Santo Domingo Project

NI 43-101 Technical Report and Feasibility Study Update

Atacama Region, Chile

Effective date: July 31, 2024

Prepared for:

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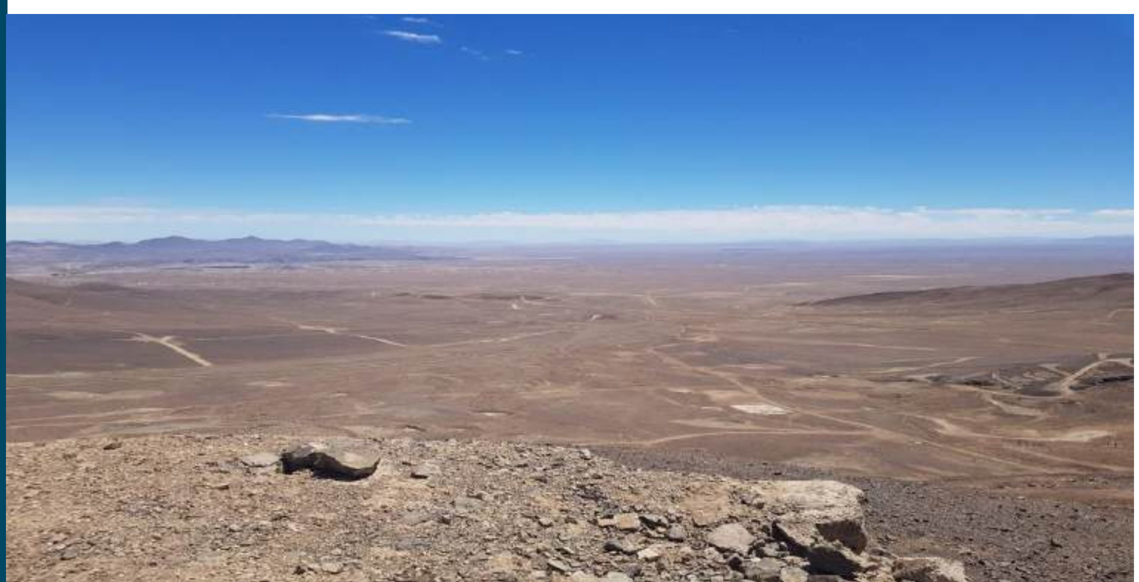

CERTIFICATE OF QUALIFIED PERSON
Peter Amelunxen, P.Eng.

I, Peter Amelunxen, P.Eng., certify that, I am employed as a Senior Vice President, Technical Services with Capstone Copper Corp. (“Capstone”), with an office address of Suite 2100, 510 West Georgia Street, Vancouver, B.C., Canada, V6B 0M3.

1. This certificate applies to the technical report titled "Santo Domingo Project, NI43-101 Technical Report on Feasibility Study Update Atacama Region, Chile", that has an effective date of July 31, 2024, and a report date of September 13, 2024 (the “Technical Report”).
2. I graduated from the University of Arizona, Tucson, Arizona, USA in 1998 with a Bachelor of Science Degree in Mining Engineering and from McGill University in Montreal, Quebec, Canada in 2004 with a Master of Engineering Degree.
3. I am a Registered Member of Engineers and Geoscientists British Columbia, P.Eng., EGBC# 59157.
4. I have practiced my profession for over 25 years. I have been directly involved in mine operations, including work as a metallurgist, senior process engineer and concentrator operations superintendent at base metals mining operations (2003-2008) in the USA and Peru, as a consulting metallurgist covering auditing, modelling, optimization and design of base metals and precious metals operations (2008-2018), and, since 2018, as an executive leading corporate technical teams in charge of geology, resource modelling, reserve estimation, mine planning, mineral processing, and tailings technical studies and strategic initiatives.
5. I have read the definition of “Qualified Person” set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects (“NI 43-101”) and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a “Qualified Person” for those sections of the Technical Report that I am responsible for preparing.
6. I visited the Santo Domingo Project Site on Thursday Nov 17, 2022, for a visit duration of 1 day.
7. I am responsible for all sections of the Technical Report.
8. I am not independent of Capstone Copper Corp. as independence is defined in Section 1.5 of NI 43-101.
9. I have been involved with the property as an employee of Capstone Copper Corp., since April 2022, overseeing continuous processing and metallurgical recovery improvements at Santo Domingo Project as part of my role as Senior Vice President, Technical Services.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: September 13, 2024

“Signed and Sealed”

Peter Amelunxen, P.Eng.

Important Notice

This report was prepared as a National Instrument 43-101 Technical Report by Capstone Copper Corp. (Capstone) with assistance from Ausenco Sustainability ULC, Ausenco Chile Limitada and Ausenco Services Pty Ltd (collectively Ausenco), collectively the Report Compilers. The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in the Report Compilers' services, based on i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report. This report is intended for use by Capstone subject to terms and conditions of its contracts with each of the Report Compilers. Except for the purposed legislated under Canadian provincial and territorial securities law, any other uses of this report by any third party are at that party's sole risk.

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1 SUMMARY

1.1 Introduction

This technical report (the Report) was compiled by Ausenco Sustainability ULC, Ausenco Chile Limitada, and Ausenco Services Pty Ltd (collectively Ausenco) for Capstone Copper Corp. (Capstone) to conform to the regulatory requirements of Canadian National Instrument (NI) 43-101 using the form 43-101 F1 Standards of Disclosure for Mineral Projects. The Santo Domingo Project involves the development of mining and processing facilities on Capstone's Santo Domingo Property (the Property), located in the Atacama Region (Region III) of the Republic of Chile.

1.2 Terms of Reference

The term "Property" is used in reference to the overall mineral tenure holdings that encompass the Santo Domingo property.

The Property is held 100% by Capstone, which prior to 2022 was Capstone Mining Corp. (Capstone Mining). The operating entity is Minera Santo Domingo Sociedad Contractual Minera (MSD), as the Chilean holding company for the Property.

A previous technical report was issued with an effective date of 19 February 2020, filed on 24 March 2020 (referred to as the 2020 Technical Report).

The Report provides an update on the project. The project includes conventional open pit mining of a copper and magnetite deposit with ore feeding a processing plant to produce separate copper–gold and iron concentrates, and thickened tailings. Tailings are stored on the Property. Filtered copper concentrate is trucked to the port and magnetite concentrate is transported by pipeline to the filtration plant at the port.

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

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

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

1.3 Property Setting

The portion of the Santo Domingo area that hosts the Santo Domingo deposits, and will host the mine and plant site areas, is located approximately 5 km southeast of the town of Diego de Almagro, 35 km northeast of Capstone's Mantoverde mine, in the Chañaral Province in the Atacama Region of northern Chile.

Current access to the planned mine and plant site area where the deposits are located is via the paved Pan-American Highway (Route 5 North) and a network of generally well-maintained paved roads. The deposits are about five hours travel time by road south from Antofagasta and two hours by road north from Copiapó.

The Atacama Region has well established infrastructure (roads and electrical transmission systems) and capacity (labour, support services) to serve the mining industry. However, there is currently no major infrastructure on the Santo Domingo property, except gravel roads for access to the concessions and drill sites and some platforms for temporary facilities. Highway C-17 connecting Diego de Almagro and Copiapó is paved and currently passes immediately east of the proposed mine–plant site area. C-17 bypass road works have been partially completed to route this public highway around the future project infrastructure. The nearby town of Diego de Almagro (population of around 7200 people in 2017 census) is connected to the regional power grid and can provide some support services for the planned operations.

The climate is generally warm, dry, and clear in all seasons. The area of the proposed mine site is classified as interior desert, whereas the proposed port location is in a coastal desert regime. Rainfall is low and concentrated in the winter months. Mining activities are expected to be possible on a year-round basis.

Elevations in the mine–plant site area range from approximately 1,000–1,300 masl. Vegetation is very sparse.

The area is likely to have high seismicity, and the site is considered to be within Zone 3 and Zone 2 according to the new Chilean standard NCh 2369, with a peak ground acceleration value of 0.4 g.

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

Minera Santo Domingo, a Capstone subsidiary, is a mining company that is legally recognized under the laws of the Republic of Chile as a "Sociedad Contractual Minera."

Capstone holds two groups of concessions with a total of 119 claims, which cover a total of 29,221 ha and includes the areas of the planned mine site, plant area and auxiliary facilities including the port facilities. The tenure consists of 99 exploitation concessions and 20 exploration concessions. All the concessions are held in the name of Minera Santo Domingo.

The total concession area is divided as follows:

- 27,597 ha of exploitation and exploration concessions that encompass the area where the mine, plant, tailings storage facilities, construction and operations camp and ancillary facilities are planned.
- 1,624 ha of exploitation and exploration concessions that encompass the port area.

Concessions are surveyed as part of the grant process and are protected under Chilean law by payment of the annual mining license fees. All concession fees are in good standing and were fully paid up to 28 February 2025 and will continue to be paid on a regular basis as due, using a formal status tracking system.

The surface land in the Communities of Diego de Almagro, Caldera and Chañaral where Santo Domingo is located is owned by the state and managed and represented by the Ministerio de Bienes Nacionales.

Capstone has developed a legal strategy to obtain the necessary surface rights to cover the planned mine, plant, camps, tailings storage facility, mine waste disposal, pipelines, port, and transmission lines.

Capstone currently possesses three registered provisional surface rights (covering 107.56 ha) and 22 definitive surface rights (covering 5,386.57 ha).

Capstone does not need to file an application for water rights for the purpose of raw water supply for the Project in consideration that desalinated water will be used during the operational phase of the project. A maritime concession has been approved which will allow the extraction of sea water for processing in the desalination plant. Water for construction will be obtained from an authorized third-party provider. However, Capstone will need to apply for water rights in the Tailings Storage Facility based on its plan to establish a shallow well pumping system to intercept infiltrating process water and transfer that contact water back to the process plant for re-use.

1.5 History

Artisanal mining activities commenced in the general mine and plant site area during the early 19th century. The major commodities targeted were gold and iron. As a result, there are a significant number of small workings and pits throughout the planned mine–plant site area. However, most of the surface workings are typical of artisanal activities, being less than a few tens of metres in length.

Modern exploration commenced in 2002. Between 2002 and June 2011, exploration work conducted by Far West Mining Ltd. (Far West) included a regional airborne geophysical survey, geological mapping, surface and drainage sampling, an induced polarization (IP) survey, transient electromagnetic surveys, core and reverse circulation (RC) drilling and resource estimation. A preliminary assessment was conducted in 2008.

Capstone Mining acquired Santo Domingo from Far West in 2011 and completed a pre-feasibility study (2011 pre-feasibility study) in the same year. In 2014, Capstone Mining completed a feasibility study. Updates to the 2014 feasibility study were completed in 2018, resulting in the January 2019 Technical Report. In March 2020 Capstone Mining filed the 2020 Technical Report which included a summary of results of a preliminary economic assessment on an alternative development option for production of battery-grade cobalt sulphate via roasting.

In March 2022, Capstone Mining merged with Mantos Copper (Bermuda) Limited forming Capstone Copper Corp., the sole owner of the Santo Domingo Project.

1.6 Geology and Mineralization

Santo Domingo is part of the iron oxide–copper–gold (IOCG) type deposits located within the Cretaceous Iron Belt (CIB), which extend approximately 630 km by 40 km in Chile´s coastal range. The main regional structural feature is the Atacama fault zone, a ductile/brittle sinistral strike-slip and dip-slip crustal scale structure that parallels the coast of Chile for over 1,200 km. The Santo Domingo deposits lie on the east side of the Atacama fault, which, in this area, consists of numerous clusters of generally north–south structural breaks in a belt approximately 30 km wide.

The main rocks exposed in Santo Domingo area include a series of volcanic, volcaniclastics rocks likely from Punta del Cobre Formation and a marine-sedimentary sequence correlated with Chañarcillo group. All rocks have been intruded by a series of small mafic and felsic plutons and dykes which produced localized garnet-bearing skarn and hornfels when in contact with sedimentary and volcanic rocks.

Hydrothermal alteration selectively affects the different lithologies and is characterized by sodic (-calcic), potassic, carbonate and calc-silicate skarn.

Fault trends at the Santo Domingo property are variably to the north, northwest, northeast and east–west and mostly correspond to high angle faults with limited displacement that generally do not affect the continuity of the mineralization. Mineralization within the deposit area consists of:

- Stratiform iron-oxide and sulphide-rich replacement mantos and breccias within volcano sedimentary rocks (e.g. Santo Domingo Sur deposit).
- Structurally-controlled iron-oxide and sulphide mineralization along the east–west Santo Domingo fault zone (e.g. Estrellita deposit).
- Small, closely spaced (100 m to 200 m) northwest-trending and moderately to steeply northeast-dipping iron oxide-copper bearing veins which range in width from a few centimetres to several metres.
- Minor copper oxide minerals disseminated in amygdule in volcanic flows and replacing volcaniclastic rocks.

Spatially, the project is divided into three large deposits, Santo Domingo, Iris Norte and Estrellita. Santo Domingo is made up of two areas, Santo Domingo Sur and Iris.

At the Santo Domingo Sur deposit drilling at 100 m centres or less has outlined a 150 m to 500 m thick copper-bearing, specular hematite–magnetite manto sequence covering an area of approximately 1,300 m by 800 m. The mantos are zoned from an outer rim of specular hematite toward a magnetite-rich core. The mantos consist of semi-massive to massive specularite and magnetite layers with clots and stringers of chalcopyrite, that range in thickness from approximately 4–20 m. Additionally, cobalt-bearing pyrite can be found mainly associated with magnetite mineralization. The upper parts of the manto sequence are frequently oxidized and contain various amounts of copper oxides and chalcocite. Drilling below a depth of 350 m is sparse and mineralization below that depth is not well defined at this time.

The Iris deposit is a narrow zone (100 m to 250 m wide) of copper-bearing iron mantos and breccias extending over 1,900 m that are hosted by andesitic tuffs and andesitic breccias. The dominating iron oxide at Iris is hematite and the main copper mineral is chalcopyrite. There are some old mine workings at the southern end of the deposit where copper oxides such as brochantite and chrysocolla were mined at surface.

Mineralization at Iris Norte is very similar to the Iris deposit. However, part of the mineralization appears to be hosted by volcaniclastic and andesitic flows. The deposit is approximately 500 m wide and has been tested over a strike length of 1,600 m. The Iris Norte deposit has been intruded by significant amounts of diorite dykes and sills that separate the deposit into two lenses. The main sulphides are pyrite and chalcopyrite, with the latter providing the copper content of the deposit.

Drilling at the Estrellita satellite deposit has outlined a tabular body of copper mineralization hosted by specular hematite-rich breccias and lesser mantos along a fault zone around the Estrellita artisanal mine workings. The east–west extent of the Estrellita deposit along the Santo Domingo fault adds up to more than 1,000 m, and the deposit remains open in both directions. The Estrellita deposit contain abundant specular hematite associated with copper oxide and sulphides. The oxide mineralization consists of chrysocolla and malachite, with lesser contents of Cu-limonite, black Cu-oxides, Cu-sulphates and Cu-clays, whereas the main sulphide mineral is chalcopyrite.

1.7 Exploration

Exploration activities have been carried out in the property since 2002. Early activities include surface field mapping focusing on the various lithologies, alteration, mineralization, structures and rock chips, and stream sediment geochemical sampling. Between 2002 and 2008 various methods, including airborne magnetics and gravity, ground induced polarization (IP), transient electromagnetic (TEM), and magnetics were completed by different specialized contractors which lead to the identification of disseminated sulphides and iron oxides, interpretation of the major structures and discrimination of lithologies with presence or absence of disseminated magnetite.

During 2013, Capstone Mining commissioned Geotech to complete a ZTEM and VTEM airborne geophysical survey. In the case of VTEM, 356 linear kilometres were flown with a line spacing of 200 m. For ZTEM, 369 linear kilometres were flown with a line spacing of 200 m. Both works covered the Santo Domingo deposit and surrounding areas.

During 2020 and 2021, a new topographic network in the processing plant area was implemented and a unified topographic network covering the entire project area with precision standards was completed. The result was a topography with contour lines every 0.5 metres. This new dataset was integrated with the previous 2012 topography to cover the entire area of the geology model used for the mineral resource estimate.

The primary exploration potential on the Property consists of oxide mineralization located above the sulphide orebody that has not been sufficiently explored. Additional opportunities to explore for copper sulfide, magnetic iron and cobalt bearing pyrite mineralization exist near in the transition between Iris Norte and Santo Domingo, within or beside the current Santo Domingo pit and at depth.

1.8 Drilling and Sampling

After the acquisition of Far West by Capstone, various drilling programs were carried out. Capstone drilled 14 twinned diamond holes for a total of 3,206 m, during 2014 and early 2015. The purpose of this drilling was to confirm previous drilling and to collect metallurgical samples. In January 2019 Capstone drilled 13 twinned diamond drill holes for a total of 3,060 m, to collect additional material for metallurgical sampling. Between April and July 2021, a brownfield program drilled 19 reverse circulation/diamond drill holes for a total of 8,546 m mainly for exploratory purposes. The brownfield program confirmed the extension of the mineralized sequence at depth between Santo Domingo and Iris Norte pits. The Geometallurgical drill program developed between August and October 2021 involved drilling 8,035 metres in 33 drill holes (twins mainly). This campaign mainly collected samples for metallurgical variability testing. Between November 2021 and February 2022, a metallurgical drill program was developed to obtain samples for the development of a pilot plant, mainly focused on the first 5 years of production. A total of 8,872 m in 41 drill holes (mainly twin holes) were drilled. Between July 2003 and February 2022, a total of 700 core and RC holes (169,692 m) were drilled over the Santo Domingo area as a whole. A total of 483 holes (128,714 m) were used to support the resource estimate and construction of the geological model.

Most drill holes are vertical. Drill cuttings and core were logged using a table of pre-set codes. All geological data were entered digitally into summary logs. Geotechnical data were also recorded. Drill collars were located using a differential global positioning system (GPS) instrument. Downhole surveying was conducted using a combination of gyroscope and accelerometer, with measurements taken every 10 m.

RC drill cuttings were collected at 2 m intervals. Core was nominally sampled at 2 m intervals. Samples for assay were marked at 1 m and 2 m intervals by technicians and subsequently adjusted by the geologist to correspond to major lithological contacts. For programs conducted prior to 2011, sample lengths were not less than 0.5 m, and most did not exceed 2 m. The shortest and longest sample lengths in 2011–2012 were 0.7 m and 2.7 m, respectively, and most samples were 2 m long.

The primary analytical laboratory was ALS Chemex and the facilities in La Serena, Chile and Antofagasta, Chile were used. Both facilities have ISO 9001:2008 accreditation and La Serena also has ISO 17025 accreditation.

Occasional overflow capacity was directed to the ALS laboratory in Lima, Peru. The ALS laboratory in Copiapó Chile was also utilized in 2015, 2019 (only for one Cu-AA05 work order), and 2021. During 2021, the sample preparation continued to be carried out in Chile, subsequently the sample was separated and sent to ALS Lima for ICP analysis and ALS Santiago for gold analysis.

Both Chilean laboratories are independent. The Copiapó laboratory is ISO/IEC 17025:2017 accredited, and the Lima laboratory is 9001:2015 accredited.

Sample preparation consisted of drying, crushing to minus #10 Tyler >70%, homogenizing and then pulverizing to minus #200 Tyler >85%. Samples were analyzed for 27 elements via ALS Chemex procedure ME-ICP61, using inductively coupled plasma (ICP). Gold assays were determined using fire assay with an atomic absorption spectroscopy (AAS) finish. Copper values over 10,000 ppm were re-assayed. Due to the ME-ICP61 method understating the iron content, 7,401 samples from the 2010 drill program were resubmitted for assay using a method with a more aggressive digestion, including all samples over 15% Fe inside the existing block model for which sample material was still available. Soluble copper analysis was conducted on 1,035 samples from 2011–2012 drilling and 1,022 samples from the 2021 geometallurgical drill program.

A total of 32,243 magnetic susceptibility measurements were recorded. Currently, the database has 7,080 density measurements under different methodologies. There were 2,229 density measurements performed by Far West personnel on core samples using the water displacement method. For the previous resource estimation, a regression formula based on specific gravity (SG) values was used to convert volumes to weights, using iron concentration as the independent variable. In 2023, SLR utilized 7,039 density samples to revise the existing formula. After excluding 1,576 samples (895 from core box weights, 295 pycnometer measurements,386 not specified), 5,463 density samples remained for use in the Mineral Resource estimate across all areas, with 4,784 samples in SD-IN. Notably, these samples do not uniformly cover the Mineral Resource volume nor oxide material. Recognizing the link between changes in density and iron content in mineralized material, Capstone opted for a regression formula based on the measured densities and rock iron content. SLR maintained the regression methodology from the prior Mineral Resource but updated it with the newly acquired density data.

The quality assurance and quality control (QA/QC) protocols remained largely consistent throughout all programs conducted by Far West and Capstone. Minor changes were implemented by Capstone to accommodate issues and recommendations from past programs and to include magnetic susceptibility measurements. Certified reference materials (CRMs), or standards, were inserted every 25th sample, constituting 4% of the total number of samples submitted. Blanks, consisting of common Portland cement, were inserted every 50th sample. Field duplicates were taken every 25th sample.

In 2022, Capstone finalized certification of the three matrix matched CRMs inserted since 2010. A review of past results from 2010 through 2015 showed that the cobalt and sulphur analyses for the CRMs were typically within reasonable ranges.

The QP considers that the drilling has been conducted in a manner consistent with standard industry practices. The spacing and orientation of the holes are appropriate for the deposit geometry and mineralization style. Sampling methods are acceptable, meet industry-standard practice, are appropriate for the mineralization style and are acceptable for Mineral Resource estimation. The quality of the analytical data is reliable, and analysis and security are generally performed in accordance with exploration best practices and industry standards.

Between March 2023 and February 2024, Capstone Copper optimized the resources and the reserves for Santo Domingo, Iris Norte and Estrellita. The results of these optimizations are those presented in this report.

1.9 Data Verification

Regular data verification programs were undertaken by third-party consultants, including RPA, from 2005 to 2019 on the data collected in support of technical reports on the Property. Between 2021 and 2023 SLR (which acquired RPA in 2019) was in charge of developing the new resource block model where the verification of the available data was carried out again.

The QP is of the opinion that Capstone's database verification procedures for the Santo Domingo Project comply with industry standards and are adequate for the purposes of Mineral Resource estimation, and that the Santo Domingo database is of sufficient quality to support a Mineral Resource estimate.

Based on the data verification and validation activities completed under the supervision of the QP, in the QP's opinion:

- Capstone's database workflows and controls are both systematic and thorough.

- Repeated verification work over the life of the project has ensured that the database was of sufficient quality to support Mineral Resource estimation.
- Capstone's database construction from the primary data, validation and verification served to confer a full understanding of the dataset, organize the source information, correct most errors, and further improve the quality of the dataset.
- SLR's reproduction of the assay hierarchy in the provided database produced no significant differences from Capstone's 'best' field.
- SLR's verification exercise for Au, Co, Cu, Fe, and S covered most of the assays in the pit shells and did not produce any material differences.
- Twin hole drilling has provided a reasonably consistent verification of the earlier drill results, particularly considering the differences in assay protocols and possible survey errors.

1.10 Mineral Processing and Metallurgical Testing

1.10.1 Overview

Metallurgical test work that was carried out from 2006 to 2023 included:

- comminution
- sulphide flotation;
- sulphide rougher concentrate regrind;
- settling and filtration tests on copper concentrate;
- iron concentration by magnetic separation and reverse flotation;
- magnetic concentrate regrind;
- settling and filtration tests on magnetite concentrate
- settling and thickening test work on final tailings.

1.10.2 Comminution

Two separate physical characterization test work programs, including mainly BWi, RWi, SMC, Axb, were conducted during 2011 and 2014. This data was further updated with additional 45 samples including mainly BWi, RWi and Axb. The compiled data delivered a 75-percentile value for Axb of 36.7, a 75-percentile value for BWi of 14.1 kWh/t, and a 75-percentile value for RWi of 16.4 kWh/t.

1.10.3 Bulk Density

Capstone chose to use regression formulae using the relationship between the measured densities and the iron content in the rock. The previously developed formulas were updated to include the information collected until October 2021. As a result, different curves have been generated for the Santo Domingo area depending on the ore type (as detailed in Section 13.1.3).

1.10.4 Copper and Gold

1.10.4.1 Pre-2018 Flotation Test Work

Bench scale flotation test work on composite and variability samples – including kinetic tests, open cycle tests (OCT) and locked cycle tests (LCT), was conducted during this period using direct sea water – as per the project definition at that time. A pilot plant using direct sea water was also completed during this period (2015) with the objective to produce concentrate and tailings for downstream testing. Due to some difficulties faced to meet iron product marketing specifications when processing the ore with direct sea water (chloride content), a decision was made to use desalinated water for the process.

Based on the decision to use desalinated water in the process, copper flotation test work was carried out in 2018 by Aminpro to develop copper and gold recovery algorithms.

1.10.4.2 2018 to 2020 Flotation Test Work

The 2018 test work program consisted of kinetic tests on composite and variability samples using desalinated sea water. The program provided desalinated water flotation results which allowed the establishment of initial copper and gold recovery algorithms. However, since samples from the 2014-2015 drill program which appeared to have aged were used, fresh drill core samples were obtained in 2019 for confirmation. These fresh drill core samples allowed the completion of a flotation test work program that included OCT and LCT on yearly composite samples (Year 1-5 and Year1, Year2, Year3, Year4 and Year 5) and SKT tests on rougher and cleaner stages on 28 variability samples, to obtain pulp zone kinetics and model the process in order to formulate new recovery algorithms for copper and gold. Test work at this stage was finalized with a copper and pyrite flotation pilot plant completed during mid 2020, using the flotation conditions established during 2019 and Year 1-5 composite sample. Challenges were faced during the pilot trials at the rougher flotation stage, mainly associated with the hydrodynamic conditions within the pilot cells and not to the effect of the ore. These hydrodynamic issues were mainly attributed to the high specific gravity of the sample.

1.10.4.3 Feasibility Update Flotation Test Work (2021 to 2023)

Several test programs were developed during this period of time, at bench and pilot scale, including optimization of flotation conditions for the main mineralized geological units and yearly composites. Some of the most relevant tests completed on the Year 1-5 composite sample were the extended 20-cycle LCT with a copper head grade sample of 0.52% Cu which reported a Cu circuit recovery of 91.3% with Cu concentrate grade of 28.3%; and the flotation pilot plant with a Cu head grade of 0.56% Cu which reported a Cu circuit recovery of 91.1% with Cu concentrate grade of 26.0% Cu.

In addition, a comprehensive flotation variability program was run on 140 variability samples (12-m drill core composites), and the results used to update the copper and gold recovery models. The results also allowed for estimation of copper concentrate grade. The resulting models are:

$$Cu\ Rec = -0.011*Co+11.967*\%CuT-13.740*\%CuT^2 +5.502*\%CuT^3-0.920*CuT^4-CuSAratio*68.714 +94.068$$

$$Cu\ Conc\ grade = -1.701*\%CuT+ CuSAratio *7.767+(CuT/S)*23.362 + 22.330$$

Where CuT is the total copper and CuSAratio is the acid soluble copper ratio, defined as the soluble copper divided by the total copper. The identification by mineralization zone according to the weathering model allowed to assign a representative acid soluble copper ratio to each zone (0.05 or 5% for the sulfide zone and 0.2 or 20% for mixed zone). The resulting equations are:

- For the Sulphide zone:

$$Cu\ Rec = -0.011*Co+11.967*\%CuT-13.740*\%CuT^2 +5.502*\%CuT^3-0.920*CuT^4-0.05*68.714 +94.068$$

$$Cu\ Conc\ grade = -1.701*\%CuT+0.05*7.767+(CuT/S) *23.362 + 22.330$$

- For the Mixed zone:

$$Cu\ Rec = -0.011*Co+11.967*\%CuT-13.740*\%CuT^2 +5.502*\%CuT^3-0.920*CuT^4-0.20*68.714 +94.068$$

$$Cu\ Conc\ grade = -1.701*\%CuT+0.2*7.767+(CuT/S) *23.362 + 22.330$$

A similar approach was used for gold and copper recovery estimation. Au and Cu head grades and acid soluble copper ratio were identified as the most relevant variables, and an initial algorithm was developed and applied for resource and reserve optimization. The model is:

$$Au\ Rec = -142.598*Au+44.752*\%CuT-10.095*\%CuT^2- CuSAratio*46.680+51$$

Inclusion of the mean representative acid soluble copper ratio for each mineralization zone (sulphides and mixed) produces the following:

- For the Sulphide zone:

$$Au\ Rec = -142.598*Au+44.752*\%CuT-10.095*\%CuT^2-0.05*46.680+51$$

- For the Mixed zone:

$$Au\ Rec = -142.598*Au+44.752*\%CuT-10.095*\%CuT^2-0.2*46.680+51$$

While this algorithm was used for resources and reserves optimization, significant late-stage work was performed to improve the gold recovery model, and this resulted in a new equation, as shown below. Because the impact on the resources and reserves were deemed to be negligible, no changes were made the project resource and reserve statement as a result of the improved expected gold recovery.

$$Au\ Rec = 49.959*\%CuT-43.778*\%CuT^2 + 41.219*(Cu/S) + 44.979$$

1.10.5 Iron

1.10.5.1 Pre-2021 Iron Concentration Test Work

DTT and LIMS magnetic separation test on composite and variability samples were conducted during this period using mainly direct sea water – as per the project definition at that time. LIMS was also used to recover magnetic iron for downstream testing, from the rougher flotation tailings during the 2015 flotation pilot plant using direct sea water. Due to some difficulties faced to meet iron product marketing specifications when processing the ore with direct sea water (chloride content), a decision was made to use desalinated water for the process.

The new water type definition led to a DTT variability program on 206 effective samples using desalinated seawater and target P_{80} of 40 µm (an approach of 80% passing 325 mesh was considered to mimic target P_{80}). A wide range of Iron head grades were tested ranging from 8.4% Fe to 61.9% Fe with an average value of 30.1% Fe, covering Santo Domingo and Iris Norte deposit (LOM). The general results indicated feasibility to achieve iron concentrate target of 65% Fe, with an average mass recovery of 22.5%. The new water type definition also led to the obtention of additional rougher flotation tailings from a separate flotation pilot plant completed during mid 2020 using desalinated seawater. The rougher magnetic separation was processed in a 48-inch diameter magnetic drum separator, using current flowsheet at the time. The program was developed in two stages: process optimization followed by performance validation. The program reported iron concentrate grade of 65% Fe with an overall mass recovery of 15.7%. In order to continue iron concentrate grade improvement, reverse flotation, screens and hydro-separation were suggested for evaluation.

1.10.5.2 Iron Concentration Test Work (2021 to 2023)

Exploratory bench scale test work including reverse flotation, screening and hydro-separation was completed mainly during 2021, on samples obtained from the iron concentration pilot plant. The results indicated that reverse flotation has the potential to improve iron concentrate grade (by 3% from 65.1% Fe to 67.1% Fe) and significantly reduce silica content (by 36% from 4.7% SiO_2 to 3.0% SiO_2) for the Santo Domingo ore tested. However, analysis of the future copper-iron processing plant and its corresponding process water recirculation finally led to discard the use of the technology (potential issues with reverse flotation reagents). Test work completed on reground magnetic separation concentrate with a hydro-separator also reported iron concentrate grade improvement (by 2% from 52.9% Fe to 54.2% Fe) and reduced silica content (by 11% from 13.9% SiO_2 to % 12.4 SiO_2) for the Santo Domingo ore tested. No issues were foreseen and therefore the incorporation of hydro-separation was established for future test work. Screening test work did not show promising results for the sample tested, and therefore it was not further considered.

During 2022, an alternative LIMS (L8 unit) study was completed to identify possibilities to further improve the iron concentrate grade by using additional cleaning stages and lower magnetic strength. The results were promising, indicating that lower magnetic strength and additional cleaning stages were an alternative to improve further the iron concentrate grade (by 3% from 65.6% Fe to 67.3% Fe for the Santo Domingo ore tested). Based on the results, this concept was adopted for future test work. Finer regrind (P_{80} of 30 µm and about P_{80} of 20 µm) was also tested during this program with promising results. However, regrind finer than P_{80} of 40 µm was not considered as a basis for the final design.

Selected conditions were then used for a second magnetic separation pilot plant where hydro-separation and additional cleaning stages were considered. This test work program reported an iron concentrate grade of 65.5% and overall mass recovery of 12.6%.

These programs were complemented by an iron concentration variability program on 39 variability samples, using LIMS L8 equipment and including all processing stages defined such as rougher magnetic concentrate, regrind at target P_{80} of 40 µm, and hydro-separation followed by 8 cleaning stages. The program reported an average iron concentrate grade of 65.2% Fe but more significantly, the variability program confirmed achievement of iron concentrate grades of 67% Fe and higher. Mass recoveries varied from 4.2% to 65.1%. In general terms, high mass recovery was related to high iron concentrate grades.

In addition to these programs, an additional DTT variability program was completed on more than 1,300 samples. The following summarises the activities:

- DTT using target P_{80} of about 40 µm (established as 80% less than 325 mesh or 45 µm) was completed on more than 1,300 samples covering magnetic iron mineralized geological units through LOM and including iron concentrate Fe grade and impurities analysis.
- DTT using finer regrind (established as 100% less than 450-mesh or 32 µm) was considered for evaluation of the iron concentrate grade.

The mass recoveries obtained from the DTT test work program at target P_{80} of about 40 µm were used to update the mass recovery algorithm. Magnetic susceptibility readings and DTT data were used to predict the mass recovery to the final concentrate. The new regression used a total of 1,324 samples for the generation of iron mass recovery models that had two different expressions and different cutoffs depending on deposit zone (Santo Domingo or Iris Norte). They included a Magnetic susceptibility cut-off by lithology. The algorithms are shown below.

- Santo Domingo:

$$MassRec = (-0.0054 * (magsus/1000)^2 + (1.3444*magsus/1000))$$

- Iris Norte:

$$MassRec = (-0.0042 * (magsus/1000)^2 + (1.1936*magsus/1000))$$

Details on iron recovery algorithm application, iron concentrate grade and cut-off by lithology are described in section 14.3.3.3.

1.10.6 Tailings Thickening

1.10.6.1 Pre-2020 Tailings Test Work

Vendor testing was completed and used to evaluate the behavior of the tailings under conditions of dynamic settling and to determine design values for the two-stage tailings thickening system. Two stages of thickening in series were selected due to the properties of thickened tailings and to avoid capital and operating costs associated with operating using non-thickened tailings.

1.10.6.2 2020 to 2023 Tailings Test Work

During 2020 to 2021, additional tailings thickening test work was completed on various tailings samples to validate a two-stage approach. It identified that the tailings could be thickened and transported with a single stage solution utilising high density or paste thickening equipment. Consequently, a trade-off study was completed in 2022 that confirmed the feasibility of a single stage solution to reduce the complexity and risk of the tailings handling system for the project. Significant upsides to reduce capital and operational expenditure were identified.

In 2023, settling and dynamic thickening testing were conducted by Tailpro Consulting and Vietti Slurrytec to verify the potential design parameters for both two-stage and single-stage tailings thickening approaches.

This test work was conducted both at a laboratory and semi-pilot stage to better understand the residence times needed and unsheared underflow densities to establish equipment torque requirements. Dynamic thickening testing was also conducted by Metso Outotec, again at both a laboratory and semi-pilot scale, to understand the vendor specified technology to dewater the tailings to design values. The 2023 tailings sample generated was a composite of the first 5 years of operation and was considered representative of the final tailings.

As part of the 2023 tailings thickening test work program, a series of beach slope predictions were developed for the design values associated with deposition to the tailings storage facility (58%, 65%, and 67% solids w/w). These predictions considered a range of simultaneous operating independent discharges to understand the impact to the beach slope and were provided to the tailings facility designers as part of the current design.

A variability program was conducted on the tailings from the 140 variability samples. Additionally, 20 final tailings composite samples were subjected to dynamic thickening and rheological measurements.

1.10.7 Byproduct Cobalt

This section is included within the context of the cobalt opportunity detailed in Section 24.4 “Other Relevant Data and Information – Optimization Opportunities” & Information” and Section 25.16“Interpretations & Conclusions - Risks & Opportunities.”

The Santo Domingo deposit is endowed with cobaltiferous pyrite. While the current study only considers the economic by-production of cobalt as a potential opportunity, considerable work has been done to evaluate this possible upside. The general processing route consists of three stages:

1. Recovery of cobaltiferous pyrite from the Santo Domingo tailings, via froth flotation
2. Oxidation of the pyrite, either via pyrometallurgical or hydrometallurgical techniques, to release the cobalt
3. Recovery and purification of the cobalt into a commercial product.

Initial cobalt test work was performed in 2018 by several laboratories supervised by Blue Coast Metallurgy Ltd. (Blue Coast). This limited test work indicated that, using the present flowsheet, it may be possible to recover approximately 80% of the cobalt, contained in pyrite, as a secondary concentrate. The initial program consisted of flotation flowsheet development at Blue Coast to produce a cobalt and copper bearing pyrite concentrate from the flotation tailings. This was later refined to focus the cobalt extraction from the cleaner scavenger tailings where the bulk of the pyrite reports from the copper flotation circuit.

In 2019 Aminpro also started cobalt flotation test work, which consisted of floating the pyrite from the cleaner scavenger tailings from the copper circuit. This work demonstrated that greater than 90% Co and 95% of the pyrite could be recovered from the cleaner scavenger tails with an effective cobalt grade in the concentrate of ~0.6% Co.

In 2019 more detailed test programs were conducted at Kingston Process Metallurgy (KPM) and SGS Lakefield to develop an oxidation and purification process to produce cobalt sulphate heptahydrate and copper sulphide precipitate as final products. The test work included biological oxidation, pressure oxidation leaching, the Albion Process, and roasting. These processes were tested under a variety of conditions to determine an optimum approach for further development.

In late 2023 the pyrite extracted from flotation tests was combined with oxide ore from Capstone Copper's Mantoverde operation and leached in columns, at Aminpro, to simulate a bio-oxidation heap leach process to recover cobalt from pyrite. This work is ongoing.

In 2024, in parallel to the column testwork at Aminpro, a counter current ion exchange pilot plant was established at Mantoverde to demonstrate the potential for ion exchange to recover the leached cobalt from simulated cobalt solutions using the raffinate from the commercial dynamic heap leach operation. This work is ongoing.

1.11 Mineral Resource Estimation

Capstone retained SLR to complete an updated Mineral Resource estimate for the Santo Domingo Sur, Iris, Iris Norte, and Estrellita areas of the Santo Domingo Project (the Project), situated in the Atacama Region, Chile. SLR updated the Estrellita (ES) model in 2022 and the combined Santo Domingo Sur – Iris – Iris Norte (SD-IN) model in 2023. In early 2024, Capstone used updated recovery curves for Cu, Au and Fe, updated metal prices and costs, and an updated NSR script to calculate updated NSR values within both SLR models. The effective date for the updated Mineral Resources for the Santo Domingo property is 31 March 2024.

In late 2020, Capstone performed extensive reviews of the geological and geochemical information for the Project, and revised the geological and oxidation models based on the consideration that the lithological units were continuous across the three previously modelled areas of the deposit. Two separate geological models were built in Leapfrog software. Dyke and diorite orientations and extents were also revised. Capstone also constructed a lithogeochemistry model using immobile elements for discrimination of lithological units. The results showed general agreement on the main lithological contacts previously defined in past modelling, however, various changes were implemented. Although the basic stratigraphy remains the same, the work resulted in a more complex model with a greater degree of confidence attributable to each material type. The new model shows greater consistency, continuity, and adherence to drill data. Based on that work, Capstone provided SLR with three new sets of geological and oxidation models for the Project. The first encloses Santo Domingo Sur and Iris, the second encloses Iris Norte, and the third encloses Estrellita.

Capstone also provided a new database, reconstructed from source logs and assay certificates. The work was internally verified as it was being completed.

Incorporating the new geological modelling and the rebuilt database, SLR constructed two new sub-blocked models. The first (SD-IN) incorporated Santo Domingo Sur, Iris, and Iris Norte into one large model. The second model (ES) was constructed for Estrellita.

Domaining strategies were revised based on the most important controls to mineralization in each area. Block grade estimates were generally performed in three passes in the order of 70 m, 140 m and 300 m, using ordinary kriging. Both models used density weighting for Au, Co, Cu, Fe, S and pyrite estimates, and employed Dynamic Anisotropy to vary search orientations and used static regression curves to estimate block density.

Magnetic susceptibility (magsus), mass recovery (massrec), iron in concentrate (FeDTT), and density estimates were not density weighted. For SD-IN, SLR estimated iron mass recovery using regression formulae developed from Davis Tube tests. SLR then generated two regular models from the sub-blocked models to generate Whittle shells, classify the estimated material, and produce the final Mineral Resource estimates.

The mineralization at the Santo Domingo project comprise polymetallic deposits in which revenue will be generated from the sale of copper-sulphide and iron-oxide concentrates. In addition, the copper concentrate contains some payable quantities of gold credits. The Mineral Resources were reported utilizing an NSR, based on preliminary metallurgical recovery test results and current long term metal pricing supplied by Capstone. The NSR ($/tonne) was assigned to each block by combining the contribution from copper, gold, and iron.

Capstone performed internal studies resulting from the upgraded modelled variable FeDTT which led to a new algorithm to determine routing codes for different mill processes associated with different anticipated upgrades to iron concentrate material, anticipated Fe price, NSR value, and processing cost for each block. The changes required by Capstone for the 2024 NSR value and processing cost calculations include:

- Four new routing codes for Fe concentrate, based on the new modelled FeDTT values, upgraded with finer regrinding when necessary
- New Cu price of US$4.10 per pound, and three new Fe prices which reflect the expected Fe grades attained in the final concentrate
- Two new processing costs for Fe concentrate types
- Mass recovery factored according to the process methodology.

1.12 Mineral Resource Statement

The Mineral Resource estimates were initially prepared by SLR in 2022 for ES and in 2023 for SD-IN. Those resources were subsequently retabulated by Capstone in early 2024 using updated NSR calculations. Mineral Resources for the Property have an effective date of March 31, 2024. Mineral Resources in Table 1-1 are reported inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves (CIM (2014) definitions) were used for Mineral Resource classification. At both SD-IN and ES the resources only include mixed and primary (sulphide) material. Oxides are not considered part of the resource at this stage of the project. The estimates for SD-IN are based on Whittle shell optimization at an NSR cut-off value of US$ 9.85 per tonne. The Mineral Resource estimates for ES are based on Whittle shell optimization at an NSR cut-off value of US$9.63 per tonne.

A portion of the deposit gets its value almost solely from iron, so it is not suitable to calculate a Copper Equivalent value for that material in the mineral resource estimate. Because of that, Capstone is reporting average NSR values (in US$/t) instead of Copper Equivalent values.

The risk factors that could potentially affect the Mineral Resources estimates include mostly the assumptions used to generate the conceptual data for consideration of reasonable prospects of eventual economic extraction. Those include:

- Commodity price assumptions;
- Exchange rate assumptions;
- Density assumptions;
- Geotechnical and hydrogeological assumptions;
- Operating and capital cost assumptions;
- Metal recovery assumptions;
- Concentrate grade and smelting/refining terms;
- Changes in interpretations of mineralization geometry and continuity of mineralization zones.

There are no environmental, permitting, legal, title, taxation, socio-economic, marketing, political or other relevant factors known to the QP, other than those discussed in this Report, which could affect the Mineral Resource estimates.

Table 1-1: Santo Domingo Project – Mineral Resource Estimates as of 31 March 2024

Category	Tonnage (Mt)	Grade NSR ($/t)	Grade Cu (%)	Grade Fe (%)	Grade Au (g/t)	Contained Metal Cu (kt)	Contained Metal Fe (Mt)	Contained Metal Au (koz)
Measured								
Santo Domingo	134	46	0.51	26.9	0.07	679	36	293
Total Measured	**134**	**46**	**0.51**	**26.9**	**0.07**	679	**36**	**293**
Indicated								
Santo Domingo	298	34	0.26	25.7	0.04	786	77	362
Iris Norte	74	28	0.14	24.0	0.02	106	18	43
Santo Domingo + Iris Norte	372	33	0.24	25.4	0.03	892	95	405
Estrellita	41	24	0.32	-	0.03	133	-	44
Total Indicated	**413**	**32**	**0.25**	**N/A**	**0.03**	1,025	**95**	**449**
Total Measured + Indicated	**547**	**35**	**0.31**	**N/A**	**0.04**	1,704	**131**	**742**
Inferred								
Santo Domingo	173	28	0.20	22.2	0.03	349	38	157
Iris Norte	30	28	0.12	23.9	0.01	35	7	14
Santo Domingo + Iris Norte	203	28	0.19	22.5	0.03	384	46	171
Estrellita	27	25	0.34	-	0.03	93	-	29
Total Inferred	**230**	**28**	**0.21**	**N/A**	**0.03**	477	**46**	**200**

Notes: **1.** The average Iron grades for the Project (Total Indicated, Total Measured plus Indicated, and Total Inferred Resources) cannot be calculated because Estrellita does not contain iron resources. **2.** Notes specific to the Mineral Resources for the Santo Domingo and Iris Norte deposits: **a**. Mineral Resources for SD include Iris. **b.** Mineral Resources are reported using a net smelter return (NSR) cut-off value of US$9.85/t. NSR is calculated using average long-term prices of US$4.10/lb Cu, US$1,600/oz Au, and Fe prices that depend on the expected grade of the Fe concentrate (US$94.75/dmt or $129.77/dmt or $140.26/dmt Fe concentrate). **c.** Mineral Resources are constrained by preliminary pit shells derived using a Lerchs–Grossmann algorithm and the following assumptions: pit slopes 36.3°- 47.9°; mining cost is calculated using a function that depends on where the material comes from (Santo Domingo or Iris Norte) and its destination (dumps, plant or stock); processing cost based on Fe concentrate routing code (including G&A costs); processing recovery based in the recovery equations for copper, gold, and iron as detailed above. Metal prices used are those listed on Note 3b. **3**. Notes specific to the Mineral Resources for the Estrellita deposit: **a.** Mineral Resources are reported using an NSR cut-off value of US$9.63/t. NSR is calculated using average long-term prices of US$4.10/lb Cu, and US$1,600/oz Au. **b.** Only copper, and gold were considered in the NSR calculation; iron, cobalt and sulphur were excluded. **c.** Mineral Resources are constrained by preliminary pit shells generated using a Lerchs–Grossmann algorithm and the following assumptions: pit slopes 43º; mining cost of US$1.55/t, processing cost of US$9.46/t (including G&A cost); processing recovery are calculated based in the recovery curves for copper and gold. **4.** Rounding as required by reporting standards may result in apparent summation differences. **5.** Tonnage measurements are in metric units. Copper and iron are reported as percentages (%) and gold as grams per tonne (g/t).

1.13 Mineral Reserve Estimation

Pit optimization, mine design and mine planning were carried out by MSD using the 2023 block model prepared by SLR. Inferred Mineral Resources were treated as waste. A block size of 12 m E x 12 m N x 15 m high was selected for the block model. The selected block size was based on the geometry of the domain interpretation and the data configuration.

The mining cost estimate for the pit optimization process was based on studies carried out by NCL during 2023 and 2024. The estimated average mining cost was separated into various components such as fuel, explosives, tires, parts, salaries, and wages, benchmarked against similar current operations in Chile. Each component was updated for third quarter 2023 prices and the exchange rate from Chilean Pesos to US dollars. This resulted in an estimated mining cost of approximately $1.57/t moved. This estimated mining cost matches well to the preliminary $1.55/t used during phase design early in the mine planning process.

Capstone Copper provided the metal prices, processing costs, refining costs, and processing recoveries.

The estimated Mineral Reserves are reported using metal prices of US$3.75/lb Cu, US$1,400/oz Au, and Fe Concentrate prices ranging from US$69/dmt to US$114.51/dmt based on the Fe grade in concentrate.

Calculations were performed in the model to determine the net smelter return (NSR) of each individual block. The internal (or mill) cut-off of $9.77/t milled incorporates all operating costs except mining. This internal cut-off is applied to material contained within an economic pit shell where the decision to mine a given block was determined by the pit optimization and was applied to all of the Mineral Reserve estimates.

Final slope angles used for the pit optimization process were a result of multiple iterations and analysis carried out by the MSD mining team and geotechnical specialists Ingeroc SpA.

The pit shells were employed as guides for the practical designs for both the Santo Domingo and Iris Norte pits. Any potential impacts to ore feed that may arise due to ore loss or dilution are not considered to be material, and no provision for ore loss or dilution is included in the mine plan or the Mineral Reserve estimate.

1.14 Mineral Reserve Statement

Mineral Reserves are summarized in Table 1-2 and have an effective date of 31 March 2024.

In the opinion of the QP, the main factors that may affect the Mineral Reserves estimate are metal prices, metallurgical recoveries, operating costs (fuel, energy and labour) and block model accuracy. However, the Mineral Reserves are not considered sensitive to moderate changes in these factors, due in part to the use of a 0.84 Revenue Factor LG shell used to guide pit design. Other factors that may impact the Reserves include land tenure or permitting, water supply and geotechnical stability of pit walls and tailings facilities.

Table 1-2: Mineral Reserves Statement (31 March 2024)

Reserve Category	Stage	Tonnage (Mt)	Ore Grade			Contained Metal		
			Cu (%)	Au (g/t)	Fe (%)	Au (koz)	Cu (kt)	Magnetite Conc. (Mt)
Proven Mineral Reserves	Santo Domingo	130.9	0.52	0.07	27.2	291	675	12.6
	Iris Norte	-	-	-	-	-	-	-
Total Proven Mineral Reserves		**130.9**	**0.52**	**0.07**	**27.2**	**291**	**675**	**12.6**
Probable Mineral Reserves	Santo Domingo	251.2	0.27	0.04	26.6	309	673	45.1
	Iris Norte	53.9	0.16	0.02	24.3	36	88	10.8
Total Probable Mineral Reserves		**305.1**	**0.25**	**0.04**	**26.2**	**346**	**761**	**55.8**
Total Mineral Reserves (Proven and Probable)	Santo Domingo	382.1	0.35	0.05	26.8	600	1,347	57.6
	Iris Norte	53.9	0.16	0.02	24.3	36	88	10.8
Total Mineral Reserves		**436.1**	**0.33**	**0.05**	**26.5**	**637**	**1,435**	**68.4**

Notes to Accompany Mineral Reserves Estimate: **1.** Mineral Reserves are reported as constrained within Measured and Indicated Resources and pit designs optimized using the following economic and technical parameters: metal prices of US$3.75/lb Cu, US$1,400/oz Au and Fe prices ranging from US$69/dmt to US$114.51/dmt based on the Fe grade in concentrate (net of Fe concentrate transport costs); average recovery to concentrate is 90.1% for Cu and 56.3% for Au, with magnetite concentrate recovery varying on a block-by-block basis; copper concentrate treatment charges of US$80/dmt, U$0.08/lb of copper refining charges, US$5.0/oz of gold refining charges, US$40/wmt and US$25.75/dmt for shipping copper and iron concentrates respectively; waste and ore mining cost of $1.55/t and process and G&A+SUSEX of US$9.77/t processed; average pit slope angles that range from 36.3° to 47.9°; a 2% royalty rate assumption and an assumption of 100% mining recovery. **2.** Rounding as required by reporting standards may result in apparent summation differences between tonnes, grade and contained metal content. **3.** Tonnage measurements are in metric units. Copper and iron grades are reported as percentages, gold as grams per tonne. Contained gold ounces are reported as troy ounces, contained copper as thousand tonnes and contained iron as metric million tonnes.,mm,n

1.15 Mining Methods

A mine plan was developed based on processing 65,000 t/d to 72,000 t/d of feed (23.7 Mt/y to 26.3 Mt/y) with a peak total mining rate of 90 Mt/y. The first year, the plant is considered Ramp up for six months with lower production rates reaching an average/y of 54.6 t/d. Then, because of the softer characteristics of the initial feed (higher copper content and lower magnetite), a period of 4 years is scheduled for a plant feed of 72,000 t/d (Years 2 to 5). This declines to average 50,000 t/d Year 16 to 19 due to concentrate pipeline limits. Initial feed to the plant is made up of material mined during Year 1. Oxide material will be identified and stockpiled separately. An 18-month pre-production period will be needed plus 6 to 12 months of early works to ensure sufficient access to the mine work front.

The mine is scheduled to work seven days per week, 365 days per year. Each day will consist of two 12-hour shifts. Four mining crews will cover the operation.

The final pit design was based on a LG shell that used a copper price of $3.75/lb, US$1,400/oz Au, and Fe Concentrate prices ranging from US$69/dmt to US$114.51/dmt based on the Fe grade in concentrate. Two pits, the Santo Domingo pit and the Iris Norte pit, were designed. The Santo Domingo pit will have eight phases; two mining phases are planned for the Iris Norte pit. These values were used only for mine design and not the economic analysis.

The Santo Domingo pit will have two exits, on the west side providing access to the stock pad area and the primary crusher. On the east side there will be another exit to access the main waste rock storage area. The Iris Norte pit will have one exit providing access to the stock pad area, the primary crusher and to access the waste rock storage area.

Mine equipment requirements were calculated based on the annual mine production schedule, the mine work schedule and equipment hourly production estimates. The study is based on operating the mine with electric shovels, hydraulic shovels and wheel loader of capacity 57, 42 and 38 m^3, respectively, and trucks with a capacity of 290 t. The fleet will be complemented with drilling rigs for ore and waste. Auxiliary equipment will include tracked dozers, wheel dozers, motor graders and a water truck. A small drill rig was also included for pre-splitting purposes.

1.16 Recovery Methods

The Santo Domingo process broadly consists of the following stages: crushing, grinding, copper flotation, magnetite recovery, copper dewatering and load-out, magnetite pumping and dewatering, tailings dewatering and storage.

1.16.1 Crushing and Grinding

The primary crushing plant processes ROM feed in a gyratory crusher. 290 t capacity trucks unload material into a 450-t live capacity dump pocket with two truck dumps stations. A rock breaker is provided for management of oversized rocks.

Primary crushing is carried out in a 60” x 89” gyratory crusher. A belt feeder transfers the ore from the crusher to the stockpile feed conveyor, and ultimately discharges fresh ore over a covered conical coarse ore stockpile.

The stockpile allows ore reclaim using four apron feeders located within the reclaim tunnels (two trains of two feeders), which feed two parallel grinding lines via dedicated conveyors. Each grinding line consists of one 18 MW autogenous grinding mill (AG mill), one pebble AG mill discharge screen, two pebble crushers (duty and standby), one cyclone cluster and one ball mill. The AG mill product slurry discharges over a horizontal single deck vibrating screen for pebble washing and transferring of pebbles to pebble conveyors and then to pebble crushing. Crushed pebbles report to the AG mill feed conveyor. There is the option of bypassing the pebble crushers and recycling uncrushed material to the AG mill using the same belt conveyor system, when required. Also, a belt plow mechanism is available on the conveyors for purging pebbles to grade to aid grind-out of mills when required.

The AG mill screen undersize feeds the 9 MW variable speed ball mill and discharges into the grinding cyclone cluster feed box, where it is mixed with the ball mill product and pumped to a cluster of 33” classification cyclones. Cyclone underflow reports by gravity to the ball mill and cyclone overflow feeds the downstream copper flotation circuit.

1.16.2 Copper Flotation

Copper flotation consists of conventional rougher flotation, regrinding and classification, with Jameson Cells used in the cleaner scalper, cleaner scavenger, and re-cleaner stages. The rougher flotation stage recovers both copper and pyrite minerals. The cleaner circuit selectively recovers copper sulphides and pyrite preferentially reports to the cleaner scavenger tail stream. There is allowance in the design for future installation of a pyrite recovery circuit to recover pyrite that contains cobalt.

The slurry product from both grinding lines flows through dedicated MSA type metallurgical samplers and then feeds a single bank of six 630 m^3 mechanical, forced air tank cells. Flotation rougher concentrate produced from the rougher cells flows by gravity to the regrind mill feed box where it is pumped to the regrind circuit that consists of a cyclone cluster operating in an open circuit and a single horizontal regrind mill.

Reground rougher concentrate of P_{80} 34 µm is conditioned in an agitated tank prior to copper sulphide recovery in the cleaner circuit.

Tails from the rougher train discharge through an MSA type metallurgical sampler to the primary flotation tails pump box and are then pumped to the magnetic separation plant. For occasions that require bypassing of the magnetite plant, tails can be directly pumped to the final tailings thickening stage.

Final copper concentrate is pumped to downstream copper thickening and dewatering prior to stockpiling and load-out.

1.16.3 Copper Concentrate Filtration

Thickened copper concentrate from the thickener underflow discharges into the copper concentrate storage tank, through a static trash screen. The storage tank allows surge management between concentrate delivery, and batch feeding of filter presses.

Copper concentrate is filtered in two pressure filters. Recovered water from these filters is sent to the copper concentrate filtrate tank, from where it is pumped to the concentrate thickener for clarification.

The filtered concentrate cake discharges at approximately 9 % moisture content and will be discharged to grade, from where mobile material handling equipment reclaim and transport concentrate to the concentrate stockpile area, located as an extension of the filter area building.

The concentrate will be loaded from the stockpile onto trucks in sealed dump containers for transport to the port; front loaders will be used for concentrate loading. Once the containers are loaded, they go through the process of extracting samples for the laboratory, capping and sealing and finally to a cleaning process to remove traces of ore.

1.16.4 Magnetic Separation

The magnetite plant is designed to produce two simultaneous products, determined by two different magnetic cleaner circuits, each tailored to favour either recovery, or concentre grade.

One of the circuits targets higher-grade concentrate and includes cleaner stages of lower magnetic intensity than the lower grade producing circuit, to favour higher overall selectivity and production of a higher-grade concentrate. The higher-grade circuit also has flexibility to report higher-grade cleaner tails to the lower grade cleaner stages, to allow scavenging of iron product to lower-grade concentrate.

The magnetic separation plant is designed to produce up to two products, which may consist of two of the following three concentrate grade categories:

- 67% iron grade (or greater) as feed for direct reduction plants;

- 65% to 67% iron grade as feed for blast furnace plants, and
- 62% to 65% iron grade concentrate as feed for iron agglomeration processes, such as a pelletization.

