their attributable interest in Minera Antamina.

The off-take agreement terms are within the reasonable range of market value treatments in similar arm’s length, with third party transactions.

The sale of lead–bismuth concentrate has been directly managed by Minera Antamina’s Commercial Department since the start of production in 2001, and includes sales by spot tenders as local sales.

Metal price assumptions are determined using a consensus approach to long-term price forecasts, have a reasonable basis, and can be used in Mineral Resource and Mineral Reserve estimation and in the cashflow analysis that supports the Mineral Reserves.

Contracts other than concentrate sales contracts are typical of such third-party contracts in Peru that the QP is familiar with.

25.14 Environmental, Permitting and Social Considerations

Environmental permits have been granted and were supported by baseline and later studies.

Key environmental management areas include the TMF, waste rock, water, soils, and air quality.

Minera Antamina holds more than 450 licenses and permits in relation to its mining activities. The licenses and permits are managed using a computerized management system with alerts on renewals and expiry dates. All permits were in good standing as at the Report effective date. Minera Antamina holds all of the key permits required to support the LOM plan.

Minera Antamina's social management is governed under the multi-stakeholder model, which promotes the role of all participants by strengthening their capacities under the Sustainable Development Goals. Minera Antamina has implemented several measures to address potential social impacts in the area of social influence generated by mining activities and to strengthen the bonds of trust between the population and the company. These are detailed in a social and community agreement plan established under the Community Relations Plan.

25.15 Capital Cost Estimates

Capital cost forecasts are based on a 12-year mine life, from 2025 to 2036, with the last year of operations being a partial year. The capital expenditures reflect funds to support operations, including funding for new facilities, growth infrastructure, equipment additions or replacements, and in-pit drilling. Capital cost estimates for infrastructure projects are primarily based on pre-feasibility and feasibility studies based on Minera Antamina's SIGIA, and includes Minera Antamina's operating experience with incorporating in-place vendor agreements and contracts.

Capital cost estimates for the LOM total US$4,022 M.

25.16 Operating Cost Estimates

Operating cost forecasts are based on a 12-year mine life, from 2025 to 2036, with the last year of operations being a partial year. Operating costs are related to the mine, concentrator, and maintenance activities. Cost estimates are based on actual mine operating costs and budgetary estimates.

Operating cost estimates for the LOM total US$14,681 M. Over the LOM, mine operating costs are estimated to average US$6.70/t mined and process costs are estimated to average US$7.90/t processed.

25.17 Economic Analysis

Teck is using the provision for producing issuers, whereby producing issuers may exclude the information required under Item 22 for technical reports on properties currently in production and where no material production expansion is planned.

Mineral Reserve declaration is supported by overall site positive cash flows and net present value assessments.

25.18 Risks and Opportunities

25.18.1 RISKS

Minera Antamina maintains a risk register using SIGRA software. This is reviewed and updated by senior management on a quarterly basis. Risks have a systematic control process in place, with periodic audits and reviews of the management for each individual risk.

Key risks identified at 31 December 2024 include:

- Accidents during personnel transport, due to such causes as vehicle mechanical failure, interactions with other, non-Minera Antamina, vehicles using the access route, severe weather conditions or reckless driving;
- Risk of collision due to intensive truck traffic and congestion in the open pit access ramps affecting haul trucks and other vehicles as a result of driver-related issues, weather conditions, poor road maintenance, or mechanical conditions of the equipment;
- Accidents during the transportation of hazardous material to site, causing spills on the public road;
- Social conflict in the Antamina area of influence, for example conflicts that could be caused by breaches of commitments, political and/or social instability;
- Risk of rocks falling from slopes and upper banks in the open pit, for example, caused by blasting effects, loose rocks in walls and slopes, plugged benches, or interaction of works in benches and phases;
- Sub-optimal Mineral Resource estimates caused by, for example, limitations in drilling, failures in the sample capture and processing, assumptions in the modelling process, and assumptions as to forecasts used in assessing reasonable prospects of eventual economic extraction;
- Sub-optimal Mineral Reserves estimation, caused by, for example, uncertainty in the metallurgical behaviour of the ore, operational dilution, deficiencies in the construction of the block model, operational assumptions, and assumptions as to forecasts used in mine planning and the economic analysis that supports the Mineral Reserves.

Risks to the production plan that were assigned a "moderate" classification as at 31 December 2024, include:

- Failure to comply with the performance assumed for loading equipment and trucks fleet, which could be caused by, for example, poor haul road maintenance, inefficiencies in the mining cycle, unexpected traffic or equipment congestion, lack of expertise on the part of the equipment operators, and labour conditions;
- Production of out-of-specification concentrate due to the unexpected higher presence of deleterious metals such as bismuth, arsenic, fluorine, and lead. Engineering studies are underway to support construction of a new filtration plant, which would allow the processing of high-lead ores;
- Failure of the grade control drilling programs because of potential interference between drill rigs and active mining.

25.18.2 OPPORTUNITIES

Opportunities include:

- There is upside potential for the Mineral Resource estimates to be converted to Mineral Reserves once the studies, engineering, economics, and permits are completed for the infrastructure required for mining support;
- There are blocks classified as Inferred within the Mineral Reserve pit that may eventually be able to be upgraded to higher-confidence Mineral Resource categories and converted to Mineral Reserves with additional drilling and studies;
- Technological initiatives are under evaluation to potentially increase productivity and metal recovery, with resulting effects on project economics if successful. These initiatives include ore sorting, coarse-particle flotation, and tailings densification.

25.19 Conclusions

Under the assumptions in this Report, the Mineral Reserve declaration is supported by overall site positive cash flows and net present value assessments, which supports Mineral Reserves. The mine plan is achievable under the set of assumptions and parameters used in this Report.

26 RECOMMENDATIONS

As Antamina is an operating mine, the QPs have no material recommendations to make.

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