The magnetite plant has the following distinctive features:

- Tailings from the rougher flotation stage are pumped to two central distributors which feed wet back-to-back type rotating magnetic drums (1,150 Gauss). Primary magnetic concentrate is collected in the mill feed hoppers.
- Rougher magnetic concentrate is pumped to the regrinding stage which consists of two parallel lines, each with one horizontal regrind mill designed to produce a P_{80} of 40 µm. Product from both mills feeds a hydro-separation stage.
- Rougher magnetic concentration tailings report to the final plant tailings stream.

Feed to hydro-separators from regrind pass through magnetic flocculators to enhance settling of magnetic iron solids, and nonmagnetic and/or colloidal material (associated with silica) is removed by hydraulic entrainment to the overflow stream. Deslimed magnetite concentrate reports to the underflow and is pumped through demagnetizing coils to the magnetic cleaner stages. Higher -grade hydro-separator underflow feeds the higher- grade magnetic cleaner circuit and lower-grade hydro-separator underflow feeds the lower-grade magnetic cleaner circuit.

The cleaning magnetic circuit consists of two parallel lines, each one with three magnetic batteries; one line is dedicated to production of higher -grade and the other to lower- grade.

The lower-grade line operates considering a magnetic field intensity of 750 Gauss for first two triplet units, and the final double drum unit uses 500 Gauss for final finishing. The high-grade line operates considering a magnetic field intensity of 500 Gauss for all units.

The final magnetite concentrate produced from each line is recovered in concentrate hoppers (one for higher -grade and one for lower- grade) and is pumped to the magnetite concentrate thickener area, where two thickeners recover water. Tailings from the cleaner magnetic stage (higher- grade and lower- grade) are combined with rougher tailings and sent to the final tailings stream.

The design has inbuilt flexibility that allows manual redirection of higher-grade cleaner tails to the lower-grade hydro separator, to allow scavenging of iron ore sufficient for recovery as lower-grade concentrate. When this flexibility is not used, higher- grade cleaner tails report to tails.

Magnetite concentrate is filtered adjacent the port area. Magnetite concentrate will be transported by pipeline from the mine site and will be received at the port in an agitated storage tank and then pumped directly to the filter plant to obtain a magnetite concentrate with a moisture content of 8% w/w. Initially there will be two ceramic disc filters (increasing to five by Year 4). The magnetite concentrate filter cake product will discharge onto a conveyor feeding the concentrate transfer tower and then the two magnetite concentrate stockpiles, one for “bulk concentrate” and another for “premium concentrate”.

1.16.5 Tailings Thickening

The tailings from the iron concentration plant are sent to dedicated tailings thickeners with the copper cleaner scavenger flotation tailings. Final thickening for disposal takes place in the vicinity of the process plant, through parallel use of three 48 m diameter high density thickeners. Flocculant will be added to aid thickening.

The underflow from the thickener (67% solids by weight) is pumped to the tailings thickener underflow reception box from where two trains of centrifugal pumps (five operating in series and five stand-by for first three years, then seven operating in series and seven stand-by) pump the tailings to the TSF.

Thickener overflow water is recovered in a process water tank located in the vicinity of the thickeners; recovered water is preferentially recycled back to the process from this tank to end users; a make-up stream flows from the process water pond to the process water tank.

1.16.6 Production Plan

The production plan obtained from the mine plan and the metallurgical models for copper and iron recovery assumes yearly average treatment rates of approximately 72,000 t/d during an initial four year period (except for year 01 which is considered Ramp up, that is to say, for years 2 to 5) then stabilizing at 65,000 t/d for the LOM, decreasing the last 4 years, with an annual peak production of 500 kt of copper concentrate for the second year of full production and an annual peak production for magnetite concentrate of 4.5 Mt that occurs in the 5th year of production.

The head grade will vary between 0.39% Cu and 0.59% Cu during the first 7 years of production. After the Seventh year, the head grade is projected to decrease to between 0.33% Cu and 0.20% Cu for nine years, before reaching about 0.12% Cu for the final three years of mining.

For the first 7 years, the head grade will average 29.0% Fe with variation over the LOM. After the seventh year, the head grade for iron is projected to vary between 21.2% Fe and 28.4% Fe, with average of 24.8 Fe%.

1.17 Infrastructure

1.17.1 Planned Facilities

The principal facilities are planned to be located at the following locations:

- The centre of the Santo Domingo property is located approximately at geographic coordinates 26° 27' 46.69” S latitude and 69° 59' 6.74" W longitude and Universal Transverse Mercator (UTM) 397,815m E and 7,073,620m N (datum: WGS 84, Zone 19S).
- Construction and Operations accommodation camp: located on site about 2.5 km north from the mine and plant.
- Tailings storage facility (TSF) designed for storing up to 361 Mt of thickened tailings: located on site about 2 km southeast from the mine and plant.

- High voltage transmission line, length approximately 9 km, from San Lorenzo substation to the main substation at the concentrator plant.
- Magnetite concentrate and water pipelines: approximately 111 km long between the Santo Domingo plant site location and the Santo Domingo port site at Punta Roca Blanca.
- Port facilities: The proposed port, Puerto Santo Domingo at Punta Roca Blanca, will be located approximately 43 km north of the town of Caldera, in the Atacama Region.
- A desalination plant located in the port area operated by a third party (a BOOT type contract).
- High voltage transmission line: from the Totoralillo substation to the Minera Santo Domingo Port and desalination plant infrastructure.

1.17.2 Access Considerations

The planned route for transporting cargo, staff and equipment to the Santo Domingo mine-plant site is from the south of the mine site by Route C-17 and from the north by Route C-13.

Capstone has commenced and partially completed construction of approximately 18.5 km of the C-17 bypass road, to reroute this public highway around the mine-plant project site.

The closest commercial airport is the Desierto Atacama Airport, 113 km south from Chañaral, which has regular scheduled flights to Santiago. The closest airport to the Santo Domingo site is the El Salvador Airport, a private airport, 44 km from the mine-plant site.

The planned port for transport and shipment of heavy machinery, equipment and materials is Punta Angamos in Mejillones, Antofagasta Region, 520 km from the plant site. This port operates throughout the year and is accessed directly from Route 5 North.

1.17.3 Waste Rock Storage Facilities

The waste rock storage facilities (WRF) are located to the west of the open pits. The WRFs were designed in 45 m lifts. Capstone concluded that the WRFs show a moderate to low potential for generation of acid rock drainage.

1.17.4 Mine Stockpile Facilities

The ore stockpiles (High and Low grade) and oxide stockpile are located between the Santo Domingo and Iris Norte pits. The stockpiles are designed with 10 m lifts and 10 m setbacks in order to facilitate later re-handling.

During the pre-production period, the ore stockpile will be constructed for later re-handling to the primary crusher. The total ore to be stockpiled during this period amounts approximately to 6.3 Mt.

1.17.5 Tailings Storage Facility (TSF)

The TSF is located approximately 2 km southeast of the process plant, to store 361 M dry tonnes of tailings in an estimated total volume of 245 Mm^3 at an overall average dry density of 1.47 t/m^3, deposited over 20 years.

The TSF will be developed on a broad alluvial plain that slopes upward to the south and east from a low point in the northwest. Tailings containment will be provided behind and upslope of a west to east earthfill and rockfill downstream dam that will be constructed across the northwest low point. This is termed the Main Dam. Over time the tailings deposit will rise and expand to the south and east and the dam will be progressively raised and extended to the east to continue providing the containment. Six stages of Main Dam development are planned through the life of mine with Stage I serving as the starter dam. Stage I is being planned to provide tailings and water storage containment for the first two years of operation. On the west side of the site, tailings containment will be provided by a high, prominent, bedrock ridge that runs from north to south. The Main Dam will have its west abutment against this ridge.

Two local low points or saddles around the upper perimeter of the site will require construction of saddle dams. Containment in the first area in the southwest side of the site will be by a dam termed the Saddle Dam while containment in the second area immediately to the east will be by a dam termed the Auxiliary Dam. These dams will be developed as earthfill and rockfill dams similar to the Main Dam but since the saddles are at higher elevations they will be smaller than the Main Dam and will not be developed initially.

The tailings will be thickened at the concentrator plant area to approximately 67% solids content, and then pumped to, and deposited in, the TSF. Tailings deposition will utilize a system of regularly rotated spigot offtake points positioned above the upper south side of the TSF with the plan to adopt or approach the sub-aerial method to maximize air drying and density. Liquid solids separation will occur on the beach and the liberated water will flow downslope to the supernatant pond against the Main Dam where it will be recirculated to the process plant.

Water management in the TSF will involve the temporary containment of all surface water in the facility including supernatant water draining from the tailings and net runoff from precipitation falling on the facility. This will be re-cycled to the process plant. A runoff diversion channel will be constructed just beyond the eastern side of the TSF and extended upslope beyond the southern limits of the TSF to divert any upgradient runoff to a point downstream of the Main Dam. The site has a very arid climate, and the month-to-month water balance will be in a significant deficit condition such that there will be no routine discharges. The water contained in the facility will be temporarily stored on top of the tailings against the Main Dam, and the dam will be sized to safely store it.

Closure of the TSF will include removal of any supernatant water, covering the surface of the tailings with non-acid generating (NAG) granular material and modifying the Stage VI emergency spillway at the east side of the Main Dam to serve as the long-term post-closure drainage outlet. The spillway will have sufficient capacity to convey the probable maximum precipitation (PMP) event throughout the post-closure phase of the facility.

1.17.6 Port and Desalination Plant

Puerto Santo Domingo will be located at Punta Roca Blanca, approximately 43 km north of the town of Caldera, in the Atacama Region. The maximum required annual port capacity is 5.4 Mt/y of magnetite concentrate and 0.72 Mt/y of copper concentrate, considering the Santo Domingo operation requirements and the future Mantoverde operation requirements.

Copper concentrate will be delivered to the port by trucks in sealed transport containers. These can be unloaded directly into a negatively pressurized storage area via conveyor belt or stored in a handling area for sealed containers. Magnetite concentrate will be received through a pipeline to the filtration plant where it will be processed for delivery to the stockpile, for subsequent ship-loading by conveyor belt. Both concentrates will be loaded onto ships using a radial loader.

The process water required by the Santo Domingo operation will be produced by a desalination plant located at the port. The process water required by the port will be obtained from the water recovered from the magnetite concentrate filtering process. Potable water for the construction phase in the port will be provided by the sanitation company Aguas Nueva Atacama.

Capstone has held detailed discussions with water supply companies to confirm interest in supplying desalinated water to the operation, from a facility at the port or from another location. The current plan is that a build–own–operate–transfer (BOOT) contractor will construct and operate the sea water intake, reverse osmosis desalination plant and brine return system at the port and the desalinated water pipeline as part of the BOOT contract. Alternatives under consideration are the purchase of desalinated water from an existing plant, from a plant that is planned to be built in the Atacama Region for multiuser supply, or, as part of a district integration opportunity, from a potential expansion to the desalination plant supporting Capstone's nearby Mantoverde operation.

1.17.7 Power and Electrical Supply

High voltage electrical power will be supplied to the mine-plant site and port site via connection to the Chilean national grid (Sistema Eléctrico Nacional or SEN). The Chilean national grid can provide a reliable and continuous supply for the project's electricity needs.

Capstone has entered into a long-term power purchase agreement (PPA) with one of the major power companies that operate on the national grid and supply several of Chile's major mining companies. This is consistent with the strategy used by other mining companies to contract for long-term electricity supply in Chile. The terms and conditions of the PPA are considered customary and competitive in the Chilean electricity market.

The connection point to feed the Mine and Plant is the San Lorenzo substation, about 9 km from the Plant, located in the municipality of Diego de Almagro. The closest connection point to Minera Santo Domingo's Port and Desalination Plant is the Totoralillo substation, about 14 km from the port area.

1.18 Markets and Contracts

Capstone requested marketing consultants David Osachoff, David J. Trotter and Braemar provide information on price projections, sales potential and shipping costs for copper concentrates and iron ore concentrates to be produced over the LOM for the 2024 feasibility study.

Capstone markets copper concentrate from its four mining operations and has established a reputation as a reliable supplier of high-quality concentrates and copper cathodes.

The Santo Domingo copper concentrate is generally considered clean; low in impurities (deleterious or penalty elements). Capstone foresees substantial demand from trading companies that specialize in blending complex materials with cleaner concentrates. These companies typically prefer concentrates like Santo Domingo's due to their compatibility with blending processes and enhanced value proposition. High-quality concentrates are coveted by both smelters and traders alike. This further supports the expected strong demand for Santo Domingo's copper concentrate in the market.

The average consensus estimates from analysts around the world expect copper prices to average $4.10 per pound in the long-term, used in the economic analysis in Section 22 of this Report.

For iron, Santo Domingo will produce three products, a 62%, 65%, and 67% iron concentrate product. The iron concentrate forecast in the production schedule is a typical pellet feed currently used in pellet plants. Magnetite is the predominant mineral. The iron concentrate grade is high, and the low alumina ($Al2O_3$) and low phosphorus (P) make the concentrate suitable for most pellet plants. Suitability and demand for this pellet feed will be considered in the context of increasing use of pellets in iron making, the increased use of higher-grade ores generally and as a premium additive to sinter plants by blending. Iron ore concentrate will be produced for shipment overseas in capesize vessels to iron and steel makers.

Each steel mill has different impurity allowances and tolerances. The impurities of concern for the magnetite iron ore concentrate are silica and copper. In Santo Domingo's forecast magnetite iron ore concentrate, copper is below the threshold but may in some circumstances represent a non-preferred feed while silica is only likely to be a cost factor or penalty element rather than a rejectable quality issue.

The specification of the Capstone pellet feed is on the higher-quality side of current and developing pellets feeds. While an iron content of 67% or higher would be the targeted quality to maximize premiums, iron ore concentrate with an iron content of 62% or more would still be marketable but would have a lower premium.

A forward pricing report prepared for Capstone by David J. Trotter in 2023 estimated that prices for 62% Fe content sinter fines (Platts Iron Ore Index or IODEX) cost-and-freight (CFR) Qingdao delivery (deemed the standard product for CFR China delivery) are expected to be $113.94/dmt (based on the average price from 2018 to 2023). Premiums for 65% Fe concentrate ($15.04/dmt), value-in-use (VIU) for 66% Fe ($17.83/dmt) and for 67% Fe ($26.32/dmt), in all cases compared to Platt 62% Fe.

Braemar reviewed its long-term estimate of shipping costs originally provided in 2018. Over the last 5 years, the market has seen extremes for freight rates for capsize vessels. The current freight C3 is $28.87/wmt. The C3 index has a yearly average rate ranging between $14.10 and $37.40 for the period 2018 to 2023. Given current bunker rates, port charges, vessel values and 60 kt/d load rate with 50 kt/d discharge rate, a projected freight rate of $25.75/wmt was determined for a Chile–China routing. Braemar estimated the freight rate in 2027 for capsize vessels from Chile to north China would be $23.53/wmt.

For the financial modelling and economic analysis in Section 22, Capstone selected a conservative approach to long-term iron concentrate pricing, particularly with respect to the Trotter 62% and 65% (CFR China) forecasts. Analyst consensus estimates were used as the basis for the long-term iron ore price forecast in this Report. As at the effective date of this Report, the long-term consensus 62% and 65% (CFR China) prices were $85/t and $110/t, respectively. An additional premium of $10/t (CFR-China) is assumed for the Project's 67% Fe product.

The financial model assumes a $25/t FOB Chile freight charge based on a Chile-to-China route, reflective of historical and forecast data from Trotter as well as Braemar, a global leader in the shipping industry.

Table 1-3 provides a summary of the key commodity prices and related assumptions used in the economic analysis of the Santo Domingo Project.

Table 1-3: Metal Prices Selected for use in Economic Analysis

Metric	Unit	Price (US$)
Copper	US$/lb	4.10
Gold	US$/oz	1,800
62% Iron magnetite Conc. Price (CFR China)	US$/t	85
65% Iron magnetite Conc. Price (CFR China)	US$/t	110
67% Iron magnetite Conc. Price (CFR China)	US$/t	120

Note: $25/t freight assumed in Iron Concentrate CFR China price

On March 25, 2021, Capstone entered a Precious Metals Purchase Agreement between Wheaton Precious Metals (WPM), Capstone Resources MSD Ltd. and Capstone with respect to the purchase and sale of gold. WPM paid an upfront cash consideration of $30M and additional deposits totaling $260M for total consideration of $290M. WPM will receive 100% of the gold production until 285,000 ounces have been delivered, thereafter dropping to 67% of the gold production. WPM will make ongoing payments equal to 18% of the spot gold price, until the deposit of $290M has been reduced to zero, thereafter increasing to 22% of the spot gold price upon delivery.

No additional contracts are in place for Santo Domingo's production for copper or iron ore concentrate. Capstone may obtain signed letters of intent (LOIs) or memorandums of understanding (MOUs) from smelters, followed by full long-term contracts, to support arrangements for project financing.

1.19 Environmental, Permitting, and Social Considerations

1.19.1 Environmental Considerations

The Santo Domingo Project obtained its environmental licences in 2015 and later, in 2020, for the Desalination Plant. The Santo Domingo Project Environmental Impact Study (EIA) included an environmental baseline conducted between 2010 and 2013 for the four defined project areas (Mine-Plant, Pipeline, Port, Electrical Transmission Line), which extend from Santo Domingo and the city of Diego de Almagro to the Punta Roca Blanca sector in Caldera. No critical environmental elements were found during the baseline studies, however, as a next step and due to the time elapsed, different environmental studies (air quality, hydrogeology, archeology, flora and fauna, among others) have begun to be updated in order to start the development of a new process for obtaining environmental and sectoral permits that will allow the engineering optimizations currently underway to proceed.

1.19.2 Permitting Considerations

Capstone currently has an environmental licence that allows starting construction and operation under the previously granted Environmental Qualification Resolutions (RCAs) 119/2015 and 122/2020. Capstone has received sectoral approvals for critical permits including the construction and operation of the tailings storage facility (TSF), tailings deposits, process plant, mining method, mine closure plan, as well as maritime concession and preliminary approval for port infrastructure and will continue to obtain all sectoral authorizations required for the Project, based on environmental approvals.

Some engineering optimizations currently in progress, such as the increased production described in this Report, will require Capstone to enter the Environmental Impact Assessment System (SEIA) through an appropriate mechanism, such as an Environmental Impact Declaration (DIA). Once the favourable Environmental Qualification Resolution (RCA) of the optimized Project is obtained, Capstone will proceed to update and obtain all the necessary sectoral permits required for the construction, operation and closure stages of the optimized project.

Capstone will pursue a strategy that includes the development of management plans and other tools for environmentally sensitive areas, which will ensure compliance with environmental, permitting and social commitments.

1.19.3 Closure Considerations

Similar to the current Mine Closure Plan (2019), the closure costs related to closure of the Santo Domingo project are estimated at US$124M. This closure cost will be reviewed in the next stage of engineering.

According to Chilean law, a bond, surety bond or insurance must be provided to the State to cover this cost.

1.19.4 Social Considerations

The communities and towns near the Project are Diego de Almagro, Chañaral and Caldera. Although there are no Colla or other indigenous lands or territories of any kind claimed on the Property, Capstone maintains open lines of communication for possible approaches or inquiries from this community.

A stakeholder plan was developed and implemented to facilitate an early community participation program that helped the local communities involved to understand the major Project components and for MSD to gather community opinions, comments and feedback. Meetings with the community, to present the Project and record their observations, were held between 2012 and 2020 in Diego de Almagro, Chañaral, Copiapó and Caldera, by means of open houses, open meetings, meetings for special interest groups such as fishermen and meetings with authorities, regional and community services as well as with professional organizations. As a result of this process, changes in the design were made to minimize impacts to the environment and surrounding communities.

Moving forward, Capstone has committed to a communications strategy that will focus on building a positive reputation and supportive environment for the development of a mining operation in the Atacama Region. A communications plan, communications committee and crisis response management plan are being developed and a

Community Office will be established in Diego de Almagro to support the deployment of the communication strategy with stakeholders.

1.20 Capital Cost Estimates

All capital costs are in Q2 2023 US$. A foreign exchange rate of 800 CLP to US$1 was used for the detailed estimate.

Capital cost estimates were prepared by the various consultants working on this Report and were based on battery limits established by Capstone. Owner costs were provided by Capstone. Estimates were based on a combination of direct quotes and benchmarking. The estimate is a Type 3 estimate according to AACE International standards, with an accuracy of -10 to +15% at the 85% confidence level.

The initial capital cost was estimated to be $2,315M. The estimated sustaining capital cost, excluding deferred stripping, totals approximately $441M. The combined initial and sustaining capital costs for the LOM were estimated to be about $2,756M (Table 1-4).

Table 1-4: Capital Cost Estimate

Description	Area	Cost ($M)
Initial Capital	Mine	370
	Process Plant	486
	Tailings and Water Reclaim	67
	Plant Infrastructure (On Site)	144
	Port & Port Infrastructure (On Site)	283
	External Infrastructure (Off Site)	151
	Indirect Costs	414
	Owner Costs	109
	Contingency	291
Total Initial Capital		**2,315**
Total Sustaining Capital (excluding deferred stripping)		**441**
Total Closure Cost		**124**

Note: Values may not sum due to rounding.

1.21 Operating Cost Estimates

All operating cost estimates are in Q4 2023 US$. Costs are based on a foreign exchange rate of CLP800 to US$1.00. For the CuEq estimate, prices of $4.10/lb Cu, $1,800/oz gold, $60/t for 62% grade magnetite concentrate, $85/t for 65% grade magnetite concentrate and $95/t for 67% grade magnetite concentrate were used. The operating cost estimate is considered to be at a feasibility study level, with an accuracy of -10% to +15%.

The operating costs are summarized in Table 1-5. The total operating cost over the projected life-of-mine is $5,846M (excluding copper concentrate land transport, which is captured as a cost in the economic model under “Copper and Magnetite Concentrate Transport and Insurance Charges”).

Table 1-5: Operating Cost Estimate

Cost Centre	LOM Total ($M)	LOM Average ($/t Treated)	LOM Average ($/lb CuEq)	LOM Average ($/lb Cu payable)
Mining	1,588	3.64	0.36	0.58
Process Plant	1,944	4.46	0.44	0.71
Plant Infrastructure	200	0.46	0.05	0.07
Tailings and Water Recovery	57.0	0.13	0.01	0.02
External Infrastructure	902	2.07	0.20	0.33
Port Infrastructure	94.6	0.22	0.02	0.03
Port	195	0.45	0.04	0.07
General	315	0.72	0.07	0.12
G&A	549	1.26	0.12	0.20
Total	**5,846**	**13.41**	**1.32**	**2.13**

Note: Values may not sum due to rounding. Cu Eq considers the gold and iron recovered in the process converted to equivalent copper based on a copper price of US$4.10/lb.

1.22 Economic Analysis

The results of the economic analysis to support Mineral Reserves represent forward-looking information that is subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here.

Forward-looking statements in this Report include, but are not limited to, statements with respect to future metal prices and concentrate sales contracts, assumed currency exchange rates, the estimation of Mineral Reserves and Mineral Resources, the realization of Mineral Reserve estimates including the achievement of the dilution and recovery assumptions, the timing and amount of estimated future production, costs of production, capital expenditures, costs and timing of the development of ore zones, permitting time lines, requirements for additional capital, government regulation of mining operations, environmental risks, unanticipated reclamation expenses and title disputes.

Additional risk can come from actual results of reclamation activities; conclusions of economic evaluations; changes in parameters as mine and process plans continue to be refined, possible variations in ore reserves, grade or recovery rates; geotechnical considerations during mining; failure of plant, equipment or processes to operate as anticipated; shipping delays and regulations; accidents, labour disputes and other risks of the mining industry; and delays in obtaining governmental approvals.

Years discussed in this sub-section are presented for illustrative purposes only, as no decision has been made on mine construction by Capstone.

The Property was evaluated using an 8% discounted cash flow (DCF) analysis on a non-inflated, after-tax basis. The cash flows consist of approximately 3 years of pre-production costs and 19 years of operation. Cash inflows consist of annual revenue projections for the mine. Cash outflows include capital costs, operating costs, royalties, and taxes, which are subtracted from the inflows to arrive at the annual cash flow projections.

To reflect the time value of money, annual net cash flow (NCF) projections are discounted back to the beginning of the project execution, using an 8% discount rate. The discount rate was determined using several factors, including the type of commodity and the level of risks (market risk, technical risk and political risk). The discounted present values of the cash flows are summed to arrive at the net present value (NPV).

An NPV sensitivity analysis to discount rates was completed using discount rates of 6%,7%, 8% (selected rate), 9% and 10%. In addition to the NPV, the internal rate of return (IRR) and payback period were also calculated. The IRR is defined as the discount rate that results in an NPV equal to zero. Cash flows are assumed to occur on an average mid-year basis of each annual period.

The financial model is based on the Mineral Reserves outlined in Section 15, the mining rates and assumptions discussed in Section 16 and the recovery and processing rates and assumptions discussed in Section 13 and Section 17, respectively.

Initial capital costs are estimated to be $2,315M. Over the LOM sustaining capital is estimated to be $1,329M including the deferred stripping.

LOM operating costs are estimated to be $5,846M.

Closure and reclamation costs are estimated to be $124M.

LOM refining and transport costs are estimated to be $884M.

For the transport and insurance charges for the magnetite concentrate, the following are included in the operating costs and the financial model:

- The magnetite concentrate is transferred to the port via pipeline with the costs included in operating costs.
- The realized price for its magnetite product is assumed to be free on board (FOB) Santo Domingo port.
- No additional transport or insurance charges are required to be included in the financial model for iron concentrate.
- Royalties of 2% NSR are payable to third parties on production of 87% total reserves. The NSR is charged on all of the metals (copper, iron and gold) recovered. The LOM royalty payments are estimated to be $288M.
- The economic analysis assumes that no inflationary adjustments are made. Capital and operating costs are based on Q1 2024 US dollars.

The pre-tax NPV discounted at 8% is US$ 2,670M, the IRR is 29.7%, and payback period is 2.9 years. On a post-tax basis, the NPV discounted at 8% is US$1.720M the IRR is 24.1%, and payback period is 3.0 years. A cash flow summary is included below in Table 1-6.

Table 1-6: Summary of Cash Flow

General	Unit	Value
Copper Price	US$/lb	4.1
Gold Price	US$/oz	1,800
62% Grade Magnetite Conc. Price (CFR China)	US$/t	85
65% Grade Magnetite Conc. Price (CFR China)	US$/t	110
67% Grade Magnetite Conc. Price (CFR China)	US$/t	120
Freight Charge (Chile-to-China)	US$/t	25
Mine Life	years	19
Production		**LOM Total / Avg.**
Total Mill Feed Tonnes	kt	436,056
Mill Head Grade Cu	%	0.33
Mill Head Grade Au	g/t	0.045
Mill Head Grade Fe	%	26.48
Mill Recovery Rate Cu	%	90.13
Mill Recovery Rate Au	%	64.74
Mill Recovery Rate Fe	%	38.76
Total Recovered Copper	kt	1,292
Total Gold Ounces Recovered	koz	411.6
Total Iron 62% Grade Tonnes Recovered	Mt	13.7
Total Iron 65% Grade Tonnes Recovered	Mt	35.38
Total Iron 67% Grade Tonnes Recovered	Mt	19.29
Total Average Annual Copper Production	Mlb/y	144
Average Year 1 to 5 Annual Copper Production	Mlb/y	240
Total Average Annual Gold Production	koz/y	19.5
Average Year 1 to 5 Annual Gold Production	koz/y	33.6
Operating Costs		**LOM Total / Avg.**
Mining Operating Cost	USD$/t moved	1.57
Processing Cost	US$/t Milled	8.50
G&A Cost	US$/t Milled	1.26
Refining and Transport Cost	US$/lb Cu Eq.	0.21
Total Operating Costs	US$/t Milled	13.41
C1 (co-product basis)*	US$/lb Cu Eq.	1.59
C1 (by-product basis)*	US$/lb Cu	0.33
Capital Costs		**LOM Total / Avg.**
Initial Capital	US$B	2.32
Sustaining Capital (includes deferred stripping)	US$B	1.33
Closure Costs	US$B	0.12
Financials - Pre-Tax		**LOM Total / Avg.**
Revenue	US$B	17.1
Royalty	US$B	0.29
NPV (8%)	US$B	2.6
IRR	%	29.7
Payback	years	2.9
Capital Intensity	US$/t Cu Eq.	21,860
NPV/Initial CAPEX	-	1.15
Financials - Post Tax		**LOM Total / Avg.**
Tax	US$B	2.02
NPV (8%)	US$B	1.72
IRR	%	24.1
Payback	years	3.0
NPV / Initial Capex	-	0.74

Note: Totals may not sum due to rounding. Possible salvage values for the mine, plant and port were not considered, due to the approximately 19-year mine life. At closure, sale of assets may present an opportunity to offset a portion of the closure and reclamation costs.

1.22.1 Sensitivity Analysis

A sensitivity analysis was performed on the financial model considering variations in:

- Metal price (copper, iron, and gold);
- Operating costs (including power);
- Capital costs;
- Cu recovery;
- Cu head grade;

The analysis shows that the Santo Domingo NPV8% is most sensitive to changes in the copper price, copper head grade and capital expenditures. The sensitivity analysis showed that Santo Domingo is less sensitive to changes in the gold and iron price.

1.23 Mantoverde-Santo Domingo District

Santo Domingo is located 35 km northeast of Capstone’s Mantoverde Mine, approximately 65 km via public roadways, and 16 km southeast of Capstone’s Sierra Norte property, comprising the Mantoverde-Santo Domingo (MV-SD) District.

The Mantoverde Mine is a copper-gold producer, with 70% ownership held by Capstone and 30% held by Mitsubishi Materials Corporation, described in the “Mantoverde Mine and Mantoverde Development Project NI 43-101 Technical Report Chañaral / Región de Atacama, Chile”, effective November 29, 2021. The Mantoverde Mine is a conventional open pit operation using truck-and-shovel technology. A concentrator plant was constructed in 2023 to treat sulphide ore; copper concentrate production is expected to begin in the second half of 2024. Oxide ores will continue to be treated in the existing solvent extraction and electrowinning plant (SX-EW). The expansion is expected to increase production from approximately 35,000 tonnes of copper as cathodes in 2023 to approximately 110,000 tonnes of copper per year combining cathodes and concentrate at full capacity. The Mantoverde Mine will be able to use the planned Santo Domingo port to ship concentrate.

The Sierra Norte property, an IOCG-type copper deposit, is located approximately 20 kilometres northwest of the Santo Domingo Project and covers over 7,000 hectares. Copper mineralization at Sierra Norte occurs in irregular tabular, bodies of specularite breccias with chalcopyrite at the Carmen-Paulina zone and copper oxidized breccias at the Esther zone. Halos of specularite-chalcopyrite vein stockworks are noted at Carmen Paulina. The deposit remains open to the south and southeast. A historical resource estimate, not compliant with NI 43-101, of approximately 100Mt at 0.45% CuT is shown in Table 1-7. Sierra Norte represents an opportunity to potentially become a future sulphide feed source for Santo Domingo, extending its higher-grade copper sulphide life, with additional upside for future exploration. Capstone entered into a binding share purchase agreement with Inversiones Alxar S.A. (“Alxar”) and Empresas COPEC S.A. (“EC”), collectively the “Sellers” to acquire 100% of Compania Minera Sierra Norte S.A. (“Sierra Norte”) on July 17, 2024. Under the terms of the purchase agreement, Capstone paid the Sellers $40M in share consideration.

Table 1-7 Sierra Norte, Historical Mineral Resource

Category	Tonnage (Mt)	Grade		Contained Metal
		CuT (%)	CuSA (%)	Cu (kt)
Carmen-Paulina				
Measured	8	0.47	0.16	35
Indicated	63	0.46	0.10	292
Total Measured + Indicated	**71**	**0.46**	**0.11**	**327**
Inferred	25	0.40	0.04	102
Esther				
Measured	1	0.42	0.26	3
Indicated	3	0.40	0.24	13
Total Measured + Indicated	**4**	**0.40**	**0.24**	**16**
Inferred	0.1	0.35	0.22	0.3

The Historical Mineral Resource was derived from the report "Actualización del Modelo Geológico y de la Estimación de Recursos Minerales del Proyecto Diego de Almagro" completed by Amec Foster Wheeler with an effective date of April 29, 2016, prepared for Alxar S.A. The historical estimates are strictly historical in nature and are not compliant with NI 43-101 and should not be relied upon. A qualified person has not done sufficient work to classify the historical estimates as current "mineral resources", as such term is defined in NI 43-101 and it is uncertain whether, following further evaluation or exploration work, the historical estimate will be able to report as mineral resources in accordance with NI 43-101. Capstone has not done sufficient work to classify the historical estimate as current mineral resources and is not treating the historical estimate as current mineral resources. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. The historical estimate is reported using a cut-off grade of 0.20% with further economic extraction parameters outlined below. Categories are based on average spacing of drillholes and levels of confidence in the grade estimation. There are no more recent estimates or data available to Capstone. The Sierra Norte deposit will require further evaluation including drilling to verify the historical estimate as current mineral resources. Readers are cautioned not to rely on the historical estimate in this section. Economic Parameters for the historical estimate include the following: Copper price: $3.00/lb; Mining cost: $1.69/t; Sulphide recovery: 91%; Sulphide processing cost: $7.26/t; Oxide (heap) recovery: 60%; Oxide (heap) processing cost: $8.12/t; Oxide (SX-EW) processing cost: $0.30/lb; Concentrate selling costs: $0.41/lb; and Cathodes selling costs: $0.04/lb.

1.24 Other Relevant Data and Information

The project execution strategy includes the engagement of an EPCM contractor (Engineering, Procurement, Construction Management). The project execution schedule considers an approximately 3-year period from execution funding approval to commencement of production ramp up.

The following future optimisation opportunities have been identified:

- Pyrite processing for cobalt recovery;
- Acid leaching of oxide material for copper recovery;
- Review of regrind mill technology and installed power requirements;
- Optimisation of the water balance around the magnetite and tailings thickening areas;

- Improve the design of the Jameson flotation cell installation;
- Integrated tailings thickeners-TSF assessment;
- Mantoverde – Santo Domingo mining district integration.

Exploration potential for copper oxides exist above the actual sulfide orebody, as well as additional opportunities to explore for copper sulfides, magnetic iron and cobalt bearing pyrite near both open pits.

Capstone's next phase of growth will involve a construction decision for Santo Domingo, considering integration of the Project with the expanded Mantoverde Mine. Capstone aims to create a 250 ktpa copper concentrate and cathode business (consolidated Mantoverde and Santo Domingo) in a world-class mining district in the Atacama region of Chile. The combination of key infrastructure already in place alongside an experienced mine build and operating team supports future development of the MV-SD district. The base case plan for MV-SD district integration includes upgrades to the existing water and power infrastructure proximal to both projects.

1.25 Interpretation and Conclusions

Based on the assumptions and parameters presented in this report, the Santo Domingo FS Update shows positive economics (i.e., $1.720M post-tax NPV (8%) and 24.1% post-tax IRR). The FS Update supports a decision to carry out additional detailed studies.

1.26 Recommendations

The list of recommendations can be completed concurrently. The total cost estimate for 3rd party services to complete the proposed work programs is $20.3M. This cost estimate excludes Capstone's in-house costs, which are to be incurred as part of the Capstone team's ongoing work on the project prior to execution approval. The recommended costs are broken down by the categories in Table 1-8 and Section 26.

Table 1-8: Recommended Work

Area	*($M)
Exploration and Drilling	6.20
Sample Preparation, Analyses and Security	0.06
Data Verification	0.16
Metallurgical Testing	0.83
Mineral Resource Estimates	0.20
Recovery Methods	0.80
Infrastructure	2.63
Markets and Contracts	0.02
Environmental, Permitting and Community	0.37
Other Relevant Information: Project Execution Plan (Interim Engineering Phase)	9.0
Total	**20.3**

*3rd Party services cost estimate summary for the recommendations, excluding Capstone's in-house costs.

2 INTRODUCTION

2.1 Introduction

This technical report was compiled by Ausenco for Capstone Copper Corp (Capstone) to conform to the regulatory requirements of Canadian National Instrument (NI) 43-101 using the form 43-101 F1 Standards of Disclosure for Mineral Projects. The Santo Domingo Project involves the development of mining and processing facilities on Capstone's Santo Domingo Property (the Property), located in the Atacama Region (Region III) of the Republic of Chile.

2.2 Terms of Reference

The term "Property" is used in reference to the overall mineral tenure holdings that encompass the Santo Domingo property.

The Property is held 100% by Capstone, which prior to 2022 was Capstone Mining Corp. The operating entity is Minera Santo Domingo SCM (MSD), as the Chilean holding company for the Property.

A previous technical report was issued with an effective date of 19 February 2020, that was filed on 24 March 2020 (referred to as the 2020 Technical Report).

The Report provides an update on the project. The project includes conventional open pit mining of a copper and magnetite deposit with ore feeding a processing plant to produce separate copper–gold and iron concentrates, and thickened tailings. Tailings are stored on the Property. Filtered copper concentrate is trucked to the port, whilst magnetite concentrate is transported by pipeline to the filtration plant at the port.

Figure 2-1: Property Location

Source: Figure courtesy Capstone, 2013.

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

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

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

2.3 Qualified Person

The Qualified Person (QP) for the report is Peter Amelunxen, P.Eng., Capstone's Senior Vice President, Technical Services. Mr. Amelunxen is not independent from Capstone. Mr. Amelunxen visited the property on 17 November 2022. During this visit he viewed the areas proposed for the mine, adjacent process plant and TSF.

Table 2-1: Qualified Person

Qualified Person	Professional Designation	Position	Employer	Independent of Capstone	Site Visit Dates
Peter Amelunxen	P.Eng. 49144	Senior Vice President, Technical Services	Capstone Copper	No	17 November 2022

2.4 Report Contributors

Table 2-2 lists Capstone and Ausenco key contributors to this report.

Table 2-2: Report Contributors

Section Number	Section Description	Report Contributor
1	Summary	Peter Amelunxen, Vivienne McLennan, Giancarlo Daroch, Guillermo Pareja, Clay Craig, Greg Lane, James Millard
2	Introduction	Peter Amelunxen, Vivienne McLennan, Giancarlo Daroch, Guillermo Pareja, Clay Craig, Greg Lane, James Millard
3	Reliance on Other Experts	Peter Amelunxen
4	Property Description and Location	Vivienne McLennan, James Millard
5	Accessibility, Climate, Local Resources, Infrastructure and Physiography	Peter Amelunxen
6	History	Giancarlo Daroch
7	Geological Setting and Mineralization	Giancarlo Daroch
8	Deposit Types	Giancarlo Daroch
9	Exploration	Giancarlo Daroch
10	Drilling	Giancarlo Daroch
11	Sample Preparation, Analyses and Security	Vivienne McLennan
12	Data Verification	Vivienne McLennan
13	Mineral Processing and Metallurgical Testing	Peter Amelunxen
14	Mineral Resource Estimates	Guillermo Pareja
15	Mineral Reserve Estimates	Clay Craig

Section Number	Section Description	Report Contributor
16	Mining Methods	Clay Craig
17	Recovery methods	Greg Lane, Peter Amelunxen
18	Project Infrastructure	Peter Amelunxen
19	Market Studies and Contracts	David Osachoff, Peter Amelunxen
20	Environmental Studies, Permitting and Social or Community Impact	James Millard
21	Capital and Operating Costs	Greg Lane, Peter Amelunxen
22	Economic analysis	Greg Lane, Peter Amelunxen
23	Adjacent Properties	Peter Amelunxen
24	Other relevant data and information	Greg Lane, Peter Amelunxen
25	Interpretation and conclusions	Peter Amelunxen, Vivienne McLennan, Giancarlo Daroch, Guillermo Pareja, Clay Craig, Greg Lane, James Millard
26	Recommendations	Peter Amelunxen, Vivienne McLennan, Giancarlo Daroch, Guillermo Pareja, Clay Craig, Greg Lane, James Millard
27	References	Peter Amelunxen, Vivienne McLennan, Giancarlo Daroch, Guillermo Pareja, Clay Craig, Greg Lane, James Millard

2.5 Effective Dates

The Report has a number of effective dates as follows:

- Date of supply of last assay data used in resource estimation: 7 Oct 2021.
- Date of Mineral Resource estimate: 31 March 2024.
- Date of Mineral Reserve estimate: 31 March 2024.
- Date of Financial analysis: 31 July 2024.

The effective date of the Report is taken to be the date of financial analysis, which is 31 July 2024.

2.6 Information Sources and References

Information sources supporting the Report include the 2020 Technical Report and other documents prepared to support the Santo Domingo FS Update.

Information used to support this Report was also derived from previous technical reports on the Santo Domingo property, from the sources listed in Section 2.7, from expert documents cited in Section 3 and from the reports and documents listed in Section 27. Additional information was sought from Capstone personnel where required.

2.7 Previous Technical Reports

Capstone has filed the following technical reports on the Santo Domingo property:

- Maycock, J., Luraschi, A., Mendoza, M., Bianchin, M., Rennie, D., Guzman, C., Amelunxen, R., Gingles, M., Kerr, T., Betinol, R., Jones, L., and Bush, G., 2020: Santo Domingo Project, Region III, Chile, NI 43-101 Technical Report: technical report prepared by Amec Foster Wheeler International Ingeniería y Construcción Limitada, Roscoe Postle Associates Inc., NCL Ltda, Aminpro Chile SPA, Sunrise Americas LLC, Knight Piésold S.A., Mplan and Brass Chile SA for Capstone Mining Corp., effective date 19 February, 2020.
- Maycock, J., Luraschi, A., Mendoza, M., Bianchin, M., Rennie, D., Guzman, C., Amelunxen, R., Gingles, M., Kerr, T., and Betinol, R., 2018: Santo Domingo Project, Region III, Chile, NI 43-101 Technical Report on Feasibility Study Update: technical report prepared by Amec Foster Wheeler International Ingeniería y Construcción Limitada, Roscoe Postle Associates Inc., NCL Ltda, Aminpro Chile SPA, Sunrise Americas LLC, Knight Piésold S.A., and Brass Chile SA for Capstone Mining Corp., effective date 26 November, 2018.
- Maycock, J., Gopfert, H., Rennie D., Guzman, C., Frost, D., Kerr, T., Betinol, R., Klimek, A., and Khera V., 2014: Santo Domingo Project, Region III, Chile, NI 43-101 Technical Report on Feasibility Study: technical report prepared by Amec International Ingeniería y Construcción Limitada, Roscoe Postle Associates Inc., NCL Ltda, Knight Piésold and Co., and Brass Chile SA, effective date 22 May 2014.

Far West Mining Ltd. (Far West), a predecessor company to Capstone, prepared the following reports to document progress on the Santo Domingo Project:

- Allen, G.J., 2004: Report on the 2003 Exploration Activities in the 4c Santo Domingo Area of the Candelaria Project, Region III, Chile: unpublished report prepared by Far West Mining Ltd.
- Allen, G.J., 2005: Report on the Exploration Activities in the 4a Santo Domingo Area of the Candelaria Project, Region III, Chile: unpublished report prepared by Far West Mining Ltd., effective date 31 July 2005.
- Allen, G.J., and Höy, T., 2005: Exploration Activities in the 4a (Santo Domingo) Area of the Candelaria Project, Region III, Chile: unpublished report prepared by Far West Mining Ltd., effective date 10 December 2005.
- Penner, R., Lacombe, P., Maycock, T., and Henry, E., 2008: Review of the Santo Domingo Sur and Iris Project, Region III, Chile: unpublished report prepared by Amec International (Chile) S.A. for Far West Mining Ltd., effective date 30 April 2008.
- Lacroix, P.A., 2006: Technical Report on the 4A (Santo Domingo) Area of the Candelaria Project, Region III, Chile: unpublished technical report prepared by Roscoe Postle Associates Inc. for Far West Mining Ltd., effective date 13 June 2006.
- Lacroix, P.A., and Rennie, D.W., 2007: Technical Report on the 4A (Santo Domingo) Area of the Candelaria Project, Region III, Atacama Province, Chile: unpublished technical report prepared by Roscoe Postle Associates Inc. for Far West Mining Ltd., effective date 19 October 2007.
- Lacroix, P.A., 2009: Technical Report on the Santo Domingo Property, Region III, Chile: unpublished technical report prepared by Roscoe Postle Associates Inc. for Far West Mining Ltd., effective date 4 June 2009.
- Rennie, D., 2010: Technical Report on the Santo Domingo Property, Region III, Atacama Province, Chile, NI 43-101 Report: unpublished report prepared by Scott Wilson Roscoe Postle Associates for Far West Mining Ltd effective date 26 August 2010.

2.8 Currency, Units, Abbreviations and Definitions

All units of measurement in this report are metric and all currencies are expressed in United States of America dollars (symbol: US$ or currency: USD) unless otherwise stated. Contained gold metal is expressed as troy ounces (oz), where 1 oz = 31.1035 g. All material tonnes are expressed as dry tonnes (t) unless stated otherwise. A list of abbreviations and acronyms is provided in Table 2-3, and units of measurement are listed in Table 2-4.

Table 2-3: Abbreviations and Acronyms

Abbreviation	Description
A	Jktech Drop-Weight Test Rock Breakage Parameter A
AA	Atomic Absorption Spectroscopy
ABA	Acid-Base Accounting
AG	Autogenous Grinding
Ai	Bond Abrasion Index
ANDE_LOWER	Lower Andesite Geological Unit
ANDE_UPPER	Upper Andesite Geological Unit
ANLT	Andesitic Lithic Tuff Geological Unit
Au	Gold
Az	Azimuth
b	Jktech Drop-Weight Test Rock Breakage Parameter b
BF	Blast Furnace
BIF	Banded Iron Formation
BWi	Bond Ball Mill Work Index
BX	Hydrothermal Breccia Geological Unit
CAD:USD	Canadian American Exchange Rate
CARB	Carbonate Rocks Geological Unit
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CIM Definition Standards	CIM Definition Standards for Mineral Resources and Mineral Reserves 2014
CIP	Carbon In Pulp
CoG	Cut-Off Grade
CRM	Certified Reference Material
CuSA	Copper Soluble in Acid
CuSAratio	Copper Solubility-In Acid Ratio
CuT	Total Copper from Both Sulphide and Oxide Mineralization
CWi	Bond Crusher Work Index
DCIP	Direct Current Resistivity and Induced Polarization
DDH	Diamond Drill Hole
DIOR	Diorite Geological Unit
DRI	Direct Reduced Iron
DT	Davis Tube
DTT	Davis Tube Test
DYKE	Dykes Geological Unit
E-GRG	Extended Gravity Recoverable Gold

Abbreviation	Description
EM	Electromagnetic
FA	Fire Assay
FET	Federal Excise Tax
FS	Feasibility Study
G&A	General And Administration
GPR	Gross Production Royalty
GQCV	Greenstone-Hosted Quartz-Carbonate Vein Deposits
GRAV	Gravimetric Finish Method
ICP	Inductively Coupled Plasma
ICP-OES	Inductively Coupled Plasma - Optical Emission Spectrometry
ID2	Inverse Distance Squared
ID3	Inverse Distance Cubed
IOCG	Iron Oxide Copper Gold
IP	Induced Polarization
IRGS	Intrusion-Related Gold System
ISO	International Organization for Standardization
LCT	Locked Cycle Test
LEIT	Low Energy Impact Test
LIDAR	Light Detection and Ranging
LIMS	Low Intensity Magnetic Separators
LTM	Local Transverse Mercator
LUP	Land Use Permit
Magsus	Magnetic Susceptibility
Masstrec	Mass Recovery
MCF	Mechanized Cut and Fill
MRE	Mineral Resource Estimate
MSD	Minera Santo Domingo
NAD 83	North American Datum Of 1983
NAG	Non-Potentially Acid Generating
NI 43-101	National Instrument 43-101 (Regulation 43-101)
NN	Nearest Neighbour
NSR	Net Smelter Return
NTS	National Topographic System
OCT	Open Cycle Test
OK	Ordinary Kriging
P_{80}	Particle Size at Which 80% Of the Material Will Pass When Screened
PAG	Potentially Acid-Generating
PEA	Preliminary Economic Assessment
PFS	Prefeasibility Study
PGE	Platinum Group Elements
QA/QC	Quality Assurance/Quality Control
QEMScan	An Automated Quantitative Electron Microscopy Technique

Abbreviation	Description
QP	Qualified Person (As Defined in National Instrument 43-101)
RC	Reverse Circulation
REE	Rare Earth Elements
ROM	Run Of Mine
RQD	Rock Quality Designation
RWi	Bond Rod Mill Work Index
SAG	Semi-Autogenous Grinding
SCC	Standards Council of Canada
SD	Standard Deviation
Sd-BWi	Micro Hardness or Bond Ball Mill Work Index on SAG Ground Material
SEDEX	Sedimentary Exhalative Deposits
SG	Specific Gravity
SKT	Simple Kinetic Test
SMBS	Sodium Metabisulphate
SMC	Semi-Autogenous Mill Competency Test
TSF	Tailings Storage Facility
UG	Underground
UTM	Universal Transverse Mercator Coordinate System
UV	Ultraviolet
V	Vanadium
VLCL	Volcaniclastic Rocks Geological Unit (Iris Norte)
VLCL_BOTTOM	Volcaniclastic Rocks Geological Unit Under ANLT (Santo Domingo)
VLCL_TOP	Volcaniclastic Rocks Geological Unit Above ANLT (Santo Domingo)
VLF-EM	Very Low Frequency Electromagnetic
VMS	Volcanogenic Massive Sulphide

Table 2-4: Units of Measurement

Abbreviation	Description
%	percent
% solids	percent solids by weight
CAD	Canadian dollar (currency)
$/t	dollars per metric ton
°	angular degree
°C	degree Celsius
µm	micron (micrometre)
B	billion
cm	centimetre
cm^3	cubic centimetre
dwt	discrete wavelet transform
ft	foot (12 inches)
g	gram
g/cm^3	gram per cubic centimetre

Abbreviation	Description
g/L	gram per litre
g/t	gram per metric tonne
h	hour (60 minutes)
ha	hectare
kg	kilogram
kg/t	kilogram per tonne
km	kilometre
km^2	square kilometre
kW	kilowatt
kWh/t	kilowatt-hour per tonne
L	litre
lb	pound
m, m^2, m^3	metre, square metre, cubic metre
M	million
Ma	million years (annum)
masl	metres above mean sea level
mm	millimetre
Moz	million (troy) ounces
Mt	million tonnes
MW	megawatt
oz	troy ounce
oz/t	ounce (troy) per tonne
oz/ton	ounce (troy) per short ton (2,000 lbs)
ppb	parts per billion
ppm	parts per million
t	metric tonne (1,000 kg)
ton	short ton (2,000 lbs)
t/d	tonnes per day
USD	US dollars (currency)
US$	US dollar (as symbol)
wmt	wet metric tonne

3 RELIANCE ON OTHER EXPERTS

3.1 Introduction

The QP has relied upon the expert reports, which provided information regarding mineral rights, surface rights, property agreements, taxes and marketing sections of this Report as noted below.

3.2 Mineral Tenure, Rights of Way, and Easements

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

- Baker McKenzie (Abogados): Mining Tenements – Minera Santo Domingo SCM: legal opinion prepared for Minera Santo Domingo, January 11th, 2024.
- Quinzio (Abogados): Informe de Títulos Servidumbres Legales Mineras Minera Santo Domingo: legal opinion prepared for Minera Santo Domingo, 25 January 2024.

This information is used in Section 1.4 and 4 of the Report. It is also used in support of the Mineral Resource statement in Section 14, the Mineral Reserve statement in Section 15 and the economic analysis result in Section 22.

3.3 Taxation

The QP has fully relied upon and disclaims responsibility for, information supplied by Capstone staff and experts retained by Capstone for information related to taxation as applied to the financial model as follows:

- Ernst and Young, 2024: Ausenco Model Review – Tax Aspects, dated July 31st, 2024, for complete revision and validation of the tributary aspects contained in Financial Model.
- Baker McKenzie, ”Santo Domingo - Memo vigencia DL 600(412104724.5)”, dated April 10th, 2024, for DL 600 validity.
- Capstone Copper, “2021-05 PMPA Gold Stream Accounting Memo - Final”, dated May 2021, and “MSD Gold Stream - transaction flow.ppt“ for methodology of inclusion of gold stream contract.

This information is used in the financial model in Section 1.22 and 22 of the Report.

3.4 Commodity Markets

The QP has fully relied upon, and disclaims responsibility for, information supplied by experts retained by Capstone for copper marketing and pricing through the following documents:

David Osachoff, Marketing Consultant to Capstone, for specialized commodity market knowledge summarized in Section 19 (March 17, 2024) (Capstone, 2024a).

This information is used in Sections 1.18, 1.22, 19 and 22 of the Report and in support of the Mineral Reserve statement in Section 15. It is also used to support the reasonable prospects for eventual economic extraction in Section 14. Metals price forecasting is a specialized business requiring knowledge of supply and demand, economic activity and other factors that are highly specialized and requires an extensive global database that is outside of the purview of a QP.

The QP considers it reasonable to rely upon the information provided by Mr. Osachoff as the review considers up-to-date, in-depth insight and analysis into all facets of the metals industry, including production supply and costs as well as consumption demand, and metal price forecasts.

The QP has fully relied upon, and disclaims responsibility for, information supplied by experts retained by Capstone for the iron fines pricing and markets in Asia, as follows:

- Dazmin, 2023, Product Analysis for Capstone Magnetite, November 2023, David John Trotter: Global Iron Ore and Commodity Marketing Consultant.

This information is used in Sections 1.18, 1.22, 19 and 22 of the Report and in support of the Mineral Reserve statement in Section 15.

The QP considers it reasonable to rely upon the information provided by Mr. Trotter for iron ore concentrate. Mr. Trotter is a global iron ore and pellet feed consultant, with significant experience in sales and marketing for major iron-producing companies, including BHP Billiton, Fortescue Metals Group and Anglo American. Mr. Trotter has been involved with sales and technology for marketing, trading and technical development in iron ore, metals and concentrate organizations across European, Chinese, Indian, American and globally developing markets. The QP was able to review Mr. Trotter’s report.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Introduction

The portion of the Santo Domingo's mining property that hosts the mine and plant site areas is located approximately 5 km southeast of the town of Diego de Almagro, 35 km northeast of Capstone's Mantoverde mine, in the province of Chañaral in the Atacama Region of northern Chile.

The centre of the Santo Domingo property is located approximately at geographic coordinates 26° 27' 46.69" S latitude and 69° 59' 6.74" W longitude and Universal Transverse Mercator (UTM) 397,815m E and 7,073,620m N (datum: WGS 84, Zone 19S).

The proposed port, Puerto Santo Domingo, will be located in the Punta Roca Blanca area, in the Atacama Region.

4.2 Property and Title in Chile

The information in this Section is based on data in the public domain and Chilean law (Chilean Civil Code, Chilean Mining Code, Chilean Tax Law, Fraser Institute, 2022). This information has not been independently verified by the QP.

4.2.1 Regulations

The mining industry is regulated by the following laws:

- Constitution of the Republic of Chile;
- Constitutional Organic Law of Mining;
- Code and Regulations governing Mining;
- Code and Regulations governing Water Rights;
- Laws and Regulations governing Environmental Protection as related to mining.

Chile's mining policy is based on legal provisions that were enacted as part of the 1980 constitution. These were established to stimulate the development of mining and to guarantee the property rights of both local and foreign investors. According to the law, the state owns all mineral resources, but exploration and exploitation of these resources by private parties is permitted through mining concessions, which are granted by the courts.

The concessions grant both rights and obligations, as defined by the Constitutional Organic Law on Mining Concessions (JGRCh, 1982) and the Mining Code (JGRCh, 1983). Some of the steps involved in the constitution of the mining concession are published weekly in Chile's official mining bulletin for the relevant region as are court processes due to conflicting claims.

4.2.2 Mineral Tenure

The concessions have both rights and obligations as defined by a Constitutional Organic Law (enacted in 1982). Concessions can be mortgaged or transferred, and the holder has full ownership rights and is entitled to obtain the rights for exploration and exploitation. A concession is obtained by a claim filing and includes all minerals that may exist within its area. A concession holder has the right to defend ownership of the concession against the state and third parties.

Exploitation concessions are valid indefinitely subject to the payment of annual fees. Once an exploitation concession has been granted, the owner can remove materials for sale.

On December 19, 2023, the Chilean legislature approved a legislative measure that revised Law No. 21,420, along with additional statutes pertaining to the mining sector, which took effect starting January 1, 2024. This measure, referred to herein as "the Amendment," updates several key regulations, including the Mining Code, the Organic Constitutional Law on Mining Concessions, and the legislation that established the National Geology and Mining Service.

The Amendment specifies the timeframe and method for exploration or exploitation mining concession holders to submit the essential geological data to the National Geology and Mining Service.

According to the Amendment, exploration mining concessions are initially valid for four years, with an option for an extension of up to four additional years, provided certain criteria are met. The Amendment prohibits concession holders from applying for a new concession in the same area from the date of the original claim until one year after its expiration. Failure to comply results in loss of rights in the area.

The Amendment changes the tax structure for mining concessions. It increases three times the tax for exploration concessions and introduces a progressive tax structure for exploitation concessions based on the duration of the concession. It outlines conditions under which reduced fees may be applied, including for small-scale mining operations and those that comply with environmental regulations.

There is a mining tax that provides protection of rights; it is calculated as a percentage of the Unidad Tributaria Mensual (UTM or monthly tax unit) and applies to each hectare of land included in the mining exploration and/or mining exploitation concessions. This tax is paid annually in a single payment before 31 March of each year.

For mining exploitation concessions, the tax rate for year 2024 is 10% of a UTM per hectare; for mining exploration concessions the tax rate is currently 3/50 of a UTM per hectare. The value of the UTM is adjusted monthly according to the consumer price index (IPC) in Chile.

At each of the stages of the claim acquisition process, several steps are required, a full description of the process is documented in Chile's mining code.

Some of the steps involved in establishing the claim are published in Chile's official mining bulletin for the appropriate region (published weekly). Capstone has a mining claim specialist to review the weekly mining bulletins and ensure that their land position is kept secure.

In addition to the mining law, the Organic Constitutional Law on Mining Concessions (1982) and the Mining Code of 1983 are the two key mechanisms governing mining activities in Chile.

4.2.3 Surface Rights

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

4.2.4 Rights of Way

The Mining Code also grants the holder of the mining concession general rights to establish a right of way (RoW), subject to payment of reasonable compensation to the owner of the surface land. A RoW is granted through a private agreement or legal decision which indemnifies the owner of the surface land. A RoW must be established for a particular purpose and will expire after cessation of activities for which the right of way was obtained. The owners of mining easements are also obliged to allow owners of other mining properties the benefit of the RoW, as long as this does not affect their own exploitation activities.

4.2.5 Water Rights

Article 110 of the Chilean Mining Code establishes that the owner of record of a mining concession is entitled to use waters found as a result of the mining work within the limits of the concession, as required for exploratory work, exploitation and processing, according to the type of concession the owner might engage in. These rights are inseparable from the mining concession (and it is called miner’s water).

Water is considered part of the public domain and is considered to be independent of the land ownership. Individuals can obtain the rights to use public water in accordance with the Water Code. In accordance with the Code (updated in 1981), water rights are expressed in litres per second (L/s) and usage rights are granted on the basis of total water reserves.

4.2.6 Environmental Regulations

All projects listed in article 10 of Law 19.300 (Environmental General Baseline Law) and in article 3 of Supreme Decree No. 40/2012 (Regulation of the Environmental Impact Assessment System, RSEIA) must enter the SEIA (Environmental Impact Assessment System) for environmental approval prior to their execution; projects such us dams, thermo-electric and hydroelectric plants, nuclear power plants, mining, oil and gas, roads and highways, ports, development of real estate in congested areas, water pipelines, manufacturing plants, forestry projects, sanitary projects, production, storage and recycling of toxic and flammable and hazardous substances. These types of projects must be approved by the Environmental Evaluation Service (Servicio de Evaluación Ambiental, SEA) and regional sectoral authorities through an Environmental Impact Declaration (DIA) or an Environmental Impact Study (EIA).

Supreme Decree No. 40/2012 was approved in 2012 but has been updated on several occasions to include additional requirements and regulations. In general terms, the regulation indicates the environmental assessment procedure and the information that must be submitted when entering an Environmental Impact Study (EIA) or an Environmental Impact Declaration (DIA). Law 19.300 and RSEIA also regulate public participation in EIAs and the Indigenous consultation process (if required), complying with ILO Agreement 169 in force in Chile since 2009. The last RSEIA update was on 1 February 2024, to include additional requirements associated with the Climate Change Framework Law and the Escazú Agreement.

4.2.7 Land Use

Chile's zoning and urban planning are governed by the General Law of Urban Planning and Construction (Ley General de Urbanismo y Construcción). This law contains several administrative provisions that are applicable to different geographical and hierarchical levels and sets specific standards for both urban and inter-urban areas.

In addition to complying with the Environmental Law (Ley Ambiental) and other legal environmental requirements, projects must also comply with urban legislation governing the different types of land use. Land use regulations are considered part of the Chilean environmental legal framework.

Land use regulatory requirements are diverse and operate at different levels, the main instruments are the inter-community regulatory plans (Planes Reguladores Intercomunales, PRI) and the community regulatory plans (Planes Reguladores Comunales, PRC). The PRIs regulate territories of more than one municipality, including urban and rural territory.

4.2.8 Closure Considerations

Law 20.551, Law of Mine Closure, enacted in October 2011, took effect in November 2012 and imposes on the mining industry the obligation to execute closure of its operations, incorporating closure as part of the life cycle of a mining project. Additionally Decree 41 covers the Regulations of the Law on Closure of Mining Works and Facilities.

To comply with these regulations, the owner of the project must submit a Closure Plan to Sernageomin, prior to starting construction of a mining project, with an approval procedure that depends on the mine capacity. There are two procedures, one for projects where the mine capacity is greater than 10,000 tonnes per month, and a simplified procedure where the mine capacity is equal to or less than 10,000 tonnes per month (including exploration projects).

The difference between these procedures is the information required to be submitted for evaluation of the Closure Plan. Closure Plans for larger operations must provide more detailed information and must also provide a monetary guarantee, through guaranteed notes, insurance or other, representative of the cost of the Closure Plan. The period granted will be accord to the useful life of the project, and the instrument that will be used. To comply with the Closure Plan, the execution of any closure activities and an update of the Closure Plan must be audited every 5 years, during mining operation, as a complementary instrument of control by Sernageomin. For smaller mining projects or exploration projects that are subject to the simplified procedure, no financial guarantee is required, and no audit of the Closure Plan is needed.

The following are the requirements for the guarantee:

- The amount of the guarantee must cover the total value of the cost for the Closure Plan including post-closure and is determined by an estimate of the current costs of the plan. The guarantee is periodically updated.
- The guarantee must be paid in full within the first two-thirds of the estimated life of the project if less than 20 years, or within a period of 15 years if the estimated life of the project is more than 20 years.
- The payment of the guarantee must begin within the first year of the start of operations and the value must be equal to 20% of the total closure cost. From the second year on, the payment must be proportional to the period which remains for the complete amount. The instruments of guarantee must be liquid and easy to execute.
- The financial guarantee can be gradually released as the Closure Plan is executed. Once the closure is complete and a certificate of final closure is issued by Sernageomin all guarantees will be released.
- Mining companies that are obliged to provide a guarantee have a period of 2 years to estimate the cost of the Closure Plan. The Closure Plan must be approved under the regulation of Mining Safety Regulations and Environmental Qualification Resolution (RCA). After this period the company must submit the cost of executing the Closure Plan as well as the guarantee to Sernageomin. Sernageomin will then confirm that the company is in compliance.

4.2.9 Foreign Investment

In Chile, foreign investors may own 100% of a company based in Chile with no limit of duration for property rights. Within the limits of the Chilean law, investors can undertake any type of economic activity.

Potential foreign investors must comply with the administrative system described in Section XIV of the Chilean Central Bank's Compendium of Foreign Exchange Regulations in order to register the entry of foreign capital into Chile. The entry of foreign capital must be registered by commercial banks which, in turn, must coordinate with the Central Bank of Chile. A minimum of $10,000 can be brought in through this mechanism in the form of currency or loans. This mechanism does not require a contract of any type. Capital entering Chile under Section XIV is not subject to any tax benefit and foreign investors using this regime are subject to the general taxation established by the Chilean Income Tax Law and the VAT Law.

On February 7, 2014, Capstone signed a foreign investment contract with the state of Chile, under the provisions of the DL 600 Foreign Investment Framework then in effect. According to the DL 600 Contract, Capstone, acting as the foreign investor entity, is entitled to make an investment in the Chilean company Minera Santo Domingo (Sociedad Contractual Minera or SCM) (MSD) the developer of the Santo Domingo property. According to the DL 600 Contract provisions, this investment can be carried out in Chile via capital contribution and debt for an authorized amount of up to $2,100 million, but in any case, not less than US$50M within 8 years from the date of the signature of the contract (this term expired on February 7, 2022). As Capstone had already contributed and effectively invested the minimum foreign investment amount required of US$50M by February 7, 2022, the foreign investment protections and tax stability rights provided in the DL600 Agreement are all available, in good standing and in effect.

Law 20.848 states that, for a period of 4 years from 1 January 2016, a foreign investor may request authorization to sign a tax invariability contract according to the terms, time frames and conditions established in Articles 7 and 11 ter of Decree Law No. 600 (DL 600 was replaced by Law 20.780 from 1 January 2016).

Article 7 of Decree Law 600 establishes a tax invariability system that grants, for a period of 10 years, a total effective tax rate of 44.45% for investments of no less than $5 million for any investment purposes in Chile.

Article 11 of Decree Law 600 establishes a tax invariability system that grants, for a period of 15 years, specific rights for investments of no less than US$50M for mining projects.

Under the provisions of the DL 600 Contract, Capstone has the following rights:

- The right to transfer its capital and net profits abroad, in accordance with the provisions of Articles 4, 5 and 6 of DL 600 and the provisions of the Income Tax Law.
- Access to the foreign exchange market, both to liquidate the foreign currency constituting the contribution and for capital and profit remittances, at the most favourable exchange rate, according to Articles 2° letter a) and 4 of DL 600.
- Tax invariability in accordance with the provisions of Article 11 ter of DL 600.
- Non-discrimination, in accordance with Articles 9 and 10 of DL 600.
- Exemption from any levy, tax or encumbrance on the net proceeds obtained by the alienation of the shares or rights representative of the foreign investment, or by the sale or liquidation of the receiving company, up to the amount actually invested under this contract, in accordance with the provisions of Article 6 of DL 600, without prejudice to the provisions of the Income Tax Law.

4.2.10 Fraser Institute Survey

Capstone has used the 2023 Fraser Institute Annual Survey of Mining Companies report (the Fraser Institute survey) as a credible source for the assessment of the overall political risk facing an exploration or mining project in Chile. Each year, the Fraser Institute sends a questionnaire to selected mining and exploration companies globally. The Fraser Institute survey is an attempt to assess how mineral endowments and public policy factors such as taxation and regulatory uncertainty affect exploration investment.

Chile has a Policy Perception Index rank of 49 out of the 86 jurisdictions in the Fraser Institute survey. Chile's Investment Attractiveness Index rating is 38 out of the 86 jurisdictions and it is ranked 26 of 58 on the Best Practices Mineral Potential Index.

4.3 Property Ownership

MSD is a mining company legally organized under the laws of the Republic of Chile. MSD holds various mining concessions in the area of Diego de Almagro and Caldera. Collectively, these properties constitute the Santo Domingo property.

The capital of MSD is owned by Capstone as follows:

- Capstone owns 100% of the issued and outstanding common shares of 0908113 BC Ltd.
- 0908113 BC Ltd owns 100% of the issued and outstanding common shares of Far West Mining Ltd.
- Far West Mining Ltd owns 100% of the issued and outstanding common shares of MSD.
- MSD owns 100% of the Santo Domingo property.

4.4 Mineral Tenure

Capstone holds two groups of concessions, totaling 119 concessions, which cover a total of 29,221 ha and include the proposed mine site, plant area and auxiliary facilities including port facilities. The tenure includes 99 exploitation concessions and 20 exploration concessions. Concessions are held in the name of MSD.

In the mine-plant area Capstone holds 96 exploitation concessions of which 82 cover all projected facilities (the group with these 82 properties is termed the "first ownership layer"). Additionally, Capstone holds 14 exploitation concessions overlapping the first ownership layer. In the port area Capstone has three exploitation concessions.

The total concession area is divided as follows:

- 27,597 ha of exploitation and exploration concessions that encompass the area where the mine, plant, construction and operations camp and ancillary facilities are planned.
- 1,624 ha of exploitation and exploration concessions that encompass the port area.

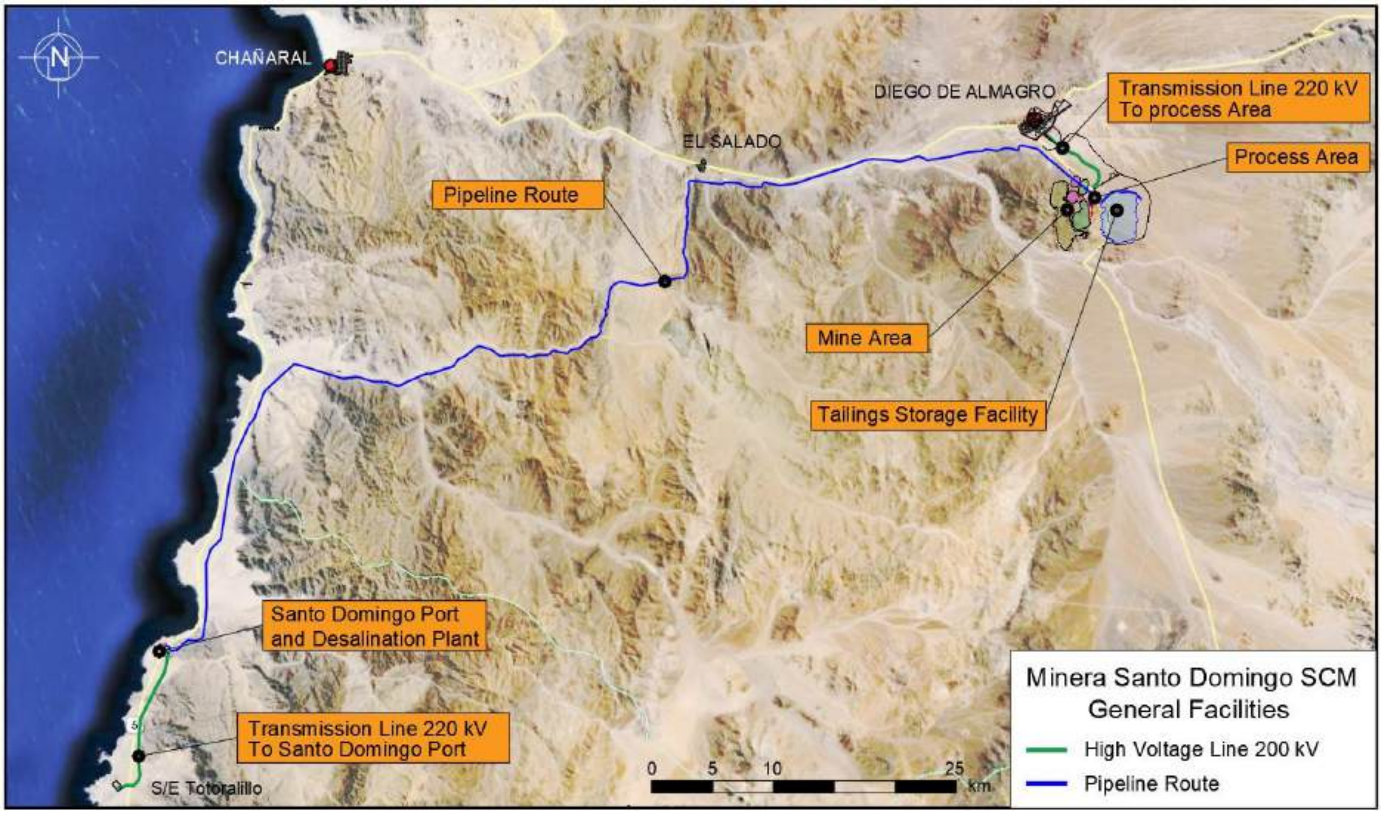
Concessions are surveyed as part of the grant process and are protected under Chilean law by payment of the annual mining license fees. Concessions are in force under Chilean law by payment of the annual mining license fees.

As part of the grant process, the concessions were surveyed by a government-licensed surveyor.

All concession fees are in good standing and were fully paid up to 28 February 2025 and will continue to be paid on a regular basis as due, using a formal status tracking system.

A simplified location plan for the contemplated infrastructure is included as Figure 4-1. A summary of the mineral tenure is provided in Table 4-1 for the exploitation concessions. Figure 4-2 shows the layout of the concessions in Table 4-1. Figure 4-3 shows the first ownership layer of properties in the mine-plant area and its NSR condition. Table 4-2 presents the exploration concessions. Figure 4-4 shows the locations of the Mineral Tenure and Surface Rights for the proposed facilities.

Figure 4-1: Proposed Facility Locations



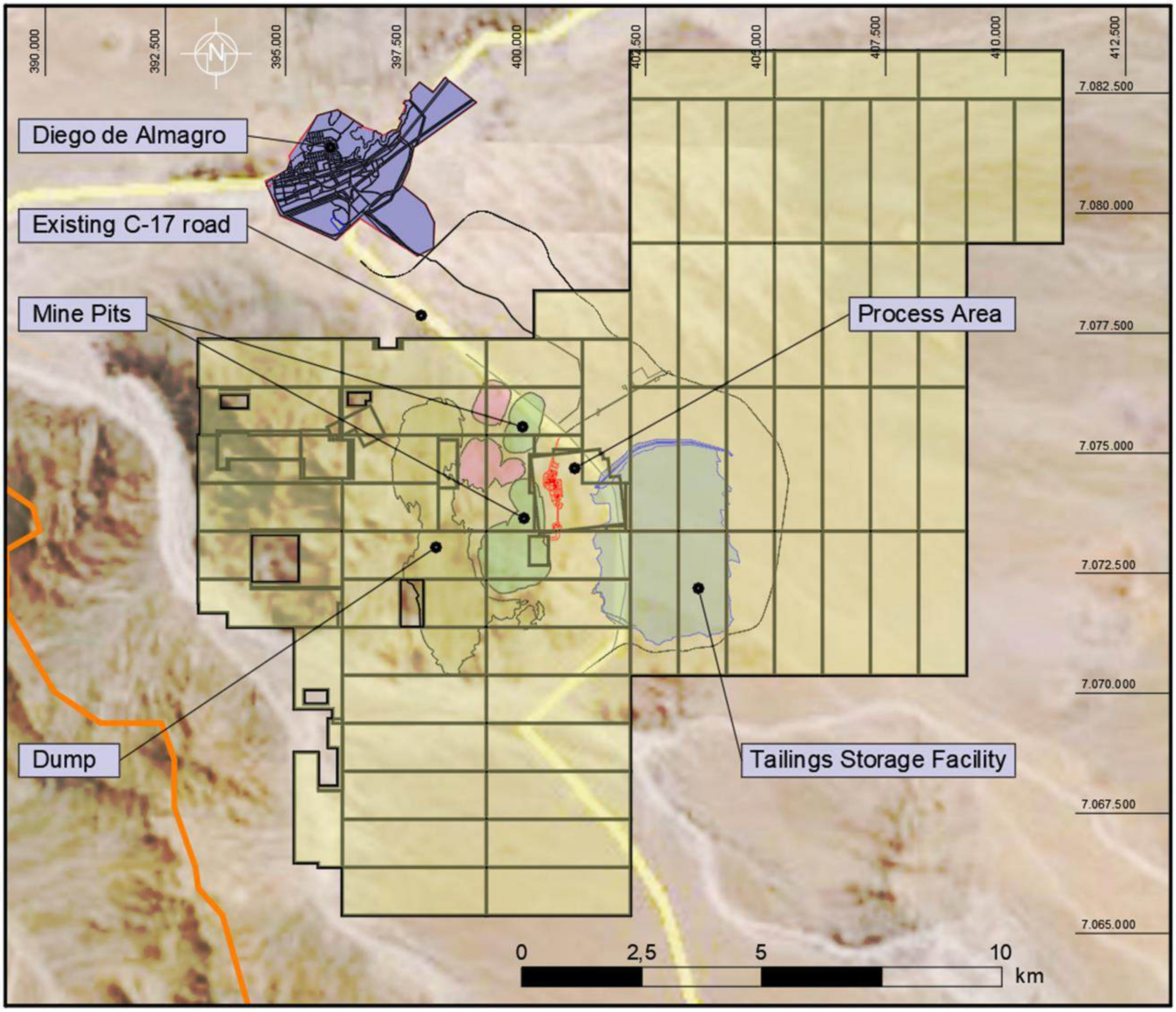
Source: Capstone, 2024. Backdrop is based on Google Earth image.

Table 4-1: Exploitation Concessions

Name	Area (ha)	Name	Area (ha)	Name	Area (ha)
SORPRESA 1/10	34	MANTA 514 1/60	300	MANTA 577 1/60	300
ESPERANZA UNO	4	MANTA 515 1/60	300	MANTA 328 B 1/2	2
ILUSION 1/2	10	MANTA 516 1/60	300	MANTA 328 C 1/2	2
MANTA 264 1/54	266	MANTA 518 1/60	300	MANTA 505 1/60	300
MANTA 270 1/28	134	MANTA 519 1/60	300	MANTA 509 1/60	300
MANTA 273 1/60	300	MANTA 520 1/60	300	MANTA 501 1/60	300
MANTA 276 1/22	110	MANTA 521 1/60	300	MANTA 502 1/60	300
MANTA 282 1/56	239	MANTA 522 1/60	300	MANTA 503 1/60	300
MANTA 330 1/58	286	MANTA 523 1/60	300	MANTA 526 1/60	300
MANTA 332A 1/38	184	MANTA 524 1/60	300	MANTA 528 1/60	300
MANTA 332B 1/2	8	MANTA 525 1/60	300	MANTA 529 1/60	300
MANTA 334 1/40	200	MANTA 517 1/60	300	MANTA 530 1/60	300
MANTA 327 1/60	300	MANTA 527 1/60	300	MANTA 578 1/39	195

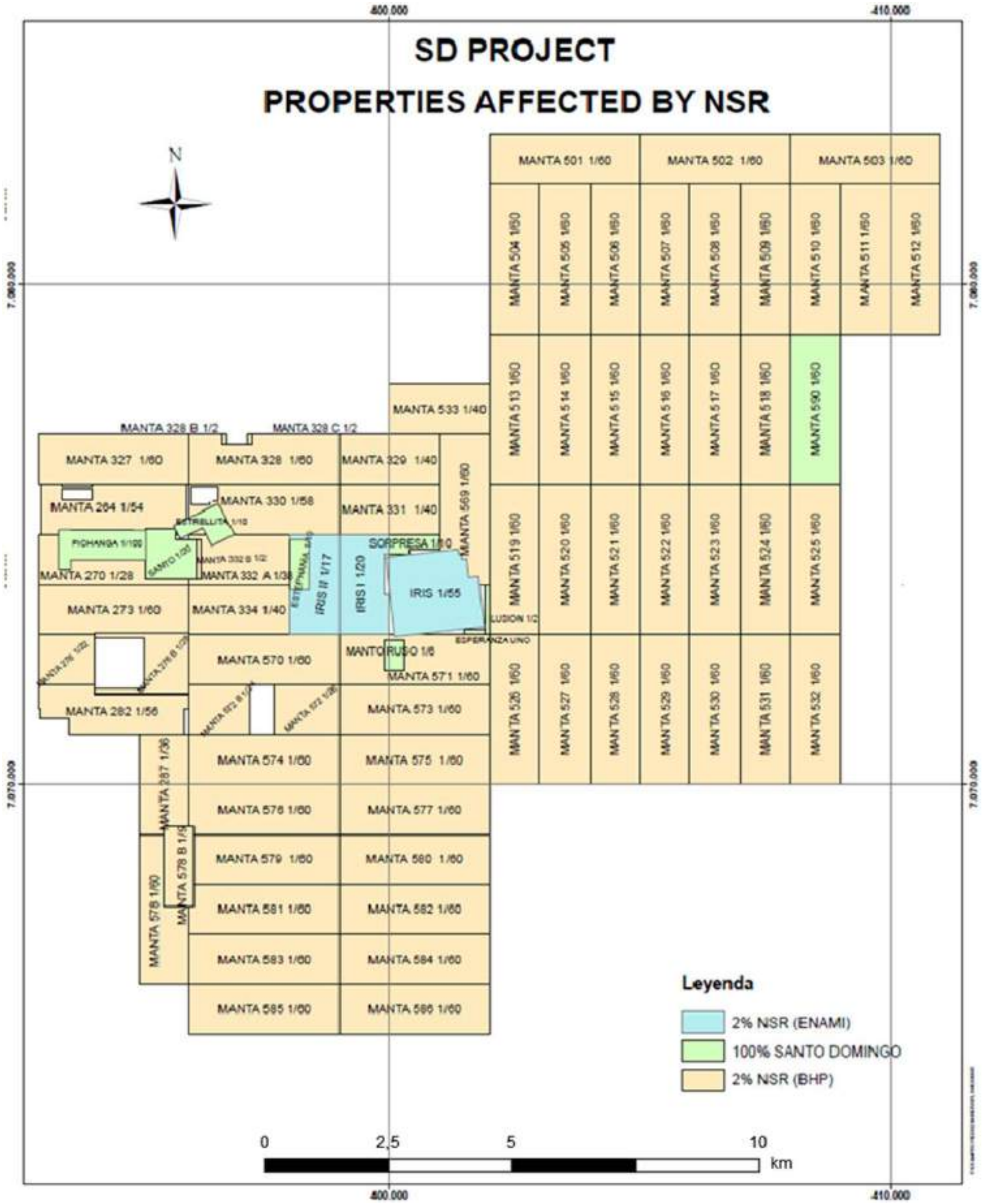
Name	Area (ha)	Name	Area (ha)	Name	Area (ha)
MANTA 328 1/60	290	MANTA 531 1/60	300	MANTA 583 1/60	300
MANTA 329 1/40	200	MANTA 532 1/60	300	MANTA 590 1/60	300
MANTA 331 1/40	200	MANTA 579 1/60	300	MANTA 578 B 1/9	31
IRIS 1/55	273	MANTA 580 1/60	300	DOMINGO 03 1/60	300
IRIS PRIMERO 1/20	192	MANTA 581 1/60	300	DOMINGO 04 1/60	272
IRIS SEGUNDO 1/17	160	MANTA 582 1/60	300	DOMINGO 07 1/60	300
ESTRELLITA 1/10	50	MANTA 584 1/60	300	DOMINGO 08 1/60	300
PICHANGA 1/100	110	MANTA 585 1/60	300	DOMINGO 09 1/60	300
SANTO 1/20	92	MANTA 586 1/60	300	DOMINGO 11 1/60	300
MANTO RUSO 1/6	24	MANTA 276 B 1/28	128	DOMINGO 12 1/60	300
ESTEPHANIA 1/10	40	MANTA 287 1/36	161	ALTO 2 1/60	300
MANTA 572 B 1/24	120	MANTA 533 1/40	200	ALTO 4 1/60	300
MANTA 504 1/60	300	MANTA 569 1/60	300	ALTO 6 1/60	300
MANTA 506 1/60	300	MANTA 570 1/60	300	ALTO 13 1/40	200
MANTA 507 1/60	300	MANTA 571 1/60	300	ALTO 14 1/50	250
MANTA 508 1/60	300	MANTA 572 1/26	130	SALADO 20 160	300
MANTA 510 1/60	300	MANTA 573 1/60	300	SALADO 27 160	300
MANTA 511 1/60	300	MANTA 574 1/60	300	OESTE 2B, 1/34	158
MANTA 512 1/60	300	MANTA 575 1/60	300	OESTE 3B, 1/26	110
MANTA 513 1/60	300	MANTA 576 1/60	300	OESTE 4B, 1/16	56

Figure 4-2: Exploitation Concessions in Relation to Proposed Infrastructure Locations



Source: Capstone, 2024. Backdrop is based on Google Earth image. Coordinate System: UTM, PSAD 56.

Figure 4-3: Location Plan, Exploitation Concessions (First Ownership Layer)

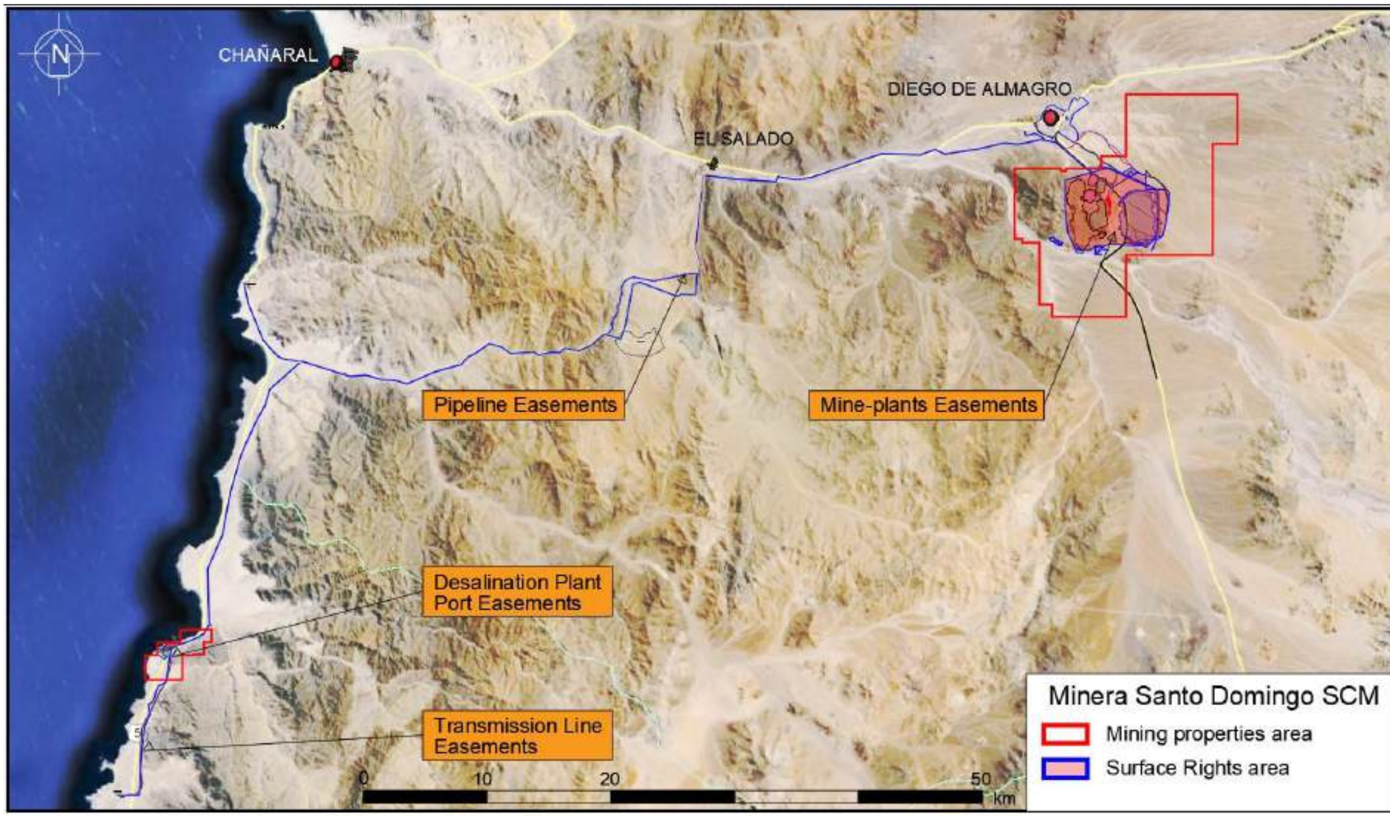


Source: Capstone, 2024. Coordinates System: PSAD 56.

Table 4-2: Exploration Concessions

Name	Area (ha)
OESTE 1C	100
OESTE 2C	300
OESTE 3C	300
OESTE 4C	300
BLANCA B18	200
BLANCA B19	100
BLOCK C1	300
BLOCK C2	300
BLOCK C3	300
BLOCK C4	300
BLOCK C5	300
BLOCK C6	300
BLOCK C7	300
BLOCK C8	300
BLOCK C9	300
BLOCK C10	300
BLOCK C11	300
BLOCK C12	300
BLOCK C13	300
DOMINGO 16 B	300

Figure 4-4: Location Plan, Mineral Tenure and Surface Rights



Source: Capstone, 2024. Backdrop is based on Google Earth image.

4.5 Surface Rights

Based on the current state of development, the existing legislation in Chile and the legal assurances necessary to safeguard the areas impacted by the proposed mine and plant development, Capstone developed a legal strategy for acquisition of surface lands sufficient to support the operation. Capstone's legal strategy includes the necessary surface rights to cover the mine, plant, camps, tailings storage facilities, pipelines, port and transmission lines.

Surface land near the community of Diego de Almagro where the mine and plant will be located is part of a larger lot owned by the State, managed and represented by the Ministerio de Bienes Nacionales. As well, land in the districts of Caldera and Chañaral where planned facilities will be located is also owned by the State and represented by the Ministerio de Bienes Nacionales.

Capstone currently possesses three registered provisional surface rights (listed in Table 4-3) and 22 definitive surface rights (listed in Table 4-4). These cover surface rights for facilities and infrastructure. All these surface rights are contracts between Capstone and the Chilean Treasury.

To date, Capstone has been granted 5,386.57 ha as definitive easements, and 107.56 ha as provisional easements. This total area covers:

- Plant and infrastructure (process plant, tailings disposal, open pit, and waste disposal);
- Pipelines;
- Temporary construction and permanent operations camp;
- Power lines;
- Port area.

Table 4-3: Provisional Surface Rights Granted to Capstone

Nº	Court	ROL (ID)	Purpose	Area (ha)	Provisional Easement Register
1	4°	1192-2021	Power line to SD	23.31	Folio 35, Nº 5, Mortgages, Liens and Prohibitions Registry, 2022.
2	4°	1313-2023	C-17 By-pass	74.76	Folio 172 vta, Nº 24, Mortgages, Liens and Prohibitions Registry, 2023
3	4°	1273-2023	Mine area	9.49	Folio 136 vta, Nº 17, Mortgages, Liens and Prohibitions Registry, 2023

Table 4-4: Definitive Surface Rights Granted to Capstone

Nº	Court	ROL (ID)	Purpose	Area (ha)	Inscription Data
1	1º	10-2012	Mine (pit. plant)	390.27	Folio 109 Nº 22. Mortgages. Liens and Prohibitions Registry. 2019
2	1º	2674-2012	Pipeline	113.73	Folio 126 Nº 23. Mortgages. Liens and Prohibitions Registry. 2019
3	1º	2675-2012	Pipeline	193.16	Folio 144 Nº 24. Mortgages. Liens and Prohibitions Registry. 2019
4	1º	620-2013	Tailings storage facility	372.20	Folio 162 vta. Nº 25. Mortgages. Liens and Prohibitions Registry. 2019
5	1º	1923-2013	Port	75.93	Folio 191 Nº 115. Mortgages. Liens and Prohibitions Registry. 2019 (Caldera)
6	1º	211-2012	Mine area	250.00	Folio 316 vta. Nº 41. Mortgages. Liens and Prohibitions Registry. 2019
7	1º	212-2012	Mine area	341.88	Folio 81 vta. Nº 13. Mortgages. Liens and Prohibitions Registry. 2021
8	1º	213-2012	Mine area	228.88	Folio 98 vta. Nº 14. Mortgages. Liens and Prohibitions Registry. 2021
9	1º	214-2012	Waste	187.75	Folio 111 vta. Nº 15. Mortgages. Liens and Prohibitions Registry. 2021
10	1º	215-2012	Mine	127.50	Folio 130, Nº 16. Mortgages. Liens and Prohibitions Registry. 2021
11	1º	2677-2012	Pipeline	65.76	Folio 333. Nº 42. Mortgages. Liens and Prohibitions Registry. 2019
12	1º	2257-2016	Mine	764.79	Folio 147 Nº 17. Mortgages. Liens and Prohibitions Registry. 2021
13	2º	768-2015	LTE. bypass	116.97	Folio 24 vta. Nº 12. Mortgages. Liens and Prohibitions Registry. 2020

Nº	Court	ROL (ID)	Purpose	Area (ha)	Inscription Data
14	2º	2303-2016	Mine	307.99	Folio 246 Nº 23. Mortgages. Liens and Prohibitions Registry. 2021
15	3º	1593-2016	Pipeline	56.29	Folio 265 Nº 158 Mortgages. Liens and Prohibitions Registry. 2019
16	4º	2282-2016	Mine	263.68	Folio 303 vta. Nº 40. Mortgages. Liens and Prohibitions Registry. 2019
17	1°	415-2013	Pipeline	166.31	Folio 19 Nº 17. Mortgages. Liens and Prohibitions Registry. 2020
18	1º	3260-2014	Borrow areas	51.84	Folio 268 Nº 27. Mortgages. Liens and Prohibitions Registry. 2021
19	2º	1861-2014	Pipeline	8.39	Folio 294 Nº 28. Mortgages. Liens and Prohibitions Registry. 2021
20	1°	3130-2013	Camp	3.33	Folio 91 vta. Nº 15. Mortgages. Liens and Prohibitions Registry. 2022
21	1º	915-2014	C-17 by-pass	132.77	Folio 42 vta. Nº 06. Mortgages. Liens and Prohibitions Registry. 2022
22	1º	3195-2012	Tailings storage facility	1,167.15	Folio 177 Nº 20. Mortgages. Liens and Prohibitions Registry. 2021

4.6 Water Rights

MSD will not need to make an application for water rights. The water for operations will consist solely of desalinated sea water. A maritime concession has been granted which will allow the extraction of sea water.

Water for construction will be obtained from an authorized third-party provider.

4.7 Royalties and Encumbrances

Government royalties are levied in the form of a mining tax.

A 2% net smelter return (NSR) royalty is payable to Enami for minerals mined from certain concessions subject to the royalty agreement in force between Enami and Capstone.

A 2% NSR royalty is payable to Ecora for minerals mined from certain other concessions subject to the royalty agreement in force between Ecora and Capstone. This Royalty right originally belonged to BHP, then was transferred to South32 and finally was transferred to Ecora.

The majority of the proposed open pits are located on concessions subject to one or the other of these two royalty agreements, with exception of the exploitation concession Manto Ruso 1/6, owned exclusively by Minera Santo Domingo. Figure 4-3 shows which NSR royalty applies to each concession.

On March 25, 2021, Capstone entered into a precious metals purchase agreement with Wheaton Precious Metals (WPM) and received an early deposit of $30 million on April 21, 2021. Additional deposits of $260M are expected during the Santo Domingo construction period, subject to sufficient financing having been obtained to cover total expected capital expenditures and other customary conditions, for a total deposit of $290M. WPM will receive 100%

of Santo Domingo's gold production until 285,000 ounces are delivered, then dropping to 67% of the gold production. As gold is delivered, Capstone receives payments equal to 18% of the spot gold price at the time of delivery for each ounce delivered to WPM, until the $290 million deposit is reduced to zero. Payments will then increase to 22% of the spot gold price upon delivery.

4.8 Permitting Considerations

The permitting status, including permits necessary to advance QP recommendations are discussed in Section 20.

4.9 Environmental Considerations

The environmental and Closure Plan status is discussed in Section 20, covering the current environmental liabilities and those associated with the life of mine (LOM) plan. Based on the information made available and reviewed, there are currently no material liabilities identified on the property.

4.10 Social Licence Considerations

The social licence status is discussed in Section 20.

4.11 Comments on Section 4

The QP was provided with legal opinion and information from Capstone staff and experts retained by Capstone that supports:

- Capstone holds 100% of the Santo Domingo property.
- Capstone is the operator.
- The mining tenure held is valid and is sufficient to support declaration of Mineral Resources and Mineral Reserves.
- Capstone has three provisional surface rights and 22 definitive surface rights. Together these easements cover 100% of the area needed for construction of facilities and infrastructure.
- Royalties in the form of the Chilean mining tax will be payable.
- Royalties to Enami and Ecora are also payable in the form of NSRs.
- Payments to WPM will occur during production for the entire LOM.
- No water rights are currently envisaged as the water will be sourced from the ocean, desalinated, and piped to the site for use.
- Capstone is not aware of any issues that may affect access, title, or the right or ability to perform work on the Property.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

Access to the planned mine and plant site areas is via the paved Route 5 North (Pan-American Highway) and a network of generally well-maintained gravel roads. The Santo Domingo deposits are roughly five hours travel time by road south of Antofagasta and two hours by road north of Copiapó.

Access to the proposed project site is via Route 5 North, heading east from the town of Chañaral for 12 km to the El Salvador turn-off and then 50 km east to the town of Diego de Almagro. C-17 southbound, a paved highway, connects Diego de Almagro with Copiapó. At 3.3 km southeast from Diego de Almagro along this highway, a secondary gravel road (the Santo Domingo access road) leads south into the property.

5.2 Climate

The proposed mine site is located in an area that is one of the driest places in the country and in the world, with high solar radiation, evaporation rates and salt concentration in the soil. Rainfall is occasional and irregular and, in some years, only during the winter period.

Because of this there are only temporary surface run-offs, except for the El Salado River which is the only permanent water course in the area of influence of the Santo Domingo property. The El Salado River has a predominantly pluvial regime and is located 7 km downstream of the planned mine and plant area. The river is highly altered and of low flow; it was used in the past to transport tailings from the El Salvador mine to the coast.

Meteorological data was collected at three different areas using monitoring stations and define two climate zones:

- Normal desert: This extends from the south of the Copiapó Valley to the southern boundary of the Region and is characterized by low annual rainfall, increasing towards the south. The average annual temperature is 15°C. The main feature in the valleys of the Region is the frost-free condition for 11 months (from August to June). Minimum temperatures occur in July and reach 5°C; maximum temperatures occur in January and reach 28°C. There is strong seasonal precipitation in the area concentrated in the period from May to August, when more than 80% of the total annual precipitation falls.
- Coastal desert: This is present in all the coastal sectors of the Region and to the north close to Chañaral. The relief does not present barriers to the marine influence; the amount of cloud depends on the presence of the Pacific Anticyclone, a high-pressure system that generates dry air masses. This type of climate is characterized by abundant and dense cloud cover that appears during the night and is dissipated during the morning; it is sometimes accompanied by heavy fog and drizzle. The ocean influence produces a moderate thermal regime with a small temperature range, both daily and annually. Precipitation is mostly associated with fronts and increases from north to south, occurring almost exclusively in winter. Chañaral receives an average of 12 mm per year.

- It is expected that it will be possible to conduct mining, processing, desalination, and port activities on a year-round basis.

5.3 Local Resources and Infrastructure

Diego de Almagro, population of 7,223 people (2017 census) located adjacent to the mine and plant area, has hotel accommodation, food, fuel and minor services available. Chañaral is a deep-sea port less than one hour's drive to the west of the property with a population of 11,073 people (2017 census). The most important logistics centre in the Region is Copiapó, two hours south of the Santo Domingo property. Copiapó has a population of approximately 175,000 people, an airport with daily scheduled flights to Santiago and companies that offer abundant services for mining and exploration. There is a private airport located at El Salvador at 44 km from the project site.

The Atacama Region has well established infrastructure (energy, water, transportation, and labour) to serve the mining industry. However, there is currently no major infrastructure on the Santo Domingo property. Some initial project infrastructure has been constructed, such as gravel roads for access to the concessions and drill sites, some earthworks platforms for temporary facilities, a 152 mm diameter industrial water delivery pipeline to the process plant site from the water treatment plant near Diego de Almagro, a high voltage connection point in the San Lorenzo Substation and part of the C-17 bypass road. The nearby town of Diego de Almagro is connected to the regional power grid and to the rail line. The rail line needs repair to the south and west of Diego de Almagro to be operational.

Proposed infrastructure is described in Section 18 of this Report.

5.4 Physiography

Elevations in the deposit area range from approximately 900–1,300 masl. Hills of gentle to moderate relief have been cut by deep gullies and are flanked by gravel-filled valleys and alluvial fans. The vegetation is very sparse. In the valleys, plant life consists of small widely spaced bushes a few centimetres high. Hillsides and peaks are generally devoid of vegetation.

The coastline in the port area is aligned along a west–southwest–east–northeast direction. The soil type is rocky soil and a lens of sand and gravel on top of the bedrock is easily recognizable.

5.5 Seismicity

Seismic maps of South America indicate that the Property has high seismicity, and the site is considered to be within Zone 3 and Zone 2 according to the new Chilean standard NCh 2369.

A seismic hazard assessment was performed by Rodolfo Saragoni (as member of S&S), a recognized Chilean seismologist, as part of the 2014 feasibility study and his recommendations are included in the current designs. An updated report was issued by Rodolfo Saragoni in 2015 for the Tailings Storage Facility area and from it the following earthquake ground motions have been adopted for the current design of the facility (see Table 5-1).

Table 5-1: Earthquake Ground Motions

Seismic Source	Magnitude	Soil Type	Peak Ground Acceleration, PGA (g)
Interplate	8.8	Hard Rock	0.36
Intraplate	8.0	Hard Soil	1.02
Intraplate	8.0	Hard Rock	0.51

An update of the seismic hazard assessment is recommended for the next stage of design. However, while this will likely change some of the ground motions for the design earthquakes, the TSF dams are not expected to be significantly affected because they will be constructed out of compacted dense rockfill using the downstream method of raising to produce inherently stable structures and the foundations will also be dense and stable.

5.6 Comments on Accessibility, Climate, Local Resources, Infrastructure and Physiography

In the QP's opinion:

- There is sufficient suitable land available within the exploitation concessions for the planned tailings disposal, mine waste disposal and mining-related infrastructure such as the open pit, process plant, workshops, and offices.
- Mining, processing, desalination, and port activities can be conducted year-round.
- The Property is likely to have high seismicity. A seismic hazard assessment was performed by a third party on behalf of Capstone and recommendations arising from the study are included in the current designs.

6 HISTORY

6.1 Regional History

Artisanal mining activities commenced in the general area of the Santo Domingo property during the early 19th century. The major commodities targeted were gold and iron. There is a significant number of small workings and pits throughout the Property. Most of the surface workings are typical of artisanal activities, being less than a few tens of metres in length. There is limited information as to the extent of underground mining activities. No production records from this activity have been located.

Modern exploration commenced in 2002 when Far West and BHP Billiton formed an alliance to explore the Chilean iron oxide-copper-gold (IOCG) belt in the Coastal Cordillera. In 2002, BHP Billiton flew a 10,700-line km Falcon™ airborne gravity gradiometer survey covering 5,145 km^2 in eight blocks along a 300 km strike length of the IOCG belt between Taltal and south of Copiapó in northern Chile. The survey outlined more than 76 target areas containing one or more distinct gravity anomalies and magnetic anomalies. Far West and BHP commenced basic exploration activity in July 2003. BHP's interest in the Property was terminated in May 2005. BHP transferred concession titles to Far West in exchange for a 2% NSR royalty.

In January 2006, Far West announced an agreement with ENAMI to acquire a 100% interest in its 673 ha Iris property. ENAMI transferred concession titles to Far West in exchange for staged payments and a 2% NSR royalty.

Between 2005 and 2011, Far West progressed with exploration activities (See table 6-1) which resulted in the discovery of Santo Domingo Sur, Iris and Estrellita deposits. An initial copper-gold mineral resource estimate was performed in 2006 for Santo Domingo Sur and updated in 2007, which included first-time estimates for Estrellita and Iris. During 2007 a ground Transient Electromagnetic (TEM) survey was conducted in the property which guided additional exploration drilling north of Iris deposit and concluded with the discovery of the covered Iris Norte deposit in 2008.

In 2008, a preliminary assessment (PA) was completed considering two open pit options for Santo Domingo and Iris deposit processing copper, gold, and iron (magnetite and hematite). During 2008 and 2009, the copper–gold resource estimates for Santo Domingo Sur and Iris were updated and the Iris Norte deposit added to the mineral resource. Iron and magnetic susceptibility were included in the updated resource estimates. A further copper–gold–iron resource update was performed in 2010, covering Santo Domingo Sur, Iris, and Iris Norte.

In June 2011, Capstone Mining (70%) and Korean Resources Corporation (KORES) (30%) acquired Far West and completed a pre-feasibility study in the same year. The study envisaged conventional open pit mining of the Santo Domingo Sur, Iris, and Iris Norte deposits; processing using a semi-autogenous grind, ball mill and pebble crushing comminution circuit (SABC), conventional copper flotation, magnetic separation; tailings disposal and storage; water and concentrates pipelines; port facilities; and associated site infrastructure requirements.

In June 2014, Capstone Mining completed a feasibility study on the property and in July 2015, the Project Environmental Impact Assessment (EIA), including the mine, infrastructure, process facilities, development of a greenfield port, and iron concentrate and water supply pipelines (as outlined in the 2014 Feasibility Study), was approved by the Chilean authorities. The feasibility study was subsequently updated in January 2019, including new pilot plant testing information, reflect changes on the current pricing and to adding cobalt into the mineral resource. In February 2020, Capstone completed an updated feasibility study that included a preliminary economic assessment on an alternative development option, consisting of conventional open pit mining, production of copper and iron concentrates via a conventional process plant and production of battery-grade cobalt sulphate via roasting.

In April 2021, Capstone Mining completed the acquisition of the 100% of Santo Domingo Property acquiring the remaining 30% stake from KORES and in March 2022, Capstone Mining merged with Mantos Copper (Bermuda) Limited forming Capstone Copper Corp. who currently owns 100% of Santo Domingo Project.

Table 6-1: Exploration History Table Summary

Year	Company	Work Program	Program Details	Reported
2002	BHP Billiton	Airborne geophysical survey	10,700-line km Falcon™ airborne gravity gradiometer survey covering 5,145 km² in eight blocks along a 300 km strike length of the IOCG belt between Taltal and south of Copiapó.	-
July 2003 to November 2005	Far West/ BHP Billiton*	Geological mapping	Approximately 50 km² of geological mapping at 1:25,000 scale	Höy and Allen, 2005
		Surface rock samples	50 samples submitted for analysis for Au and a 27-element ICP suite. Samples were generally taken where copper oxides were apparent and hence most samples contained anomalous levels of copper.	
		Sediment samples	47 sieved (106 µm) samples, submitted for analysis for Au and a 27-element ICP suite.	
		IP survey	17.6-line km	
		RC drilling	67 holes (20,592 m) analyzed for Au and a 27-element ICP suite	
November 2005 to May 2006	Far West	Geophysical data interpretation	Falcon™ gravity and magnetic susceptibility plots were produced for data from Quantec Geofisica Limitada. The gravity anomalies define a north–south-oriented cluster of northwest-trending features up to 5 km long within the Property.	Lacroix, 2006
		RC drilling	15 holes (5,176 m) analyzed for Au and a 27-element ICP suite	
		Core drilling	4,057 m in eight holes; analyzed for Au and a 27-element ICP suite	
May 2006 to September 2007	Far West	RC drilling	215 holes (51,909.5 m); analyzed for Au and a 27-element ICP suite	Lacroix and Rennie, 2007
		Core drilling	15 holes (2,649.75 m); analyzed for Au and a 27-element ICP suite	
September 2007 to December 2008	Far West	Transient Electromagnetic (TEM) surveys	81-line kilometre of data collected. Copper exploration program consisted of 35 survey lines, 65.5-line kilometre. Water exploration program consisted of 8 survey lines, 15.5-line kilometre.	-
		RC drilling	37 holes (10,376.5 m); analyzed for Au and a 27-element ICP suite	Lacroix, 2009
		Core drilling	One hole (495.25 m); analyzed for Au and a 27-element ICP suite	
December 2008 to May 2010	Far West	RC drilling	Nine holes (2,557 m); analyzed for Au and a 27-element ICP suite	Rennie, 2010
		Core drilling	26 holes (9,073 m); analyzed for Au and a 27-element ICP suite	

6.2 Production Area

There has been no formal production from the Santo Domingo property area.

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

The Santo Domingo deposit is part of the Chilean Iron Belt (CIB), which is defined as north–south belt stretching for over 2,000 km parallel to the Chilean coast, from approximately 25ºS to 31ºS.

The main rocks at regional scale belong to a Jurassic – Early Cretaceous volcanic-plutonic arc and back-arc basin that produce thick volcanic and volcaniclastic rocks represented by La Negra and Punta del Cobre formations and the Bandurrias Group (Parada et al., 2007). Relatively contemporaneous marine sediments deposition occurred within the back-arc basing represented by the Chañarcillo Group (Figure 7-1). All these rocks coexisted with a number of tabular-shaped mafic to felsic plutonic complexes between approximately 132 Ma to 106 Ma, including the Sierra Dieciocho plutonic complex located in the vicinity of Mantoverde and Sierra Aspera plutonic complex, 10 km northwest of Santo Domingo deposit. These rocks collectively produce favorable host rocks for iron oxide-copper-gold (IOCG) deposits.

The dominant structural feature of the belt is a complex sinistral strike-slip and dip-slip fault system known as the Atacama fault zone, which is interpreted to be related to an oblique subduction of a Jurassic to early Cretaceous magmatic arc. Initial faulting took the form of strike-slip, causing mylonite development and ductile deformation. This gave way to dip-slip fault movement and brittle deformation during later extensional tectonism, which accommodated a number of volcanic- or intrusive-hosted breccia zones which became sites for the formation of a number of Fe-oxide apatite (IOA) and iron-oxide–copper–gold (IOCG) deposits.

The CIB in northern Chile hosts various deposits including the Fe-oxide-apatite and iron-oxide-copper-gold (IOCG) deposits (Figure 7-1):

- The iron-rich end members have magnetite as main ore mineral and can be classified as Kiruna-type magnetite–apatite deposits with associated actinolite–apatite and calcic-sodic alteration. Host rocks are typically brecciated volcanic materials, or brecciated intrusions. Some examples of the larger Kiruna-type deposits in the CIB include Romeral, Cerro Negro Norte and Cerro Iman.
- The copper-rich members contain abundant hydrothermal hematite and/or magnetite with supergene and hypogene copper mineralization. Alteration varies, but commonly, these deposits are associated with potassic, extensive sodic, and sodic calcic mineral assemblages. Carbonates are typically common. Copper-bearing end members include Mantoverde, Candelaria-Punta de Cobre, Sierra Norte and Casualidad.

7.2 Property Geology

7.2.1 Lithologies

The Santo Domingo property lies 30 km to the east of the Atacama fault complex, which, in this area, consists of numerous clusters of generally north–south structural breaks in a belt approximately 30 km wide. The main rocks exposed in Santo Domingo area comprehend a series of volcanic, volcaniclastics rocks probably from Punta del Cobre Formation and a marine-sedimentary sequence correlated with Chañarcillo group. All rocks have been intruded by a series of small mafic and felsic plutons and dykes which produced localized garnet-bearing skarn and hornfels when in contact with sedimentary and volcanic rocks.

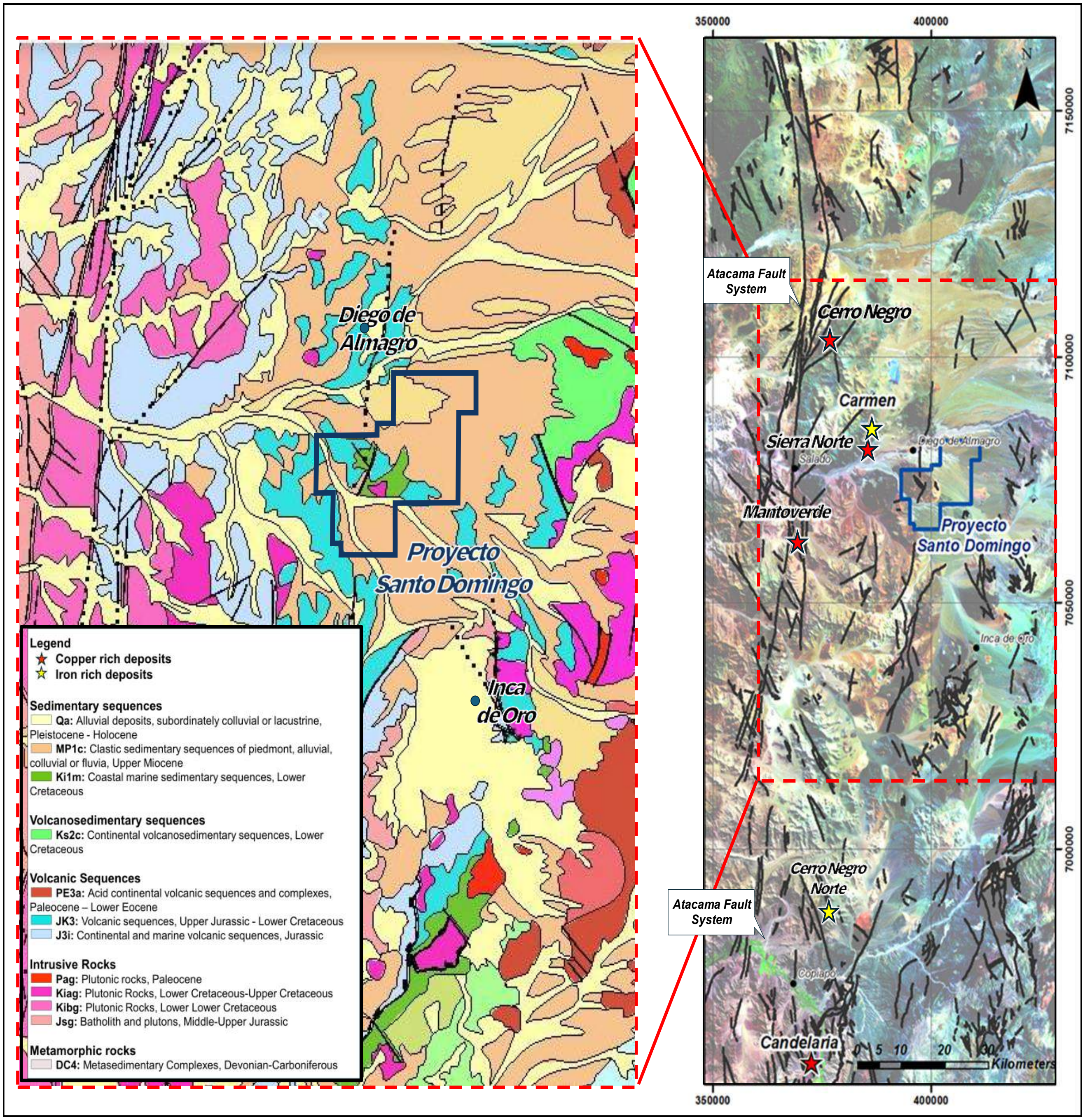
The volcanic andesite flows are exposed in both Santo Domingo (Santo Domingo Sur area mainly) and Estrellita (Figure 7-2). They exhibit varying textures from near aphanitic to coarse-grained feldspar-phyric, but are generally medium grained, with 20% to 30% euhedral, white, prismatic plagioclase (±minor hornblende) phenocrysts in a grey to brownish aphanitic groundmass with some containing amygdales filled with quartz, calcite, and copper oxides. These rocks can be found both at the base and at the top of the stratigraphic column of the area. The top of the ore body is located within the upper Andesite. The mineralization in this unit is located within andesitic breccias, especially in contact with the underlying volcaniclastic rocks, and within amygdaloidal andesites. Also, the entire sequence is crossed by mineralized veins associated with structures.

The volcaniclastic rocks outcrop mainly in the central sector between the area of Santo Domingo Sur and Iris (Figure 7-3, volcaniclastic rocks), stratigraphically between the upper and lower andesite units. These rocks comprise primarily tuffaceous sediments or crystal tuffs, lithic tuffs, massive to poorly bedded, fine to medium grained and commonly challenging to distinguish from fine-grained, massive flows. The sequence also contains laminated and welded tuffs that usually work a marker horizon. Individual units reach thicknesses of up to 50 m but can comprise the bulk of the stratigraphy, reaching over 400 m in thickness within the orebody with minor intervals of limestone and andesite lavas. In the deposit, the complete unit reaches approximately 400 m in thickness, with a maximum length and width of 5,500 m and 2,000 m, respectively, considering the entire area from Santo Domingo to Iris Norte. This sequence contains the bulk of the iron oxide-copper-gold mineralization within the deposit.

In the SD area, the volcaniclastic sequence is cut by a layer of andesitic lithic and crystalline tuffs that divides the volcaniclastics, and the mineralized body within it, in two. These andesitic tuffs are poorly mineralized; any mineralization present is discontinuous and occurs mainly as veins.

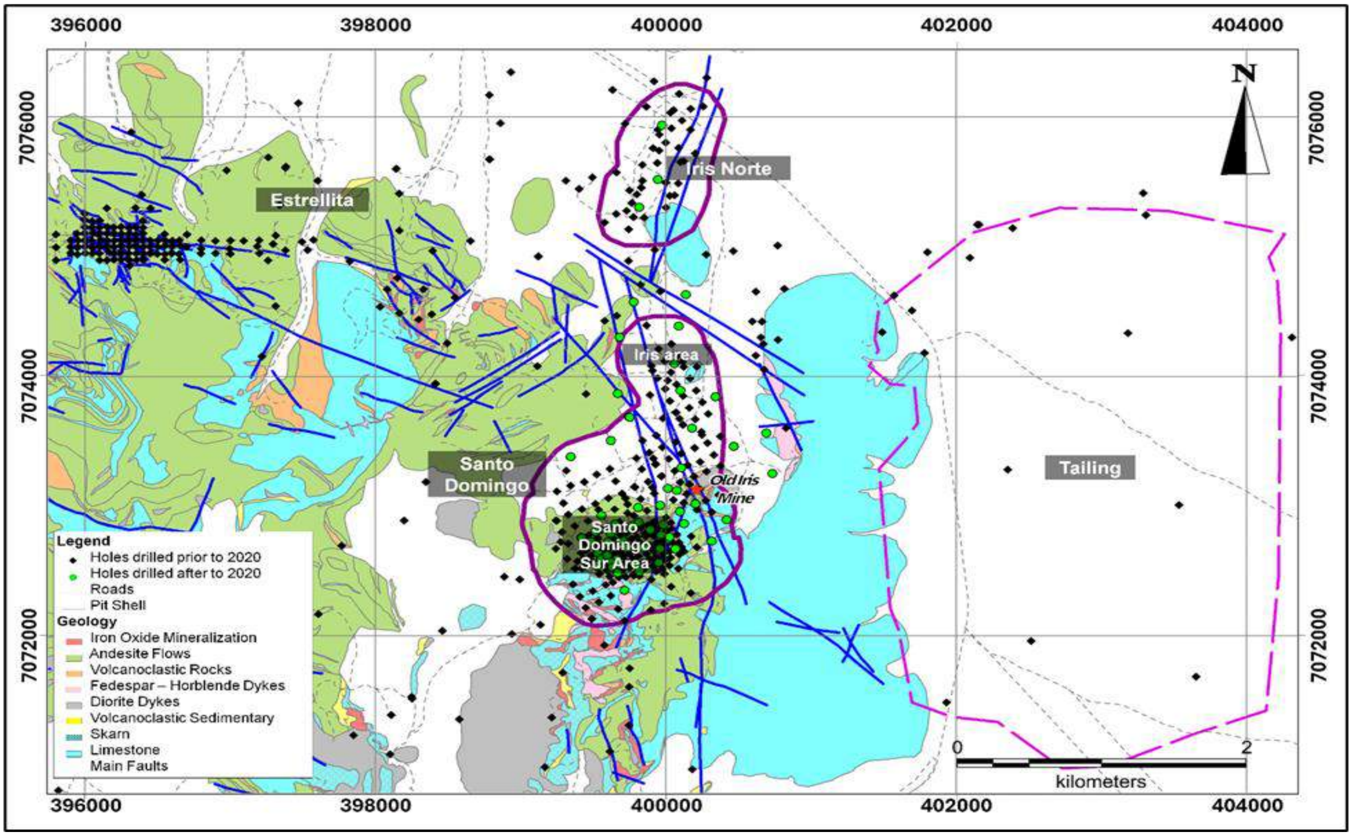
Within the area of the deposit, the volcaniclastic sequence transitions laterally within relatively short distances into carbonate-rich sediments and limestones, which are generally massive to thickly bedded, fine grained and dark to light grey. Several drill intercepts and the continuation of a marker horizon has allowed Santo Domingo geologists to support this interdigitation between mineralized volcaniclastics and the relatively barren limestones and carbonate sediments. This sedimentary sequence varies in thickness from a few metres to over 350 m but can be the predominant rock type across several hundred metres of stratigraphy. These rocks outcrop mainly east of Santo Domingo, between the Santo Domingo projected pit and the tailing and also forming the top parts of many prominent hills in the Estrellita and in the area between Santo Domingo and Iris Norte (Figure 7-2).

Figure 7-1: Regional Geology and Fault Structures in the Greater Santo Domingo Property Area



Source: Figure prepared by Capstone, 2024. Deposits shown as red stars outside the Property area. Grid coordinates indicate map north and figure scale.

Figure 7-2: Deposit and Prospect Layout



Source: Figure prepared by Capstone, 2024.

A relatively small breccia pipe with sub-vertical geometry is located southeast of Santo Domingo and cuts all the geological units of the model, from the lower andesites through the volcaniclastic rocks and even reaching the upper andesites. The breccia cut the mineralized sequence, exhibiting monolithic clasts, sometimes mineralized whereas the matrix has mixtures of iron-oxides, sulfides, and gangue minerals.

A series of small plutons and sub-horizontal dykes appear to be late and are cutting the stratigraphic sequence of Santo Domingo and are distributed across the area the entire deposit, from Santo Domingo to Iris Norte. The intrusives have been separated into two main units in the model, diorites, and dykes (Figure 7-2). The first unit is represented by an intermineral andesite/diorite porphyry, which cuts the volcanic sequence, while the second is represented by post-mineral andesites/diorites intrusive, rich in feldspars, inequigranular. More abundant dyke swarms and pluton can be found at the southern portion of Santo Domingo area generating garnet-bearing skarn and hornfels when in contact with sedimentary and volcanic rocks.

7.2.2 Structural Geology

Fault trends at the Santo Domingo property are variably with a main northwest trend associated with the mineralization and of secondary importance, north-east, and more rarely, east-west orientation.

These faults are complex and seem to have been active repeatedly through time. Most faults recognized in the area, either through mapping or drill intersections, appear to be high angle faults with limited displacement in both dip-slip and strike-slip movement. The faults appear to have been active while the deposition of the different rock units occurred.

The most obvious structure located 2km west of the deposit, crosses the Estrellita and Estefánia areas. It is a large east-west trending, steeply north dipping, north-side-down block fault, with a probable right lateral strike-slip component. Most of the historic copper production in the area comes from or near this structure.

The most prominent fault set closer to Santo Domingo deposit, trends northwest and has fault separations of approximately 1 km. Several others northwest-trending faults associated with mineralization are also recognized in the Santo Domingo Sur area.

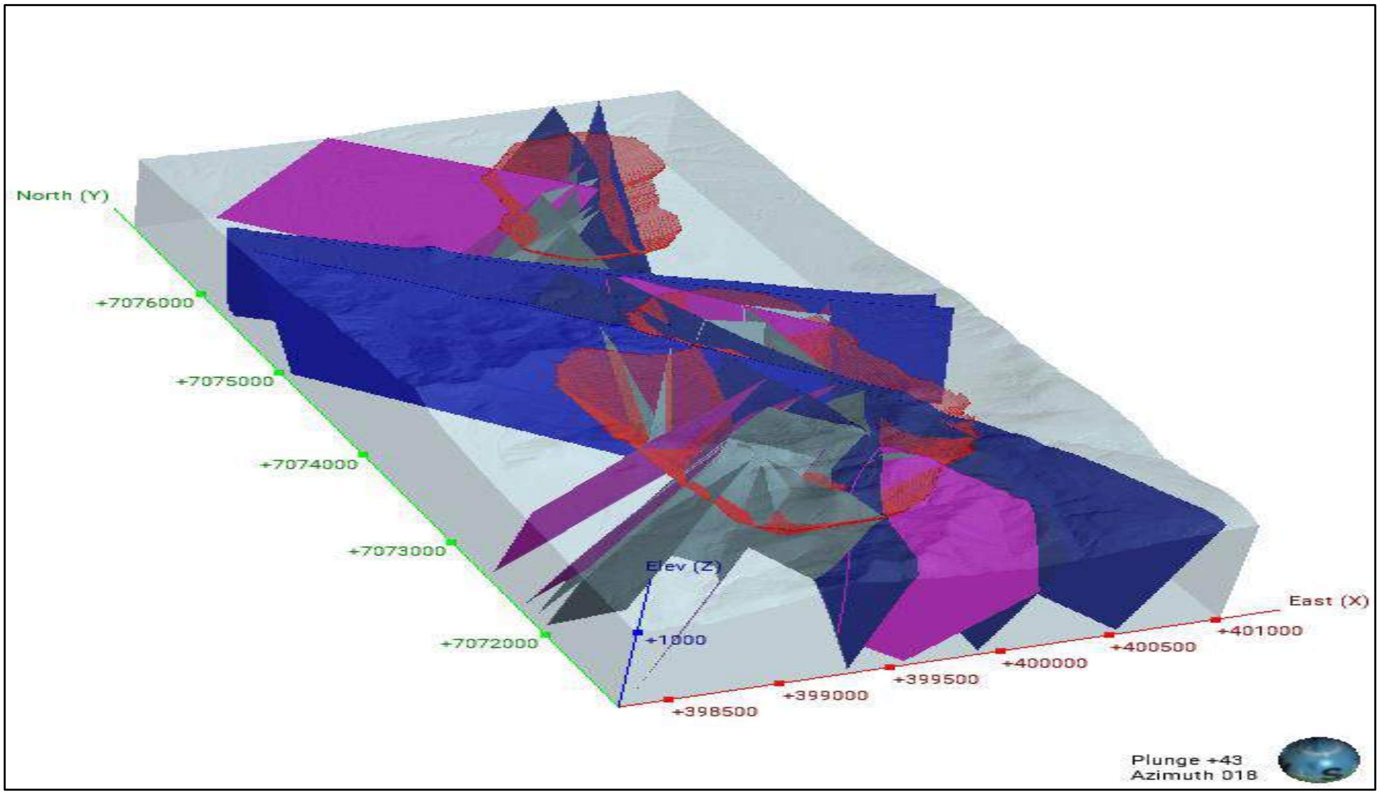
High-angle block faulting played an important role in localizing manto- and fault-related iron oxide–copper mineralization in the Property. These faults have probably uplifted the central part of the Santo Domingo Sur area in Iris area, bringing the manto succession close to surface.

At project scale, the main structural arrangement can be seen in Figure 7-3. The structural model, developed during 2022 for Santo Domingo, shows preferential major faults NS-N35°W/70°SW for Santo Domingo, and N5°-20°E/70°-85°NW for Iris Norte. Additionally, a structural jog type configuration is observed in Santo Domingo and Iris Norte (Figure 7-3).

Based on indirect methods of deep geophysics and overburden thickness, it has been interpreted that between the deposits of Santo Domingo (Iris area) and Iris Norte, a NW fault was developed with two major faults trending N55°W/80°SW, which separated both environments through normal faulting and rotation of the South block. This discontinuity still needs to be studied and validated.

Limited mapping and recognition of outcrop-scale, open folds indicate that the rocks have been gently folded along north–northeast-trending axes.

Figure 7-3: Structural Geology 3D Viewing Showing Structures in the Santo Domingo and Iris Norte Areas



Source: Figure prepared by Capstone, 2024. The figure shows the structures recognized in the Santo Domingo deposit. The major faults are shown in blue, the main faults in magenta and the minor faults in grey. The 2024 resource pit is shown in red for reference.

7.2.3 Alteration

Hydrothermal alteration and mineralization in the Santo Domingo area affects all rocks and exhibits numerous styles and events with multiple overprinting components. At the deposit and district scales four styles of alteration are recognized: sodic (-calcic), potassic, carbonate and calc-silicate skarn. A clear hydrothermal zoning occurs from proximal to distal assemblages at deposit scale (Santo Domingo Sur) and apparently at district scale at depth and towards the diorite intrusive complex.

Main sodic (-calcic) alteration minerals are albite, actinolite, chlorite, epidote and titanite that replace mainly volcanic and intrusive rocks. Scapolite–actinolite–pyroxene veins can be found at the southern portions of the area close by and within the diorite stocks and dykes. At surface, actinolite, chlorite and carbonate typically occur as infilling amygdule's and open spaces. Pink albite replaces plagioclase in the more porphyritic rocks.

Potassium silicate alteration is less common but is found as K-feldspar–chlorite–carbonate–quartz mineral assemblages. Patchy K-feldspar mainly replaces plagioclase (albite) and is also found in veins with carbonate and quartz. This alteration is mainly located within the copper-iron mantos.

Carbonate rich assemblages are widespread and overprint the previous mineral associations. In addition, carbonate (calcite, ankerite, siderite)–chlorite–quartz veins and stockwork are commonly found cutting all rock types of the area.

Calc-silicate skarn minerals are found south of Santo Domingo where carbonate rich rocks and lesser volcanic rocks are in contact with diorite intrusive units. Main alteration minerals are garnet (andradite), epidote, pyroxene, actinolite and carbonate.

7.2.4 Weathering and Supergene Development

Supergene processes are weakly developed in the Santo Domingo area. Oxidation is shallow, ranging from 70 m to 90 m below surface, and enrichment is weak, consistent with the low total sulphide contents and the calcareous and feldspathic nature of the host rock. The iron–copper–gold mineralization in Santo Domingo is almost entirely hypogene; the proportions of sulphides to oxides are approximately 13:1. At shallow levels, typical copper–iron mineralization includes small veins, hydrothermal breccias and mantos. Specular hematite ±magnetite is commonly altered to earthy hematite and goethite and usually is found with mixtures of copper oxides. QEMScan data suggest the oxides in Santo Domingo mainly correspond to a mixture between minerals such as chrysocolla, black oxides, Cu-limonite, Cu-clays and malachite in order of occurrence. On the other hand, Estrellita orebody contain ~40% of oxides which are characterized by chrysocolla, malachite, Cu-limonite, black oxides, Cu-sulfates and Cu-clays mainly. A narrow and weak enrichment zone with digenite ±chalcocite ±covellite is locally present, partially replacing fractures and rims of bornite and chalcopyrite crystals. Little amounts of native copper can also be found locally interstitial to the matrix of hydrothermal breccias or associated with faults in the shallower portions of the central part of the Santo Domingo Sur area.

7.3 Deposits

To date, three deposits, two of which support Mineral Reserves, and a number of prospects, have been identified in the Property (Figure 7-2) Santo Domingo, Iris Norte and Estrellita. The Santo Domingo deposit itself has two areas: Santo Domingo Sur and Iris.

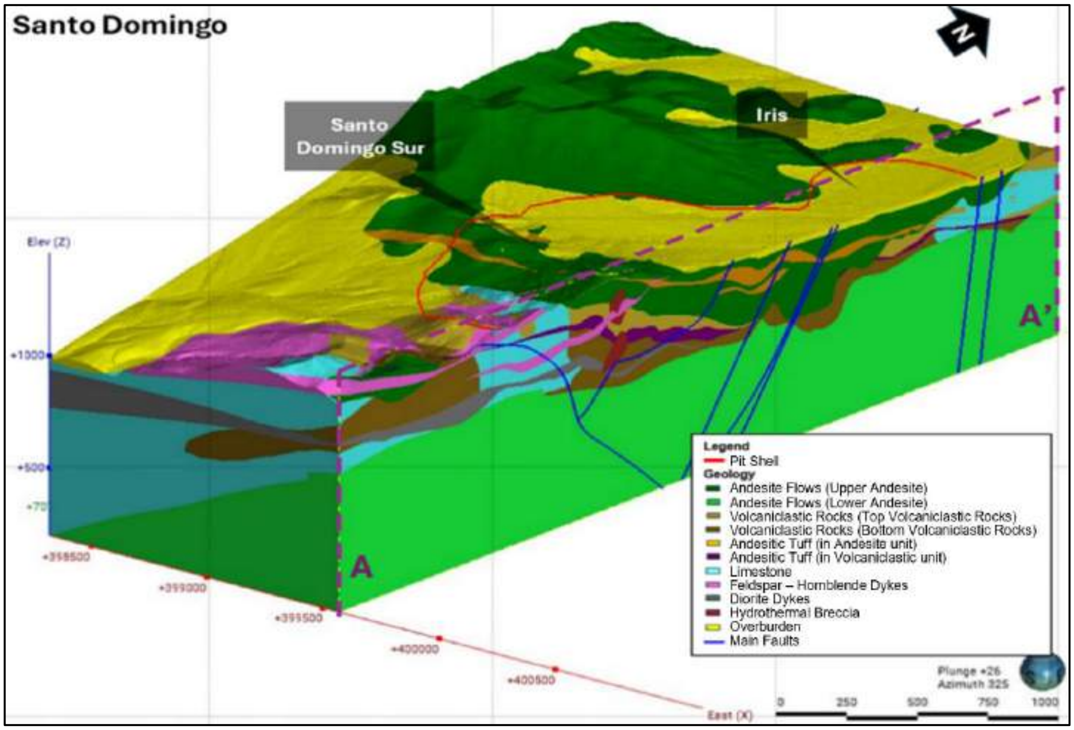
7.3.1 Santo Domingo

7.3.1.1 Santo Domingo Sur

The andesitic flows and volcaniclastic tuffs are the primary hosts to mineralization at Santo Domingo Sur (Figure 7-2, Figure 7-4, and Figure 7-5). The stratigraphic sequence of andesitic flows and tuffs dips gently (at an angle of approximately 15°) towards the north–northwest under gravel cover.

Figure 7-4 presents a geology and structural 3D view of the Santo Domingo deposit. Figure 7-5 is an example cross-section through the Santo Domingo deposit showing the location and orientation of the mineralization.

Figure 7-4: Geology and Structure 3D View, Santo Domingo



Source: Figure prepared by Capstone, 2024.

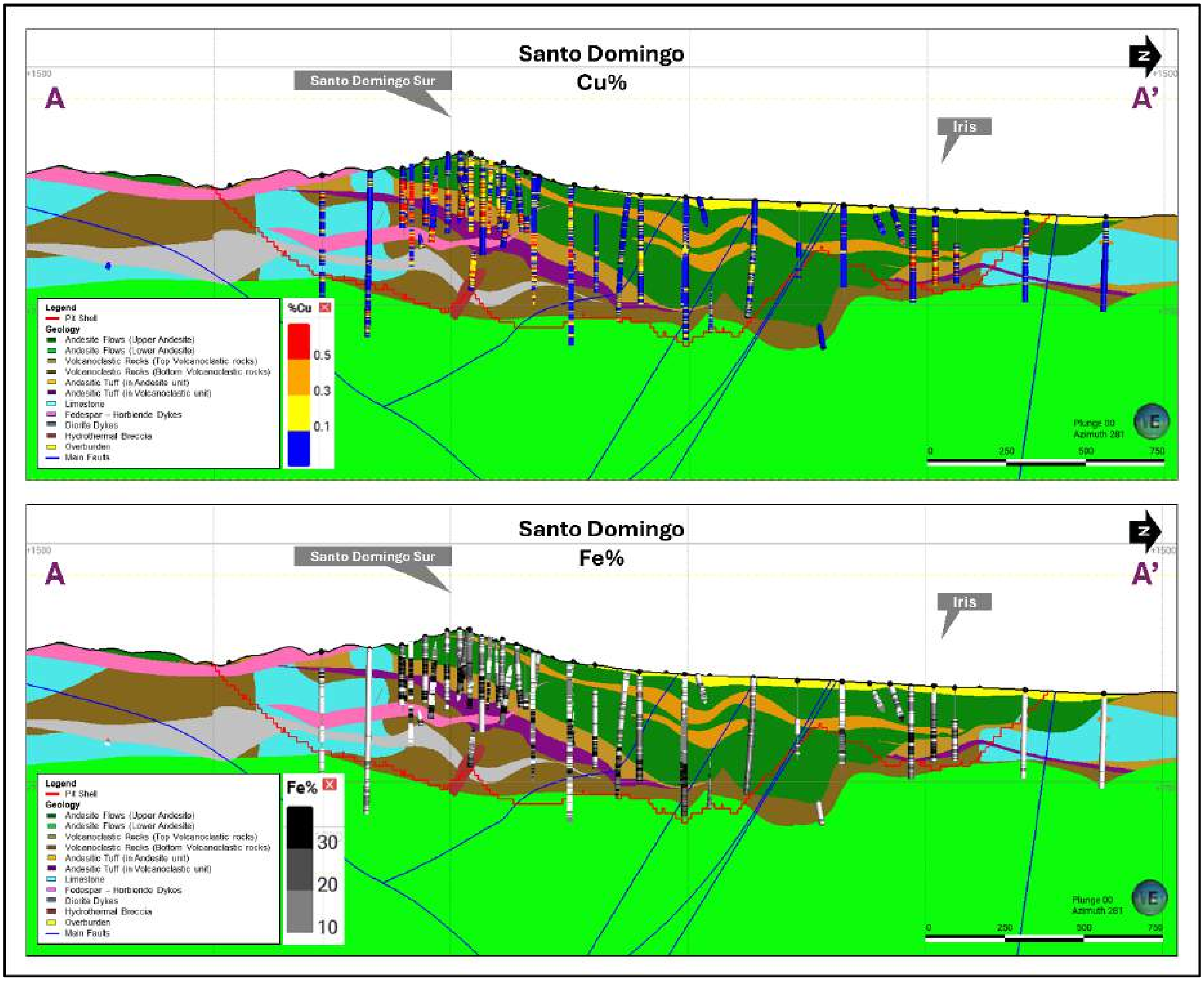
Mineralization occurs in the form of copper-bearing semi-massive to massive iron oxide mantos that have replaced the tuffaceous rocks, with minor veins and breccias. The mantos are zoned from an outer rim of specular hematite toward a magnetite-rich core.

The tuff sequence has been intruded by fine-grained diorite sills that are present in almost all drill holes at Santo Domingo Sur, varying in drilled thickness from a few metres to more than 60 m. Similar diorites have been intersected in the Iris area and have been observed in outcrop to the south of Santo Domingo Sur area.

The diorites are typically altered and in rare cases contain copper mineralization. These observations suggest that the diorite intrusions are more or less contemporaneous with the mineralizing event and may in fact have been the heat engine for the formation of the deposit. The last intrusive events in the area are feldspar–hornblende porphyry dykes.

Drilling has identified a 150 m to 500 m thick, copper-bearing, specularite–magnetite sequence covering an area of approximately 1,000 m by 1,200 m and traced to a depth of approximately 525 m below surface. Mineralization consists of stacked semi-massive to massive specularite-magnetite mantos replacing tuff and tuffaceous sediments overlain by andesitic flows. The mantos contain clots and stringers of chalcopyrite and cobalt-rich pyrite, and range in thickness from approximately 4 m to 20 m. The upper parts of the manto sequence directly below the overlying andesite flows are frequently oxidized and contain various amounts of copper oxides and chalcocite.

Figure 7-5: Simplified Geology Cross-Section, Santo Domingo



Source: Figure prepared by Capstone, 2024. Drill traces show total Cu and Fe grades.

Mineralization in the deposit is strongest in the southern part and in the upper levels. Copper grade and intensity of the mineralization weaken towards the northern part of the deposit as well as with depth. The high-grade core of the deposit is located along the southern margin and close to surface. The oxidized mineralization at surface becomes gradually less oxidized with depth and transitions through an enrichment zone of secondary sulphides (chalcocite) and lesser oxides into a sulphide zone where the main copper mineral is chalcopyrite.

A zone of hydrothermal brecciation has been recognized in the southeast of the area. The mineralized breccia consists of andesite and volcaniclastic rocks fragments in a fine-grained matrix of iron oxides sulphides and other gangue minerals. The most superficial part of the breccia contains mainly andesite fragments while at depth the fragments mainly correspond to rocks of mineralized volcaniclastic origin, which is consistent with the host rock. The upper part of the breccia is oxidized with both limonite, which is the dominant iron oxide and copper oxides. Native copper has also been observed. The lower part of the breccia contains regular sulphide mineralization. In the proximity of the major structures that have produced the brecciation, areas with copper oxides are observed even at depth, surrounded by an enrichment zone with a strong presence of secondary sulphides, passing in its most distal part to a zone of primary sulphides. The breccia has been intercepted by multiple drill holes, establishing a complex geometry that forms a narrow body at depth, but which widens toward the surface.

7.3.1.2 Iris

The Iris area is blind, covered by a sequence of Quaternary gravel (Figure 7-4 and Figure 7-5). The elongated shape of the deposit and textures observed in core drill holes indicate that the Iris area has formed in a north–northwest-striking fault zone that is bounded by a west-dipping fault that can be traced along most of the deposit's western side.

The Iris area footprint, when projected to surface, is approximately 500 m wide, has a strike length of 1,200 m and has been traced from surface to a depth of approximately 500 m below surface. When the dip and plunge of the zones is considered, the real width of the deposit is of the order of 200 m.

The deposit consists of iron oxide mantos and breccias developed along a north–northwest-striking fault zone. The dominant iron oxide at Iris is hematite and the main copper mineral is chalcopyrite.

Mineralization occurs close to surface at the southern end and plunges gently towards the north. The distribution of copper mineralization in the Iris area is more erratic and irregular when compared to the Santo Domingo Sur area. This is attributed to structural controls playing a greater role in the formation of the Iris area as contrasted with the more continuous stratiform replacement style mineralization at Santo Domingo Sur.

There are some old mine workings at the southern end of the deposit where copper oxides such as brochantite and chrysocolla were mined at surface. The oxide mineralization is hosted by a specularite manto that is cut by steeply dipping structures. The extent of oxide mineralization at surface is approximately 100 m by 60 m (Figure 7-2, Old Iris Mine, red star on the map).

The Iris Mag zone is located between the Iris and the Santo Domingo Sur area. Mineralization in the zone consists of magnetite and chalcopyrite with a very high magnetite content (40% and more) and typically low copper content (approximately 0.1% Cu on average).

The host rocks are andesitic flows and andesite breccias with a much smaller tuff component than the other zones. It appears that this part of the deposit has been subject to the initial high temperature magnetite event but shows little evidence of a later oxidizing overprint that has introduced high grade copper and gold values elsewhere.

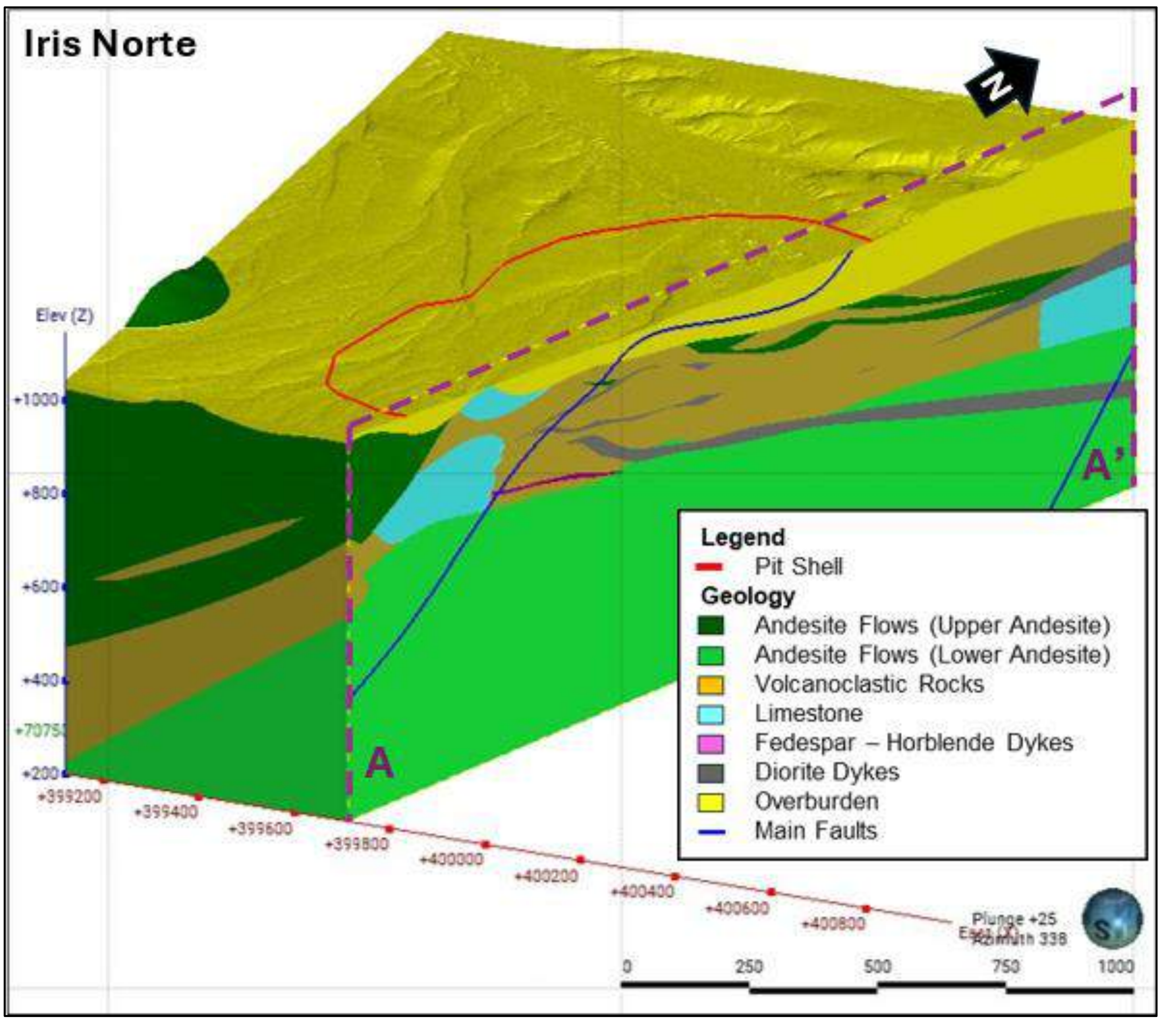
7.3.2 Iris Norte

The Iris Norte deposit (Figure 7-2) is located 600 m to the north of the Santo Domingo deposit and is also blind, being entirely covered by a gravel sequence. The deposit is very similar in character to Iris. The deposit is approximately 1,000 m wide and has been tested over a strike length of 1,600 m and to a depth of 300 m below surface.

Figure 7-6 is a geology 3D View of the Iris Norte deposit, and Figure 7-7 is a cross-section through the deposit. Mineralization is hosted within andesitic flows and volcaniclastics, which differs to the mainly tuff host rock at Santo Domingo. The deposit displays a north-easterly strike which is a rotation of approximately 55° clockwise versus the strike of the Santo Domingo deposit. The Iris Norte deposit has been intruded by significant numbers of diorite dykes and sills and presents an intercalation of andesitic lava flows with volcaniclastic andesitic rocks that make the mineralization more discontinuous, generating different mineralized levels.

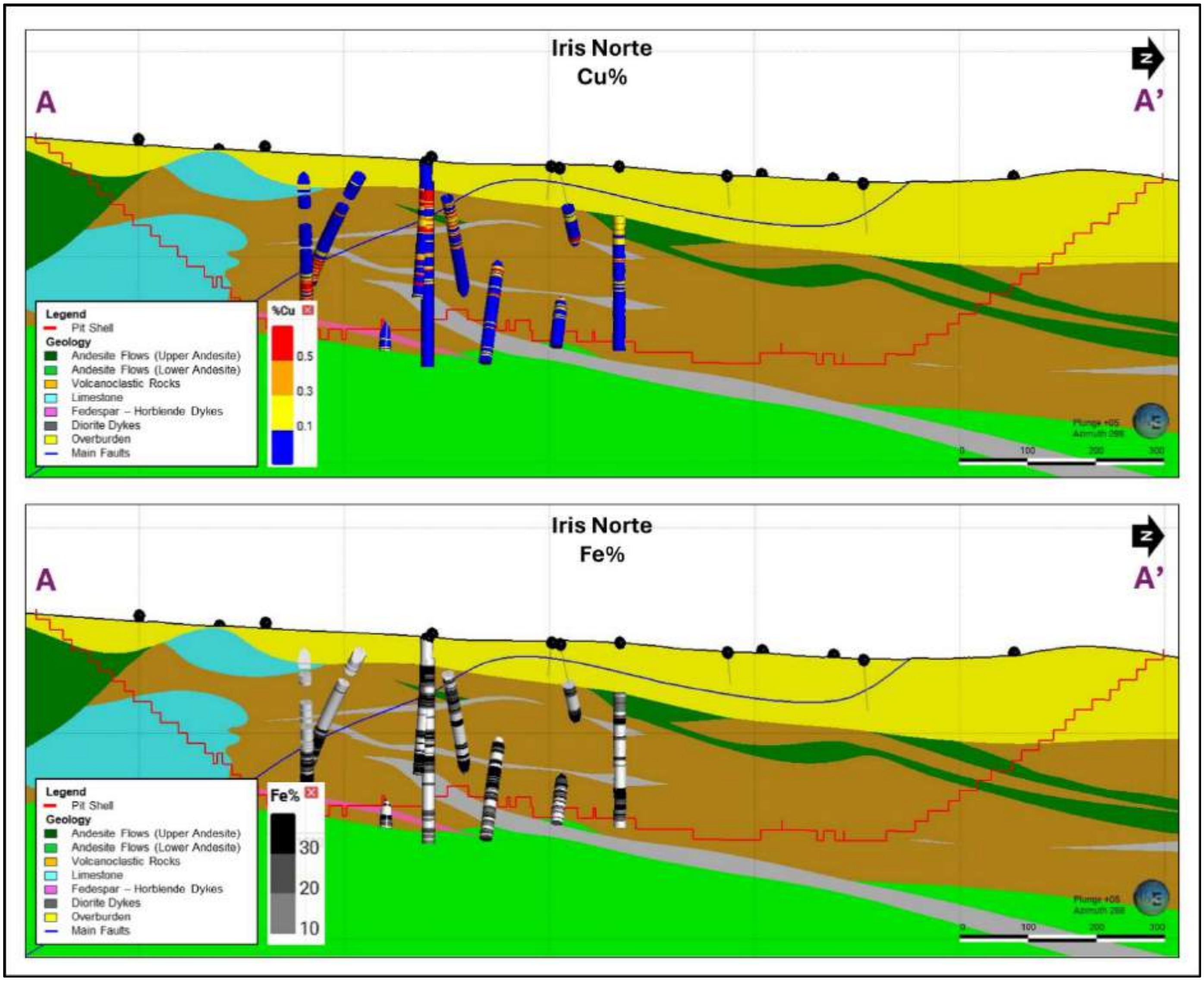
Mineralization consists of mixed magnetite and hematite-rich mantos. The main sulphides in Iris Norte are cobalt-rich pyrite with lesser chalcopyrite, with the latter providing the copper content of the deposit. Iris Norte contains a higher proportion of magnetite than the Iris area and there are a higher proportion of intrusive rocks.

Figure 7-6: Geology and Structure 3D View, Iris Norte



Source: Figure prepared by Capstone, 2024.

Figure 7-7: Simplified Geology Cross-Section, Iris Norte



Source: Figure prepared by Capstone, 2024. Drill traces show total Cu and Fe grades.

7.3.3 Estrellita

Estrellita (Figure 7-2) is an east–west-striking, flat-lying to shallow north-dipping tabular body lying approximately 3.5 km northwest of Santo Domingo Sur. Mineralization is interpreted by Santo Domingo geologists to occur at a higher stratigraphic level than Santo Domingo and Iris Norte, which are hosted in tuff sequences below the level of mineralization at Estrellita.

Figure 7-8 is a 3D view through the Estrellita deposit. Figure 7-9 is a cross-section showing a simplified geological interpretation.

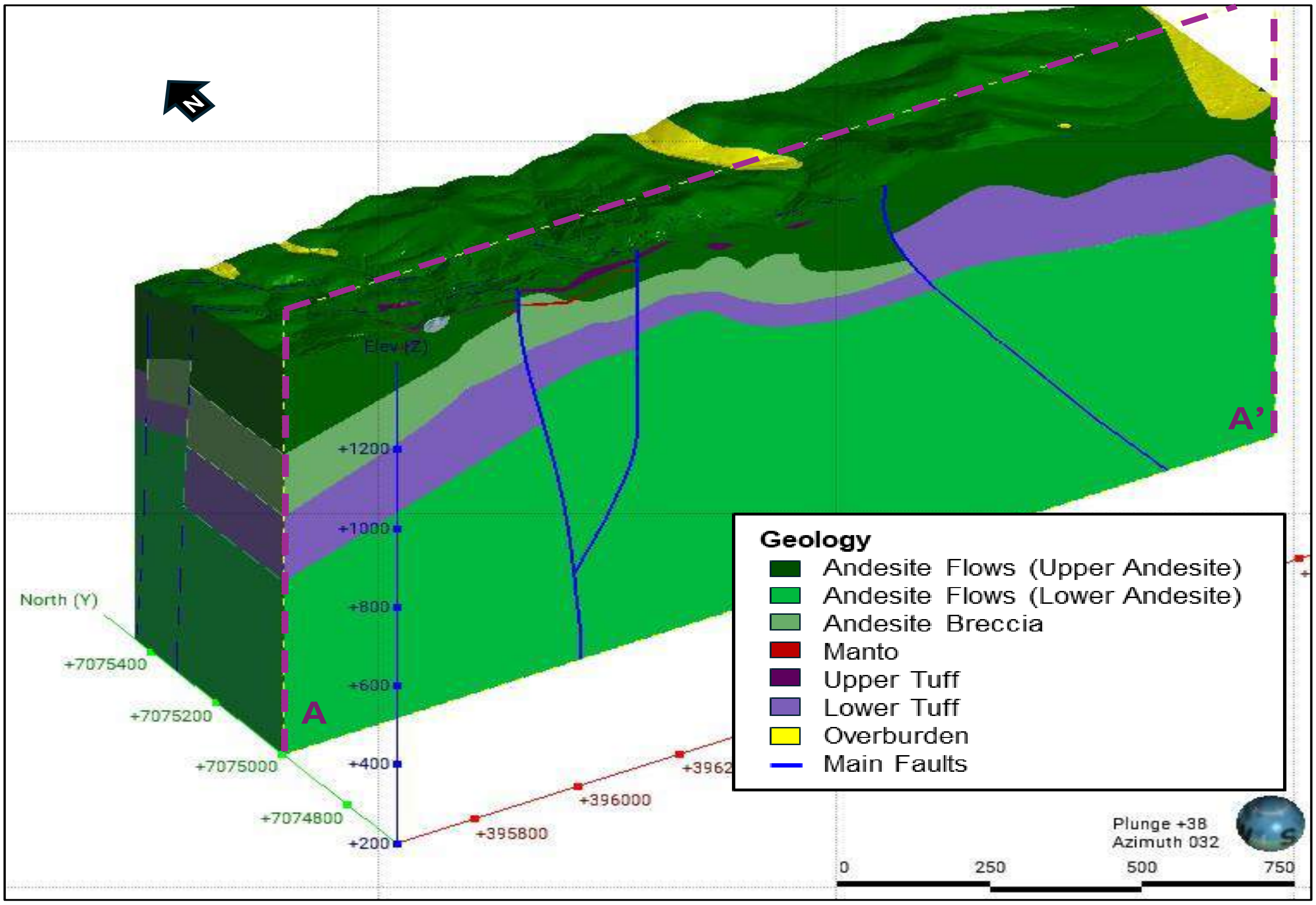
Drilling at Estrellita has shown that the host package of andesitic porphyries and flows has a thickness of up to 200 m. In the Estrellita area, this package is underlain by a sequence of volcaniclastics with minor intercalations and interbeds of andesite porphyry, limestone and altered tuff.

Estrellita has been faulted into a series of four blocks which step downwards to the north, with displacement across the faults ranging up to approximately 75 m. The overall footprint of the zone measures 900 m long by 450 m wide and is up to 100 m thick. The deepest drill intersections are in the order of 250 m below surface. The zone is thickest in the middle and narrows somewhat towards the periphery. There are narrower zones of limited lateral extent in the footwall of the main zone, but it is open ended to the east and west.

The character of mineralization at the Estrellita deposit is a mixture of manto-style, iron oxide and structurally controlled, vein and breccia-style mineralization. The central part of the Estrellita deposit consists of a more or less horizontal tabular body of iron oxide manto that appears to have formed at the intersection of a flat-lying and a steeply dipping set of specularite structures.

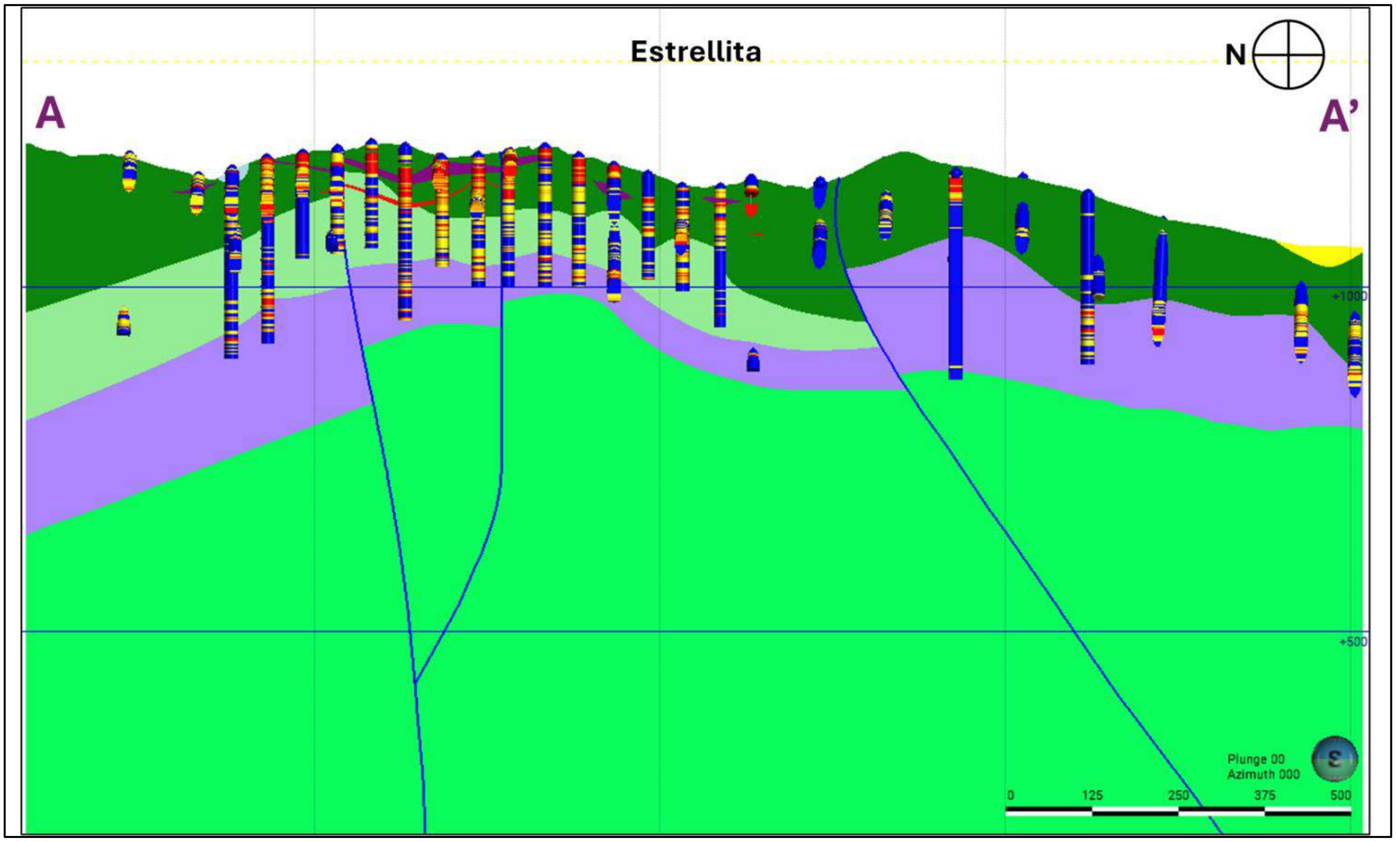
Copper mineralization typically consists of copper oxides such as chrysocolla, malachite, Cu-limonite, black oxides, Cu-sulphates, and Cu-clays. In the same way as in Santo Domingo the oxidized mineralization at surface becomes gradually less oxidized with depth and transitions through a mixed and enrichment zone with oxides and secondary sulphides into a sulphide zone where the main copper mineral is chalcopyrite.

Figure 7-8: Geology and Structure 3D View, Estrellita



Source: Figure prepared by Capstone, 2024.

Figure 7-9: Simplified Geological Cross-Section, Estrellita



Source: Figure prepared by Capstone, 2024. Drill traces show total Cu and Fe grades.

7.4 Comments on Geological Setting and Mineralization

The QP considers that the understanding of the geology and mineralization of the property is adequate for use in the Mineral Resource estimation.

8 DEPOSIT TYPES

8.1 Deposit Model

Information discussed in this section is from public domain sources and has been summarized from Barton, 2014, Williams et al, 2005 and references therein.

IOCG deposits, comprehend a diverse group with an empirical definition based on geochemical features, which brings together ore systems featuring Cu, Fe with or without Au, as main economic metals.

IOCG deposits, while sharing an empirical definition, exhibit considerable geological diversity and ages. Notable IOCG deposits globally include Carajás in Brazil, Gawler Craton and Cloncurry districts in Australia, and the coastal batholithic belt in Chile and Peru during the Jurassic-Cretaceous period. The geological and tectonic settings where these deposits form are diverse, from rift systems to continental margins, highlighting the intricate interplay of tectonic and magmatic processes.

The bulk composition of IOCG deposits defines the iron oxide (–Cu–Au–REE–P–Ag–U–Co) clan. These deposits, marked by >10% low-Ti Fe oxides, exhibit elevated contents of Cu, Au, REE, P, U, Ag, and Co. The geochemical specialization challenges understanding of the processes leading to this composition. The family of deposits lacks globally coherent tectonic or magmatic correlations, presenting a distinctive challenge for geological classification.

IOCG orebodies display intense and voluminous hydrothermal alteration in proximity, reflecting variable pressure-temperature conditions. All IOCG systems exhibit extensive Na-Ca-K(-Fe) exchange associated with clastic and igneous rocks and are commonly zoned. Typical alteration types are sodic-calcic, potassic, hydrolytic (acid), and skarn and Fe-oxide replacement when carbonate rocks are present.

Mineralization occurs over a broad depth range, and structural/stratigraphic controls play a pivotal role. Magnetite and hematite are the main iron oxides, commonly associated with pyrite and chalcopyrite and to a lesser extend bornite, chalcocite, pyrrhotite and sphalerite. Barite and anhydrite occur in some deposits. Hematite dominated assemblages in the form of specularite, commonly have higher sulfide content than magnetite ores, and may be replaced by magnetite (Musketovite). Gangue phases associated with hematite are quartz, chlorite, sericite, and sparse apatite. Magnetite dominated assemblages, has actinolite and apatite and lesser sulfides (typically Py>Cpy) and variety of accessory minerals including Titanite. Associated alteration includes sodic plagioclase, marialitic scapolite, calcic amphiboles or pyroxenes and epidote.

Structural and/or stratigraphic controls are pronounced, with deposits characteristically localized on fault bends and intersections, shear zones, rock contacts, or breccia bodies, or as lithology-controlled replacements.

Globally, several IOCG deposits stand out for their size and economic significance. In Brazil, the Carajás district hosts prominent deposits such as Salobo, Cristallino, Sossego, and Alemão. The Gawler Craton and Cloncurry districts in Australia boast the substantial Olympic Dam and Ernest Henry deposits. Meanwhile, Chile contributes to the list with the notable Candelaria-Punta del Cobre and Mantoverde deposits. The largest and most extensively mined IOCG-type deposits in Chile occur within the Chilean Iron Belt (CIB) which it extends in a north-south direction from La Serena to

Taltal in an area of 600 km by 50 km. The Atacama Fault System, a structurally complex zone, is the main structural feature along the belt and associated with these deposits. Deposits within the CIB have two general end members: a magnetite–apatite–actinolite mineral assemblage similar to the Kiruna deposit in Sweden, and a copper-rich type (IOCG) similar to the Olympic Dam deposit in Australia. The magnetite-rich deposits in Chile have been mined for iron since the early 1800s and the Los Colorados mine south of Copiapó is still in production. Examples of copper rich IOCG deposits in the belt include Candelaria and Mantoverde.

The Santo Domingo deposit contains many features associated with an IOCG type of deposit, including magnetite-hematite-pyrite-chalcopyrite as main mineral association with sodic-calcic, potassic and carbonate alteration styles. The orebody is hosted in volcaniclastic and volcanic rocks from the Punta del Cobre Formation from Cretaceous age, and exhibits a strong lithological-stratigraphy control, generating sub-horizontal replacing mantos.

The descriptions of the Candelaria and Mantoverde deposits as examples of other IOCG deposits in the CIB are described below.

8.2 Candelaria Deposit

Lundin Mining's Candelaria mine, located 20 km south of Copiapó, is hosted in altered volcanic and volcaniclastic rocks of the Punta del Cobre Formation, deposited in an Early Cretaceous continental volcanic arc and marine back-arc basin terrane. The stratigraphy includes the Geraldo Negro Member and the Algarrobo Member, consisting of andesitic volcaniclastic and dacites rocks. Candelaria's mineralization comprises magnetite-chalcopyrite-pyrite, primarily in the upper Geraldo Negro andesite and overlying Algarrobo Member. The strata-bound mineralization is constrained by an impermeable scapolite-rich skarn at the base of the Chañarcillo Group. Host rocks exhibit strong alteration, including biotite-quartz-magnetite assemblages in deeper areas, potassium feldspar with chlorite and/or biotite higher up, and a sodic alteration zone with albite-chlorite-calcite-hematite in the upper part. Sulphide stringers, mainly chalcopyrite and pyrite, post-date alteration events. Iron oxide mineralization dates back to 116 Ma to 114 Ma, followed by copper mineralization at 112 Ma to 110 Ma (Marschik et al., 2000). Ca-amphibole has been dated at 111.7 ±0.8 Ma (Ullrich and Clark, 1998) and hence is closely associated with the copper mineralizing event. These ages are broadly coincident with the age of the adjacent granitoid pluton which is therefore thought to be genetically related to mineralization.

8.3 Mantoverde Deposit

Capstone Copper´s Mantoverde mine is situated approximately 100 km north of the Candelaria deposit and 25 km southwest of the Santo Domingo deposit.

The main rocks are diorite plutons and altered andesitic volcanic rocks that belong to the La Negra Formation form Jurassic age (Zamora and Castillo (2000) and the Quebrada Salitrosa geological map by Godoy and Lara (1998)).

The Atacama fault zone, with three main branches (the eastern, central, and western faults), passes through the mine area, causing structural deformation. The Mantoverde deposit, located along the Mantoverde fault, exhibits tabular breccia bodies containing Iron oxides and copper minerals.

The hydrothermal alteration affecting the andesite and diorite host rocks involves chloritization, potassic metasomatism, and a series of mineral assemblages; Chlorite–quartz, Calcite–sericite–hematite–magnetite and K-feldspar–quartz–specularite.

Oxidation occurs to depths of over 200 m, with copper mineralogy in the oxide zone including sulphates, carbonates, silicates, chlorides, and pitchy copper ore. Supergene enrichment is present at the oxide-sulphide transition, with chalcocite and cuprite as dominant copper minerals. Hypogene sulphides are mainly chalcopyrite and pyrite. Gold is included in both sulfide phases, and gold contents in composite samples show a positive correlation with the Cu grades.

The mineralization at Mantoverde coincides with the age of the Sierra Dieciocho pluton, around 121 to 117 Ma.

8.4 Comment on Section 8

The QP considers that exploration programs that use copper rich IOCG deposit models are appropriate to the Property.

9 EXPLORATION

9.1 Geological Mapping

Field mapping activities have been carried out in the property since 2003. Far West completed approximately 50 km^2 of geological mapping at 1:25,000 scale between 2003–2005, focusing on the various lithologies, alteration, mineralization and structures. More recently in 2023, an oriented structural mapping was completed to support the updated structural model for the project.

9.2 Geochemical Sampling

Far West collected a total of 50 rock chip samples and 47 stream sediment samples (sieved to 100% passing 106 µm in the field) and generated copper and gold plots to assist exploration efforts. Most rock chip samples were collected near copper showings and copper bearing veins and hence contain anomalous copper values and relatively weakly anomalous gold. Drainages in the areas underlain by andesite flows, especially in the north and northwest part of the Property, are generally anomalous for copper. These values form a broad anomaly corresponding to northwest-trending specularite–copper oxide mineralized veins that cut the andesite rocks. Gold values in sediments are generally low.

9.3 Geophysical Survey

Several airborne and ground geophysical surveys were carried out in the property. Between 2002 and 2008, various methods, including airborne magnetics and gravity, ground induced polarization (IP), transient electromagnetic (TEM), and magnetics were completed by different specialized contractors. Historical geophysical datasets were also reprocessed. The induced polarization and TEM survey has been used to identify areas with disseminated sulphides and iron oxides that replace volcaniclastic rocks. Other survey such magnetics were used for district targeting, interpretation of the major structures and discrimination of lithologies with presence or absence of disseminated magnetite.

During 2013, Capstone Mining contracted Aeroquest Airborne to complete a VTEM and ZTEM helicopter airborne survey in a portion of the Property. The survey, spanning 356-line kilometres, employed geophysical sensors such as the versatile time domain electromagnetic (VTEMplus) system and a caesium magnetometer. The survey was performed in a southeast to northwest direction with a traverse line spacing of 200 meters, and a single line flown in an east to west direction. The ZTEM survey utilized a Z-Axis Tipper Electromagnetic (ZTEM) system and a cesium magnetometer, capturing 369-line kilometres of geophysical data. Geotech Ltd. oversaw the data processing, generating final digital data and maps. The processed survey results were presented as Reduced to Pole (RTP) Total Magnetic Intensity, In-Phase Total Divergence at 25 Hz, 75 Hz, and 300 Hz, Tzx and Tzy In-line and Crossline In-Phase and Quadrature Profiles over a 75 Hz Phase Rotated Grid, and a 3D View of In-Phase Total Divergence versus Skin Depth. The survey results included detailed profile plans, contour maps, and 3D views constructed from the collected data which include a series of potential exploration targets.

9.4 Grids and Survey

The coordinate system in use for the deposits is UTM Zone 19S, PSAD-56 datum.

The topography used was from a detailed aerial survey of the planned plant site area using a scale of 1:1,000 and 1 m contour spacing, prepared by Fugro Interra S.A. (Fugro) for Capstone in April 2012.

During 2020, Aguayo TCM implemented a topographic network in the Plant area, increasing the detail for this specific zone. In 2021, a unified topographic network that covers the entire project area with precision standards suitable for the construction of engineering works was consolidated. In this latter work, the topographic network was expanded to cover the study area of 5,700 hectares. For the survey, a DJI drone, Phantom 4 RTK model, with a 20MP RGB sensor, mechanical shutter, and dual-frequency GPS with a time-event marker for photographs, was used. These characteristics make it ideal for flights over small but challenging terrain. The result was a topography with contour lines every 0.5 metres.

For the resource estimate, SLR examined the data and observed a significant enhancement in resolution within the drilled areas. However, it was noted that the LiDAR data did not encompass the entire block model area, specifically in the northern sector of Iris Norte. To address this, SLR integrated the new data from 2021 with the older topography dataset from 2012, ensuring coverage of the entire model. The results were validated against the previous topography to detect and rectify any unexpected errors. Minor artifacts at the margins, stemming from the data merge, were corrected. Additionally, two surfaces were generated to encompass the SD-IN and ES block model extents.

In 2023, a topographic survey of the tunnels and workings of Estrellita tied to the reference points of Santo Domingo on 3 fronts of the area was carried out by Aguayo TCM. For georeferencing, 3 mobile GNSS LEICA GS14 units and a TOPCON HIPER V GNSS were used, with the latter serving as the base at Vertex SD28 of the Geodetic Network of the Santo Domingo Project. Given this setup, geodesy of the location was conducted in reference to the SIRGAS FUGRO network of Santo Domingo Epoch 2002.0, with vertical component correction applied to the Santo Domingo Project by FUGRO.

9.5 Petrology, Minerology, and Research Studies

Detailed petrography and mineralogy studies have been completed on selected areas within the property. These studies have been completed to identify and quantify ore and gangue minerals and for the descriptions of textural variations in several rocks. Modal analysis studies (QEMSCAN) were performed on various mineralization types at Santo Domingo Sur to determine mineral species and their compositions for recovery tests and determining grinding parameters. Additional QEMSCAN analysis were carried out between 2021 and 2022, from 140 samples obtained for Cu and Fe variability samples in sulphide and mixed zones, and in 29 samples in the oxide zones.

Two theses have been completed on the Santo Domingo deposit:

- Daroch, G., 2011: Hydrothermal Alteration and Mineralization of the Iron Oxide (Cu–Au) Santo Domingo Sur Deposit, Atacama Region, Northern Chile: unpublished M.Sc. thesis, University of Arizona, Tucson, Arizona, United States, 90 p

- Duran, M., 2008: Paragenesis of the Santo Domingo Sur Iron Oxide–Copper–Gold Deposit, Northern Chile: Unpublished M.Sc. thesis, Queen's University, Kingston, Ontario, Canada, 100 p.

9.6 Exploration Potential

IOCG style mineralization at Santo Domingo occurs over a large area of approximately 8 km wide and 6 km long. Limited drilling and geophysics suggest Santo Domingo has potential for additional sulfides in the vicinity of the Santo Domingo and Iris Norte resource extents, and at depth, specifically along a sparsely drilled mineralized volcaniclastic sequence that extends over 1.5km from Santo Domingo until Iris Norte area.

Copper oxide mineralization has been identified in Santo Domingo, Iris Norte and Estrellita deposits above the sulphide orebodies but has not been specifically targeted to date. The current drilling and metallurgical data suggest oxide mineralization is relatively continuous and amenable for leaching.

The Estrellita satellite deposit contains both oxide and sulfides potential and the deposit remains open towards the west and east along their major E-W mineralized trend.

Further potential exists for iron mineralization without copper, which so far has been deemed uneconomic once an iron processing capacity has been built on the Property. The main iron potential is located around Iris Norte and the southern area of San Domingo Sur, where several magnetite-rich skarns of unknown size were identified.

9.7 Prospects/Exploration Targets

9.7.1 Santo Domingo – Iris Norte

The area between Iris Norte and Santo Domingo exhibits potential for near-surface Cu-Fe oxide and sulphide mineralization. The relatively low drilling density in this area does not give a clear indication of the continuity of the mineralization between the Iris Norte and Santo Domingo pits. This sector has a strike length of 500 m and 650 m wide and requires more drilling to understand the possible potential and to improve the geological understanding of the sector.

9.7.2 South of Santo Domingo

An area of approximately 500 and 400 m wide with volcaniclastic sequences interdigitated with limestone and cut by dikes was identified towards the south, beyond the limits of the currently recognized mineralized area in Santo Domingo. The characteristics of this sector are very similar to those of the center of the deposit, but on a smaller scale, with volcaniclastic sequences with iron and lesser copper mineralization. Additional drilling is required to better understand the geological contacts, geometry of the sector and test for economic mineralization in the area.

9.7.3 Deeper Mineralization.

The Santo Domingo Sur, Iris and Iris Norte deposits have been explored to a depth of approximately 350 m and mineralization remain open. Drilling below this level is very sparse, but deep drill holes at San Domingo Sur have intersected mineralization as deep as 650 m. The character and extent of deep mineralization has not been fully established and potential for additional mineralization exists.

The latest drilling program completed during 2021, confirmed the extension of the mineralized sequence of the deposit at depth with an extension of more than 1.5 km between Santo Domingo and Iris Norte deposit. While several drillholes showed low levels of copper, a notable discovery was the presence of high concentrations of magnetic iron and cobalt bearing pyrite that extended for more than 200 m thickness within the volcaniclastic sequence. These findings suggest substantial potential to expand the current resource through further drilling.

9.7.4 Santo Domingo/Iris Norte/Estrellita Oxides

Oxide mineralization has been recognized with drilling in the shallower portions and above the Santo Domingo, Iris Norte and Estrellita sulphide bodies. Mineralization is discontinuous, hosted in amygdules, fractures, veins and breccias within the volcanic andesites and as a replacement within volcaniclastic units. However, insufficient drilling and soluble copper analysis has been performed to define a mineral resource for the oxide mineralization. Uncertainty remains regarding the continuity and the quality of the oxide mineralization but a combined exploration target between 80 to 100 Mt between 0.3 and 0.4% total copper (CuT) is expected based on limited drilling information.

9.7.5 Estrellita Area

In the Estrellita area, several gently north-dipping, stratabound, iron oxide ±copper horizons up to 12 m thick occur in roughly the same 200 m stratigraphic interval, traced with drilling or extrapolated across 3 km of strike length. Additionally, hydrothermal breccias with specularite-rich matrix and chalcopyrite and pyrite intersect the mineralized horizons along east-west fault zone.

Mineralization in the area to the west of Estrellita is not fully delineated due to limited drilling and may extend for additional 300 m. Similarly, to the east of the orebody, mineralization narrows but only limited drilling has tested the continuity to the east.

10 DRILLING

10.1 Introduction and History

Between July 2003 and February 2022, a total of 700 drill holes (169,692 m) were completed over the Property as a whole. Of these, 160 are core drill holes (DDH), 117 are holes collared as reverse circulation (RC) and then finished as core holes, and 423 are RC holes. Thirty holes were drilled as twins. Most of the drill holes are vertical or near vertical, with 183 holes collared at a dip shallower than -80°. Drill hole lengths vary widely, but are typically in the range between 200 m and 400 m.

Geotechnical drilling was conducted by Far West Mining Ltd. (Far West) between 2006 and 2010 and comprised a total of 28 oriented core drill holes (26 with geotechnical core logging), representing more than 7,000 m of core. The 2010 geotechnical campaign (four drill holes totaling 1,158 m) was supervised by AMEC. During 2011–2012, additional drilling was conducted by Capstone to gather geotechnical data to complete slope calculations for the Santo Domingo Sur/Iris pit and the Iris Norte pit. The 2011–2012 geotechnical/hydrological drilling campaign was designed and supervised by AMEC and consisted of 17 bore holes, for a total of 2,375 m, for Iris Norte, Santo Domingo Sur/Iris, and the proposed TSF area. An additional 22 core holes totalling 849 m were completed in 2012 to support geotechnical studies in the proposed plant area (Table 10-1).

Table 10-1: Geotechnical Drilling

Year	Purpose	Drill Hole Type	Count	Total Length (m)
2006	Geotechnical	DDH	1	645.95
2009	Geotechnical	DDH	12	4,574.48
2010	Geotechnical	DDH	4	1,157.55
2010	Geotechnical	DDH	11	3,638.10
2011	Geotechnical	DDH	8	614.15
2012	Geotechnical	DDH	9	1,761.10
2012	Geotechnical	DDH	22	849.28

Condemnation drilling was conducted by Far West from late 2010 to early 2011 and by Capstone from 2012 to 2013 (Table 10-2). A total of 3,576 m in 13 RC holes were drilled in the proposed waste rock facility (WRF), process plant, and tailings areas. The condemnation drilling was in addition to 5,627 m in 20 past exploration drill holes that had been drilled within the boundaries of the proposed mine site installations (WRF and process plant areas).

Table 10-2: Condemnation Drilling

Year	Purpose	Drill Hole Type	Count	Total Length(m)
2010	Condemnation	RC	1	150
2011	Condemnation	RC	11	3,308
2012	Condemnation	RC	1	150
2013	Condemnation	RC	5	1,242

The drilling history of the Project is summarized by purpose, by hole type, and by area in Table 10-3.

Figure 10-1 provides a regional-scale collar location plan for this drilling.

Table 10-3: Drilling History by Purpose, Hole Type, and Area

	DDH		RC		RC-DDH		Total	Total Length
	Count	m	Count	m	Count	m	Count	(m)
2004	**-**	**-**	**21**	**6,510.0**	**-**	**-**	**21**	**6,510.0**
Estrellita Resource	-	-	5	1,174.0	-	-	5	1,174.0
Iris Norte Resource	-	-	5	1,500.0	-	-	5	1,500.0
Iris Metallurgy	-	-	1	300.0	-	-	1	300.0
Exploration	-	-	10	3,536.0	-	-	10	3,536.0
2005	**-**	**-**	**51**	**15,884.0**	**-**	**-**	**51**	**15,884.0**
SD Resource	-	-	38	12,328.0	-	-	38	12,328.0
Iris Norte Resource	-	-	6	1,616.0	-	-	6	1,616.0
Estrellita Resource	-	-	5	1,346.0	-	-	5	1,346.0
Exploration	-	-	2	594.0	-	-	2	594.0
2006	**15**	**2,929.5**	**146**	**35,963.5**	**9**	**4,273**	**170**	**43,165.9**
SD Resource	-	-	45	12,167.0	8	4,057	53	16,224.4
Iris Resource	2	563.2	40	10,264.5	1	216	43	11,043.2
Iris Norte Resource	-	-	4	1,080.0	-	-	4	1,080.0
Estrellita Resource	13	2,366.3	56	12,152.0	-	-	69	14,518.3
Exploration	-	-	1	300.0	-	-	1	300.0
2007	**-**	**-**	**90**	**20,403.5**	**-**	**-**	**90**	**20,403.5**
SD Resource	-	-	29	9,577.5	-	-	29	9,577.5
Iris Resource	-	-	2	500.0	-	-	2	500.0
Estrellita Resource	-	-	47	7,650.0	-	-	47	7,650.0
Exploration	-	-	8	2,310.0	-	-	8	2,310.0
Hydrology	-	-	4	366.0	-	-	4	366.0
2008	**-**	**-**	**41**	**10,856.0**	**-**	**-**	**41**	**10,856.0**
SD Resource	-	-	7	1,978.0	-	-	7	1,978.0
Iris Resource	-	-	4	1,426.0	-	-	4	1,426.0
Iris Norte Resource	-	-	18	5,298.0	-	-	18	5,298.0
Estrellita Resource	-	-	6	956.0	-	-	6	956.0
Hydrology	-	-	6	1,198.0	-	-	6	1,198.0
2009	**-**	**-**	**-**	**-**	**13**	**4,757**	**13**	**4,757.1**
Iris Resource	-	-	-	-	1	267	1	267.0
SD Metallurgy	-	-	-	-	6	2,502	6	2,501.7
Iris Metallurgy	-	-	-	-	3	1,176	3	1,176.3

	DDH		RC		RC-DDH		Total	Total Length
	Count	m	Count	m	Count	m	Count	(m)
Iris Norte Metallurgy	-	-	-	-	3	812	3	812.2
2010	**12**	**3,626.0**	**29**	**6,444.0**	**6**	**2,349**	**47**	**12,418.7**
SD Resource	1	500.0	6	1,320.0	-	-	7	1,820.0
Iris Resource	-	-	6	1,140.0	-	-	6	1,140.0
Iris Norte Resource	-	-	3	650.0	-	-	3	650.0
SD Metallurgy	7	1,968.4	11	2,750.0	6	2,349	24	7,067.1
Exploration	-	-	2	434.0	-	-	2	434.0
Geotech - Pit Slopes	4	1,157.6	-	-	-	-	4	1,157.6
Condemnation	-	-	1	150.0	-	-	1	150.0
2011	**12**	**1,257.1**	**14**	**4,138.0**	**22**	**4,370**	**48**	**9,765.4**
SD Metallurgy	-	-	-	-	22	4,370	22	4,370.4
Exploration	-	-	3	830.0	-	-	3	830.0
Geotech	6	349.2	-	-	-	-	6	349.2
Geotech - Pit Slopes	2	265.0	-	-	-	-	2	265.0
Hydrology	4	642.9	-	-	-	-	4	642.9
Condemnation	-	-	11	3,308.0	-	-	11	3,308.0
2012	**34**	**2,914.4**	**9**	**1,020.4**	**42**	**8,325**	**85**	**12,260.3**
SD Metallurgy	-	-	2	290.0	37	7,279	39	7,568.8
Iris Norte Metallurgy	-	-	-	-	5	1,047	5	1,046.7
Exploration	2	204.1	-	-	-	-	2	204.1
Geotech	22	849.3	-	-	-	-	22	849.3
Geotech - Pit Slopes	9	1,761.1	-	-	-	-	9	1,761.1
Hydrology	1	100.0	6	580.4	-	-	7	680.4
Condemnation	-	-	1	150.0	-	-	1	150.0
2013	**1**	**90.0**	**13**	**1,862.5**	**-**	**-**	**14**	**1,952.5**
Condemnation	-	-	5	1,242.0	-	-	5	1,242.0
Hydrology	1	90.0	8	620.5	-	-	9	710.5
2014	**11**	**2,519.0**	**-**	**-**	**-**	**-**	**11**	**2,519.0**
SD Metallurgy	11	2,519.0	-	-	-	-	11	2,519.0
2015	**3**	**687.1**	**-**	**-**	**-**	**-**	**3**	**687.1**
SD Metallurgy	3	687.1	-	-	-	-	3	687.1
2019	**13**	**3,059.8**	**-**	**-**	**-**	**-**	**13**	**3,059.8**
SD Metallurgy	13	3,059.8	-	-	-	-	13	3,059.8
2021	**2**	**867.4**	**9**	**2,234.0**	**8**	**5,445**	**19**	**8,545.9**
Exploration	2	867.4	9	2,234.0	8	5,445	19	8,545.9
2021	**16**	**3,457.42**	**-**	**-**	**17**	**4,577.61**	**33**	**8,035.03**
SD Metallurgy	16	3,457.42	-	-	17	4,577.61	33	8,035.03

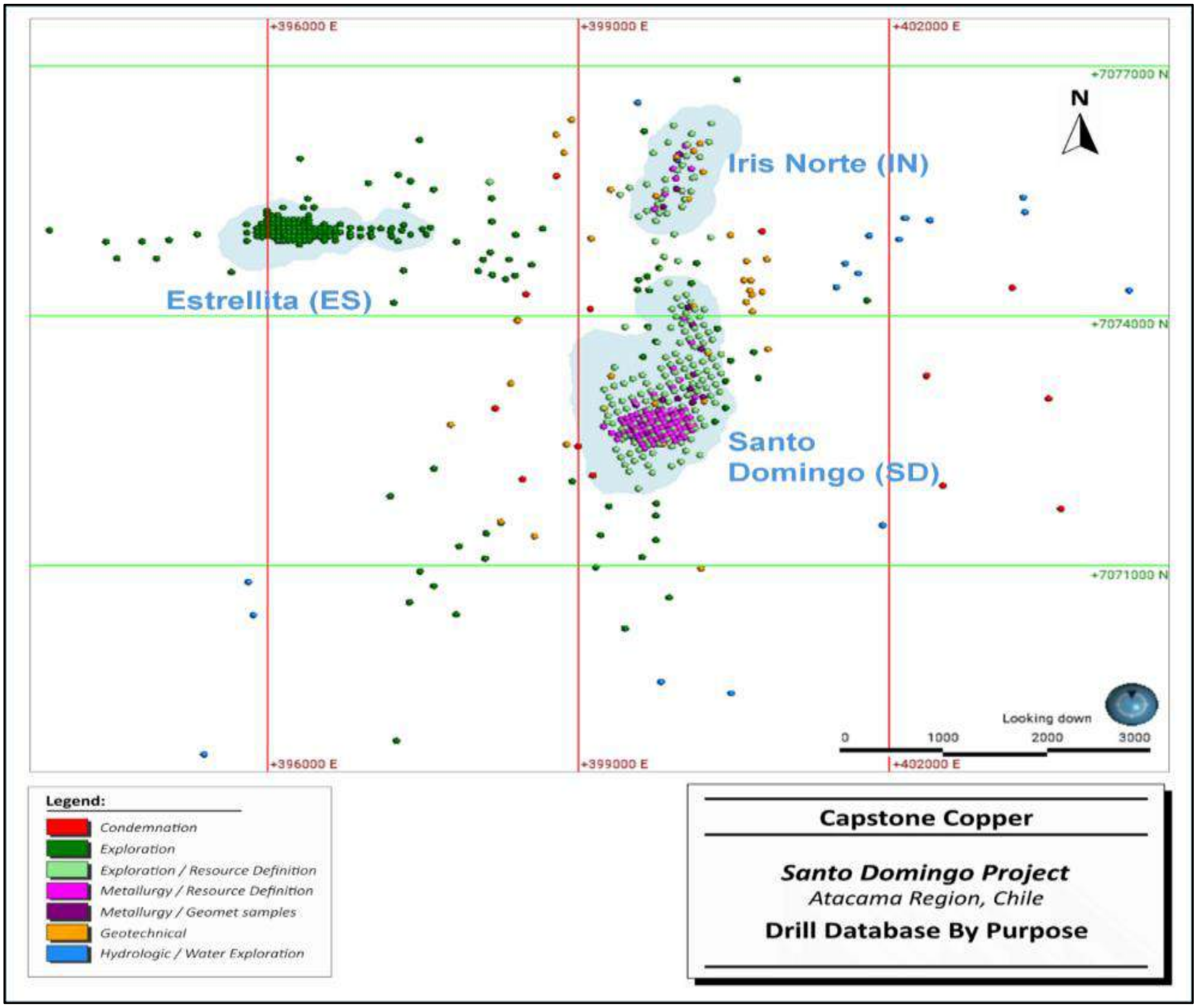
	DDH		RC		RC-DDH		Total	Total Length
	Count	m	Count	m	Count	m	Count	(m)
2021-2022	41	8,872.74	-	-	-	-	41	8,872.74
SD Metallurgy	41	8,872.74	-	-	-	-	41	8,872.74
Total	**-**	**-**	**-**	**-**	**-**	**-**	**700**	**169,692.97**

As outlined in Section 14, the 2022 Santo Domingo geological and oxidation models were divided into two separate models. The first enclosed Santo Domingo Sur and Iris (called SD now), and the second enclosed Iris Norte (termed IN or Area 2). Additional models for geology and oxidation were added on the west margin of the IN-block model volume.

The SD (or Area 1) block model area contains 352 drill holes (213 RC, 48 DDH, and 91 RC holes with core tails), completed on an approximate 100 m spacing and reducing to 50 m spacing in the centre of the deposit. The IN (or Area 2 and 4) block model area is defined by 57 drill holes (43 RC, six core, and eight RC holes with core tails) drilled on approximately 100 m spacing. The SD-IN drilled areas appear to be contained to the north and south, though there may be additional potential for mineralized volumes at depth.

The Estrellita area, including holes outside of the Mineral Resource volume, contains a total of 149 drill holes (137 RC and 12 core holes). The deposit remains open to the east and to the west. The zones are interpreted to be flat lying, hence down-dip extensions are unlikely. However, there is potential for additional mantos to occur below the presently drilled area.

Figure 10-1: Drill Database by Purpose



Source: Figure prepared by Capstone Copper, 2024.

10.2 Drill Methods

Over the exploration period, Chilean-based drill companies Harris y Cia., Major Drilling, Geo Operaciones, Captagua, and Terraservice have undertaken drilling operations. The majority of the RC drilling was conducted using a truck mounted Schramm Rotadrill, a centre return hammer and a 5.5 in. (13.97 cm) carbide button bit. Core drilling used various drill rig types. HQ-size core (63.5 mm diameter) was typically drilled to a depth of approximately 300 m, below which NQ-size core (47.6 mm diameter) was drilled. In addition, PQ-size core (85 mm) was used for metallurgical holes.

The drill programs were originally designed to target gravity and magnetic anomalies for mineralization of the Candelaria or Mantoverde IOCG style. Later programs consisted of core drill holes that were designed to provide information on geology, mineralization and structure, and material for metallurgical test work. RC and DDH drill holes were designed to tighten the drill spacing within the initial proposed mining areas and to provide sample material for metallurgical test work.

10.3 Collar Surveys

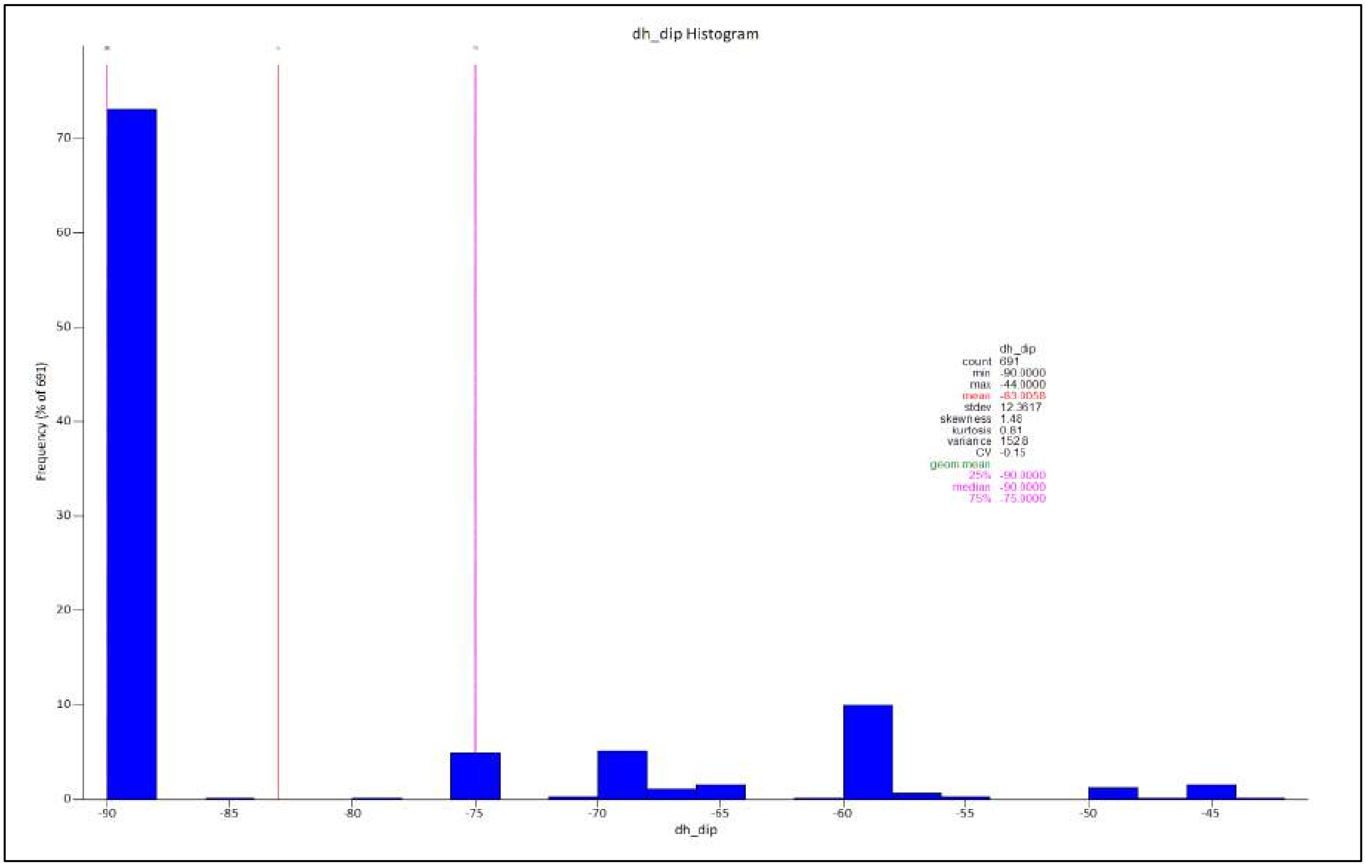
Drill collars were located using a differential global positioning system (DGPS). These coordinates can be considered to be accurate to within one metre or less (SLR, 2023). Relative elevations between holes in close proximity (such as at Santo Domingo Sur) were determined using a tight chain and clinometer. SLR validated the collar surveys against the topographic surfaces provided and found that the collars conformed to the topography within acceptable tolerances.

10.4 Downhole Surveys

Downhole surveying was conducted by Comprobe Ltda. (Comprobe) using a combination of gyroscope and accelerometre, with measurements taken every 10 m. During the drilling programs developed after 2019, the surveys were measured by the Terraservice company using the same type of equipment (SPT Gyromaster) and using the same methodology as in previous years. SLR notes that the downhole survey instruments were not affected by magnetic interference.

Figure 10-2 shows the statistics for the measured dip at hole collar, at the closing of the database considered by SLR for the mineral resources estimate (February 2021). The histogram represents the frequency of dip data at the collar from the drillholes and it shows that most holes were drilled vertically or close to vertical.

Figure 10-2: Downhole Survey Dips



Source: Figure prepared by SLR, 2023.

10.5 Logging Procedure

Drill cuttings and core were logged using a set of codes similar to those used for surface mapping. All geological data were entered digitally into summary logs. All digital data (collar and downhole surveys, assays, geotechnical and geological logs) were subsequently entered into an MS Access database for the Property. CSV tables were then exported for use in various software including Leapfrog and Vulcan, as required for presentation and interpretation purposes.

Core was placed into wooden core boxes by the drilling contractor at the drill site. The depth of each interval of core pulled was marked on a wooden block and placed in the core box. The core was then transported to a logging facility by personnel of the company at the time of drilling.

Geotechnical staff performed core photography and geotechnical logging at the logging facility. Geotechnical data recorded included recovery, rock quality designation (RQD), fracture frequency, rock alteration and weathering, structure type, angle and roughness, joint compressive strength (JCS) and bulk density. Cut core samples with a length of 15 cm or 20 cm were also collected and stored for subsequent triaxial and point load tests.

Geological staff then logged the geology, noting lithology, mineralogy, and other characteristics using the same codes employed for logging of the RC cuttings. Structural information was recorded for drill core only.

10.6 Core Recovery

Overall sample recoveries have been good throughout all drilling programs. SLR reviewed the recovery data for the database and found that the recovery was well within acceptable limits. Recovery was calculated as a ratio of the actual core length in the box to the drilled length indicated on the metre blocks. SLR noted that some intervals had recoveries greater than 100%, which is not realistic. These intervals, just over 100%, are, in SLR's opinion, probably due to slight gaps between pieces of core that caused inaccurate measurements. After normalizing all of these spurious values to 100%, the length-weighted average recovery was 96.9%.

SLR also noted the following:

- 63 intervals recorded recovery of 0%, with 18 intervals greater than 0.5 m;
- 96 intervals had values greater than 100% recovery;
- 312 out of the 15,410 measured intervals (0.2%) were below 50% recovery; 76 of these low recovery intervals were greater than a metre in length.

10.7 Sample Length and True Thickness

Most holes are vertical or near vertical because the mineralization on the Property tends to be horizontal or gently dipping. Approximately 25% of the holes included in the Mineral Resource estimate were drilled at angles shallower than -80°.

Inclined holes, particularly core holes, were drilled in order to establish the limits of mineralization at the edges of the deposits as well as to establish the structural framework at Estrellita, Iris, and Iris Norte.

Drill sections in Figure 7-5, Figure 7-7, and Figure 7-9. show the orientations of selected drill holes in relation to the mineralization at each deposit.

10.8 Exploration Drilling

Additional holes have been drilled to test other gravity and magnetic features in the Santo Domingo area (Figure 10-1) and intersected widespread but discontinuous copper and iron mineralization around the deposits. A full discussion of exploration drilling at Santo Domingo is presented in Section 9.

10.9 Drilling

Three programs of twin core drilling, both for confirmation purposes and to collect material for metallurgical testing, have been conducted by Capstone. These programs were carried out in 2010, 2014–2015, and 2019. All programs were carried out in the area of the proposed Santo Domingo Sur open pit. Discussions of twin drilling are provided in Section 12 and Section 14.

10.10 Conclusions

The QP is of the opinion that:

- Historical and Capstone drill programs were performed according to industry standards
- Collar and downhole survey work and results are acceptable
- Logging procedures are well designed and implemented
- Magsus measurements and instrument calibrations are well planned, researched, and understood
- Collar positions conform well to topography surfaces

The QP has not identified any drilling, sampling, or recovery factors that could materially impact the accuracy and reliability of the results.

10.11 Recommendations

To improve the quality of the drilling data, the QP recommends the following work to be performed:

- Perform additional twin drilling, especially to add sequential copper data to the model areas at a cost of approximately US$6.2M that would allow an improvement in the definition of the oxide, mixed and sulphide zones and the testing and extension of the mineralized zones.

11 SAMPLE PREPARATION, ANALYSES, AND SECURITY

During RC and core drilling campaigns from 2004 through 2021, Far West and Capstone Copper subsidiary Minera Santo Domingo (MSD) submitted 62,629 samples for analysis. Sampling included QA/QC steps as described in this section.

In the QP's opinion, the sample preparation, analysis, security, and QA/QC procedures at Santo Domingo are adequate for use in the estimation of Mineral Resources.

11.1 Sampling Methods

11.1.1 Geochemical Sampling

A total of 47 sediment samples were collected from drainages within and immediately peripheral to the Santo Domingo area. Most drainage channels in the area were sampled. Approximately 200 g of -106 µm material was collected from each sample site using an Endecott No. 140 sieve (or equivalent) and simple bubble plots of copper and gold in sediments were produced. The samples were analyzed by ALS laboratory, in Chile, for gold and a 27-element inductively coupled plasma (ICP) package.

11.1.2 Reverse Circulation Sampling

RC drill cuttings were blown into a cyclone and collected every two metres from top to bottom of each hole, regardless of lithology changes. This material was dumped directly into a riffle splitter with a bar separation of approximately one centimetre. Both parts of the initial split were reintroduced to the splitter and divided a second time to ensure adequate mixing of the entire sample. Half of this initial split was re-split and then split again. These three consecutive splits resulted in a final sample one-eighth the size of the initial complete sample.

All samples were sent for analysis except overburden and obviously barren bedrock intervals.

A 2 kg to 3 kg portion of the final split was bagged and ticketed with a unique assay number, ready to be sent to the laboratory for analyses. A second sample of 3 kg to 4 kg was collected from the other half of the final split and stored (covered) at or near the drill site.

11.1.3 Core Drilling

Samples for assay were marked at one metre and two metre intervals by technicians and subsequently adjusted by the geologist to correspond to major lithological contacts. For programs conducted prior to 2011, sample lengths were not less than 0.5 m, and most did not exceed two metres. The shortest and longest sample lengths in 2011–2012 were 0.7 m and 2.7 m, respectively, and most samples were two metres long. In 2019 and 2021, samples ranged from 0.6 m to 3 m, and most samples continued to measure two metres. Sampled intervals were cut in half along the drill axis using a diamond saw. Half of the sample was returned to the core box and stored at the core facility. The other half was bagged and shipped (via ALS courier) to the ALS laboratory at La Serena, Antofagasta or Copiapó, Chile, for preparation and analysis.

11.1.4 Metallurgical Sampling

Metallurgical sampling is discussed in Section 13.

11.2 Analytical and Test Laboratories

The primary analytical laboratory was ALS, and its facilities in La Serena, Chile and Antofagasta, Chile were principally used. Both facilities have ISO 9001:2008 accreditation and La Serena also has ISO 17025 accreditation. The ALS laboratories are independent of Capstone.

Occasional overflow capacity was directed to the ALS laboratory in Lima, Peru. The ALS laboratory in Copiapó Chile was also utilized in 2015, 2019 (only for one Cu-AA05 work order), and 2021. Both laboratories are independent. The Copiapó laboratory is ISO/IEC 17025:2017 accredited, and the Lima laboratory is 9001:2015 accredited.

The independent check laboratory used in 2011-2012 was Andes Analytical Assay Ltda. (Andes) in Santiago, which also holds ISO 9001:2008 accreditation.

The independent SGS laboratory in Santiago, Chile and ALS laboratories in Antofagasta and La Serena, Chile were utilized for density determinations.

11.3 Sample Preparation and Analysis

Upon arrival at the laboratory, the RC and core samples were organized, recorded, and prepared for analyses using ALS's Prep-31 process. This process consists of:

- Drying at 60°C
- Crushing (jaw crusher) to minus #10 ASTM (2mm) >70%Homogenizing and splitting to 1,000 g with a Jones splitter
- Storage of reject material (over 500 g)
- Pulverizing 500 g sample with a ring pulverizer to minus #200 Tyler (75 µm) >85%
- Storage in 250 g envelopes

Crushers and pulverizers are cleaned with compressed air after processing each sample and cleaned with crushed quartz at regular intervals. Following preparation, a 30 g split of the sample is delivered to the adjacent analytical laboratory where a 30 g fire assay with an ICP-atomic absorption spectroscopy (ICP-AAS) finish is performed.

All samples were analyzed for a suite of elements including copper, iron, cobalt, and sulphur using ICP methods. From 2007 to 2009 and 2011 to 2015, samples were analyzed using ALS procedure ME-ICP61, which is ICP-atomic emission spectrometry (ICP-AES) following four acid, total digestion (HF-HNO_3–$HClO_4$ acid digestion, HCl leach) covering 33 elements. Beginning in 2010, samples were analyzed using ALS procedure ME-ICP81, which included analysis of 17 elements by ICP-AES following sodium peroxide fusion. From 2019 to 2021, ALS procedure ME-MS61 including ICP-AES or ICP-mass spectrometry (ICP-MS) following four acid, total digestion was used for analysis of 48 elements with lower detection limits than ME-ICP61 or ME-ICP81.

Copper values over 10,000 ppm were assayed using ALS method Cu-AA62 or Cu-OG62, which involved total digestion and an ICP-AAS or ICP-AES finish. Gold content was determined using method Au-AA23 (30 g sample, fire assay with an ICP-AAS finish). These analytical procedures align with typical industry practices.

The ME-ICP61 protocol was recognized as understating the iron content, particularly in high grade samples. The upper limit for ME-ICP61 is 50% Fe, which resulted in a downward bias in the block model iron grades in previous resource estimates. From the 2010 program onwards, the ALS ME-ICP81 protocol was implemented, which incorporated a more aggressive digestion (peroxide fusion) and had a 70% Fe upper limit. A total of 7,401 previously drilled samples were submitted for re-analysis using ME-ICP81 in 2009, including all samples over 15% Fe inside the existing block model for which sample material was still available.

Soluble copper analysis was conducted at ALS in Chile on 5,762 samples from oxidized areas, particularly in the upper mineralized areas, from 2005-2019 drilling. The assay protocol included the ALS Cu-AA05 method for analysis of non-sulphide copper by dilute sulphuric acid leach and AAS finish on a 1 g sample.

The QP is of the opinion that the sample preparation and analytical techniques are consistent with industry standards and that Capstone has taken appropriate steps to ensure that the analytical methods are suitable for the style of mineralization at the Project.

Table 11-1 presents a summary of the different analytical methods used throughout the life of the project.

Table 11-1: Assay Method Summary

Element	Analytical Method	Method Limits	Years used	Description
Copper	ME-ICP61	1-10,000 ppm	2004-2009, 2011-2015	0.25 g is digested with perchloric, nitric, hydrofluoric, and hydrochloric acids. The residue is leached with dilute hydrochloric acid and diluted to volume. This solution is analyzed by ICP-AES with results corrected for spectral interferences.
	ME-ICP81	0.002-30%	2009-2012	Sodium peroxide fusion of a 0.2 g sub-sample with ICP-AES.
	ME-MS61	0.2-10,000 ppm	2019-2021	4-acid digestion of a 0.25 g sub-sample followed by either ICP-MS or ICP-AES instrument finish. This method has lower detection limits than ME-ICP61.

Element	Analytical Method	Method Limits	Years used	Description
Overlimit Copper	Cu-AA62	0.001-50%	2004-2008	Overlimit copper analysis for samples >10,000 ppm. 4-acid digestion of a 0.4 g sub-sample with an ICP-AAS instrument finish.
	Cu-OG62	0.001-50%	2014-2021	Overlimit copper analysis for samples >10,000 ppm. 4-acid digestion of a 0.4 g sub-sample with an ICP-AES instrument finish.
Gold	AuAA23	0.005-10 g/t	2004-2021	Only samples with more than 0.5% Cu are typically analyzed for Au. 30 g of pulp is fused with a mixture of lead oxide, sodium carbonate, borax, silica, and other reagents as required, inquarted with 6 mg of gold-free silver and then cupelled to yield a precious metal doré bead. 0.5 mL of dilute nitric acid is added to the doré bead to remove Ag, then 0.5 mL hydrochloric acid is added to decompose the Au, with each step including heating via microwave oven. The digested solution is cooled, diluted to a final volume of 4 mL with de-mineralized water, and analyzed by ICP-AAS against matrix-matched standards.
Iron	ME-ICP81	0.05-70%	2010-2021	Plus 2009 re-analysis of samples within the resource domains from previous drilling. Sodium peroxide fusion of a 0.2 g sub-sample with ICP-AES.
Cobalt	ME-ICP61	1-10,000 ppm	2004-2009, 2011-2015	4-acid digestion of a 0.25 g sub-sample with ICP-AAS finish.
	ME-ICP81	0.002-30%	2009-2012	Sodium peroxide fusion of a 0.2 g sub-sample and ICP-AES finish.
	ME-MS61	0.1-10,000 ppm	2019-2021	4-acid digestion of a 0.25 g sub-sample followed by either ICP-MS or ICP-AES instrument finish. This method has lower detection limits than ME-ICP61.
Sulphur	ME-ICP61	0.01-10%	2004-2009, 2011-2015	4-acid digestion of a 0.25 g sub-sample and ICP-AAS finish.
	ME-ICP81	0.01-60%	2009-2012	Sodium peroxide fusion of a 0.2 g sub-sample and ICP-AES finish.
	ME-MS61	0.01-10%	2019-2021	4-acid digestion of a 0.25 g sub-sample followed by either ICP-MS or ICP-AES instrument finish. This method has lower detection limits than ME-ICP61.
Non-sulphide Copper	Cu-AA05	0.01-10%	2005-2021	Samples from oxidized areas used for non-sulphide Cu analysis. At room temperature, 1.0 g of pulp in 5% sulphuric acid is shaken using an automatic shaker for 60 minutes. The solution is subsequently filtered into a flask ensuring the residue is well washed with warm water. The filtrate is diluted to volume with water. Non-sulphide Cu content is measured by ICP-AAS.

Notes: All samples were prepared at ALS in Chile, then analyzed in Chile or periodically in Lima, Peru to take advantage of analytical capacity. ALS in Antofagasta was used in 2007-2008, 2012, 2015 and 2019, in La Serena in 2007-2008, 2009-2013 and 2021, in Copiapó in 2015, 2019 (only for one Cu-AA05 work order), and 2021. The selection of the facilities depended on capacity and turnaround times, as well as proximity to the Project site.

11.4 Density Determinations

Density determinations were performed by Far West personnel on 2,292 core samples from 13 drill holes for different lithologies in the Estrellita deposit and 2,015 determinations from 11 drill holes in the Santo Domingo deposit.

Far West made direct measurements on core samples using the water displacement method and calculated specific gravity using the formula:

$$Specific\ Gravity\ (S.G.) = Mair/(Mair\text{-}Mw)$$

Where:

Mair = weight of the dry sample in air

Mw = weight of the sample in water

For the Santo Domingo deposit, a total of 498 determinations from 46 holes were completed at ALS in La Serena, Chile and the 219 determinations from 20 holes were completed at the SGS laboratory, in Santiago Chile. Both laboratories used a paraffin coating on the core and the water displacement method. For Iris Norte, a total of 37 determinations from 13 drill holes were completed at ALS in La Serena, Chile. The bulk density formula was:

$$Bulk\ Density = A\ /\ (C - [(B\text{-}A)/Dwax])$$

Where:

A = Weight of Sample (g)

B = Weight of Sample covered with paraffin wax (g)

C = Volume of Displaced Water (cm^3)

Dwax = Density of paraffin wax (g/cm^3) = 0.89 g/cm^3

In addition, 295 determinations were completed on RC samples from two Santo Domingo drill holes at ACME Analytical Laboratories (ACME) using the pycnometer method on pulps prepared from the samples. Another 219 determinations were completed by the pycnometer method at the SGS laboratory, in Santiago, Chile, on the same samples from 20 drill holes that were used for density determinations by the water displacement method.

The specific gravity (SG) was determined using the following formula:

$$SG = Ws/Wds\ x\ SGs$$

Where Ws is the weight of the sample; Wds is the weight of the displaced solvent; and SGs is the specific gravity of the solvent. The most common solvent is acetone, although methanol can also be used. A summary of the SG data by lithological unit is included in Table 11-2.

Table 11-2: Density Data by Lithology – Santo Domingo

Lithology	Count	Water Displacement (g/cm³)	Count	Pycnometer (g/cm³)
Andesite	1,911	2.83	74	2.96
Andesite Tuff	580	3.00	102	3.09
Basalt	45	2.84	-	-
Breccia	17	3.09	7	3.74
Carbonate-rich	12	2.97	-	-
Diorite	37	2.82	10	2.85
Dyke	26	2.63	44	2.73
Faulted Rock	69	2.92	-	-
Iron Oxide – Magnetite	258	3.64	17	3.59
Iron Oxide – Mixed	329	3.47	39	3.69
Iron Oxide – Specularite	322	3.44	14	3.50
Limestone	109	2.72	-	-
Manto	3	4.60	3	4.82
Overburden	2	3.05	-	-
Sedimentary	525	2.88	5	3.11
Tuff	114	3.04	32	3.28
Volcaniclastic	167	3.30	167	3.61

SG determinations specifically performed as part of infrastructure studies are not included in the tabulations above.

11.5 Magnetic Susceptibility

Magnetic susceptibility (magsus) is the degree to which a substance can be magnetized. Measurements of magsus can be used as a proxy for the magnetite content of a sample given the ferromagnetic character of this mineral.

The iron mineralization at SD-IN is predominantly magnetite, which can be recovered by low intensity magnetic separators (LIMS), and hematite, which generally cannot be recovered by LIMS. Consequently, the assays for total iron do not provide a basis for estimation of the recoverable iron component.

Starting in 2007, the Santo Domingo project began to record magnetic susceptibility on drill and rock samples to evaluate the potential for magnetic iron recovery as magnetite (Daroch, 2020b). The instrument used at the time was a GMS-2 handheld unit from Fugro.

In 2008, a bulk sample was collected by blending drill cuttings from a number of Santo Domingo Sur and Iris drill holes. This sample was subject to bulk flotation to remove the sulphide components and the tailings from this process were subject to iron recovery testing. The results of the test work indicated that LIMS would produce a good quality magnetite iron concentrate. Davis Tube (DT), Satmagan, and magsus tests were conducted on a set of 22 sub-samples from the bulk composite. A strong linear relationship was observed between the magnetic susceptibility and both

Satmagan and DT mass recovery readings. Capstone expanded the program to confirm the observed relationship and develop a reliable regression equation relating magnetic susceptibility readings to mass recovery.

During Q1 2009, Far West took magsus measurements on samples available from previous drilling (2005-2009) for use in estimation of mineral resources. Field personnel took measurements from core stored in the Diego de Almagro core yard using the Fugro GMS-2 instrument, supervised by Santo Domingo geologists. Far West excluded samples containing less than 15% Fe on the first pass to reduce the number of measurements needed and to meet the deadline for inclusion in the estimation of mineral resources that year. In drill programs after 2009, all samples within mineralization domains were measured from infill drilling and all mineralized samples from other drilling were measured.

Measurements were completed on coarse sample laboratory rejects, crushed to 70% less than 2 mm size stored in plastic bags. Four readings were taken on every sample; the average was used in the data for the estimation of mineral resources. If a significant deviation between readings occurred, the measurements were repeated.

Where no coarse laboratory rejects were available, measurements were taken on pulp samples. The laboratory prepared a pulp for every drill sample. Pulps generally weighed between 100 g and 300 g and were crushed to 85% less than 75 µm (i.e., #200 mesh), much finer than the coarse laboratory rejects. Pulp samples showed lower absolute magsus values than coarse rejects, therefore, a correlation factor (Factor A) was developed using measurements from 798 pulp and coarse reject samples and subsequently applied to make these two subsets consistent (Wood et al., 2020, page 12-4). The factor (Factor A) used to adjust the pulp dataset was:

$$MS_{coarse\ reject}=(-0.00001\ x\ MS_{pulp}{}^{2})+2xMS_{pulp}-132.08$$

Where: MS = magnetic susceptibility reading average x 100

During the 2010 drill program, magsus readings were collected only in coarse laboratory reject samples using the Fugro GMS-2 instrument. Several twin holes were drilled to acquire magsus data in areas for where sample material was no longer available for testing.

During the 2011-2012 drill program, Fugro's GMS-2 was replaced with a new instrument, KT-10 V2 from Terraplus.

During 2021, two new KT-10 V2 devices were acquired from Terraplus, these were used to take readings of the coarse rejects from the geometallurgy drilling campaign.

To February 2022, a total of 32,243 magsus readings were collected for the Project, of which 2,078 (approximately 7.5%) were taken from pulps as no coarse reject material was available. Drill programs in 2015 and 2019 included only twin holes for metallurgical purposes, therefore, their magsus was not used in the Mineral Resource estimate. In 2021, an additional 2,934 measurements were taken from 2006 through 2012 coarse rejects, focused on samples in andesite units. As of June 2021, there were 27,742 magsus measurements available for use in the estimation of the mineral resource.

11.5.1 Instrumentation Calibration and QA/QC

From Q4 2009 until September 2011, as part of a quality assurance and quality control (QA/QC) effort, Far West carried out calibration tests on a set of 13 reference samples taken from reject material at Santo Domingo to confirm the performance of the GMS-2 magnetic susceptibility metre. Results from these calibration tests were plotted in chronological order to check for deviation.

By the end of the 2010 program, it was noted that some reference samples, in particular the higher ranges at 40,000 and 70,000 and above (as measured in SI units), were trending downwards through time. Although most measurements for Santo Domingo were in the range of 2,000 and 35,000, the GMS-2 instrument was replaced.

A new instrument, KT-10 V2 from Terraplus, was acquired and all samples from the 2011-2012 drill program (hole 4a3-10-399DD and beyond) were measured with the new instrument. Magsus readings from the new instrument were relatively higher compared with GMS-2 measurements. A set of 552 samples selected from 10 different drill holes, covering a wide range of magnetic susceptibility, were measured to develop a regression line between the GMS-2 and the new KT-10 V2 instrument (Figure 10 3). The results obtained were used to estimate a factor (Factor B) to correlate magsus data between the old and new magsus datasets:

$$MS=(-0.000001373 \text{ x } MS^2)+(0.8363xMS)$$

Where: MS = magnetic susceptibility reading average x 100

During the 2011-2012 drill program, a total of 191 pulp samples were submitted to ALS Chemex in Perth, Australia, for percent magnetite analysis under laboratory-controlled conditions. Capstone reported a correlation factor of 0.943 between the average of four magsus readings and percent of magnetite (Wood et al., 2020, page 11-3).

During the metallurgical drilling programs in 2015 and 2019, magsus measurements were carried out with a new Terraplus model KT-10 S/C V2, with similar brand and specifications to the 2012 model. A total of 223 control samples from four drill holes were re-measured with the KT-10 S/C V2 instrument to confirm the previous relationship (Factor B) used to adjust the new dataset with the historic database. Results were almost identical to the 2012 model results, therefore, the same regression factor (Factor B) continued to be used for further magsus measurements.

Starting in November 2021, with the use of the new KT-10 V2 instruments, a remeasurement program of historical samples was implemented to verify the operation of the new equipment. No significant bias between measurements from the previous and new instrument was observed. Additionally, a QA/QC system was implemented for magnetic susceptibility measurements, where the calibration pads used for the susceptibility measurements are routinely measured as reference materials to ensure the correct operation of the equipment. The pads correspond to low and high grades of magnetic susceptibility.

Low Susceptibility Calibration Pad:

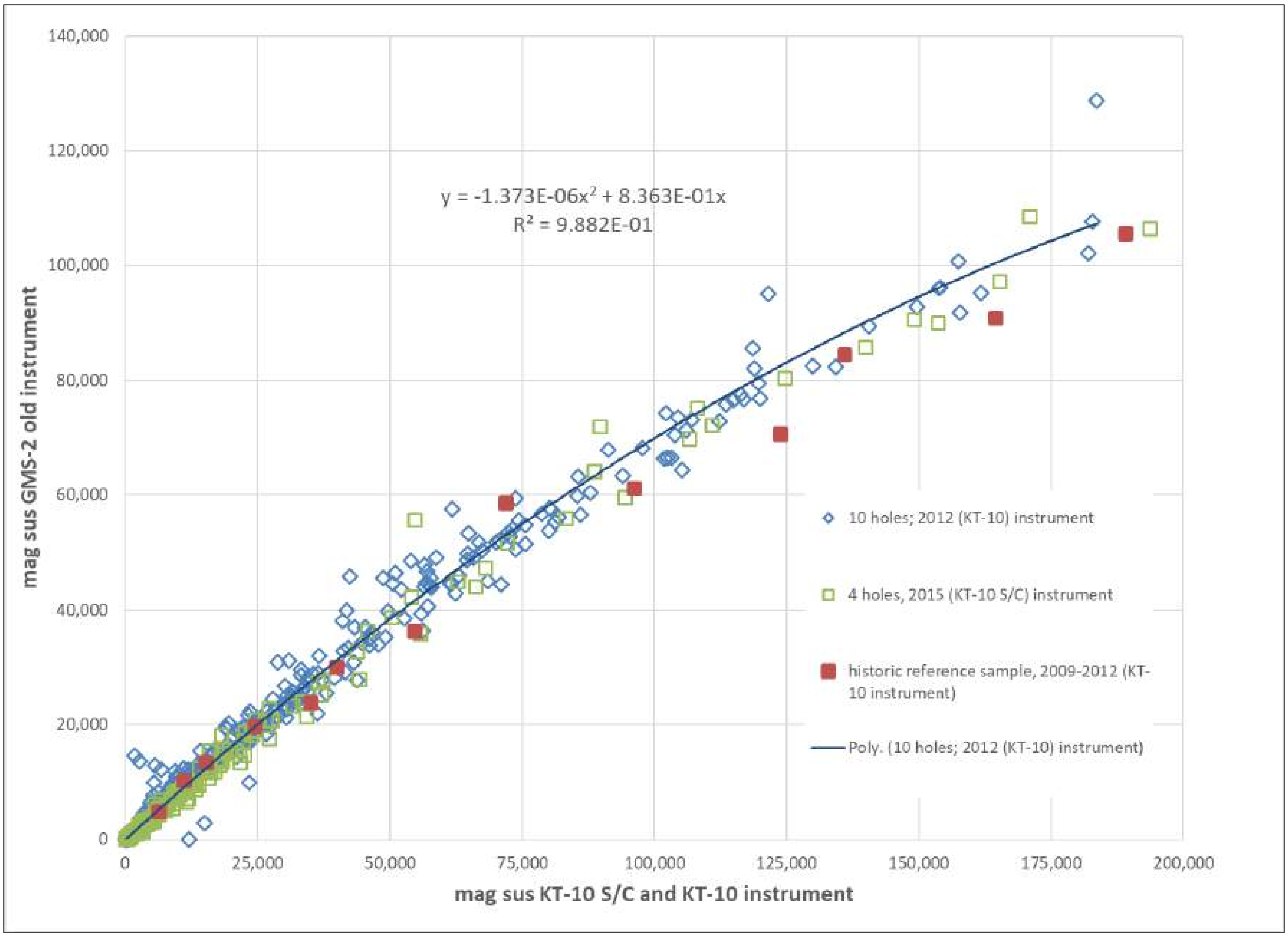
- Serial: 277/2021
- Nominal value: 29.3*10-3 SI

High susceptibility calibration pad:

- Serial: 260/2020
- Nominal value: 2135*10-3 SI

A pad was measured at least once within each group of 20 samples, resulting in an overall reading rate of 6%. When the pad reading fell outside of ±2.5% of the mean, the batch of 20 samples was re-read and all readings replaced in the database. Duplicate readings were completed once every hundred samples for a rate of 1%.

Figure 11-1: 2015 (KT-10 S/C V2) and 2012 (KT-10 V2) Magsus Metre vs. the Old (GMS-2) Magnetic Susceptibility Instrument



Source: Figure prepared by Capstone, 2020.

11.5.2 Magsus Field Procedures

As described above, magnetic susceptibility is read with a handheld instrument on drilled coarse laboratory reject samples (or, in one set of samples, pulps) returned by the assay laboratory. Samples, placed in a plastic bag, typically weigh three to four kilograms for RC drilling and seven to 10 kg for diamond core. Plastic bags of sample reject material from the laboratory are shaken to homogenize the material then laid flat on a table.

Magsus readings are carried out by technicians under geological supervision using KT-10 instruments according to the user guide. Four readings are taken per sample and recorded on paper sheets prepared by the geologist. If there is a significant deviation between readings, the readings are repeated until consistency is achieved between the four points. The data collected is later checked and input into an MS Excel sheet prepared by the geologist, where an average of the four readings is calculated. This average number is the final value entered in the field for that sample. Since 2021, two magnetic susceptibility pads, a low value and a high value, are used as QA/QC reference materials. A pad was read at least once within each group of 20 samples. When the pad reading fell outside of ±2.5% of the mean, the batch of 20 samples was re-read and all readings replaced in the data. Further, the MS Excel data is later reviewed and samples with large variances are re-read. Finally, magsus data is entered into the database and final values are calculated using a correction factor to normalize values between instruments. A total of 27,742 magsus determinations were made to the effective date of the Mineral Resource estimate.

Of the readings, 2,093 were conducted on pulps owing to the lack of remaining reject material. Readings on pulps routinely yield lower values than those taken on rejects.

For quality assurance, 191 pulp samples from the 2011–2012 drilling campaign were submitted to ALS in Perth, Australia, for percent magnetite analysis. Capstone reported that a correlation coefficient of 0.943 between the average of four magsus readings and percent magnetite was achieved.

The QP is of the opinion that the magsus data capture, handling, and analysis are of sufficient quality to support the Mineral Resource estimate.

11.6 Sample Security

The logging facility is fenced, locked when not occupied, and is secure. Samples at the logging facility are handled only by company employees or company designates (i.e., ALS personnel). Boxes of drill core and bags of RC chips are brought from the drill site to the logging facility daily by Capstone personnel or by contractors. Courier services, supplied through ALS, collect samples from the logging facility periodically and deliver them directly to ALS. When sending prepared sample pulps to its analytical laboratory in Lima, Peru, ALS uses tracked courier services.

Once leaving the logging facility, sample security could not be confirmed, however, Capstone states that, in virtually all cases, copper estimates in logged drill holes correlate well with analytical results.

The QP is of the opinion that sample security is adequate. It is recommended that some form of sample sealing procedure be implemented to deter any potential for tampering after the samples leave the logging facility.

11.7 Quality Assurance and Quality Control

Quality assurance (QA) consists of the designed and documented workflows and procedures necessary to demonstrate that the assay data has precision and accuracy within generally accepted limits for the sampling and analytical methods used, to ensure sufficient confidence in the results to support the Mineral Resource estimate. Quality control (QC) consists of procedures used to ensure that an adequate level of quality is maintained in the process of sampling, preparing, and assaying the drill core samples. In general, QA/QC programs are designed to prevent or detect contamination and allow analytical precision and accuracy to be quantified. In addition, a QA/QC program can disclose the overall sampling–assaying variability of the sampling method itself. The QC controls used comprise blanks, certified reference materials (CRM), and duplicate samples. QC sample insertion rates through the drill program history are outlined in Table 11-3.

Table 11-3: QC Sample Insertion Rates: 2004 to 2021

Sample	2004 to 2011		2012		2015		2019		2021		2004 to 2021	
	#	%	#	%	#	%	#	%	#	%	#	%
Drill Samples	48,447	4,340	675	790	8,377	62,629	-	-	-	-	-	-
QC Controls	6,921	14	736	17	132	20	221	28	975	12	8,895	14
Blank	1,483	3	232	5	38	6	45	6	175	2	1,973	3
CRM	2,521	5	293	7	48	7	104	13	542	6	3,508	6
Duplicates	2,937	6	211	5	46	7	72	9	258	3	3,524	6
Field Duplicate	1,839	4	142	3	0	-	24	3	86	1	2,091	3
Coarse Reject Duplicate	352	1			23	3	24	3	86	1	485	0.8
Pulp Reject Duplicate	439	1			23	3	24	3	86	1	572	0.9
Between-Lab Field Duplicate	307	1	-	-	-	-	-	-	-	-	307	0.5
Between-Lab Pulp Duplicate	-	-	69	2	-	-	-	-	-	-	69	1

The QA/QC protocols have remained largely consistent throughout all programs conducted by Far West and Capstone. Minor changes have been implemented by Capstone to accommodate issues and recommendations from past programs, and to include magsus measurements.

Prior to 2019, no CRMs were certified for cobalt or sulphur, although the CRMs were analyzed for both elements. SLR also notes that the blank material used prior to 2008 was found to contain significant amounts of sulphur, so blank QC sample results for sulphur from the earliest drilling could not be used to assess cross contamination for sulphur during sample preparation.

In 2022, Capstone finalized certification of the three matrix matched CRMs inserted since 2010. A review of past results from 2010 through 2015 show that the cobalt and sulphur analyses for the CRMs were typically within reasonable ranges, although some results during periods of bias or outside the acceptable ranges may warrant re-analysis, including the surrounding samples in areas critical to the Mineral Resource estimate.

In the QP's opinion, the QA/QC program as designed and implemented by MSD is adequate and the assay results within the database are suitable for use in a Mineral Resource estimate.

11.7.1 Blanks

The regular submission of blank material is used to assess contamination during sample preparation and to identify sample numbering errors. Capstone has used pulverized quartz and crushed quartz for blank material since 2008. From 2004 through 2007, Capstone had used powdered cement.

A blank sample is entered into the assay stream every 25th sample. The current database contains 1,973 blanks. In 2008, it was discovered that the blank material in use up to that time contained significant amounts of iron. The powered cement material is now noted to also contain significant amounts of sulphur. As a result, 867 pre-2008 blank results are considered unreliable to assess cross contamination for sulphur, and somewhat reliable to assess cross contamination for iron while remaining effective for copper, gold, iron and cobalt. The remaining 1,106 blank values from 2008 onwards were typically acceptable and yielded satisfactory results.

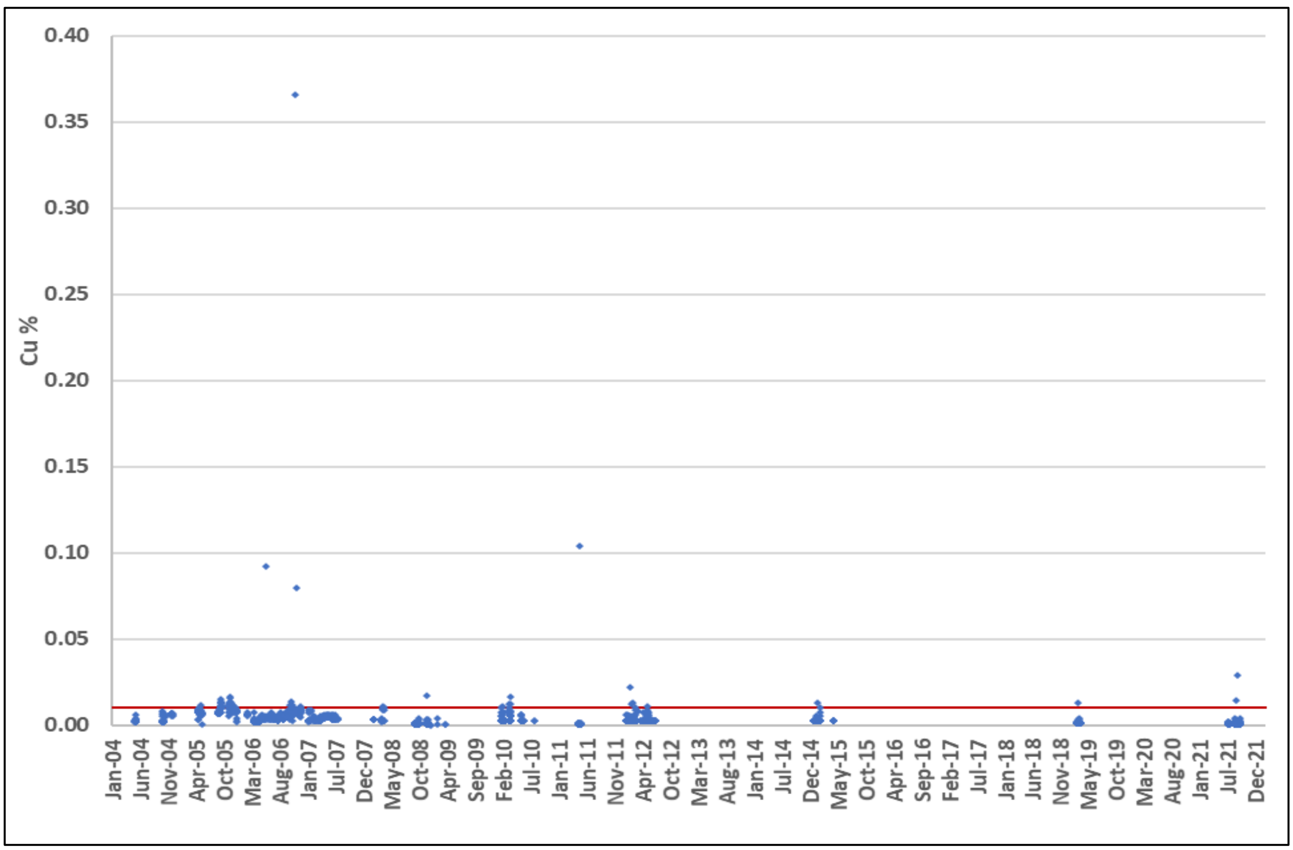
Capstone analyzed and prepared charts depicting the performance of the blank submissions. The QC protocols accept results returning up to 10 times the detection limit of the ore grade method as a pass, shown as a red line in Figure 11-2 through Figure 11-6 In the case of iron, the dashed red line in Table 11-4, denotes the blank is monitored for contamination relative to typical values only, as the quartz material used naturally contains iron. Table 11-4 presents a summary of the blank sample submissions by year. Figure 11-2 through Figure 11-6 show the performance of blank material for copper, gold, iron, cobalt, and sulphur, respectively. Results indicate a negligible amount of sample contamination. When a failure occurred within a batch of samples with no significant mineralization, no re-analysis was requested, however, the laboratory was contacted to review sample preparation procedures with staff.

Table 11-4: Blank Submission Summary

Year	Count	Blank Type	Laboratory
2004	61	FINE	ALS-LASERENA
2005	155	FINE	ALS-LASERENA
2006	36	FINE	ALS-ANTOFAGASTA
2006	413	FINE	ALS-LASERENA
2007	123	FINE	ALS-ANTOFAGASTA
2007	103	FINE	ALS-LASERENA
2008	26	COARSE	ALS-ANTOFAGASTA
2008	39	COARSE	ALS-LASERENA
2008	26	FINE	ALS-ANTOFAGASTA
2008	39	FINE	ALS-LASERENA
2008	1	FINE	CHECK LAB
2008	19	NOT SPECIFIED	ALS-ANTOFAGASTA
2008	23	NOT SPECIFIED	ALS-LASERENA
2009	2	COARSE	ALS-LASERENA
2009	2	FINE	ALS-LASERENA
2010	187	COARSE	ALS-LASERENA

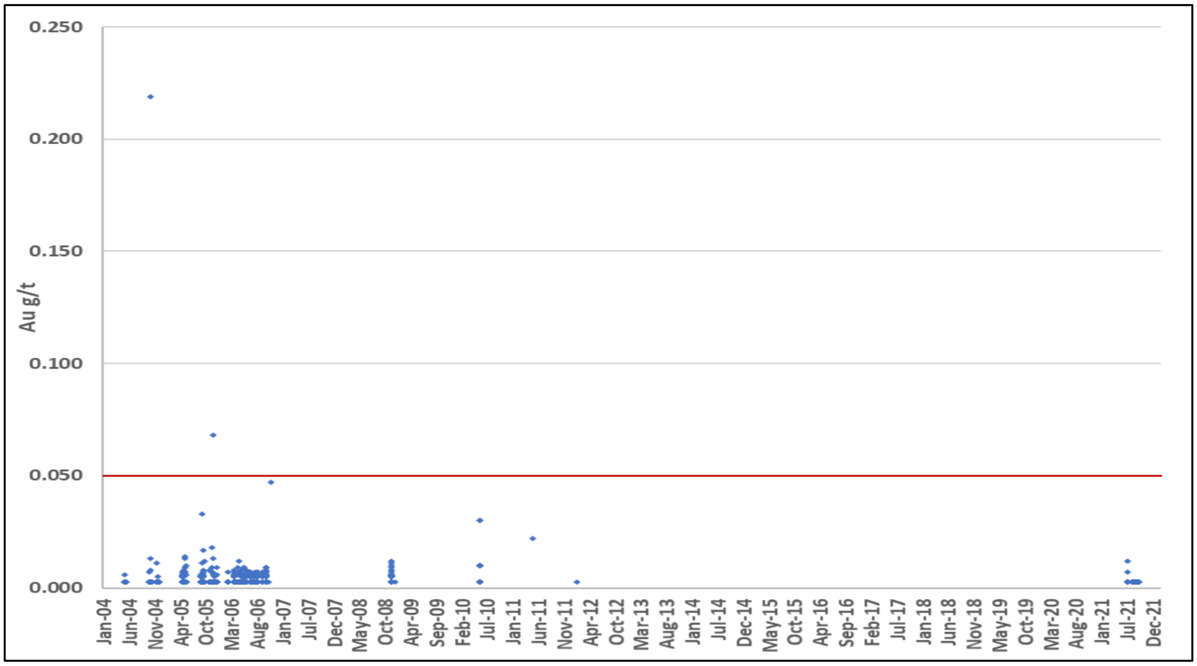
Year	Count	Blank Type	Laboratory
2010	14	COARSE	CHECK LAB
2010	160	FINE	ALS-LASERENA
2010	14	FINE	CHECK LAB
2011	16	COARSE	ALS-LASERENA
2011	17	FINE	ALS-LASERENA
2012	20	COARSE	ALS-ANTOFAGASTA
2012	98	COARSE	ALS-LASERENA
2012	15	FINE	ALS-ANTOFAGASTA
2012	99	FINE	ALS-LASERENA
2015	23	COARSE	ALS-COPIAPO
2015	15	FINE	ALS-COPIAPO
2019	24	COARSE	ALS-ANTOFAGASTA
2019	21	FINE	ALS-ANTOFAGASTA
2021	83	COARSE	ALS-COPIAPO
2021	92	FINE	ALS-COPIAPO

Figure 11-2: Blank Control Sample Assays Copper



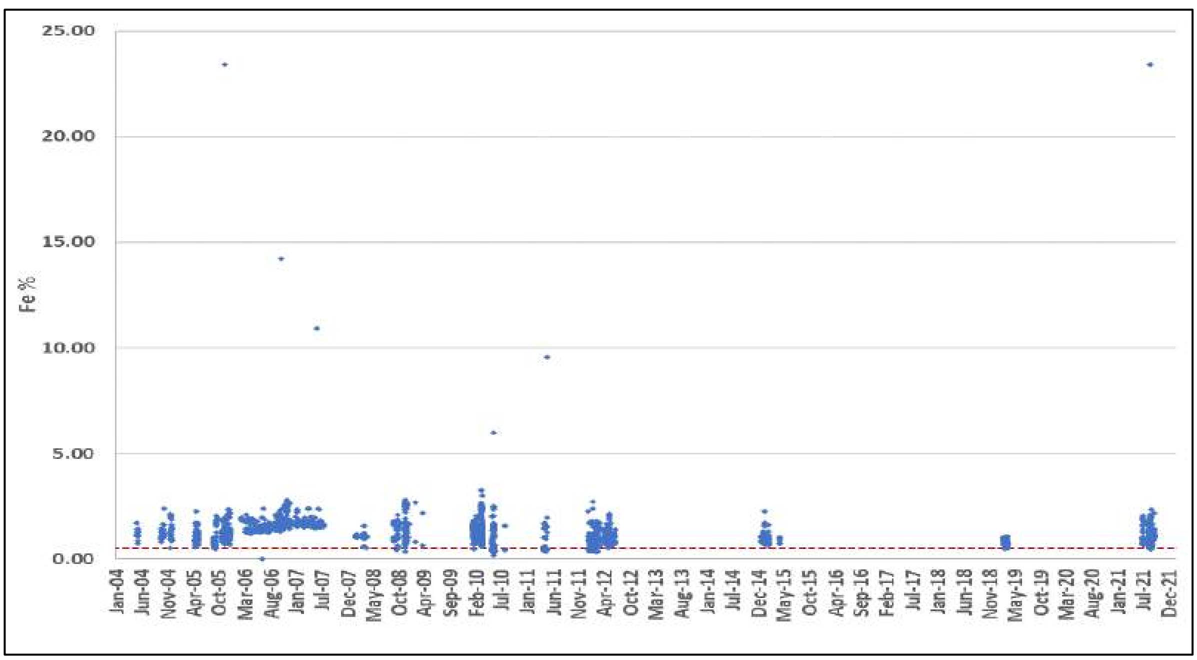
Source: Figure prepared by Capstone Copper, 2023.

Figure 11-3: Blank Control Sample Assays Gold



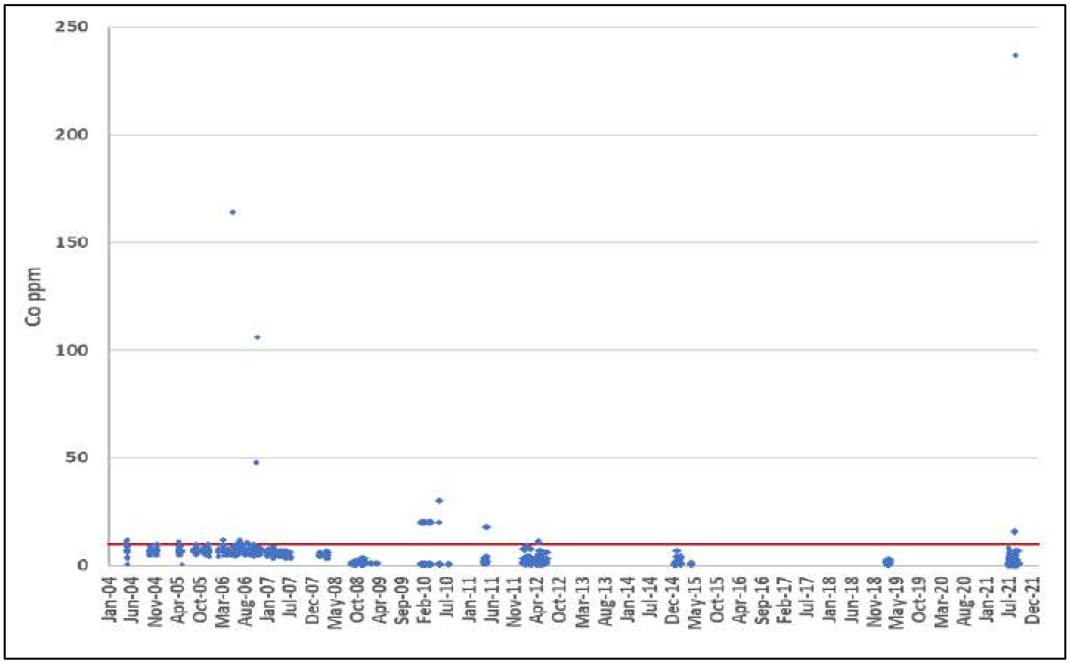
Source: Figure prepared by Capstone Copper, 2023.

Figure 11-4: Blank Control Sample Assays Iron



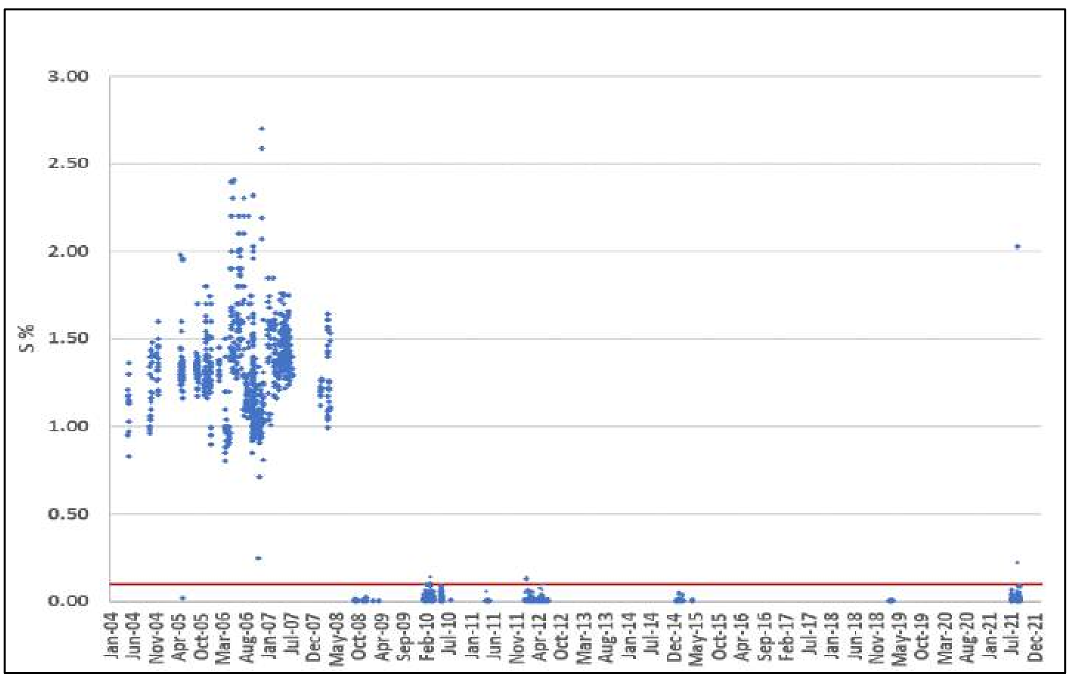
Source: Figure prepared by Capstone Copper, 2023.

Figure 11-5: Blank Control Sample Assays Cobalt



Source: Figure prepared by Capstone Copper, 2023.

Figure 11-6: Blank Control Sample Assays Sulphur



Source: Figure prepared by Capstone Copper, 2023.

11.7.2 Certified Reference Materials

Results of the regular submission of CRMs are used to identify issues with a specific batch of samples, and to detect long term biases associated with the primary assay laboratory. Capstone analyzed the results of the CRMs and plotted them in control charts, with failure rates, defined as assay values reporting more than three standard deviations (SD) from the expected value, and warning rates, defined as assay values reporting more than two SD, but less than three SD from the expected values.

CRMs were also added to the sample stream at a rate of one in 25. Over the history of drilling at Santo Domingo, at least 14 different standards were used, although the protocol since 2009 has been to use three: a high, medium, and low-grade standard certified by ALS for copper, iron, and gold (high and medium only) and, since 2022, certified by CDN Resource Laboratories Ltd. (CDN) for cobalt (high and medium only) and sulphur. These standards are matrix-matched to the mineralization at Santo Domingo. Four commercial standards used in 2019 and 2021 were added for CRM coverage for cobalt and sulphur. The database contains 3,511 CRMs analyzed at a wide range of grades. The range of CRMs used from 2004 through 2021 is summarized in Table 11-5.

SLR reviewed the performance charts for the CRMs, with examples shown in Figure 11-7 through Figure 11-11 and is of the opinion that there is no evidence of a persistent bias in the results. There is evidence of decreased accuracy and precision in assay results for some standards over certain periods. According to Capstone, the severity of these occurrences is not deemed to be critical to the use of the assay data for estimation of Mineral Resources. The QP concurs with this conclusion.

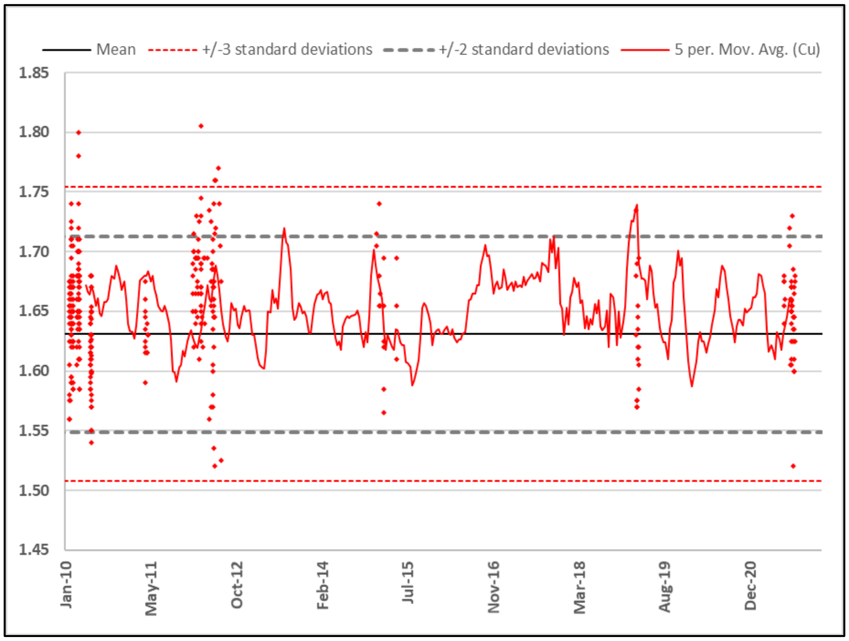
Table 11-5: CRM Performance Summary

CRM	Year	Certified Mean	1 Std Dev	Count	Performance Mean	Bias[3]
			Copper			
CDN-CGS-1	2004	0.596	0.029	31	0.59	-1.3%
CDN-CGS-2	2004-2006	1.177	0.046	247	1.18	0.6%
CDN-CGS-3	2004-2006	0.646	0.031	211	0.63	-2.6%
CDN-CGS-5	2004-2006	0.155	0.006	272	0.15	-2.3%
CDN-CGS-7	2006-2008	1.01	0.035	377	0.99	-1.8%
CDN-CGS-8	2006-2009	0.105	0.004	396	0.11	0.3%
CDN-CGS-91	2006	0.473	0.0125	53	0.49	3.1%
CDN-CGS-11	2006-2008	0.683	0.013	324	0.68	-0.6%
MSD-HIGH	2010-2021	1.631	0.041	354	1.65	1.3%
MSD-MED	2010-2021	0.472	0.01	448	0.47	-0.2%
MSD-LOW[2]	2010-2021	0.072	0.003	474	0.07	-1.2%
O520	2019-2021	0.293	0.008	152	0.30	1.0%
O521	2019-2021	0.607	0.015	86	0.61	0.4%
O522	2019-2021	0.916	0.026	39	0.92	0.5%
O523	2019-2021	1.72	0.038	14	1.71	-0.6%
O504b	2021	1.11	0.042	3	1.12	0.7%
O502b	2021	0.773	0.02	11	0.76	-1.2%
O501b	2021	0.26	0.011	19	0.26	0.5%
			Gold			
CDN-CGS-1	2004	0.53	0.068	31	0.55	3.1%
CDN-CGS-2	2004-2006	0.97	0.092	247	0.97	0.2%
CDN-CGS-3	2004-2006	0.53	0.048	211	0.54	1.0%

CRM	Year	Certified Mean	1 Std Dev	Count	Performance Mean	Bias[3]
CDN-CGS-5	2004-2006	0.13	0.02	272	0.14	4.8%
CDN-CGS-7	2006-2008	0.95	0.04	377	0.96	0.6%
CDN-CGS-8	2006-2009	0.08	0.006	396	0.08	6.0%
CDN-CGS-91	2006	0.34	0.017	53	0.35	2.6%
CDN-CGS-11	2006-2008	0.73	0.034	324	0.719	-1.5%
MSD-HIGH	2010-2021	0.299	0.022	354	0.29	-3.6%
MSD-MED	2010-2021	0.098	0.016	448	0.120	22.7%
O520	2019-2021	0.176	0.008	152	0.17	-1.3%
O521	2019-2021	0.376	0.019	86	0.37	-0.8%
O522	2019-2021	0.574	0.018	39	0.56	-1.6%
O523	2019-2021	1.04	0.027	14	1.02	-1.6%
O504b	2021	1.61	0.04	3	1.62	0.8%
O502b	2021	0.495	0.015	11	0.50	0.0%
O501b	2021	0.248	0.01	19	0.247	-0.4%
Iron						
MSD-HIGH	2010-2021	49.9	0.8245	354	49.9	0.1%
MSD-MED	2010-2021	30.9	0.622	448	29.3	-5.3%
MSD-LOW[2]	2010-2021	17.05	0.344	474	17.5	2.5%
O520	2019-2021	16.43	0.922	152	16.67	1.4%
O521	2019-2021	20.71	1.115	86	20.64	-0.3%
O522	2019-2021	24.63	0.998	39	24.67	0.2%
O523	2019-2021	28.9	1.402	14	28.44	-1.6%
Cobalt						
MSD-HIGH	2010-2021	353	11.5	354	365	3.5%
MSD-MED	2010-2021	194	5.5	448	185	-4.7%
MSD-LOW[2]	2010-2021	17.6	-	474	18.2	3.4%
O520	2019-2021	203	6	152	201	-0.8%
O521	2019-2021	386	14	86	386	0.0%
O522	2019-2021	550	9.5	39	555	0.9%
O523	2019-2021	728	28	14	731	0.4%
O504b	2021	20.9	1.62	3	22.1	5.6%
O502b	2021	20.2	1.89	11	20.4	1.0%
O501b	2021	15.8	1.39	19	16.4	3.9%
Sulphur						
MSD-HIGH	2010-2021	1.71	0.09	354	1.856	8.5%
MSD-MED	2010-2021	1.45	0.04	448	1.468	1.2%
MSD-LOW[2]	2010-2021	0.11	0.005	474	0.129	16.9%
O520	2019-2021	1.01	0.085	152	0.987	-2.3%
O521	2019-2021	1.8	0.089	86	1.814	0.8%
O522	2019-2021	2.5	0.103	39	2.560	2.4%
O523	2019-2021	3.82	0.149	14	3.874	1.4%
O504b	2021	1.31	0.061	3	1.333	1.8%
O502b	2021	0.95	0.025	11	0.962	1.3%
O501b	2021	0.354	0.028	19	0.358	1.2%

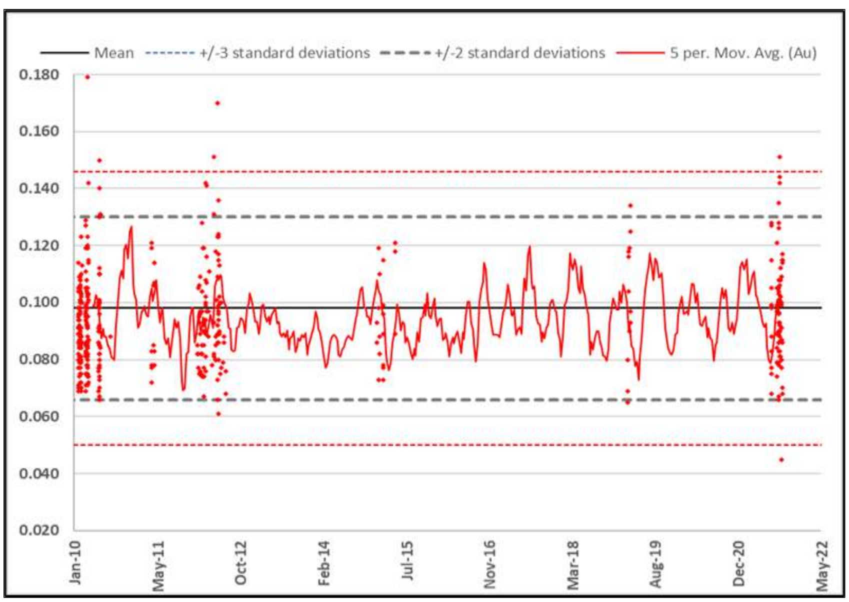
Notes: CDN labelled CRMs were sourced from, or certified by, CDN Resource Laboratories Ltd., Delta, BC, Canada; MSD labelled custom CRMs were certified for Cu, Au, Fe by ALS, La Serena, Coquimbo, Chile and for Co and S by CDN; and O labelled CRMS were sourced, or certified, by OREAS Ore Research & Exploration Pty Ltd., Bayswater North, VIC, Australia. 1. CDN-CGS-9 has a provisional value for Au. 2. MSD-Low has an indicated value for Co, a provisional value for S to be used as a guideline only and is not certified for Au. 3. Bias is calculated as (Performance – Certified Value)/Certified Value * 100%.

Figure 11-7: Example Control Chart: Copper MSD-High



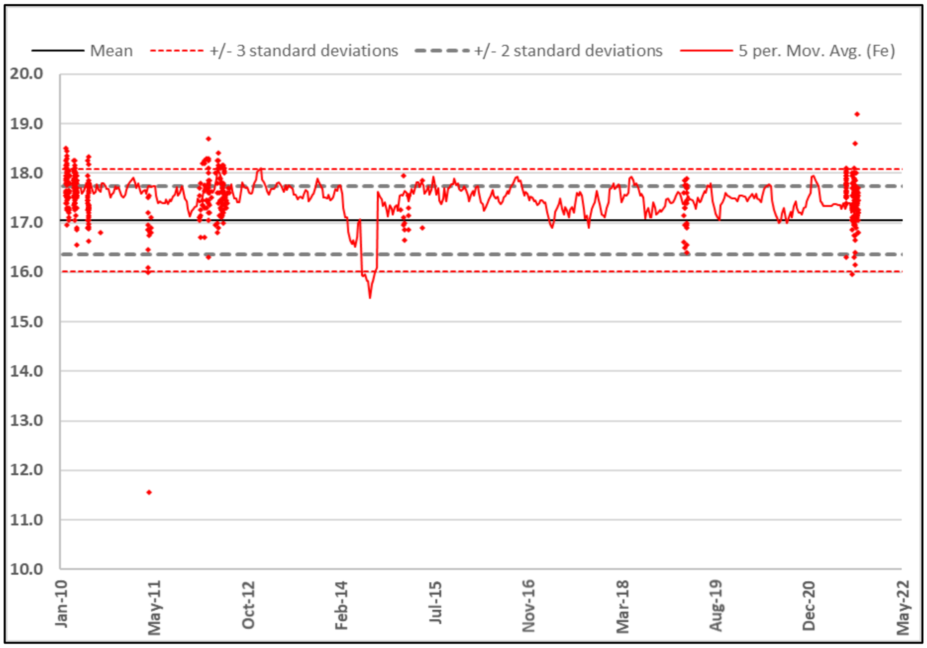
Source: Figure prepared by Capstone Copper, 2023.

Figure 11-8: Example Control Chart: Gold MSD-Med



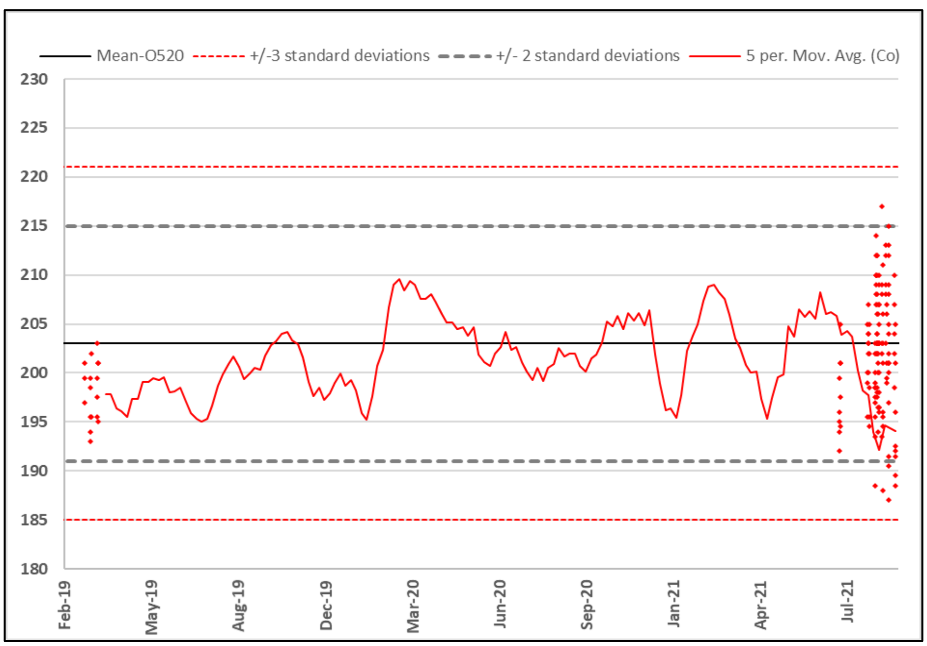
Source: Figure prepared by Capstone Copper, 2023.

Figure 11-9: Example Control Chart: Iron MSD-Low



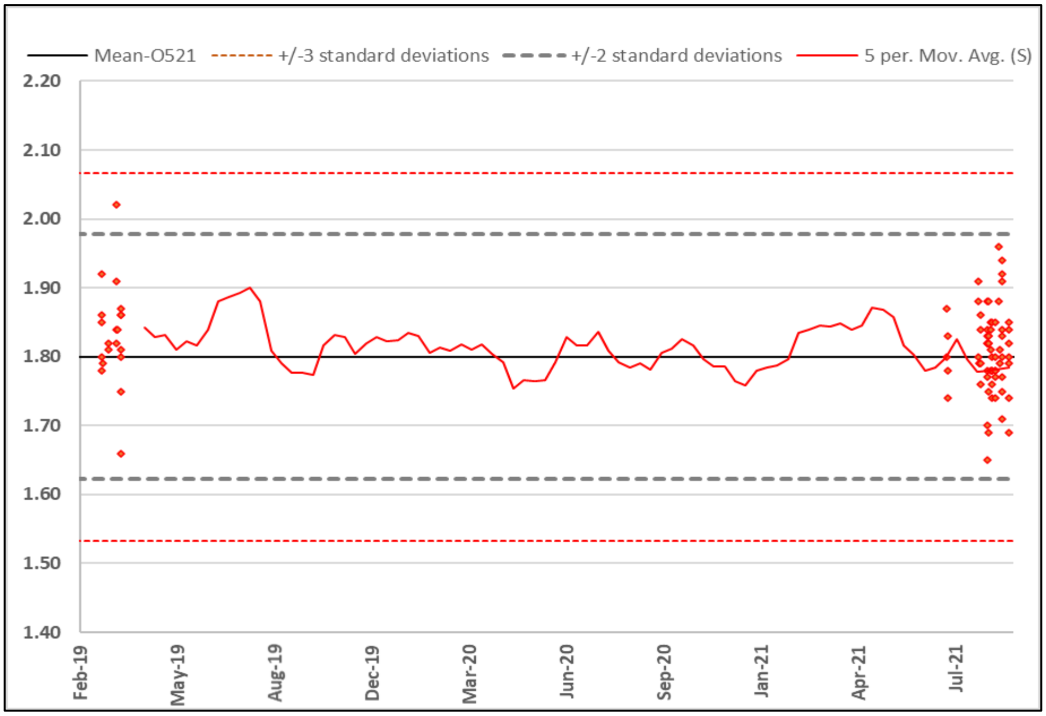
Source: Figure prepared by Capstone Copper, 2023.

Figure 11-10: Example Control Chart: Cobalt OREAS 520(O520)



Source: Figure prepared by Capstone Copper, 2023.

Figure 11-11: Example Control Chart: Sulphur OREAS 521



Source: Figure prepared by Capstone Copper, 2023.

11.7.3 Duplicates

Duplicate samples are inserted to monitor preparation and assay grade precision and variability as a function of sample homogeneity and laboratory error. Field duplicates include the natural variability of the original core sample, as well as all levels of error including core splitting, sample size reduction in the preparation laboratory, sub-sampling of the pulverized sample, and the analytical error. Coarse reject and pulp duplicates provide a measure of the sample homogeneity at different stages of the preparation process (crushing and pulverizing).

The following general duplicate performance guidelines were used:

- Acceptable difference value for field duplicates is < 30%
- Acceptable difference value for coarse reject duplicate is < 20%
- Acceptable difference value for pulp is < 10%
- Typically, coarse reject and field duplicates are viewed using an absolute relative error of 20% (equates to ±10% precision level).

Duplicates were taken, at minimum, every 50th sample and included pulps, coarse rejects, and field samples. Field duplicates were taken as either a split of the primary sample in RC cuttings or a quarter cut core. The database contains results for 572 pulp, 485 coarse reject, and 2,252 field duplicates (1,626 RC and 626 core). Capstone personnel conducted statistical analyses of the paired duplicate assays and plotted the results on a suite of plots to compare originals and duplicates.

Duplicate performance in RC and core drilling is summarized in Table 11-6 and illustrated in Figure 11-12 through Figure 11-16. Correlations for copper were very strong in RC field duplicates but moderate in quarter-core duplicates. Correlations for gold were moderate in RC field duplicates and fair in quarter-core duplicates, however, the coarse reject and pulp correlations for gold preparation duplicates were poor. This demonstrates the nuggety nature of gold in the Santo Domingo deposits. The remaining elements – iron, cobalt, and sulphur – demonstrated very strong correlations between original and duplicate values for all duplicate types.

Copper, iron, cobalt, and sulphur do not demonstrate apparent strong bias. RC gold coarse reject and pulp duplicates demonstrate a high bias (duplicates are higher value than original). Sorting of the denser gold bearing particles in the coarse reject and pulp samples can cause this bias, rather than sampling or analytical error. Performance of gold in the preparation duplicates should be monitored closely and concerns communicated to the laboratory quickly.

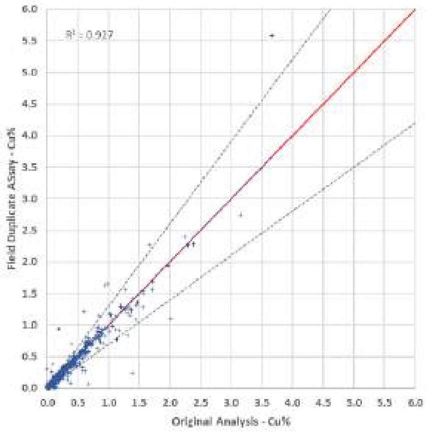
Generally, scatter can be considered high across all elements and duplicate types. In the case of field duplicates, this corresponds to the natural high variability observed in mineralization during core logging. For copper, iron, and sulphur in RC drilling, samples demonstrated the expected increasing homogenization/reduction of variability moving from a field duplicate through a coarse reject duplicate to a pulp duplicate. Cobalt and gold values did not demonstrate the expected homogenization, but rather remained highly variable throughout the sample homogenization process in RC drill holes and, to a lesser extent, in core drilling. Capstone should continue to track the behaviour of duplicates closely and review possible changes to the analytical methods to improve homogenization. In particular, cobalt and gold may benefit from an increase in the sub-sample size analyzed.

This high variability affects the local accuracy in block grades, especially for gold and cobalt, however, no material impact on global grades would be expected.

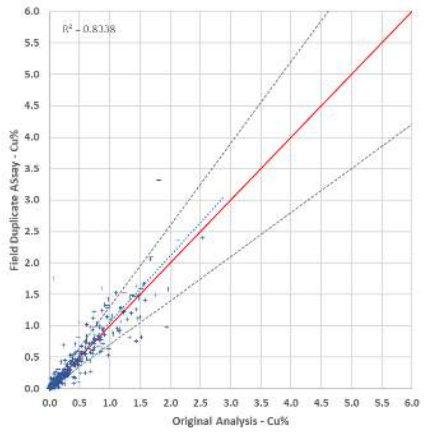
Table 11-6: Duplicate Performance Summary

	Duplicate Type	RC		Core	
		R^2	Performance Comment	R^2	Performance Comment
Copper	Field	0.9270	88% within 30%	0.8338	67% within 30%
	Coarse Reject	0.9919	86% within 20%	0.9844	80% within 20%
	Pulp Reject	0.9982	71% within 10%	0.9918	77% within 10%
Gold	Field	0.8114	64% within 30%	0.7815	59% within 30%
	Coarse Reject	0.9886	36% within 20%	0.8330	47% within 20%
	Pulp Reject	0.9350	22% within 10%	0.9517	44% within 10%
Iron	Field	0.9581	98% within 30%	0.9648	98% within 30%
	Coarse Reject	0.9799	98% within 20%	0.9885	99% within 20%
	Pulp Reject	0.9668	94% within 10%	0.9949	95% within 10%
Cobalt	Field	0.9464	89% within 30%	0.9395	83% within 30%
	Coarse Reject	0.9938	72% within 20%	0.9848	85% within 20%
	Pulp Reject	0.9895	56% within 10%	0.9728	78% within 10%
Sulphur	Field	0.9521	84% within 30%	0.9384	78% within 30%
	Coarse Reject	0.9924	76% within 20%	0.9846	84% within 20%
	Pulp Reject	0.9849	63% within 10%	0.9827	76% within 10%

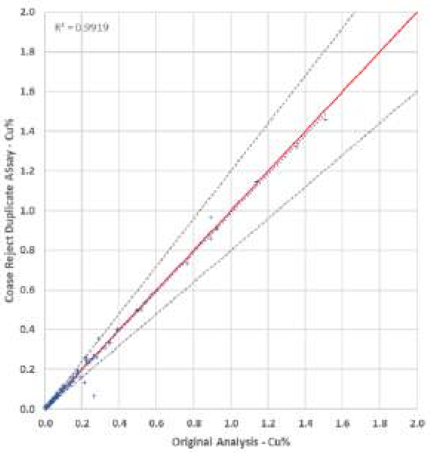
Figure 11-12: Duplicate Performance: Copper



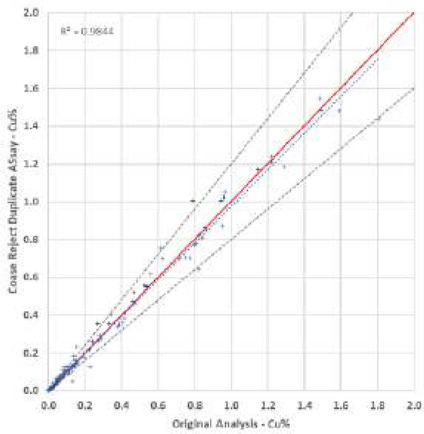
a) RC Field Duplicate – Cu% with ±30% guidelines



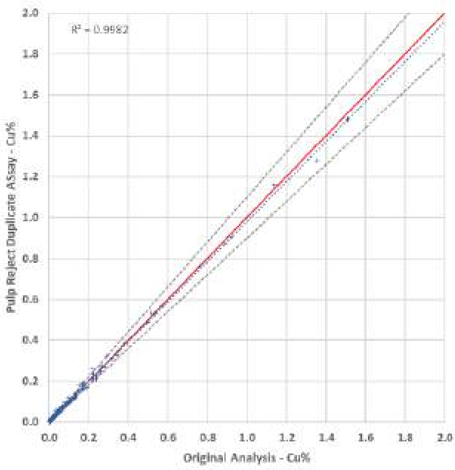
b) ¼ Core Field Duplicate – Cu% with ±30% guidelines



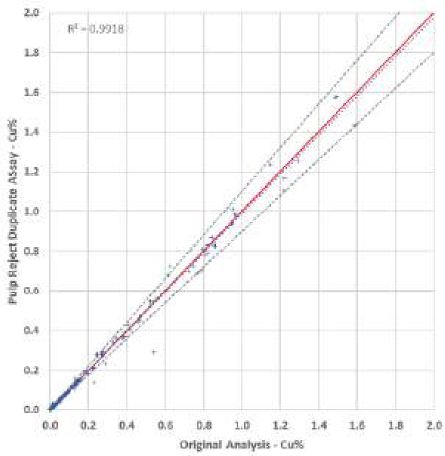
c) RC Coarse Reject Duplicate – Cu% with ±20% guidelines



d) Core Coarse Reject Field Duplicate – Cu% with ±20% guidelines



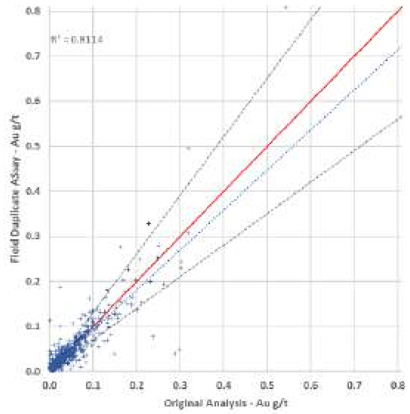
e) RC Pulp Duplicate – Cu% with ±10% guidelines



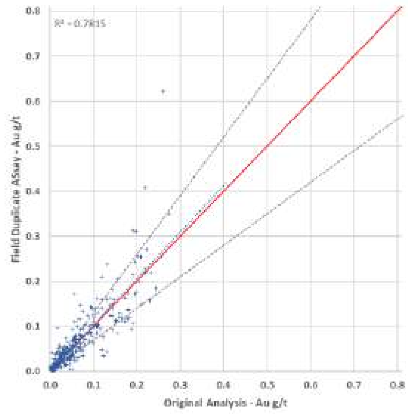
f) Core Pulp Duplicate – Cu% with ±10% guidelines

Source: Figure prepared by Capstone Copper, 2023.

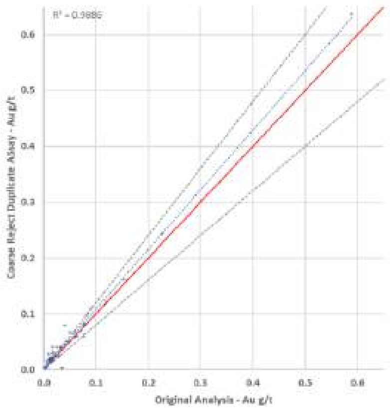
Figure 11-13: Duplicate Performance: Gold



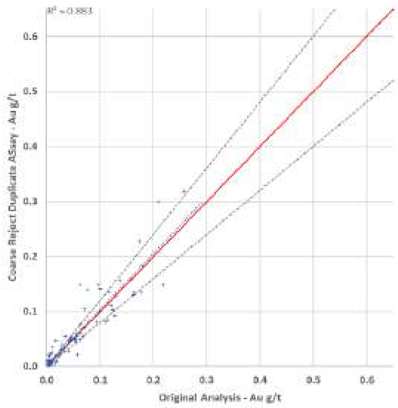
a) RC Field Duplicate – Au g/t with ±30% guidelines



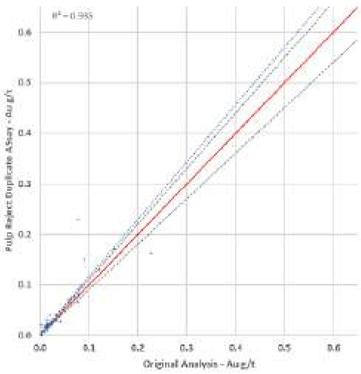
b) ¼ Core Field Duplicate – Au g/t with ±30% guidelines



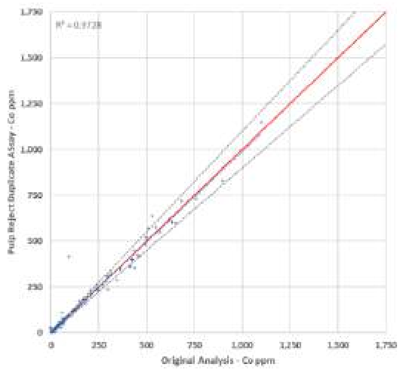
c) RC Coarse Reject Duplicate – Au g/t with ±20% guidelines



d) Core Coarse Reject Duplicate – Au g/t with ±20% guidelines



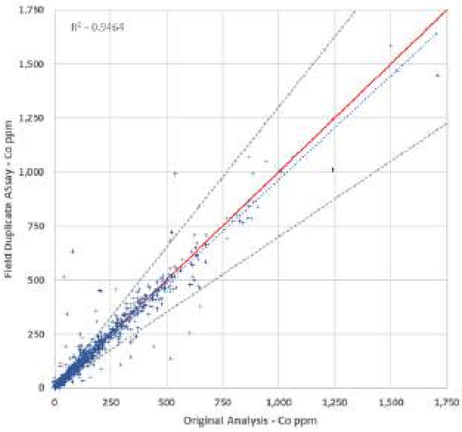
e) RC Pulp Duplicate – Au g/t with ±10% guidelines



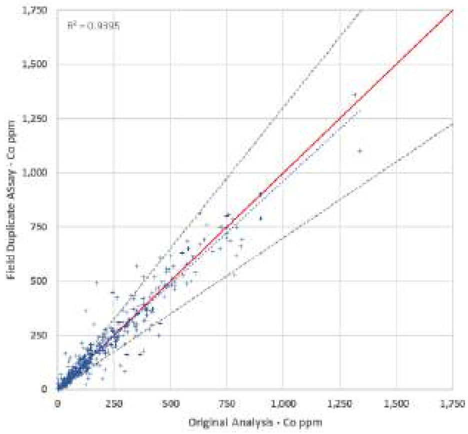
f) Core Pulp Duplicate – Au g/t with ±10% guidelines

Source: Figure prepared by Capstone Copper, 2023.

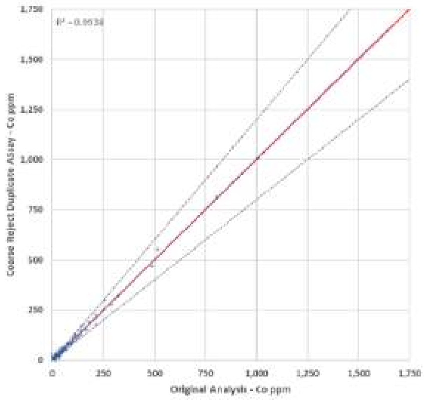
Figure 11-14: Duplicate Performance: Iron



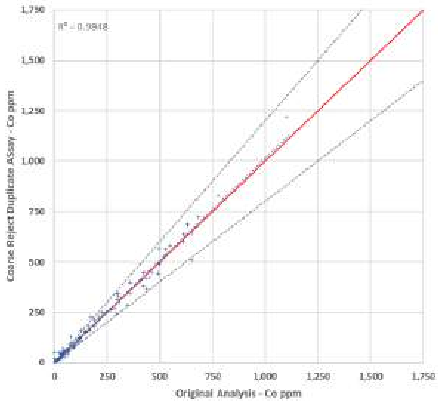
a) RC Field Duplicate – Co ppm with ±30% guidelines



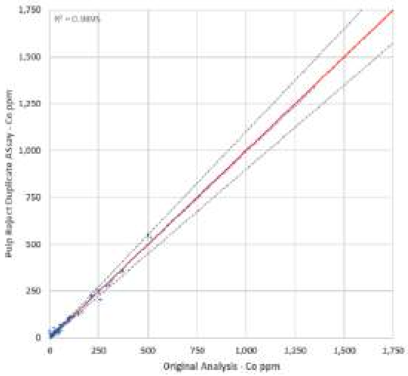
b) ¼ Core Field Duplicate – Co ppm with ±30% guidelines



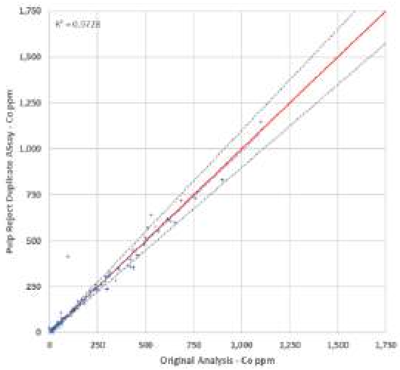
c) RC Coarse Reject Duplicate – Co ppm with ±20% guidelines



d) Core Coarse Reject Duplicate – Co ppm with ±20% guidelines



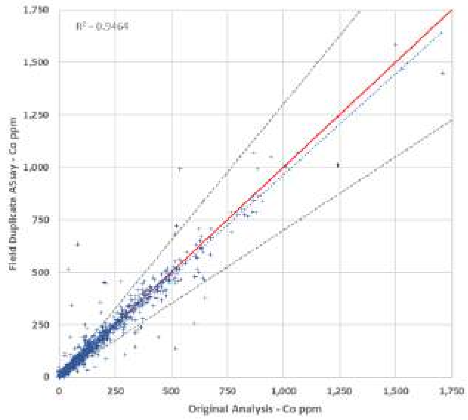
e) RC Pulp Duplicate – Co ppm with ±10% guidelines



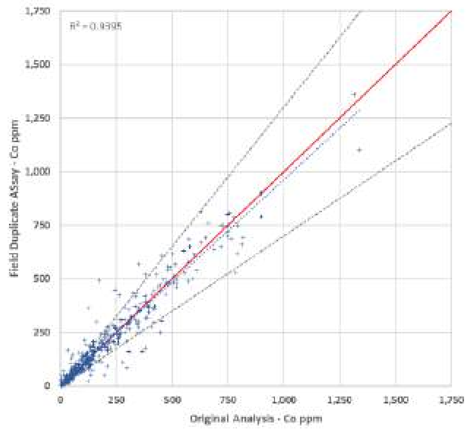
f) Core Pulp Duplicate – Co ppm with ±10% guidelines

Source: Figure prepared by Capstone Copper, 2023.

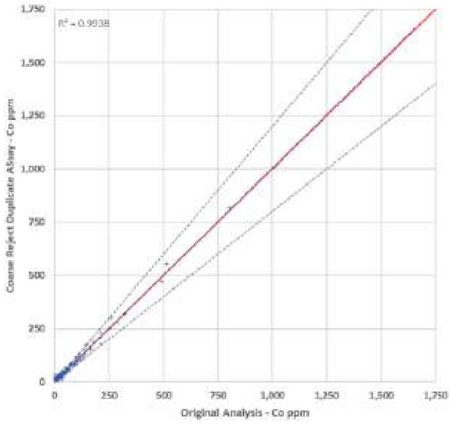
Figure 11-15: Duplicate Performance: Cobalt



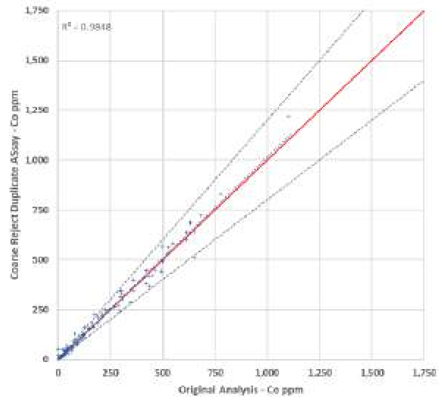
a)RC Field Duplicate – Co ppm with ±30% guidelines



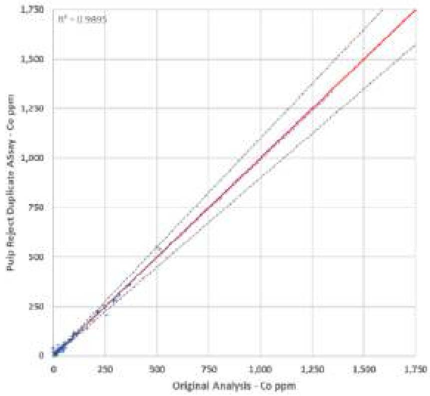
b) ¼ Core Field Duplicate – Co ppm with ±30% guidelines



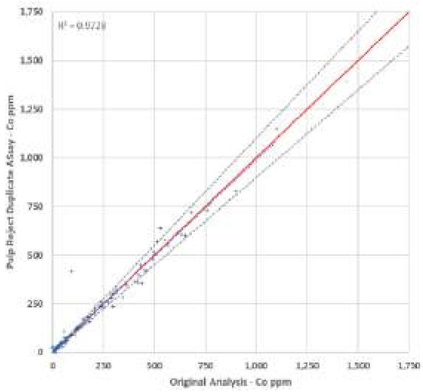
c) RC Coarse Reject Duplicate – Co ppm with ±20% guidelines



d) Core Coarse Reject Duplicate – Co ppm with ±20% guidelines



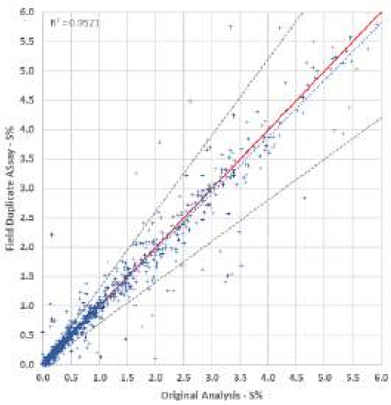
e) RC Pulp Duplicate – Co ppm with ±10% guidelines



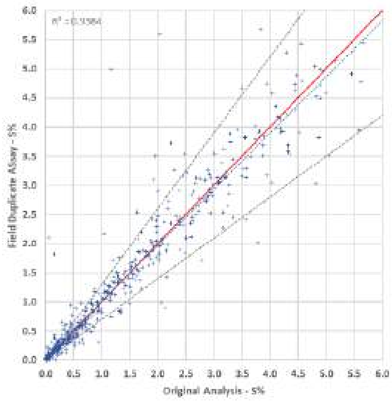
f) Core Pulp Duplicate – Co ppm with ±10% guidelines

Source: Figure prepared by Capstone Copper, 2023.

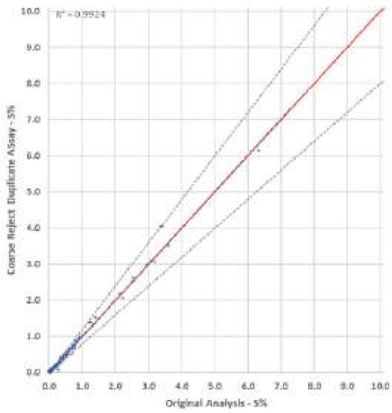
Figure 11-16: Duplicate Performance: Sulphur



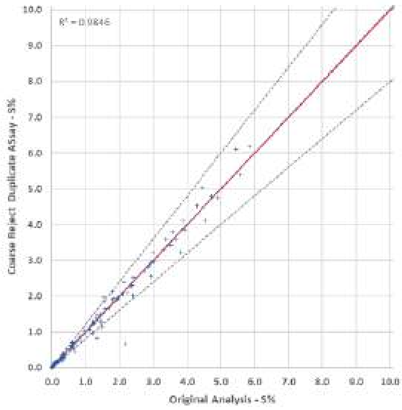
a) RC Field Duplicate – S% with ±30% guidelines



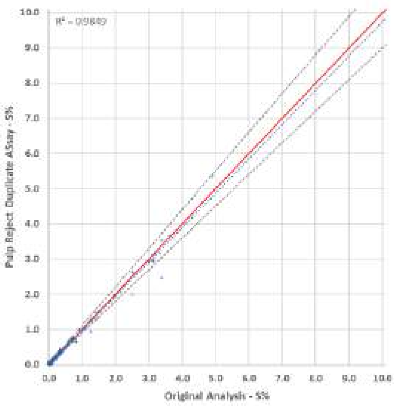
b) ¼ Core Field Duplicate – S% with ±30% guidelines



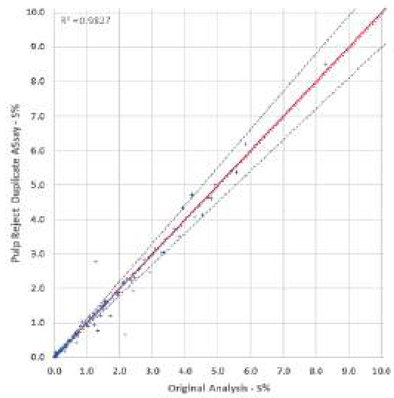
c) RC Coarse Reject Duplicate – S% with ±20% guidelines



d) Core Coarse Reject Duplicate – S% with ±20% guidelines



e) RC Pulp Duplicate – S% with ±10% guidelines



f) Core Pulp Duplicate – S% with ±10% guidelines

Source: Figure prepared by Capstone Copper, 2023.

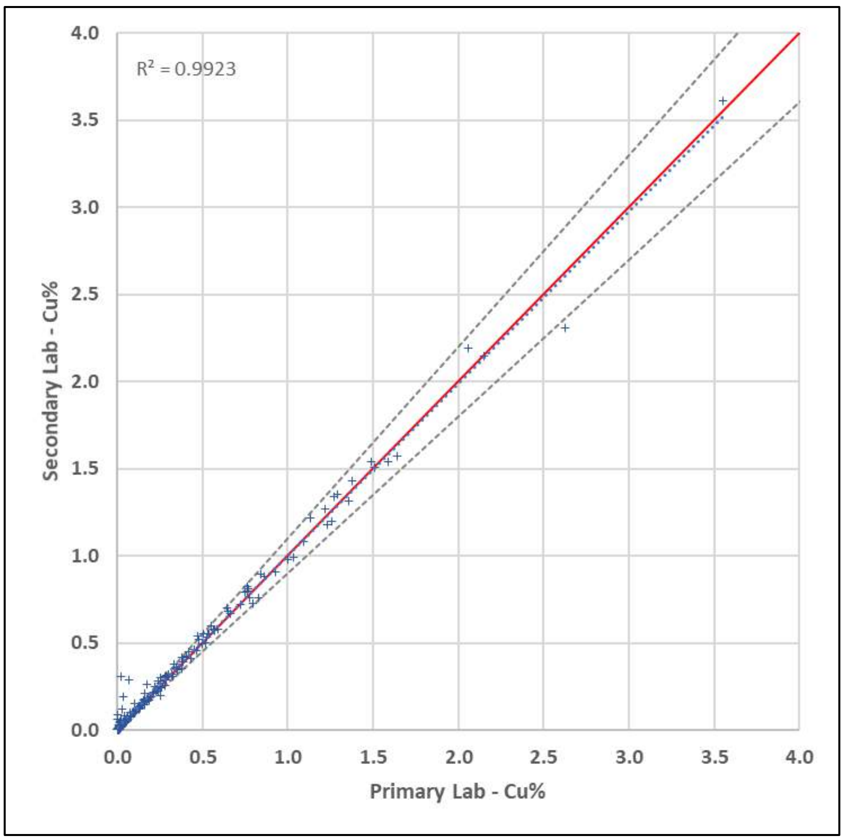
11.7.4 External Check Sampling

As part of the QA/QC program, sample pulps were periodically submitted to a secondary laboratory in 2008, 2010, and 2012. Check assays consist of submitting pulps that were assayed at the primary laboratory to a secondary laboratory and re-analyzing them by using the same analytical procedures. This is done primarily to improve the assessment of bias, in addition to the submission of CRMs to the original laboratory.

A total of 215 check assays were sent for analysis for copper and iron, and 203 were sent for analysis for gold. Only five check samples were analyzed for cobalt and sulphur (2008), a sample count too small to effectively compare inter-laboratory performance. Figure 11-17 through Figure 11-19 show the performance of the check assays for copper, gold, and iron.

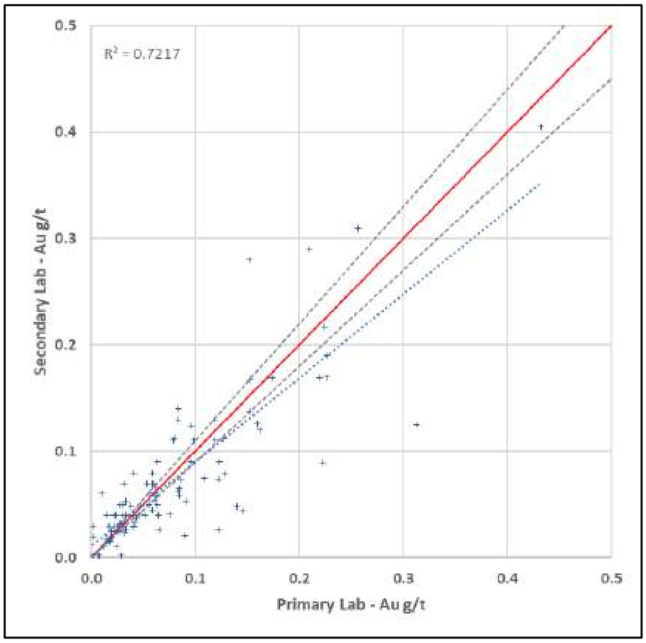
The QP recommends that Capstone continues to perform a check sample program at regular intervals to determine if any significant bias has been introduced into the sample assay programs at the main laboratories.

Figure 11-17: Check Assay Performance: Copper with ± 10% Guidelines



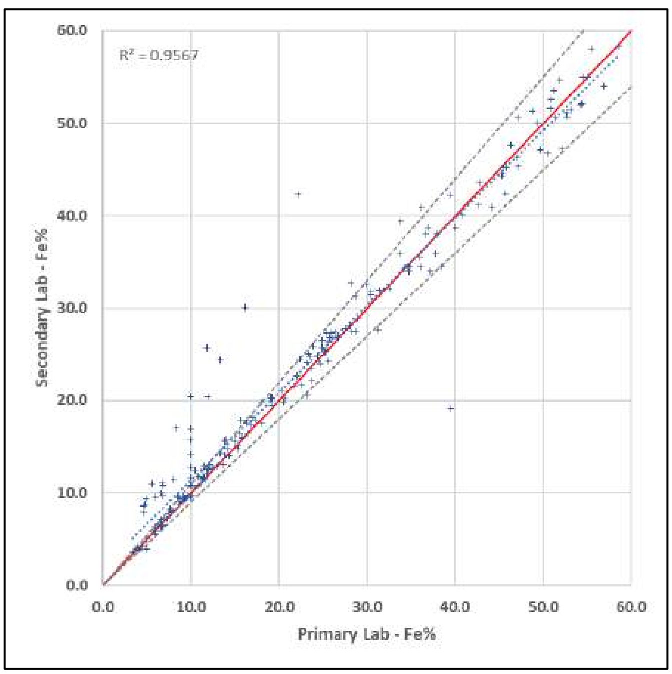
Source: Figure prepared by Capstone Copper, 2023.

Figure 11-18: Check Assay Performance: Gold with ± 10% Guidelines



Source: Figure prepared by Capstone Copper, 2023.

Figure 11-19: Check Assay Performance: Iron with ± 10% Guidelines



Source: Figure prepared by Capstone Copper, 2023.

11.7.5 Sulphur Assay QA/QC

From 2004 to 2015, Capstone collected sulphur assays along with QC data comprising blanks, reference materials, and duplicates. The reference materials were not certified for sulphur during the drilling campaigns in 2004 to 2015. However, CRMs were used in 2019 and 2021. During 2004-2015, sulphur was not considered to be a critical component of the assay database. Therefore, no monitoring of sulphur QA/QC results was carried out and no remedial steps were taken for values now known as failures with the CRM. Capstone collected, analyzed, and plotted the sulphur QA/QC results to provide a basis for assessment of the quality of the sulphur database for inclusion in the previous Mineral Resource estimate effective February 13, 2020.

Analytical work was carried out by ALS of La Serena, Antofagasta, and Copiapó, Chile. The assay method for sulphur was ICP-AES and ICP-MS following either four-acid or peroxide digestion. The lower detection limit (DL) for sulphur is 0.01% S. The QP notes that lower DL values in the database have been replaced with half the DL, or 0.005% S, which is common industry practice. The upper DL for most, but not all, of the database is 10.0% S and these values have been replaced with 10.1% S, which, in SLR's opinion, may impart a low, conservative bias to the grade estimates. SLR notes that there are ten samples in the database with values higher that 10.1% S. These were all from holes drilled in the period 2009 to 2010, during which time the peroxide fusion ICP protocol was used.

In the QP's opinion, the number and distribution of sulphur analyses are effectively equivalent to the data for copper, iron, gold, and cobalt. As such, sulphur data represents an appropriately sized database for Mineral Resource estimation. The analytical protocols used are commonly accepted in the industry, and ALS is a certified commercial laboratory. Given that sulphur is not of primary economic interest in the current Mineral Resource estimate, the QP is of the opinion that the lack of 2004 to 2015 QC coverage is not likely to have a significant impact on the Mineral Resources. If sulphur does become more important to Mineral Resource disclosure, then Capstone could implement re-assay programs wherever sulphur assay QA/QC is deemed to be insufficient, pending further study.

11.7.6 Magnetic Susceptibility QA/QC

From 2010 through 2019, QA/QC of magsus measurements included re-reading a fixed suite of past samples on a weekly basis to ensure the instrument was producing reproducible readings over time. The magsus metre was replaced in 2012, 2015, and 2019 when changes in the readings were observed. In 2012, a regression formula was developed to level the readings from the new metre back to the older dataset. This formula did not require modification in 2015 or 2019.

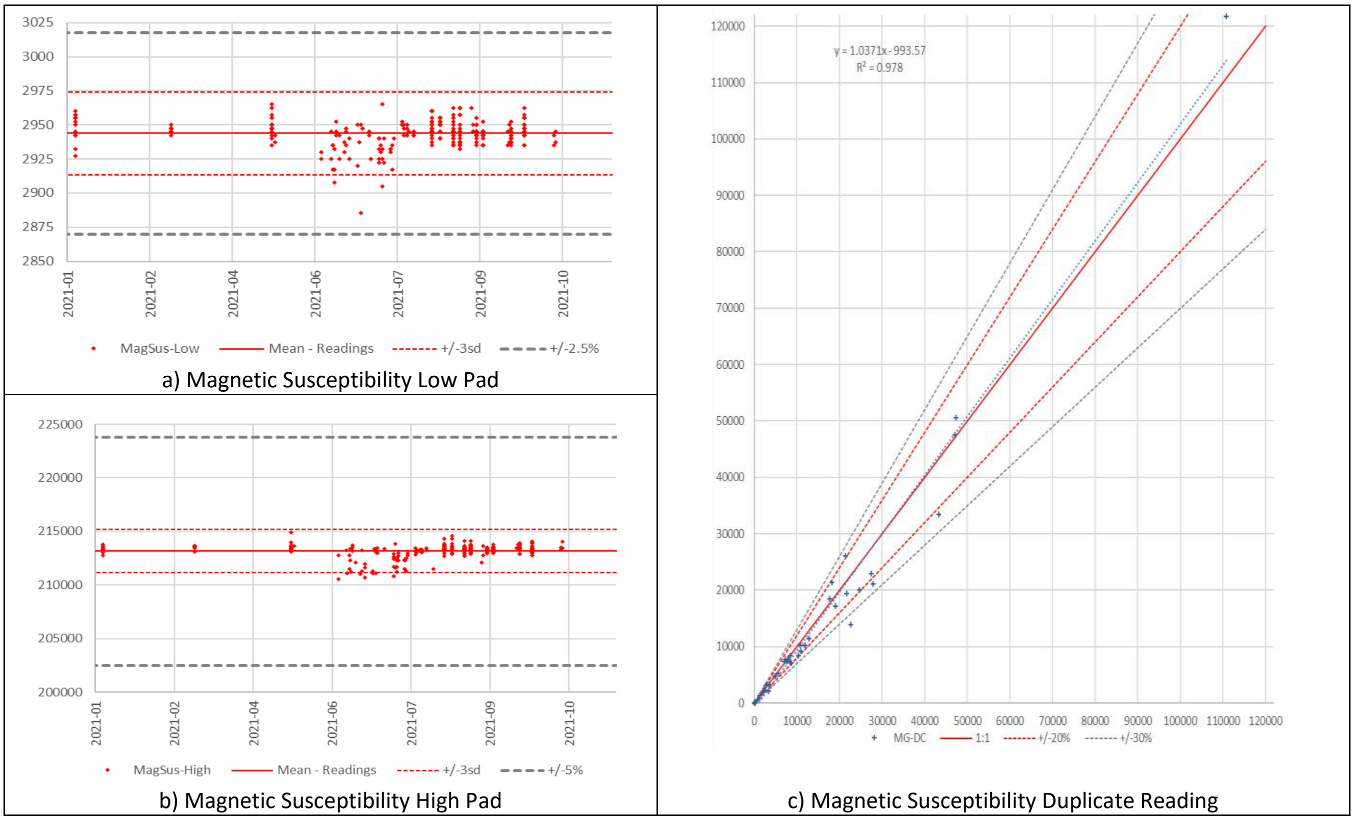
In 2021, two magsus pads, a low value and a high value, were purchased to replace the suite of past samples. A pad was measured at least once within each group of 20 samples, resulting in an overall reading rate of 6%. Duplicate readings were completed once every hundred samples for a rate of 1%. Performance plots for the QA/QC measures implemented in 2021 are shown in Figure 11-20.

When the pad reading fell outside of ±2.5% of the mean, the batch of 20 samples was re-read and all readings replaced in the database. Overall, 98% of final readings fell within 3SD of the dataset mean; all readings were within ±2.5% of the mean.

Duplicate readings of the sample showed a strong correlation with an R2 of 0.978 with 83% of the values within 20% of each other. Overall, duplicates showed a 4% low bias. The reason for the bias is under investigation. However, it is suspected that reading the duplicate immediately after the original could result in lower values because of possible inconsistencies in zeroing between readings (GeoResults, 2023). The bias concern will continue to be monitored for future drilling (McLennan, 2022b).

In the QP's opinion, the magsus QA/QC programs are well implemented and executed, lending solid support to the Mineral Resource estimation.

Figure 11-20: Magnetic Susceptibility 2021 QA/QC Performance



a) Magnetic Susceptibility Low Pad

b) Magnetic Susceptibility High Pad

c) Magnetic Susceptibility Duplicate Reading

Source: Figure prepared by Capstone Copper, 2023.

11.7.7 Low Cobalt Values QA/QC Check

To reduce uncertainty in areas where Santo Domingo's first years of production are planned, a suite of 1,577 sample pulp rejects from 2005 through 2012 drillholes was analyzed for cobalt and sulphur with CRM that covered a lower range of cobalt values. The CRM were inserted at a rate of 10% (158 CRM). To cover the lower cobalt values, commercially purchased OREAS 161 and OREAS 924 were used along with MSD-High and MSD-Med CRM made from Santo Domingo material at higher cobalt values. Overall, the original cobalt and sulphur analysis were reproduced, with correlation coefficients of 0.975 and 0.976 respectively.

Review of CRM performance was acceptable, with cobalt in the CRM biased below the mean for OREAS and biased above the mean in the Santo Domingo CRM, within acceptable limits. The check confirmed the accuracy of cobalt and sulphur at lower values.

11.7.8 Review of Potential Bias between the Drill Hole Assays and Metallurgical Lab Assays

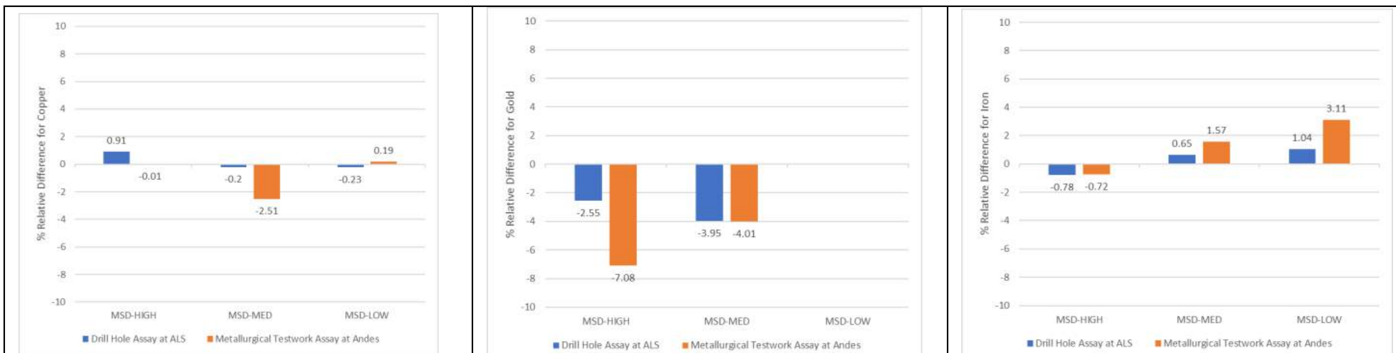
Capstone evaluated the potential for bias between ALS assays in the drill holes used in the block model versus the Andes metallurgical test work by comparing performance of CRMs that were inserted with the samples. Overall, bias between the drill hole assays and metallurgical test work assays was assessed as within acceptable limits using the pulp duplicate performance guideline of ± 10%, shown in Table 11-7 and Figure 11-21. Both sets of results were generally within acceptable limits for the CRM for certified values of copper, iron and gold (excludes MSD-LOW which is not certified for gold), and the datasets were assessed as not significantly different from each other (AsGeoMin, 2023). The QP agrees with this conclusion.

Table 11-7: Summary of Bias between the Drill Hole Assay Dataset and Metallurgical Test Work Dataset

	MSD-HIGH		MSD-MED		MSD-LOW	
	ALS	Andes	ALS	Andes	ALS	Andes
Copper	0.91	-0.01	-0.20	-2.51	-0.23	0.19
Gold	-2.55	-7.08	-3.95	-4.01	Not certified	
Iron	-0.78	-0.72	0.65	1.57	1.04	3.11

Note: Bias is shown as relative difference in %, calculated as Bias = (Average of CRM Result – CRM Certified Mean)/CRM Certified Mean *100%

Figure 11-21: Bias between the Drill Hole Assay Dataset and Metallurgical Test Work Dataset within ± 10%



a) Copper Between-lab Relative Difference %
b) Gold Between-lab Relative Difference %
c) Iron Between-lab Relative Difference %

Source: Figure prepared by Capstone Copper, 2023.

11.8 Database Management

RC drill cuttings and core were logged, and data collected entered into an MS Excel spreadsheet. Each geologist was responsible for entering his/her own logs. Data from these individual "unproofed" logs were printed out and then checked line by line against the original handwritten log by a team of two geologists. Corrections were made and a "proofed" version of the individual log was saved. Each individual "proofed" geology log was then added to a "master geology" log. This master file was then available for further analysis and/or display by exporting the data in the required format.

A separate assay ledger is also kept for each hole. Initially, sample intervals and numbers are entered manually into the ledger and then transcribed into an MS Excel spreadsheet. The initial ledgers or logs are completed by the samplers at the drill site for RC cuttings and at the core logging facility for core. Inserted blanks, standards, and duplicates are also recorded in this ledger. Assay results, when available from the laboratory, are cut and pasted into the digital ledger from an MS Excel file provided by the laboratory. Once complete, data from the ledger are imported to a master MS Access database containing all the drill assays.

One person is responsible for management of the database, posting of final results, and controlling user access. A copy of the database is imported to GEMS for interpretation and presentation purposes.

Data for density, magnetic susceptibility, and surveys are also captured in spreadsheets and then imported to the master MS Access database.

Capstone has a corporate policy on data backup and the database is subject to regular backup procedures.

11.9 Conclusions

In the QP's opinion:

- The sampling methodologies employed by Far West and Capstone are consistent with industry best practice and appropriate for the mineralization style. The sampling is configured such that it is representative of the deposit overall.
- The database is reasonably free from error and suitable for use in Mineral Resource estimation.
- The CRM assays were carried out at an acceptable insertion rate, were reviewed in a timely fashion, and the results triggered reasonable and appropriate responses. The CRM results indicate that the assaying was generally of good quality and acceptable for use in Mineral Resource estimation.
- The results of the QA/QC programs support the database used for Mineral Resource estimation.

11.10 Recommendations

The QP makes the following recommendations with respect to sample preparation, analysis, and security:

- Complete further review of past performance of CRMs recently certified for cobalt and sulphur to evaluate a possible impact of values outside acceptable ranges. If reanalysis of batches with newly recognized QA/QC failures is needed, approximately 3,500 analyses for cobalt and sulphur could be required at a cost of approximately US$45K.
- Continue to closely monitor duplicate performance and investigate solutions to improve homogenization of samples or reduce variability through analysis of larger sub-samples for cobalt and gold. The cost of an external study is estimated at US$8K.

- Continue to perform a check sample program at regular intervals to determine if any significant bias has been introduced into the sample assay programs at the main laboratories. Cost will vary with the size of subsequent drill campaigns, estimated at US$5K to US$10K.
- Fully investigate bias in duplicate magnetic susceptibility readings.
- Implement a stronger sample sealing procedure to reduce potential for tampering after the samples leave the logging facility.
- Ensure that density sampling in oxide and mixed material is carried out at regular intervals in all future drilling.
- The work without associated costs can be performed in-house by Capstone.

12 DATA VERIFICATION

Results of data verification and validation activities conducted under the supervision of the author and Qualified Person, Peter Amelunxen, P. Eng., Capstone's Senior Vice President, Technical Services, are described in this section.

12.1 Database and Geological Model Verification

The database was queried for overlapping intervals, collar maximum depth exceedances, and comparison of collar elevation with topography. Few collar elevations were found to exceed the topography by more than five metres, however, SLR noted that these holes were condemnation holes located outside the Mineral Resource area. Nevertheless, the QP recommends reviewing and resurveying these holes.

The drill hole traces were reviewed in three-dimensional (3D) view, level plan, and vertical sections and no unreasonable geometries were identified.

During the site visit, SLR:

- Inspected the core handling facility and reviewed the geological logging and sampling procedures, and the magnetic susceptibility determination procedures.
- Reviewed core and logging records for holes 4a3-06-188DD, 4a3-06-077DD, 4a3-10-077B, and 4a3-09-360DD in comparison with the lithology and weathering models. The main lithological contacts, weathering features, and samples were appropriately recorded in the database. SLR concluded that there was good agreement between the geological interpretations, log records, digital information, and resultant geological models.
- Used a hand-held GPS unit to check collar positions of 4a3-06-162, 4a3-06-169, 4a3-11-399DD, and 4a3-12-456DD.

No issues were identified.

12.2 Assay Database Verification

12.2.1 RPA (prior to 2012)

Data verification was reportedly conducted by Höy and Allen (2005) in 2005. However, details of this work are no longer available. Prior to 2012, a number of verification studies were undertaken by RPA in support of technical reports on the Santo Domingo property by Lacroix (2006), Lacroix and Rennie (2007), AMEC (2008), and Rennie (2010). Work completed included verification of the database at the time, independent witness sampling, drill collar location checks, downhole survey reading checks, and investigation of the potential for downhole contamination in RC drill holes. No material errors, inconsistencies, or discrepancies were noted.

12.2.2 RPA (2012)

RPA undertook additional data verification steps in support of the 2012 Mineral Resource estimate (AMEC et al., 2014). This work consisted of inspection and validation of drill hole data collected since Rennie (2010). A few errors were captured during the import and validation; these were corrected. RPA also reviewed the validation work conducted by Capstone personnel. No material errors, inconsistencies, or discrepancies were noted.

12.2.3 RPA (2018)

The cobalt database was provided to RPA in MS Excel format. RPA verified cobalt entries in the MS Excel sheet with the assay certificates available. No errors, inconsistencies, or discrepancies were noted.

RPA also checked the results of the analytical QA/QC programs, reviewing standard, blank, and duplicate data.

12.2.4 Capstone (2021-2022)

This subsection is taken from an internal validation memorandum describing the process and results of data verification by Capstone (McLennan, 2022).

In preparation for the 2022 Mineral Resource update, Capstone rebuilt the entire database from source logs and certificates (Kral, 2017). Capstone then validated the drill data from 2004 to 2021 in four passes (McLennan, 2022): (1) the 2021 data only, (2) the rebuilt assay database, (3) a comparison of the shared records in the rebuilt assay database versus the original database used in the resource estimation, and (4) a check on drill collars and downhole surveys. Results were as follows:

- No errors were found for collar information;
- No significant errors were found in the downhole survey records;
- Assay crosscheck of approximately 12% of the entire database resulted in less than 0.01% error rate
- Approximately 80% of the magsus readings were checked against source data and showed an error of 0.91% for any data type. Erroneous records were very similar to the values in the database, so it appears the readings may have been redone while the updated source file was not properly archived.

Capstone took the following corrective actions:

- Estrellita – review of post-2007 resource estimate drill holes for inclusion in the update by Capstone;
- Survey – correction of errors and import of three missing certificates;
- Assays – correction of one copper error, addition of Fe-ICP81 data omitted, and review initiated to standardize handling of below detection limit values;
- Magnetic Susceptibility – levelling formulae differences were resolved

Capstone determined that the final overall error rate for the 2004-2021 analytical data was within the generally allowed limit of 1%. The error rate was 0.01% for assays (12 errors in 57,858 records checked) and 0.91% for magsus readings (214 records that could not be confirmed against a source document).

12.3 SLR (2022) Assay Verification

SLR received updated databases as Capstone's validation was being completed, performing cross checks of the data in tandem with Capstone's work.

SLR imported the comma delimited (CSV) drill hole database export files to an SQL database, and then performed a cross-check on the hierarchical supersedence of the assay methods performed on each sample, generating a compiled final 'SLR' field for Au, Co, Cu, Fe, and S to compare with the 'best' fields generated by Capstone. SLR found no significant errors in the compiled assay fields.

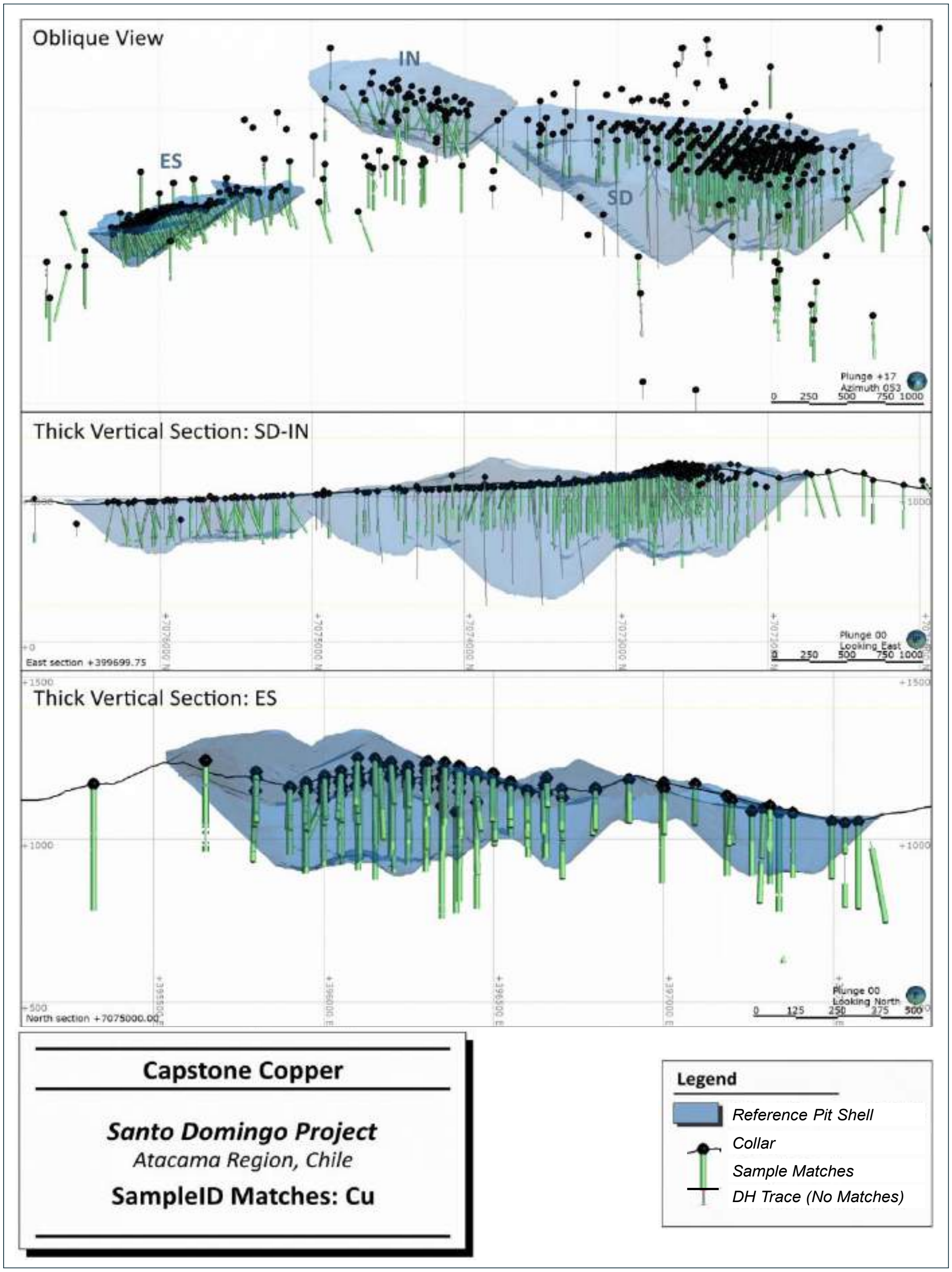
SLR received the assay certificates spanning 2004-2021 in PDF and CSV format. SLR read each CSV, and compiled and imported the sample information to an SQL database for further processing and final matching by sampleID and BatchName. SLR notes that while this technique captures most of the assay certificate information and directly compares it to the Mineral Resource assay database, there are currently some limitations to this process. For instance, the technique does not supersede assays with re-analyses in other certificates. Also due to the changing format of assay information from laboratory to laboratory and year to year, SLR notes that it was unable to capture the date information for all of the assay certificates, and that internal laboratory duplicates may have also been captured from the same file.

Table 12-1 presents the results of SLR's certificate assay matching exercise. There are 78,888 assay records in the Capstone drilling database. From the certificate files, SLR captured 93,063 copper assays, 108,949 iron assays, 91,926 cobalt assays, 58,624 gold assays, and 95,284 sulphur assays. From these compiled assays, SLR matched sampleIDs for 48,041 copper assays, 45,674 iron assays, 39,533 cobalt assays, and 25,692 gold assays in the Mineral Resource database. The matched sample IDs represent approximately 37.5% to 51.6% of the Mineral Resource database assays.

Most of the discrepancies were accounted for by sample overlimit/re-analyses and lower detection limits (LDLs) set to half the LDL in certificates and to the LDL in the database. Spot checks and questions about individual samples were sent to Capstone and explained or corrected as the work progressed.

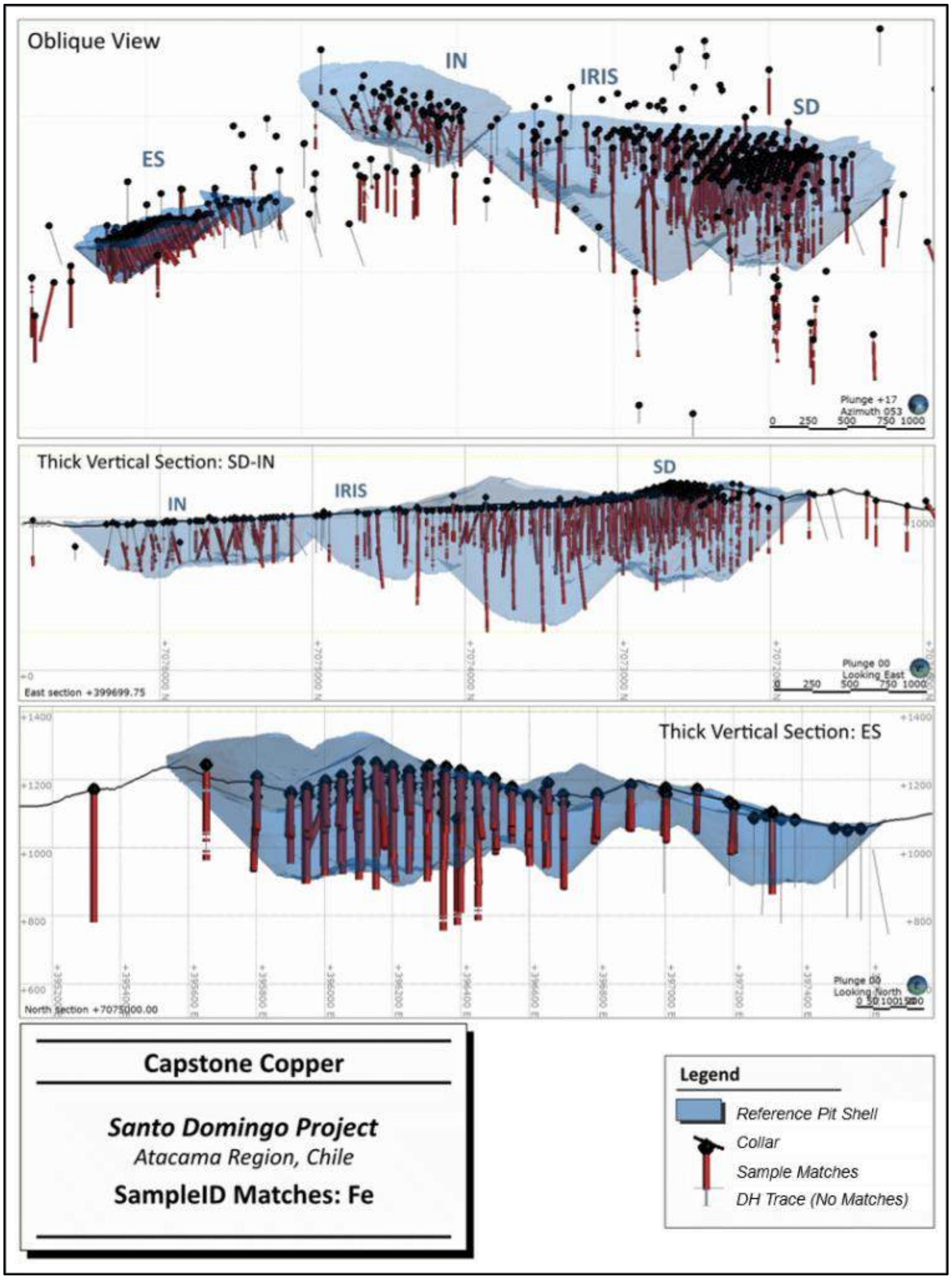
SLR concluded that only a few of the matched sample IDs may be legitimate discrepancies and would have no significant impact on the Mineral Resource estimate. Since the small number of discrepancies remaining in SLR's verification exercise are subject to uncertainty because of limitations in compiling re-analyses accurately, no quantitative table of discrepancies is provided in this section. A qualitative summary of SLR's work is provided in Table 12-1. Examples of the coverage of the matching sampleIDs are provided in Figure 12-1 through Figure 12-5.

Figure 12-1: Matching SampleID; Cu



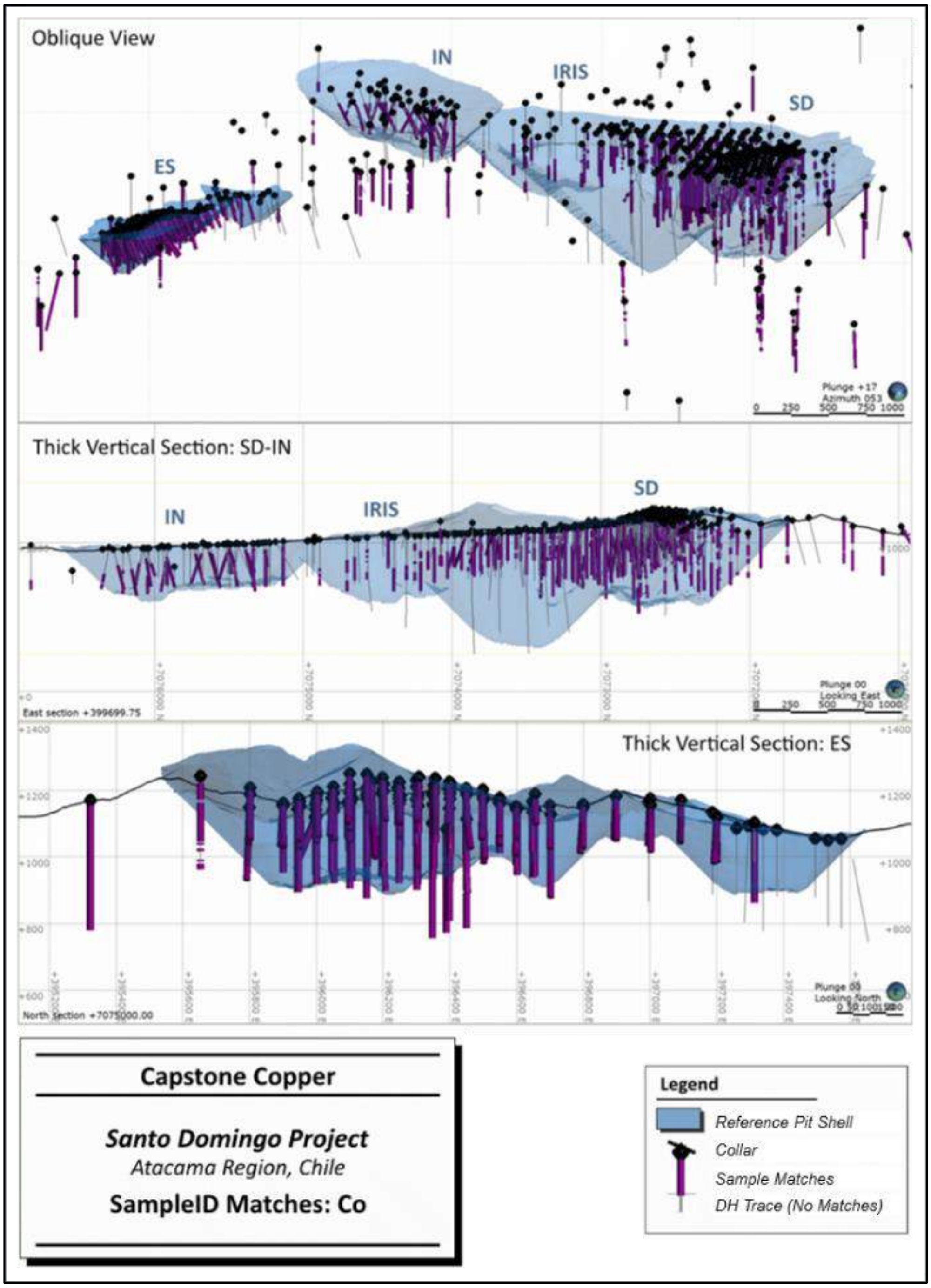
Source: Figure prepared by SLR, 2023.

Figure 12-2: Matching SampleID, Fe



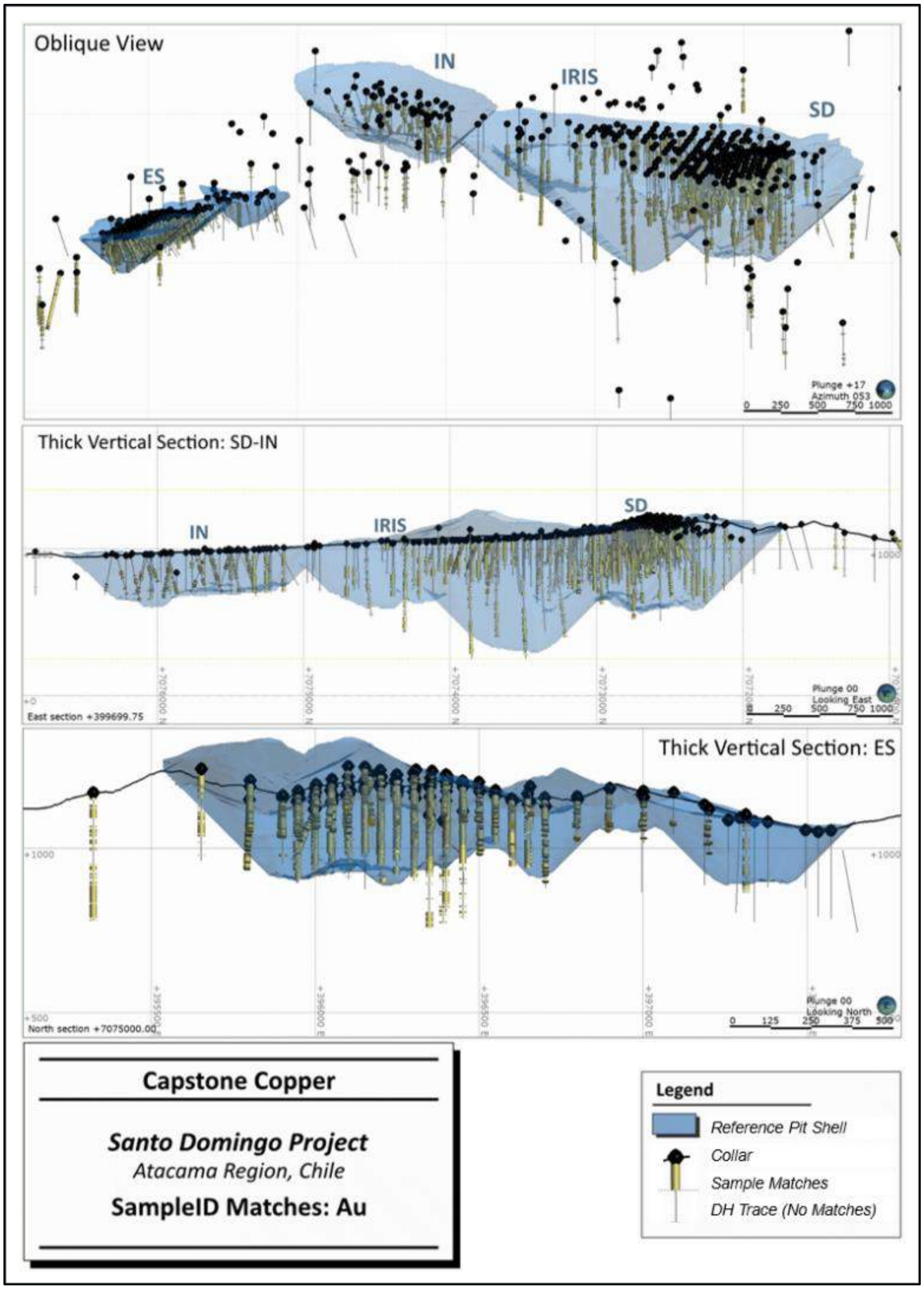
Source: Figure prepared by SLR, 2023.

Figure 12-3: Matching SampleID, Co



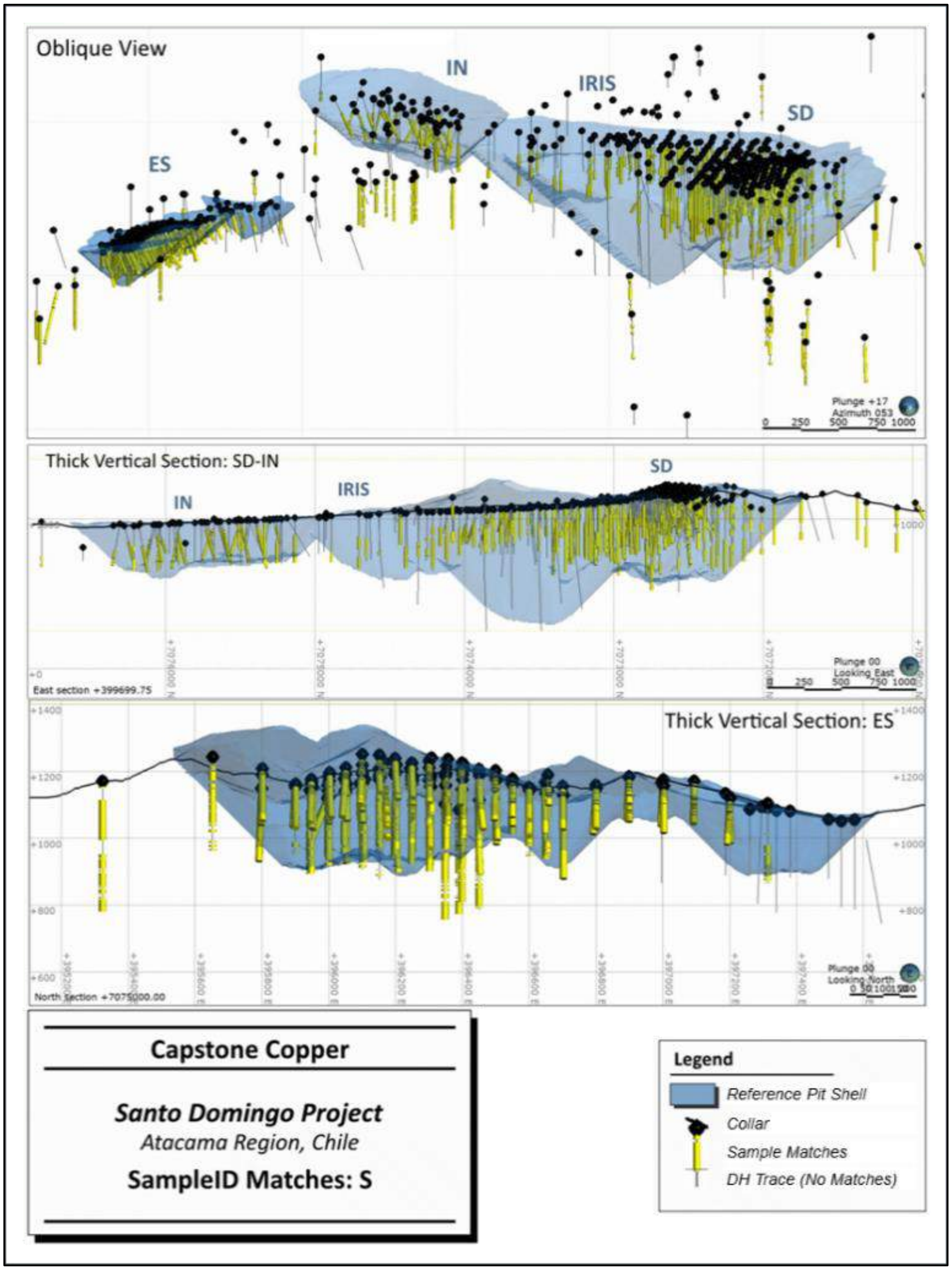
Source: Figure prepared by SLR, 2023.

Figure 12-4: Matching SampleID, Au



Source: Figure prepared by SLR, 2023.

Figure 12-5: Matching SampleID, S



Source: Figure prepared by SLR, 2023.

Table 12-1: Summary of SLR 2004-2021 Assay Verification Exercise Capstone Copper – Santo Domingo Project

Element	Units	Certificate Count	Cert-DB	Element	Units	Certificate Count	Cert-DB	Element	Units	Certificate Count
Au	g/t	58,624	21,622	36.9	21,495	99.4	127	0.10	34	0.1
Co	ppm	91,926	39,533	43.0	39,528	100.0	5	50.00	2	0.0
Cu	%	93,063	48,041	51.6	45,849	95.4	75	0.05	32	0.1
Fe	%	108,949	45,674	41.9	44,932	98.4	742	2.00	611	1.3
S	%	95,284	35,721	37.5	35,715	100.0	6	0.10	3	0.0

Notes: **1**. Certificates were filtered for the Mineral Resource areas. **2**. Table represents only the matches achieved with SLR's certificate compilation methodology and is not exhaustive. **3**. Some Sample IDs in certificates may be repeated. **4**. SampleID mismatches may not reflect errors in the database. Many differences are likely re-assayed in other certificates. **5**. Matches are only by SampleID and Assay Method.

12.4 Density Verification

Capstone staff sent SLR a collection of density certificates covering 539 assays, representing approximately 10% of the 5,457 density measurements in the Mineral Resource database. Density certificate assays were compiled into a table and compared them to the densities in the Mineral Resource database. A total of 498 out of the 539 assays produced sampleID matches within the Mineral Resource database. All of the matching certificate sampleID density values matched the density values contained in the Mineral Resource database. The QP considers this result sufficient to support the Mineral Resource estimate and recommends continuing to compile all density certificates to a folder for future verification exercises.

12.5 Twin Drilling

For the 2010 twin drilling campaign, SLR matched intervals of four metre composites for each of the pairs and plotted the grades for gold, copper, and iron to compare the results. SLR found that, for most of the pairs, the assay results compared reasonably well. The data were observed to be quite noisy at the four-metre resolution; however, it was generally noted that high- and low-grade zones matched and that the grades tended to cluster in the same ranges. Only one pair of twinned holes (4a3-06-099/4a3-10-099-B) displayed significant differences that could not be attributed to hole deviation.

In 2018, SLR compared copper, gold, iron, and cobalt assay values from the drilling completed in 2014–2015 with the corresponding twin hole in the database. SLR assessed that most of the pairs of assay results compared reasonably well.

Capstone drilled another 13 twin holes in January 2019 to acquire additional material for metallurgical test work. Some of these holes were drilled at the same locations as in the 2014–2015 program. SLR compared the results from the 2019 twin drilling program with those from the 2014–2015 and earlier programs. Assay results for gold, copper, iron, magnetic susceptibility, and cobalt were generated across the intervals bounded by the resources wireframe models. The grades and overall thicknesses of the zones intersected were observed to be generally consistent, although some significant local variability in grades were noted. There did not appear to be a consistent bias between the newer and older generation of drill results (Wood et al., 2020).

12.5.1 SLR (2022)

SLR reviewed the twinned holes in Leapfrog and in the drill hole database and excluded redundant holes or portions of redundant holes to simplify the grade estimate and avoid unwanted artifacts. Unsampled "stems" and twins were excluded from the Mineral Resource database. An analysis of three "twins" and a list of excluded intervals is provided in Section 14.3.3.2.

12.6 Conclusions

Based on the data verification and validation activities completed, the QP is of the opinion that the data is adequate to support this Technical Report, and:

- Capstone's database verification procedures for the Santo Domingo Project comply with industry standards and are adequate for the purposes of Mineral Resource estimation, and that the Santo Domingo database is of sufficient quality to support a Mineral Resource estimate.
- Capstone's database workflows and controls are both systematic and thorough.
- Repeated verification work over the life of the project has ensured that the database was of sufficient quality to support Mineral Resource estimation.
- Capstone's database construction from the primary data, validation and verification served to confer a full understanding of the dataset, organize the source information, correct most errors, and further improve the quality of the dataset.
- SLR's reproduction of the assay hierarchy in the provided database produced no significant differences from Capstone's 'best' field.
- SLR's verification exercise for Au, Co, Cu, Fe, and S covered most of the assays in the pit shells and did not produce any material differences.
- Twin hole drilling has provided a reasonably consistent verification of the earlier drill results, particularly considering the differences in assay protocols and possible survey errors.

12.7 Recommendations

To improve the quality of exploration, drilling, and analytical data, the QP recommends that Project personnel undertake the following as part of their regular duties:

- Continue with systematic verification and validation of the database to ensure that sampleID supersedence and assay supersedence are captured from the same and separate certificate IDs in ways that are transparent and facilitate reproduction outside of the database software.
- Set the LDL values in the compiled 'best' fields to half of the '<' value in the database.
- Set the upper detection level (UDL) values in the compiled 'best' fields to the '>' value in the database + 0.0001.
- Ensure that the final sampleIDs and certificate IDs match the source material.
- Ensure that the 'best' fields clearly indicate which assay and certificate ID was chosen, and that a flag field serves to mark superseded assays.
- Store the reasons for re-assaying in separate fields.
- Add a 'Year' field to the collar table.
- Continue to monitor for negative values.

All of the recommended work can be performed in-house by Capstone.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Metallurgical Test Work

13.1.1 Summary

A summary of the metallurgical test work performed from 2006 to 2023 is provided in Table 13-1.

Table 13-1: Metallurgical Test Work Summary Table

Date	Test work Type	Laboratory/ Test work Facility	Work Performed
2006	Comminution	SGS Santiago	Grindability response test work on two drill core samples.
2008	Comminution	SGS Mineral Services	Grindability response test work on five composite drill core samples. Included BWi; Bond ball mill modified tests; RWi; and SAG power index (SPI) tests.
	Cu flotation	SGS Santiago	Two master composites (MC-A and MC-B). Copper rougher kinetic, Cu and pyrite rougher kinetic and Cu cleaner and pyrite rougher flotation tests; pyrite rougher flotation on copper rougher tailings to maximize recovery of sulphur from the flotation rougher tailings.
2009	Magnetite concentrate	SGS Lakefield	Response of composite samples to magnetic separation using DT laboratory tests.
		Studien-Gesellschaft für isenerz-Aufbereitung (SGA)	LIMS (Low Intensity Magnetic Separators) testing to develop a marketable magnetite concentrate.
		Compañía Minera del Pacifico (CMP)	Testing of magnetic concentrate.
2010	Comminution	SGS Santiago	Grindability program on 128 samples; ball mill calibration program on four samples.
		Phillips Enterprises (Advanced TerraTesting Inc.)	Crushing and geotechnical program on 128 samples.
	Cu flotation	SGS Santiago	Cu mineralogy.
		Aminpro	Chemical and mineralogical analysis (QEMSCAN) and rougher kinetic flotation tests on five samples; all rougher flotation tests were conducted using sea water.
		SGS Lakefield	Cu flotation performance and recoveries on five samples of the same composite; Cu performance and recovery test work on three different composite samples from Santo Domingo Sur; testing used three different water types: SGS Lakefield water, Capstone-supplied saline water and synthetic sea water; five flotation tests to determine the effect of primary grind on Cu recovery.
	Magnetite concentrate	SGA	Magnetite recovery tests on five whole ore samples of the same composite at a final grind of less than 63 µm.

Date	Test work Type	Laboratory/ Test work Facility	Work Performed
2011	Comminution	Ammtec (now ALS-Ammtec)	SAG mill competency; confirmatory ball mill tests; 19 samples tested.
		SGS Santiago	RWi, BWi, Ai and SMC tests on 58 samples.
	Cu flotation	SGS Lakefield	Cu flotation performance testing on four composite samples (Eight-Year, Hematite, Magnetite and Oxide) and a set of 38 variability samples; investigated optimized use of seawater.
	Magnetite concentrate	SGS Lakefield	Bulk flotation tests at optimum conditions to produce Cu rougher concentrate for regrind power testing (Special Jar Mill Grindability Test); Cu rougher flotation tailings of the Hematite and Magnetite composites for LIMS recovery studies; and Cu cleaner concentrate from the Eight-Year composite for concentrate filtration and washing tests.
		SGA	Optimization test work for LIMS Fe recovery; fresh water and sea water used; variability sample testing to determine the correlation between DT test results and LIMS cleaner tests; and correlation between Satmagan/magnetic susceptibility head grade and DT test recovery; iron recovery variability tests on 35 samples that represented five defined ore zones.
		Metso	Cu and LIMS Fe rougher concentrates to determine the specific power required for regrinding.
		Ausenco PSI	Rheology tests on a LIMS magnetite concentrate sample.
		SGS Santiago	DT test and Satmagan tests using 52 rougher tailings generated from rougher kinetics tests.
	Filtration	Outotec	Concentrate filtration tests to determine filtration equipment (Larox PF and Ceramic).
2012	Comminution	JKTech	SMC test on 91 samples to verify the comminution circuit throughput capacity.
	Cu flotation	SGS Santiago	Flotation test work program using 51 variability samples for rougher kinetic test and 15 variability samples for open cycle test (OCT).
	Physical characterization	Jenike and Johanson (Chile)	Size characterization, flow properties, cohesive strength and bulk density for modelling the operation of the stockpile and material handling including the filter hopper.
	Settling (concentrate and tailings)	FLSmidth	Tailings thickener tests and concentrate thickener tests on tailing samples and magnetite concentrate samples respectively; different flocculant and thickener technologies tested to achieve the highest settling rate; tailings thickener tests and concentrate thickener tests on tailings samples and magnetite concentrate samples respectively; different flocculants and doses tested; rheological tests.
		Outotec	Tailings thickener tests and concentrate thickener tests on tailing samples and magnetite concentrate samples respectively; 10 tests completed for magnetite concentrate and 11 tests completed for tailings; two additional tests completed to compare two different flocculants.
	Filtration	Outotec (Chile)	Filtration tests for magnetite concentrate at 65% w/w solids; eight tests completed to obtain a chloride concentration lower than 300 ppm Cl.
2013	Magnetite Regrind	Moly-Cop Chile	Regrinding test program using rougher magnetic concentrate samples provided by Capstone to determinate conditions for the design and operation of the magnetite regrind mill.
	Cu flotation	SGS Lakefield	Rougher kinetics, cleaner flotation, LIMS tests and rougher magnetic circuit tests were completed on three composites (SD1, Hematite and Magnetite); tailings produced used for settling and rheological testing.
	Physical characterization	Patterson & Cooke (Chile)	Rheology test program using two tailings samples to determinate the physical properties of the tailings.

Date	Test work Type	Laboratory/ Test work Facility	Work Performed
	Settling	Outotec (Chile)	Tailings thickener test using Hematite and Magnetite tailings provided by SGS Lakefield; two stages of thickening tested using different flocculants and doses; rheological tests.
		FLSmidth (USA)	Sedimentation and rheology testing programs for Hematite and Magnetite tailings provided by SGS Lakefield to determine the sizing and operational parameters for tailings thickeners considering both composites.
		Tenova Delkor (Canada)	Settling and thickening tests on two composites (Hematite and Magnetite) provided by SGS Lakefield; sizing for the tailings thickeners completed considering two separate stages of thickening.
2014	Cu flotation	SGS Santiago	Rougher, cleaner kinetics and locked cycle tests (LCT) completed on three selected composites; variability samples (open circuit and locked cycle) tested to validate the algorithm developed during the 2011 pre-feasibility study.
	Magnetite Concentrate	ALS Ammtec	Magnetite recovery program using three composite samples; optimum magnetic rougher and cleaning grind sizes and conditions including washing and magnetic strengths was confirmed for design purposes.
2015	Pilot plant	ALS Santiago	Pilot plant test work included: hardness testing of six composites for BWi, Rwi, Ai and SMC. Low-Impact Crusher work index and JK Drop Weight test on two samples. The pilot plant generated concentrate for further testing including thickening and filtration testing for Cu and magnetite concentrates.
	Bench scale	Aminpro	Confirmed conditions in roughing and cleaning using sea water. In all cases the kinetics were determined, which allowed for circuit simulation work. LCT tests with recycle water (base case for previous tests) and without recycle water, indicated that the latter requires more than three times longer residence time. Model simulations of the ALS pilot plant confirmed the results of the LCT with recycled water.
2018	Cu flotation	Aminpro	Rougher and cleaner kinetics with desalinated water for modelling the flotation circuit. Selection of samples for the development of a recovery algorithm for Cu and Au with desalinated water. Rougher kinetics on samples with different head assays. LCT with desalinated water to determine a recovery factor for the cleaning stage and validate the algorithm developed by Aminpro.
	Cu/pyrite flotation	Blue Coast	Cu/pyrite separation for sample generation using a Years 1-5 composite sample from the 2015 testing program.
	Mineralogical characterization	AuTec Innovative Extractive Solutions Ltd (AuTec)	Mineralogical characterization of pyrite concentrates, and pyrite flotation tailings samples produced from the Years 1-5 composite Co mineral deportment. Pyrite particle scanning to determine variability of Co concentrations within the pyrite. Pyrite particle depth profiling to distinguish Co in solid solution from Co-hosted on particle surfaces.
	Oxidation process scoping	SGS Lakefield SGS Malaga	Pressure Oxidation (POX) and Bio-Oxidation (BiOX) tests using the pyrite concentrate generated from the Years 1-5 composite. Pyrite POX leach scoping to test the amenability of pyrite concentrates to pressure oxidation leach recovery of Co. BiOX leach scoping to test the amenability of bio-oxidation recovery of Co and Co recovery in relation to degree of sulphur oxidation.

Date	Test work Type	Laboratory/ Test work Facility	Work Performed
2019	Bulk pyrite flotation	Blue Coast	Bulk pyrite sample generation from 450 kg of Years 1-5 composite from the 2015 testing program. The bulk pyrite concentrate produced was subsequently used in all the 2019 Co testing programs.
	BiOX leaching	SGS Lakefield	Pyrite fine grinding and BiOX leach testing to demonstrate the amenability of the pyrite concentrates to Co recovery.
	POX leaching	SGS Lakefield	Pyrite POX leach condition scoping. Testing included six POX leaching tests at varying temperature and pressure conditions. Pressure oxidation leach of a high-temperature dead roast calcine produced at Kingston Process Metallurgy.
	Albion leaching	Kingston Process Metallurgy	Albion leach testing consisting of ultra-fine grinding and two Albion leach tests at different grind sizes.
	Roasting/leaching	Kingston Process Metallurgy	Roasting conditions scoping consisting of seven roasting tests in laboratory stationary and circulating fluid bed reactors testing roasting conditions from 600–840ºC. Calcine leaching scoping consisting of 12 calcine leaching tests on the products of the seven roasting tests. Testing conditions varied the acid concentration, oxygen injection and temperature.
	Leach solution purification	SGS Lakefield	Solution neutralization of POX solutions combined from all the POX leaching tests. Four solution neutralization tests of roaster calcine leach solutions generated at Kingston Process Metallurgy. One Cu precipitation test of a partially neutralized solution. One Mn removal test on neutralized calcine leach solution using Caro's acid.
	Pyrite flotation	Aminpro	LCT conducted for the copper flotation circuit on a composite for each of the first 5 years of the mine plan and on a combined Years 1-5 composite. These tests were part of the Cu/Fe metallurgical confirmation testing in fresh water but were assayed and balanced for Co. Pyrite cleaning testing treating the cleaner tailings from the Cu flotation locked cycle testing.
	Cu flotation	Aminpro	Rougher and cleaner kinetics with desalinated water for modelling the flotation circuit. OCT and LCT with desalinated water for a representative Years 1-5 composite sample. LCT with desalinated water for each composite per year for Years 1 to 5. Variability tests for algorithm update. Simulation and modelling of the flotation circuit. Pyrite/cobalt flotation tests.
	Magnetite concentrate	SGS Santiago	Magnetic separation tests using LIMS including rougher and cleaner stages with desalinated water, to confirm circuit performance and design criteria.
		Polimin	Magnetic separation test using LIMS including rougher and cleaner stages.
2020	Pilot plant	Aminpro	First pilot plant test work using desalinated water for Cu and pyrite flotation (2019 – 2020) with a representative First 5 years composite sample. Production of rougher flotation tailings for magnetic separation pilot plant.

Date	Test work Type	Laboratory/ Test work Facility	Work Performed
	Magnetite concentrate	SGS Santiago	Davis Tube Test program using 206 variability samples for metallurgical response evaluation.
	Settling and thickening (copper concentrate)	Tailpro Consulting, Vietti Slurrytec	Characterization, settling, and dynamic thickening of a copper concentrate sample.
2021	Comminution	Asmin	Comminution test on 45 variability samples including BWi, RWi and SMC.
	Flotation	SGS	Cu and pyrite flotation at mini pilot plant scale using desalinated water, with the objective to demonstrate the feasibility to obtain a commercial grade – copper concentrate.
	Pilot plant	Aminpro / Polimin	First magnetic separation pilot plant using rougher flotation tails and desalinated water, performed in a fully scalable LIMS industrial diameter magnetic drum with aim to obtain design parameters. Obtention of metallurgical sample for thickening and filtration testing, as well as small scale testing for downstream silica reduction processing alternatives evaluation.
	Settling (iron concentrate)	Tailpro	Independent iron concentrate characterization, settling and dynamic thickening test using concentrate obtained from magnetic separation pilot plant developed by Aminpro-Polimin. Rheology measurement.
		Metso:Outotec (Chile)	Iron concentrate characterization, settling and dynamic thickening test using concentrate obtained from magnetic separation pilot plant.
	Filtration	Metso:Outotec (Chile)	Iron concentrate filtration test using concentrate obtained from magnetic separation pilot plant, performed with pressure filtration technology and vacuum (ceramic plate) technology filtration.
	Magnetite concentrate	Clariant	Reverse flotation laboratory scale test performed on iron concentrate samples for metallurgical response exploration.
		Derrick	Small scale test provided to vendor for preliminary high vibrating screen technology evaluation.
		San Lorenzo	Extended Davis Tube Test campaign approaching 1318 samples, covering the Santo Domingo and Iris Norte deposit through all magnetic iron mineralized geological units, for metallurgical response variability evaluation, including assessment of regrind effect. Analysis of Iron concentrate grade and impurities.
	Settling and thickening (tailings)	Tailpro	Characterization, settling, and dynamic thickening of tailings samples. Semi-pilot thickening testing with aim to confirm / optimize tailings thickening design. The tailings sample was obtained from the pilot Magnetic Separation program developed by Aminpro-Polimin during the year 2021.
2022	Cu/pyrite flotation	Blue Coast	Rougher kinetic and cleaner test including copper flotation reagents assessment with the objective to confirm pyrite (cobalt) deportment to cleaner tails thus confirming flotation circuit strategy. Open cycle test for yearly samples – covering LOM, and mineralized geological unit samples, including volcaniclastic top and bottom, ANLT, andesite and breccia zones, for overall variability assessment. Locked cycle test on main volcaniclastic units, for overall metallurgical performance evaluation. Extended 20-cycles locked cycle test to simulate a close loop reclaim water system. Rougher and cleaner Simple Kinetic Test (SKT) and Full Kinetic Test (FKT) on volcaniclastic top and bottom, ANLT, andesite and breccia units as well as yearly samples covering LOM for modelling the copper and pyrite flotation circuit.

Date	Test work Type	Laboratory/ Test work Facility	Work Performed
	Mineralogical characterization	Memorial University of Newfoundland	Geological composite feed samples and cobaltiferous pyrite concentrate samples study using a Mineral Liberation Analyzer (MLA XBSE).
	Magnetite concentrate	Aminpro	Magnetite concentrate test work using a representative first five years composite sample for additional cleaning stages evaluation, considering magnetic separation and reverse flotation.
	Cu oxide leaching	Asmin	Exploratory oxide zone testing for copper leaching, considering bottle roll test using 28 variability samples and column test using samples from volcaniclastic (3) and andesite (3) geological units as well as a referential sample from Mantoverde deposit. Testing conditions as per Mantoverde operation. Mineralogical study on volcaniclastic and andesite samples showing chrysocolla as main copper oxide mineral overall.
2023	Flotation pilot plant	Asmin	Pilot plant test work using desalinated water for Cu and pyrite flotation with a representative Year 1-5 composite sample and flotation conditions previously established (Blue Coast 2022), for copper and pyrite flotation strategy demonstration. Production of rougher flotation tailings for magnetic separation pilot plant, copper concentrate sample for thickening and filtration vendor testing, rougher concentrate sample for regrind evaluation. Pyrite concentrate obtention for testing.
	Magnetic separation pilot plant	Aminpro	Magnetic separation pilot plant using rougher flotation tails and desalinated water performed in a fully scalable LIMS industrial diameter magnetic drum and the incorporation of hydroseparation stage and additional cleaning stages, with aim to obtain design parameters. Obtention of metallurgical sample for regrind, thickening and filtration testing.
	Cu and iron concentrate regrind	Vendor testing	Cu regrind testing completed on Cu rougher concentrate obtained from the flotation pilot plant (Year 1-5 composite sample); Iron regrind testing completed on iron rougher concentrate obtained from the magnetic separation pilot plant (Year 1-5 composite sample).
	Cu and iron concentrate settling and thickening	Vendor testing	Cu concentrate characterization, settling, and dynamic thickening testing completed on Cu concentrate obtained from the flotation pilot plant (Year 1-5 composite sample); Iron concentrate characterization, settling, and dynamic thickening testing completed on Iron concentrate obtained from the magnetic separation pilot plant (Year 1-5 composite sample).
	Cu and iron concentrate filtration	Vendor testing	Cu concentrate filtration testing completed on thickened Cu concentrate obtained from the flotation pilot plant (Year 1-5 composite sample): Iron concentrate filtration testing completed on thickened iron concentrate obtained from the magnetic separation pilot plant (Year 1-5 composite sample).
	Tailings settling	Tailpro Consulting, Vietti Slurrytec	Characterization, settling, and dynamic thickening of tailings samples. Semi-pilot thickening testing with aim to confirm / optimize tailings thickening design, using final tailings from flotation pilot test work and magnetic separation pilot test work for a Year 1-5 composite sample. Variability samples study on 20 composite samples.
		Metso	Vendor semi-pilot thickening testing with aim to confirm / optimize tailings thickening design, using final tailings from flotation pilot test work and magnetic separation pilot test work for a Year 1-5 composite sample.
	Mineralogical characterization	CEM Geoatacama	Mineralogical characterization of 140 variability samples covering LOM mineralized geological units via SEM (TIMA).
	Cu flotation	Aminpro	140 SKT flotation test work completed on rougher and cleaner stages for copper and gold algorithm update, including circuit simulation and modelling for Cu concentrate algorithm development.

Date	Test work Type	Laboratory/ Test work Facility	Work Performed
	Dilution test	SGS Santiago	Cleaner dilution flotation test work completed on Year 1 – 5 composite sample and 38 variability samples.
	Magnetite concentrate	Aminpro	Magntite concentrate test work on 39 variability samples covering Santo Domingo and Iris Norte deposit for mineralized geological units through LOM. Processing via LIMS rougher stage, regrind and classification to target P_{80} of 40 µm, hydroseparation and 8- LIMS cleaning stages, using desalinated sea water.
2024	Cobalt column leach	Aminpro	Leach columns to confirm leachability of Mantoverde and Santo Domingo pyrite to extract cobalt.
	Cobalt ion exchange pilot	Mantoverde site	Simulated cobalt raffinate solutions prepared and passed through a continuous ion exchange unit to confirm ability of resins to recover and concentrate cobalt.
	Iron concentrate pelletization	Natural Resources Research Institute (NRRI)	Evaluation of pellet quality produced from Santo Domingo iron concentrate, using sample obtained from the iron pilot plant program (Table 13-1, Year 2023), including several bench – and full – scale pot grate test.

13.1.2 Comminution

13.1.2.1 2011 Pre-Feasibility Study Phase Test Work

During the 2011 pre-feasibility study phase, A total of 128 core samples were selected. BWi, RWi, Ai and SPI tests were completed (see Table 13-1, year 2010); 110 samples defined as DJB Consultants Inc or Main Drill Core Samples were used as a preliminary estimate for comminution requirements.

SMC testing was completed on 19 of the Main Drill samples (see Table 13-1, year 2011). These results were used to finalize the 2011 pre-feasibility study comminution requirements for the grinding circuit.

13.1.2.2 2014 Feasibility Study Phase Test Work

Two comminution circuit sizing exercises were completed as part of the 2014 feasibility study phase:

- SMC tests were performed on the remaining 91 of the 110 Main Drill Core Samples from 2011. Circuit design was re-evaluated using the SMC test methodology to verify the comminution circuit throughput capacities (see Table 13-1, year 2012).
- A second SMC testing campaign using 58 samples (Infill Samples) was completed to increase the data set and confidence level of the mill power requirements. The 58 Infill Samples were taken within the area of the proposed Santo Domingo open pit and were obtained from within the area planned to be the source of material processed in the first three years of production (see Table 13-1, year 2011).

The Main Drill Core Samples returned the following results:

- For the Iris and Iris Norte ores there is no significant difference in competency between magnetite and hematite ores. The Iris ores are the softest materials, with an Axb value of about 80.
- The BWi showed no significant differences between the hematite and magnetite zones for the Iris and Iris Norte areas.

- The Santo Domingo material shows significant differences between the hematite and magnetite feed types, with a 25% variance in the Axb values of the ore types. The hematite zone is the most competent with the lower Axb value of 39.3.
- In the Santo Domingo deposit, the hematite zone is 15% harder than the magnetite zone with respect to BWi, with average BWi indices of 14.1 kWh/t and 12.1 kWh/t, respectively.

The Infill Samples had the following characteristics:

- The hematite ores have an Axb value of 42.2 and are the most competent ores in the first 3 years of operation. The magnetite ores are 10% less competent with an Axb of 46.5.
- There is a similar trend with respect to the BWi with the hematite ores being 15% harder than the magnetite ores with BWi values of 13 kWh/t and 11.2 kWh/t, respectively.
- The RWi showed differences of around 10%, with averages of 14.4 kWh/t and 13.4 kWh/t for the hematite and magnetite ores, respectively.

13.1.2.3 2015 Pilot Plant Sample

During the 2015 pilot plant test work, eight composites were included for hardness analysis (see Table 13-1, year 2015). These composites represented each of the first 5 years of operation as identified at the time, plus a combined composite; they were tested mainly for BWi, RWi, Ai and SMC.

13.1.2.4 2021 Feasibility Study Optimization Test Work

An additional 45 variability samples were selected during 2021 (see Table 13-1, year 2021). Tests completed included mainly BWi, RWi and SMC.

The results of the SMC tests reported an Axb value of about 61, classified as soft ore. The RWi reported an average value of around 13.5 kWh/t. Direct measurement of BWi indicated a lower work index value than expected from the Santo Domingo project full dataset (by around 15%). An investigation concluded that it was related to an uncalibrated new piece of laboratory equipment and a correction factor was used to correct for this bias, resulting in an average corrected value of around 11.9 kWh/t.

13.1.2.4.1 Overall Data Analysis

The analysis of the Santo Domingo comminution data set completed by Ausenco delivered a 75%-percentile value of 36.7 for Axb; a 75%-percentile value of 14.1 kWh/t for BWi and a 75%-percentile of 16.4kWh/t for RWi. This percentile was lately used for the design.

13.1.3 Bulk Density

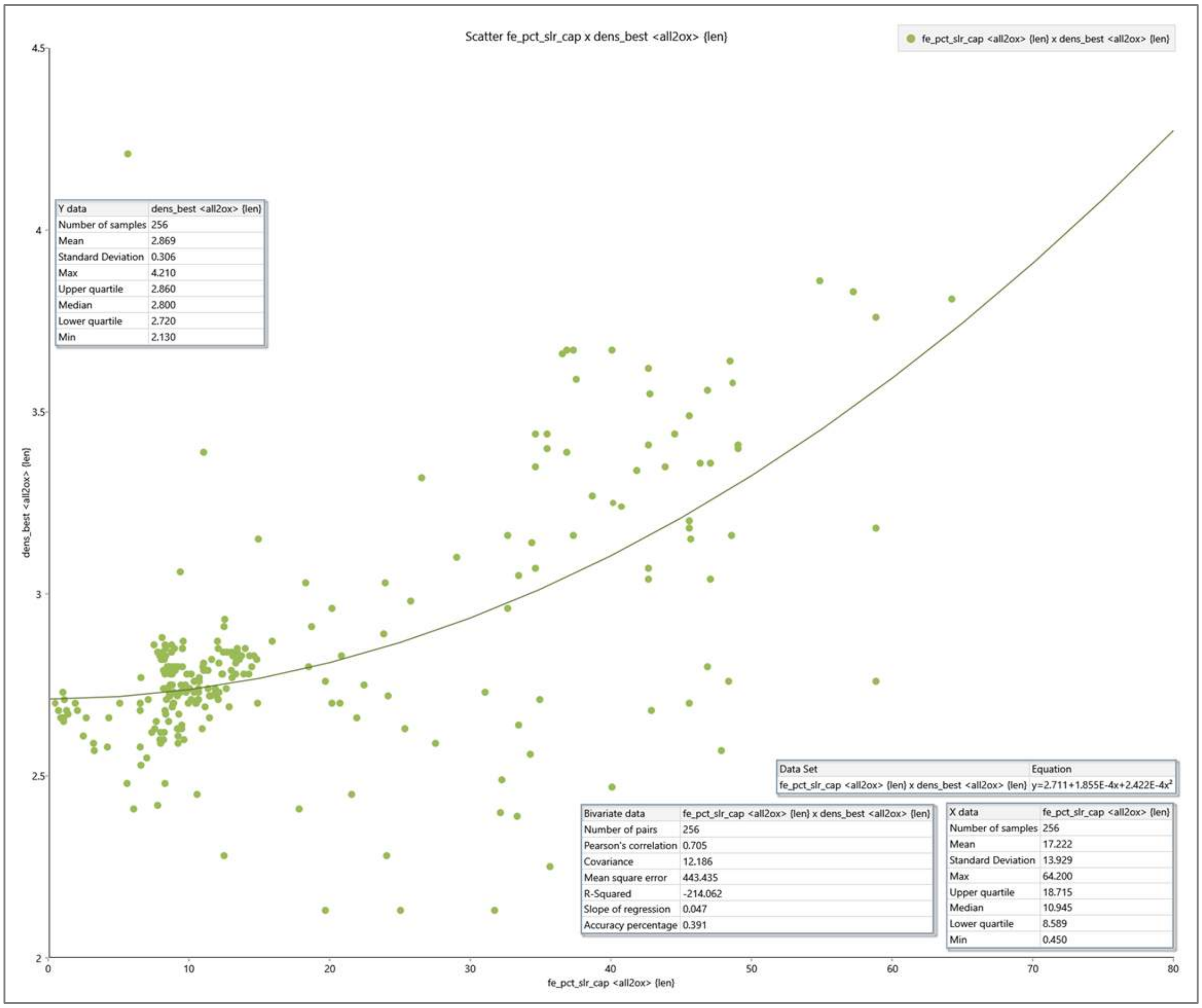
13.1.3.1 Density Regression Analyses and Application

The Capstone density database provided to SLR in October 2021 contains 7,039 density samples. Of these, 1,576 were discarded (895 densities from core box weights, 295 pycnometer measurements, 386 unknown), leaving 5,463 density samples available for use in the Mineral Resource estimate for all areas. There are 4,784 samples in SD-IN. These density samples do not cover the volume of the Mineral Resource estimate evenly, or adequately in several areas, especially in oxide material. Since changes in density in the mineralized material are driven by changes in iron content, Capstone chose to use regression formulae using the relationship between the measured densities and the iron content in the rock to estimate the density in all samples with Fe assays. SLR thus elected to continue using the regression methodology from the previous Mineral Resource, updated with the new density data provided. Also, having density values in each sample allows its use as a weight in interpolation (Section 14.3.7 and Figure 14-16).

SLR attempted to produce density curves for each area and ore type, but found that there was not enough data to produce separate curves for SD and IN. SLR then decided to group SD and IN together for three different regression formulae to apply to the Mineral Resource database assay table: one for oxide, one for mixed, and one for sulphide material. SLR also found that quadratic equations fit the density datasets better than the previously used linear equations. The regression curves are shown in Figure 13-1 to Figure 13-3.

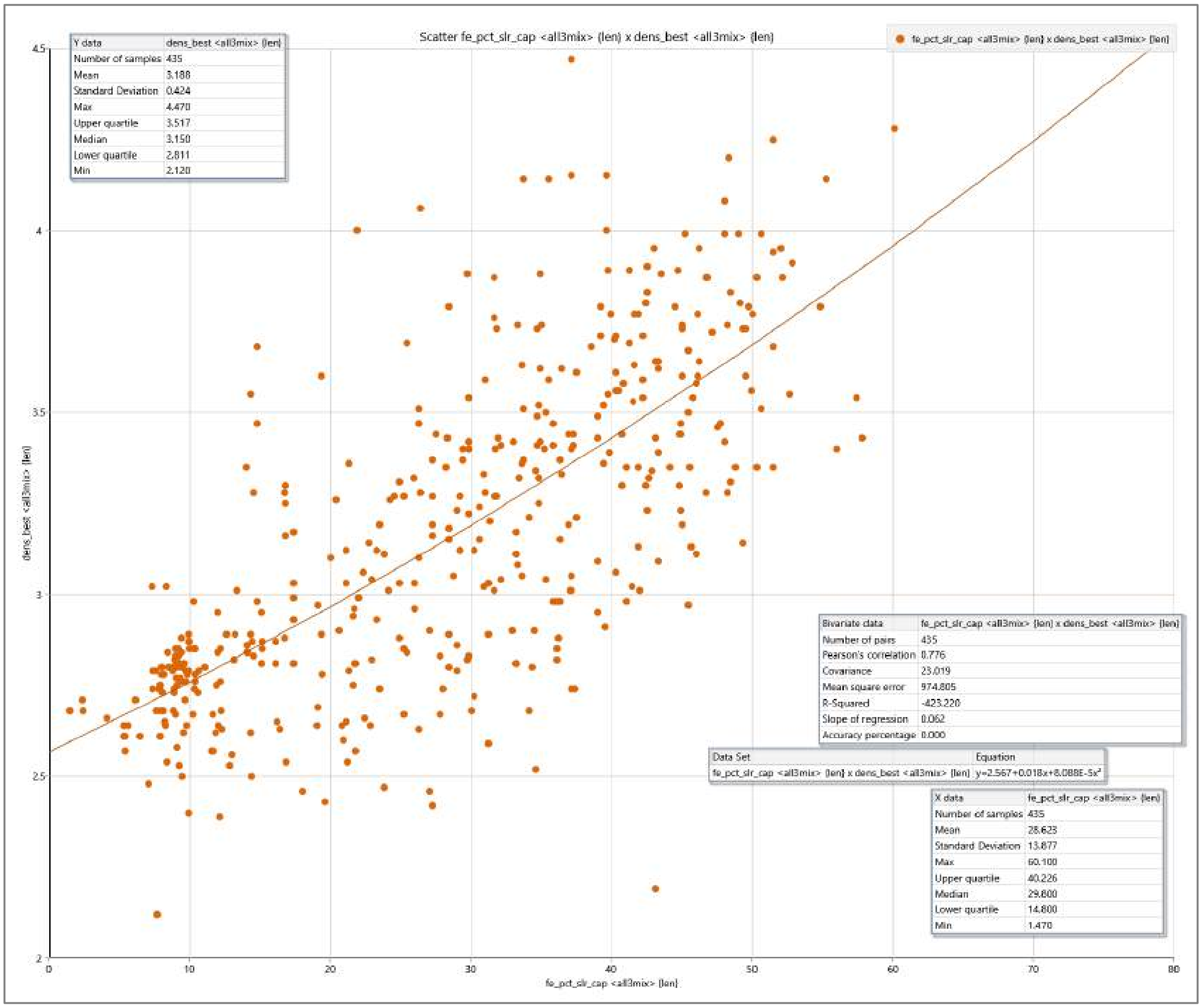
Density estimates were run for all redox zones based on the regression formulae, and the remaining un-estimated blocks in each lithology were set to the average density for each lithology.

Figure 13-1: Density Regression Curve for Oxide Material: SD + IN



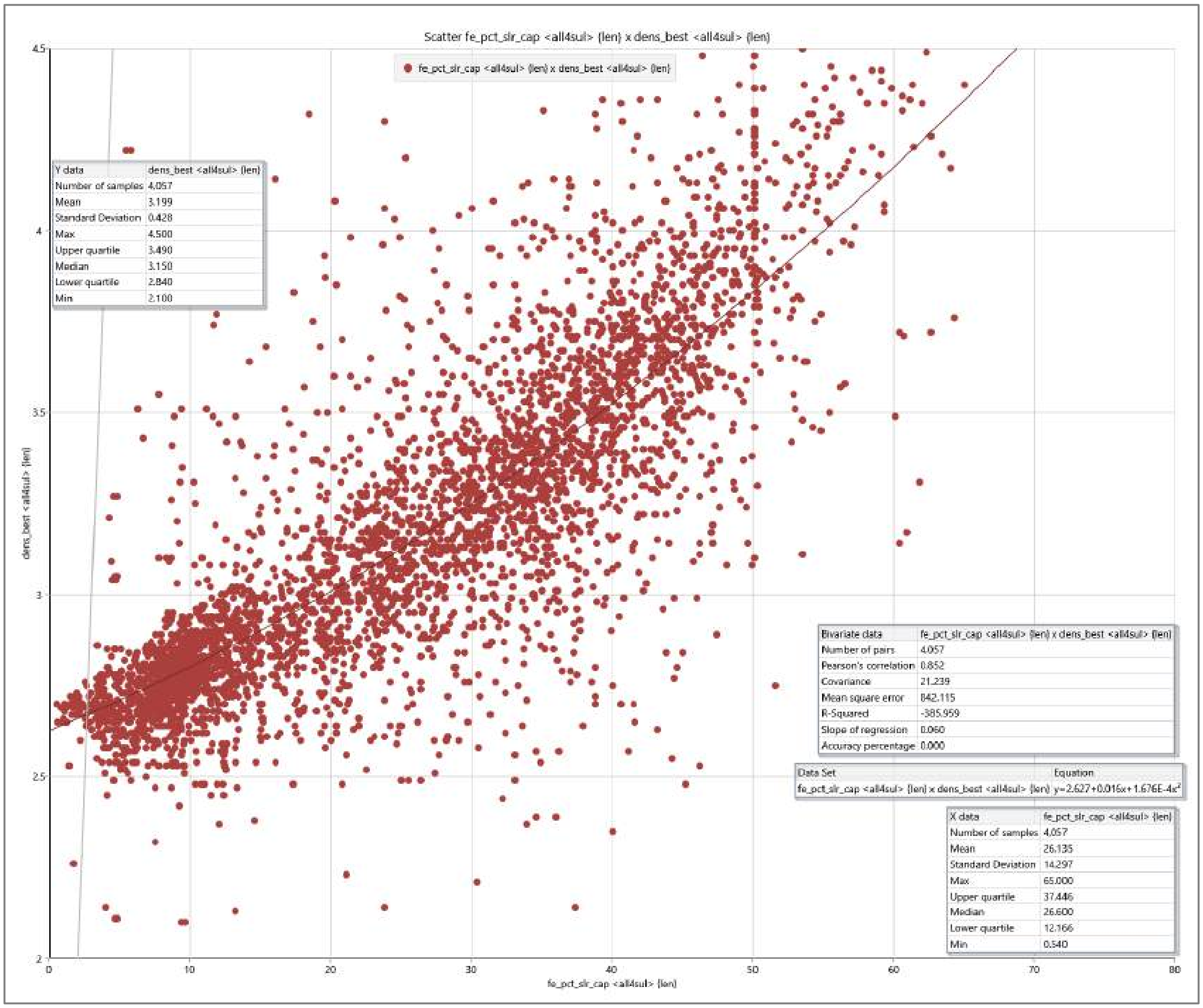
Source: Figure prepared by SLR, 2023.

Figure 13-2: Density Regression Curve for Mixed Material: SD + IN



Source: Figure prepared by SLR, 2023.

Figure 13-3: Density Regression Curve for Sulphide Material: SD + IN



Source: Figure prepared by SLR, 2023.

13.1.4 Copper Flotation

13.1.4.1 Pre-2018 Flotation Test Work

Bench scale flotation test work on composite and variability samples – including kinetic tests, OCTs and LCTs – was conducted during this period using direct sea water – as per the project definition at that time (see Table 13-1, from year 2008 to 2015. A pilot plant using direct sea water was also completed during this period (see Table 13-1, year 2015), with the objective to produce concentrate and tailings for downstream testing. Due to some difficulties with meeting iron product marketing specifications when processing the ore with direct sea water, the decision was made to use desalinated water for the process.

13.1.4.2 2018 to 2020 Flotation Test Work

The 2018 flotation test program (see Table 13-1, year 2018) consisted of rougher and cleaner kinetic tests on composite and variability samples from the 2014-2015 drill program using desalinated sea water. The program provided the first desalinated water flotation results, which were used to establish initial copper and gold recovery models.

Fresh drill core samples, obtained in 2019, were used to confirm the results. These fresh drill core samples were used for a test program including OCT and LCT on composite samples representing the first five years of production, and rougher and cleaner SKT tests on 28 variability samples, to obtain kinetics models of the process for development of recovery equations for copper and gold including rougher and cleaner flotation stages (see Table 13-1, year 2019); these equations were used for the cobalt section at the time. Full head samples characterization including mineralogy was completed on all composite and variability samples.

The results showed that rougher flotation at natural pH in desalinated water yielded good results and that lime gave excellent pyrite depression in the cleaners. Therefore, lime was established as the pyrite depressant instead of sodium metabisulfite as previously established.

A copper and pyrite flotation pilot plant were completed during 2020 (see Table 13-1, year 2020), using desalinated seawater, the flotation conditions established during 2019 and a Year 1-5 composite sample. Initial challenges were faced during the pilot trials at the rougher flotation stage, mainly associated to hydrodynamic issues that were attributed to the high specific gravity of the sample. Modifications to the equipment were made towards the end of the operation allowing the achievement of copper recovery of 91.5% and concentrate grade of 21% Cu.

13.1.4.3 Feasibility Update Flotation Test Work (2021 to 2023)

An additional and comprehensive flotation geometallurgical test program was completed during 2022 and 2023 (see Table 13-1, from year 2022 to 2023). The program was conducted with desalinated sea water and included:

- Bench scale test work on composite samples covering LOM mineralized geological units,
- Bench scale tests on yearly composite samples covering LOM,
- Bench scale extended 20-cycle LCT on Year 1-5 composite sample to simulate a similar closed loop reclaim water system,
- Flotation pilot plant using a Year 1-5 composite sample,
- Bench scale tests on 140 variability samples covering LOM.
- Flotation dilution tests with objective to assess alternative for a copper flotation scalping design.

The objective of the initial work completed on the composite samples was to optimize flotation conditions considering as the base case, a bulk Cu and Pyrite rougher flotation approach, followed by selective Cu flotation in cleaner stages. Design criteria was reviewed and updated based on the results of this program.

The objective of the variability test was the obtention of updated recovery equations for copper and gold. Copper concentrate grade was also derived from these tests.

13.1.4.3.1 Composite Samples Covering Mineralized Geological Units

Chemical and mineralogical content of the composite samples used for the test work are presented in Table 13-2 and Table 13-3.

Table 13-2: Composite Samples Head Grade

Composite ID	Assays				
	Cu (%)	Au (g/t)	Co (%)	Fe (%)	S (%)
VCLC_TOP	0.41	0.04	0.03	35.7	2.40
VCLC_TOP_2	0.56	0.09	0.04	43.0	3.02
VCLC_BOT	0.30	0.04	0.03	33.0	2.62
VCLC_BOT_2	0.32	0.04	0.027	33.3	2.30
ANDESITE COMP	0.27	0.03	0.02	16.8	1.20
ANLT COMP	0.36	0.05	0.02	25.4	1.82
BRECCIA COMP	0.34	0.05	0.01	21.0	1.25
MC YR0-1	0.64	0.10	0.04	36.0	3.70
MC YR1-5	0.46	0.06	0.03	30.9	2.32
MC YR6-10	0.30	0.03	0.03	37.0	2.67
MC YR11-18	0.17	0.02	0.02	28.2	2.11

Source: Blue Coast Research, Canada, 2022.

Table 13-3: Composite Samples Mineralogical Composition

Mineral	VCLC-Top	VCLC-Bot	Andesite Comp	ANLT Comp	Breccia Comp	MC YR 1-5	MC YR 6-10	MC YR 11-18
Pyrite	4.46	5.28	1.91	3.25	1.94	3.87	5.08	3.42
Chalcopyrite	0.78	0.65	0.6	1.03	0.58	1.29	0.69	0.3
Bornite	0.08	0.06	0	0.03	0.05	0.11	0.04	0
Tetrahedrite-Tennantite	0	0	0	0	0.01	0	0	0
Chalcocite	0.02	0.02	0.07	0.02	0.02	0.03	0.02	0.06
Covellite	0	0	0	0	0	0	0	0
Copper (native)	0	0	0	0	0	0	0	0
Sphalerite	0	0.02	0	0	0	0	0	0.03
Galena	0	0	0	0	0	0	0	0.01
Iron oxide	41.97	37.11	16.38	26.53	22.44	35.73	45.09	33.26
Magnetite-Ti	0.08	0.03	0.39	0.02	0.24	0.06	0.02	0.06
Ilmenite	0.03	0.01	0.02	0.04	0.17	0.03	0.01	0

Mineral	VCLC- Top	VCLC- Bot	Andesite Comp	ANLT Comp	Breccia Comp	MC YR 1-5	MC YR 6-10	MC YR 11-18
Rutile	0.08	0.04	0.11	0.13	0.5	0.09	0.06	0
Dolomite	0.06	0	0	0.02	0.23	0.05	0	0
Ankerite	2.9	2.29	0.06	2.47	7.07	3.25	1.86	0.01
Siderite	5.85	3.47	0.1	3.14	6.26	4.09	3.97	0.02
Calcite	2.68	3.36	1.78	3.61	5.46	3.9	2.65	1.33
Other	0.09	0.04	0.17	0.05	0.1	0.1	0.07	0.1
Quartz	1.2	0.9	0.77	1.13	1.36	1.17	0.87	0.6
Plagioclase	6.71	4.67	39.53	10.38	19.66	8.24	3.44	17.3
Orthoclase	16.81	17.14	11.43	19.4	16.69	18.11	16.73	9.42
Pyroxene	2.66	7.15	9.76	4.43	1.84	4.81	3.13	13.69
amphibole	1.61	1.59	2.3	3.29	1.32	1.57	1.43	4.56
Zircon	0	0	0	0	0	0	0	0
Titanite	0.14	0.29	3.12	1	0.41	0.29	0.4	1.37
Chlorite	6.41	8.85	6.53	11.1	4.95	7.4	9.61	8.95
Biotite	1.22	0.43	0.29	0.39	0.27	0.26	0.46	0.42
Muscovite/Sericite	1.28	1.19	1.08	1.98	2.18	1.15	0.9	1.07
Clays	0.03	0.02	0	0.06	0.06	0.03	0.01	0
Misc. Silicates	1.46	1.64	1.59	2.07	3.05	1.69	1.18	0.47
Unknown	0.84	0.73	1.29	1.17	1.97	0.8	0.68	0.46
Epidote	0.36	2.7	0.16	3.02	0.4	1.61	1.28	2.68
Gypsum/Anhydrite	0	0	0	0	0.25	0	0	0
Barite	0	0	0.01	0.02	0.05	0.01	0	0
Apatite	0.2	0.32	0.55	0.23	0.47	0.26	0.3	0.39
Total	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**

Source: Blue Coast Research, Canada, 2022.

Additional sample was prepared for the volcaniclastic units (labeled as: VLCL_TOP_2 and VLCL_BOT_2), as required during the metallurgical test work program. Also, a composite sample to represent the period Year 0-1 was prepared. The chemical assays of these samples are presented in Table 13-4.

Table 13-4: Composite Samples Head Grade

Composite ID	Assays				
	Cu (%)	Au (g/t)	Co (%)	Fe (%)	S (%)
VLCL_TOP_2	0.56	0.09	0.04	43.0	3.02
VLCL_BOT_2	0.32	0.04	0.027	33.3	2.30
MC YR0-1	0.64	0.10	0.04	36.0	3.70

Source: Blue Coast Research, Canada, 2022.

Initial flotation test work focused on confirming flotation reagents and pH conditions using the main mineralized geological units (volcaniclastic rocks top and bottom (see Table 13-5 and Table 13-6).

Several tests were completed using different reagents and doses, including rougher and cleaner kinetic tests for pulp zone kinetic parameter establishment, open and locked cycle tests (six cycles with water recirculation). Test work focused on copper rougher, copper cleaning, and pyrite flotation. The following tables present the OCT and LCT results for the final optimized reagent and pH conditions. The results indicate that a successful copper / pyrite separation was achieved. It should be noted that recovery to concentrate reported on OCTs within section 13 only relates to third cleaning concentrate copper content versus feed copper content and therefore it does not incorporate contribution from intermediate streams (metallurgical laboratory procedure); in this sense, LCT and pilot plant results are to be considered as more representative of metal recovery which consider contribution from intermediate streams in a continuous-closed mode.

Table 13-5: Results of Open Cycle Test on Volcaniclastic Rock Sample

Variable	Grade					Recovery (%) to Concentrate				
Circuit/Element	Au (g/t)	Co (%)	Cu (%)	Fe (%)	S (%)	Au (%)	Co (%)	Cu (%)	Fe (%)	S (%)
Cu Circuit Concentrate	2.73	0.15	26.9	32.1	34.7	73.0	7.4	87.8	1.4	21.3
Py Circuit Concentrate	0.17	0.73	0.14	45.7	50.6	10	78.9	1.0	4.3	68.3
Calculated Head	0.07	0.037	0.56	42.1	2.96	-	-	-	-	-

Source: Blue Coast Research, Canada, 2022. OCT recoveries not representative of a full-scale, continuous-closed circuit.

Table 13-6: Results of LCT on Volcaniclastic Rock Sample (Average Last Three Cycles)

Variable	Grade					Recovery (%) to Concentrate				
Circuit/Element	Au (g/t)	Co (%)	Cu (%)	Fe (%)	S (%)	Au (%)	Co (%)	Cu (%)	Fe (%)	S (%)
Cu Circuit Concentrate	2.88	0.08	28.8	31.1	32.7	74.0	3.9	90.7	1.3	18.9
Py Circuit Concentrate	0.26	0.73	0.14	46.6	51.0	17.3	89.2	1.2	5.0	76.9
Calculated Head	0.07	0.037	0.54	42.1	2.97	-	-	-	-	-
Volcaniclastic top rock (sample identification: VLCL_TOP_2)										

Source: Blue Coast Research, Canada, 2022.

The optimum flotation conditions established are shown in Table 13-7. Based on the test program, the approach of a Cu and Pyrite bulk rougher flotation stage followed by selective Cu flotation cleaner stage was confirmed to be viable.

Table 13-7: Rougher and Cleaner Optimal Flotation Conditions

Stage/Parameter	P_{80} (µm)	pH	Collector Ethyl Xanthate (g/t)
Copper flotation			
Primary grind	150	—	25
Rougher	—	9.5	20
Regrind	34	—	-
Cleaner	—	12	5
Pyrite Flotation			
Pyrite flotation	34	9.0	111

Source: Blue Coast Research, Canada, 2022.

The same flotation conditions were used to perform OCT´s on composite samples representing the rest of the mineralized geology units with the objective to confirm reagent and pH conditions selected. The results, presented from Table 13-8 to Table 13-10, confirm the viability of the flotation conditions selected.

Table 13-8: Results of OCT for Andesite Rock Composite Sample

Variable	Grade					Recovery (%) to Concentrate				
Circuit/Element	Au (g/t)	Co (%)	Cu (%)	Fe (%)	S (%)	Au (%)	Co (%)	Cu (%)	Fe (%)	S (%)
Cu Circuit Concentrate	2.04	0.26	24.8	31.2	35.1	55.3	10.9	84.0	1.7	27.0
Py Circuit Concentrate	0.41	1.01	0.33	45.1	50.6	14.4	54.0	1.4	3.1	50.0
Calculated Head	0.03	0.02	0.27	16.9	1.19	-	-	-	-	-
Andesite rock (sample identification: Andesite comp)										

Source: Blue Coast Research, Canada, 2022. OCT recoveries not representative of a full-scale, continuous-closed circuit.

Table 13-9: Results of OCT for Andesitic Lithic Tuff Rock Composite Sample

Variable	Grade					Recovery (%) to Concentrate				
Circuit/Element	Au (g/t)	Co (%)	Cu (%)	Fe (%)	S (%)	Au (%)	Co (%)	Cu (%)	Fe (%)	S (%)
Cu Circuit Concentrate	2.42	0.16	27.4	33.8	34.8	60.0	7.2	89.5	1.6	22.8
Py Circuit Concentrate	0.28	0.79	0.21	48.9	51.0	11.8	62.7	1.2	4.0	57.0
Calculated Head	0.05	0.03	0.38	25.8	1.88	-	-	-	-	-
Andesitic lithic tuff rock (sample identification: ANLT comp)										

Source: Blue Coast Research, Canada, 2022. OCT recoveries not representative of a full-scale, continuous-closed circuit.

Table 13-10: Results of OCT for Breccia Rock Composite Sample

Variable	Grade					Recovery (%) to Concentrate				
Circuit/Element	Au (g/t)	Co (%)	Cu (%)	Fe (%)	S (%)	Au (%)	Co (%)	Cu (%)	Fe (%)	S (%)
Cu Circuit Concentrate	2.51	0.10	27.7	27.5	29.7	54.4	7.2	84.8	1.6	27.3
Py Circuit Concentrate	0.67	0.74	1.04	44.6	49.2	17.8	62.8	3.9	3.1	55.7
Calculated Head	0.05	0.02	0.38	20.7	1.27	-	-	-	-	-
Breccia rock (sample identification: Breccia comp)										

Source: Blue Coast Research, Canada, 2022. OCT recoveries not representative of a full-scale, continuous-closed circuit.

13.1.4.3.2 Yearly Composite Samples Covering LOM

Bench scale tests were also completed using selected flotation conditions, on yearly composite samples covering LOM, considering samples representing the periods indicated below – as per the mine plan at the time. The calculated copper head grade of the samples varied from 0.66% Cu (Year 0-1) to 0.16% Cu (Year 11-18). Sample (Year 1-5) reported 0.55% Cu and sample (Year 6-10) reported 0.28% Cu. These values are within an appropriate range compared to the updated mine plan.

- Initial mine plan period (Year 0-1)
- First 5 years of the mine plan (Year 1-5)
- Mid mine plan period (Year 6-10)
- Final mine plan period (Year 11-18).

These tests were completed with the objective to confirm reagent and pH conditions selected on yearly composites, also demonstrated the viability of the flotation conditions selected. Results are presented from Table 13-11 to Table 13-14.

Table 13-11: Results of OCT on Year 0-1 Sample

Variable	Grade					Recovery (%) to Concentrate				
Circuit/Element	Au (g/t)	Co (%)	Cu (%)	Fe (%)	S (%)	Au (%)	Co (%)	Cu (%)	Fe (%)	S (%)
Cu Circuit Concentrate	2.74	0.07	28.5	30.6	30.4	55.2	2.9	83.4	1.7	16.0
Py Circuit Concentrate	0.41	0.72	0.14	46.6	52.2	21.3	82.0	1.0	6.7	71.3
Calculated Head	0.10	0.04	0.66	34.9	3.68	-	-	-	-	-

Source: Blue Coast Research, Canada, 2022. OCT recoveries not representative of a full-scale, continuous-closed circuit.

Table 13-12: Results of OCT on Year 1-5 Sample

Variable	Grade					Recovery (%) to Concentrate				
Circuit/Element	Au (g/t)	Co (%)	Cu (%)	Fe (%)	S (%)	Au (%)	Co (%)	Cu (%)	Fe (%)	S (%)
Cu Circuit Concentrate	2.45	0.10	28.7	30.1	32.5	58.4	4.3	78.4	1.5	19.0
Py Circuit Concentrate	0.26	0.83	0.28	47.7	51.7	13.1	75.7	1.6	5.1	64.0
Calculated Head	0.06	0.04	0.55	29.7	2.57	-	-	-	-	-

Source: Blue Coast Research, Canada, 2022. OCT recoveries not representative of a full-scale, continuous-closed circuit.

Table 13-13: Results of OCT on Year 6-10 Sample

Variable	Grade					Recovery (%) to Concentrate				
Circuit/Element	Au (g/t)	Co (%)	Cu (%)	Fe (%)	S (%)	Au (%)	Co (%)	Cu (%)	Fe (%)	S (%)
Cu Circuit Concentrate	3.05	0.13	27.8	32.7	33.1	67.0	3.1	89.3	0.8	11.4
Py Circuit Concentrate	0.14	0.74	0.06	47.5	51.0	13.2	78.5	0.8	5.0	76.3
Calculated Head	0.04	0.04	0.28	37.2	2.63	-	-	-	-	-

Source: Blue Coast Research, Canada, 2022. OCT recoveries not representative of a full-scale, continuous-closed circuit.

Table 13-14: Results of OCT on Year 11-18 Sample

Variable	Grade					Recovery (%) to Concentrate				
Circuit/Element	Au (g/t)	Co (%)	Cu (%)	Fe (%)	S (%)	Au (%)	Co (%)	Cu (%)	Fe (%)	S (%)
Cu Circuit Concentrate	2.32	0.23	23.2	32.7	36.6	48.5	5.3	82.6	0.7	10.3
Py Circuit Concentrate	0.12	0.64	0.33	46.3	51.4	11.7	67.0	5.5	4.5	66.8
Calculated Head	0.03	0.03	0.16	27.7	2.07	-	-	-	-	-

Source: Blue Coast Research, Canada, 2022. OCT recoveries not representative of a full-scale, continuous-closed circuit.

13.1.4.3.3 Extended 20-cycles LCT

Bench scale extended 20-cycles Locked Cycle Test was also completed on a representative First five years composite sample (Year 1 -5 composite sample), to simulate a close loop reclaim water system. The results obtained during stability of test confirmed the metallurgical performance reporting 28.3% Cu concentrate grade and 91.3% Cu recovery.

Table 13-15: Results of 20-cycles LCT on Year 1-5 Composite Sample

Variable	Grade					Overall Recovery (%)				
Circuit/Element	Au (g/t)	Co (%)	Cu (%)	Fe (%)	S (%)	Au (%)	Co (%)	Cu (%)	Fe (%)	S (%)
Cu Circuit Concentrate	2.80	0.05	28.3	29.6	29.2	70.1	2.7	91.3	1.6	21.2
Py Circuit Concentrate	0.41	0.69	0.68	44.8	47.6	22.4	87.8	4.8	5.4	75.8
Calculated Head	0.07	0.03	0.52	30.7	2.32	-	-	-	-	-

Source: Blue Coast Research, Canada, 2022.

The analysis of the final Cu and pyrite concentrates obtained from this test work is presented in Table 13-16.

Table 13-16: Full Results of Final Cu Concentrate and Pyrite Concentrate from 20-Cycles LCT

Element	Unit	Final Cu Conc	Final Py Conc
Ag	g/t	21.6	7.5
Al	%	0.68	0.46
As	%	0.02	0.01
Au	g/t	2.92	0.36
Ba	ppm	109.61	49.45
Be	ppm	<0.2	<0.2
Bi	ppm	<2	9
C_{tot}	%	0.58	0.35
Ca	%	0.77	0.49
Cd	ppm	8.0	6.9
Cl	%	< 0.01	< 0.01
Co	ppm	467	7019
Cr	ppm	18	15
Cu	%	27.92	0.68

Element	Unit	Final Cu Conc	Final Py Conc
Ag	g/t	21.6	7.5
F	%	< 0.01	< 0.01
Fe	%	30.21	47.11
Ga	ppm	<20	20.97
Ge	ppm	<20	21
Hf	ppm	<20	<20
Hg	ppb	824	93
In	ppm	<20	<20
K	%	0.2	0.1
Li	%	9	5
Mg	%	0.39	0.24
Mn	ppm	390	238
Mo	ppm	68	54
Na	%	0.08	0.07
Nb	ppm	<10	<10
Ni	ppm	31	261
P	%	0.269	0.018
Pb	ppm	236.22	31.20
Pd	g/t	<0.01	0.0125
Pt	g/t	0.01	0.01
Rb	ppm	<20	<20
Re	ppm	53.55	96.83
S_{tot}	%	28.92	47.58
Sb	ppm	46.55	15.95
Se	ppm	31	<10
Si	%	1.99	1.48
Sn	ppm	34	<10
Sr	ppm	24	11
Ta	ppm	21	25
Te	ppm	45	37
Ti	%	0.08	0.05
Tl	ppm	3	17
V	ppm	25.37	28.76
W	ppm	29	<10
Zn	ppm	694	130
Zr	ppm	37	31

Source: Blue Coast Research, Canada, 2022.

13.1.4.3.4 Copper and Pyrite Flotation Pilot Plant

Once the bench scale testing was finalized, a continuous flotation pilot plant test program was completed using 61 tonnes of Year 1 – 5 composite sample and desalinated water with conventional flotation cells. The objective was to confirm the capability to demonstrate production of saleable Cu concentrate and to obtain samples of representative concentrate and tailings for characterization and testing of downstream processes.

The pilot plant was initiated with a commissioning sample which allowed for the equipment to be calibrated for the design grind size, pH and reagent settings, and flotation stage residence times.

The continuous flotation pilot plant consisted of the following equipment and configuration:

- Ball mill (1) equipped with a spiral classifier (1),
- Conventional rougher flotation cells (5),
- Regrind ball mill treating rougher concentrate (1),
- Conventional first cleaner (4), second cleaner (2) and third cleaner flotation cell (1).

The reagents type and dose used were those established during the optimization flotation test program (Table 13-7) with minor adjustment during the trials (ethyl xanthate increased to 125 g/t for the pyrite circuit).

The pilot plant was initiated with a commissioning sample which allowed for the equipment to be calibrated for the design grind size, pH and reagent settings, and flotation stage residence times. Once commissioned, the 61-tonne composite representing years one through five was piloted. The results are shown in Table 13-17. Full chemical assays and ICP of the final Cu concentrate is presented from Table 13-18 to Table 13-20. Those of the pyrite concentrate are shown from Table 13-21 to Table 13-23. Particle size distribution of the concentrate samples was also measured, reporting P_{80} of 43 µm for the Cu concentrate and average P_{80} of 70 µm for the pyrite concentrate.

Table 13-17: Results from Copper and Pyrite Flotation Pilot Plant

Variable	Grade					Overall Recovery (%)				
Circuit/Element	Au (g/t)	Co (%)	Cu (%)	Fe (%)	S (%)	Au (%)	Co (%)	Cu (%)	Fe (%)	S (%)
Cu Circuit Concentrate	2.06	0.07	26.0	-	29.3	53.8	4.9	91.1	-	27.3
Py Circuit Concentrate	0.33	0.67	0.60	-	48.9	13.5	78.9	2.8	-	64.6
Calculated Head	0.07*	0.03	0.56	-	2.29	-	-	-	-	-
*Measured value										

Source: Asmin Industrial Limited, Chile, 2023.

Table 13-18: Chemical Assay Results for Final Cu Concentrate from Flotation Pilot Plant

Cu T	Fe T	S	Au	Pd	Pt	Ins.	Ag	Co	Mn	B	Hg	Se
(%)	(%)	(%)	(g/t)	(g/t)	(g/t)	(%)	(ppm)	(%)	(%)	(ppm)	(ppm)	(ppm)
26.0	26.81	28.78	1.9	<0.10	<0.10	9.4	20	0.064	0.04	<100	<50	<100

Source: Asmin Industrial Limited, Chile, 2023.

Table 13-19: ICP Results for Final Cu concentrate from Flotation Pilot Plant

Al	As	Ba	Be	Bi	Ca	Cd	Co	Cr	K	La	Mg	Mo
(%)	(ppm)	(ppm)	(ppm)	(ppm)	(%)	(ppm)	(ppm)	(ppm)	(%)	(ppm)	(%)	(ppm)
0.8	155	188	<10	<50	1.5	<10	704	<10	0.3	<100	0.39	<100

Source: Asmin Industrial Limited, Chile, 2023.

Table 13-20: ICP Results or Final Cu concentrate from Flotation Pilot Plant (cont'd)

Na	Ni	Pb	Sb	Sr	Te	Ti	Tl	V	Zn
(%)	(ppm)	(ppm)	(ppm)	(ppm)	(ppm)	(%)	(ppm)	(ppm)	(ppm)
0.67	<50	>5	<100	21	<100	<0.1	<200	<20	>5

Source: Asmin Industrial Limited, Chile, 2023.

Table 13-21: Chemical Assay Results for Final Pyrite Concentrate from Flotation Pilot Plant

Cu T	Fe T	S	Au	Pd	Pt	Ins.	Ag	Co	Mn	B	Hg	Se
(%)	(%)	(%)	(g/t)	(g/t)	(g/t)	(%)	(ppm)	(%)	(%)	(ppm)	(ppm)	(ppm)
0.782	41.73	48.50	1.00	<0.10	<0.10	3.80	3.00	0.672	0.01	<100	<50	<100

Source: Asmin Industrial Limited, Chile, 2023.

Table 13-22: ICP Results for Final Pyrite Concentrate from Flotation Pilot Plant

Al	As	Ba	Be	Bi	Ca	Cd	Co	Cr	K	La	Mg	Mo
(%)	(ppm)	(ppm)	(ppm)	(ppm)	(%)	(ppm)	(ppm)	(ppm)	(%)	(ppm)	(%)	(ppm)
0.40	<100	<100	<10	<50	0.40	<10	6780	97	<0.1	<100	0.20	120

Source: Asmin Industrial Limited, Chile, 2023.

Table 13-23: ICP Results or Final Pyrite Concentrate from Flotation Pilot Plant (continuation)

Na	Ni	Pb	Sb	Sr	Te	Ti	Tl	V	Zn
(%)	(ppm)	(ppm)	(ppm)	(ppm)	(ppm)	(%)	(ppm)	(ppm)	(ppm)
<0.10	590	<0.05	<100	14	<100	<0.1	<200	<20	>5

Source: Asmin Industrial Limited, Chile, 2023.

13.1.4.3.5 Bench Scale Dilution Cleaner Test

The bench scale dilution cleaner test was completed on a Year 1-5 composite sample using desalinated seawater, with the objective to assess alternative for a copper flotation scalping design.

Table 13-24: Results from Dilution Cleaner Test on Year 1-5 Composite Sample

Sample ID	Cu Head Grade (%)	Fe Head Grade (%)	S Head Grade (%)	Co-Head Grade (%)	2nd Dilution Cu grade (0.5 min) (%)
Year 1-5 Composite	0.535	28.84	2.41	0.0240	29.12

Source: SGS Minerals S.A., Chile, 2023.

The result of the dilution test with Cu concentrate grade of 29.12% Cu for Year 1-5 composite sample, is favorable for the alternative to consider a copper flotation scalping design. Dilution cleaner test on variability samples was also performed with similar results (see 13.1.4.3.8 Variability Samples Dilution Cleaner Test).

13.1.4.3.6 Variability Test Work

The variability test work completed during 2023 consisted of testing 140 samples using SKT tests on rougher and cleaner stages to develop a kinetics flotation model for deriving the recovery and concentrate grade equations for copper, and the recovery equation for gold.

A range of copper head grades was tested, from 0.02% Cu to 1.29% Cu with an average value of 0.48% for the set. The recovery equations developed from the results are discussed in Section 13.2 Recovery Estimates.

13.1.4.3.7 Variability Samples Characterization

Copper, gold, cobalt, iron, and sulphur content was chemically analyzed for the 140 variability samples – in addition, soluble copper content, full ICP, and mineralogy was completed on each sample. Table 13-25 to Table 13-27 show the key results. The number of samples tested for each geological unit is also included. Volcaniclastic rocks (top and bottom) are more abundant within the reserves and, therefore, mores samples from these lithologies were considered. Andesite rock was split between upper (ANDE UPPER) and lower (ANDE LOWER), while volcaniclastic within Iris Norte (VLCL) is also presented.

Table 13-25: Cu and Au Head Grade Statistics of the 140 Variability Samples

Item	Quantity (N°)	Cu (%)			Au (g/t)		
		Minimum (%)	Maximum (%)	Average (%)	Minimum (%)	Maximum (%)	Average (%)
VLCL TOP	50	0.035	1.287	0.654	0.013	0.199	0.085
VLCL BOTTOM	49	0.018	1.031	0.382	0.009	0.134	0.055
VLCL	11	0.027	0.789	0.247	0.008	0.093	0.047
ANDE UPPER	5	0.233	0.834	0.444	0.025	0.140	0.084
ANDE LOWER	2	0.047	0.213	0.130	0.012	0.031	0.022
ANLT	14	0.116	0.701	0.402	0.017	0.154	0.056
BRECCIA	9	0.075	0.920	0.452	0.024	0.141	0.069
Total	**140**	-	-	-	-	-	-

Source: Aminpro Chile Spa, Santiago, Chile, 2023.

Table 13-26: Co and Fe and S Head Grade Statistics of the 140 Variability Samples

Item	Quantity (N°)	Co (%)			Fe (%)			S (%)		
		Minimum (%)	Maximum (%)	Average (%)	Minimum (%)	Maximum (%)	Average (%)	Minimum (%)	Maximum (%)	Average (%)
VLCL TOP	50	0.003	0.073	0.034	22.090	57.790	37.780	0.070	6.930	2.960
VLCL BOTTOM	49	0.002	0.055	0.030	16.500	46.200	32.449	0.140	5.180	2.611
VLCL	11	0.012	0.046	0.027	20.320	44.860	32.965	1.700	5.430	3.311

Item	Quantity (N°)	Co (%)			Fe (%)			S (%)		
		Minimum (%)	Maximum (%)	Average (%)	Minimum (%)	Maximum (%)	Average (%)	Minimum (%)	Maximum (%)	Average (%)
ANDE UPPER	5	0.004	0.027	0.014	11.510	15.590	13.650	0.470	2.330	1.278
ANDE LOWER	2	0.001	0.025	0.013	19.380	20.510	19.945	0.070	1.050	0.560
ANLT	14	0.003	0.085	0.025	9.380	48.370	24.852	0.250	5.750	1.895
BRECCIA	9	0.005	0.029	0.015	11.610	37.870	23.022	0.410	2.620	1.477
Total	**140**	-	-	-	-	-	-	-	-	-

Source: Aminpro Chile Spa, Santiago, Chile, 2023.

Table 13-27: Average Copper Mineral Statistics of the 140 Variability Samples

Item	Quantity (N°)	Chalcopyrite	Bornite	Chalcocite/Digenite	Enargite/Tennantite
		Average (%)	Average (%)	Average (%)	Average (%)
VLCL TOP	50	1.514	0.073	0.056	0.003
VLCL BOTTOM	49	0.822	0.045	0.035	0.001
VLCL	11	0.619	0.040	0.085	0.002
ANDE UPPER	5	0.761	0.069	0.118	0.002
ANDE LOWER	2	0.140	0.002	0.020	0.000
ANLT	14	0.943	0.029	0.032	0.007
BRECCIA	9	0.711	0.077	0.108	0.010
Total	**140**	-	-	-	-

Source: Aminpro Chile Spa, Santiago, Chile, 2023.

13.1.4.3.8 Variability Samples Dilution Cleaner Test

Bench scale dilution cleaner tests were completed on 38 selected variability samples with the objective of assessing the variability of the ore with respect to the alternative of a copper flotation scalping design. The results confirm that the copper flotation scalping design remains favorable. Low Cu head grades of around 0.1%Cu reported the lower results and may require additional cleaning (re-cleaning).

Table 13-28: Results on Dilution Cleaner Test for Variability Samples

Sample ID	Cu Head Grade (%)	Fe Head Grade (%)	S Head Grade (%)	Co Head Grade (%)	Geological Unit	2nd Dilution Cu grade (0.5 min) %
Sample 121	0.123	39.79	2.89	0.040	VLCL_TOP	18.88
Sample 66	0.363	36.53	2.73	0.033	VLCL_BOTOM	24.90
Sample 38	1.132	37.34	3.68	0.030	VLCL_TOP	32.42
Sample 42	1.318	31.85	2.72	0.025	VLCL_TOP	33.00
Sample 120	0.753	44.98	3.02	0.033	VLCL_TOP	30.73
Sample 97	0.514	29.72	2.55	0.028	VLCL_BOTOM	27.06
Sample 81	0.478	42.10	3.35	0.028	VLCL_BOTOM	28.96
Sample 110	0.355	39.51	2.21	0.025	VLCL_BOTOM	26.74

Sample ID	Cu Head Grade (%)	Fe Head Grade (%)	S Head Grade (%)	Co Head Grade (%)	Geological Unit	2nd Dilution Cu grade (0.5 min) %
Sample 136	0.907	43.11	3.72	0.042	VLCL_BOTOM	29.37
Sample 74	0.939	40.95	2.09	0.017	VLCL_TOP	30.79
Sample 29	0.340	52.69	2.64	0.035	VLCL_TOP	21.45
Sample 79	0.607	39.40	2.79	0.024	VLCL_TOP	29.11
Sample 55	1.031	39.54	3.68	0.031	VLCL_BOTOM	30.01
Sample 26	0.335	28.77	2.26	0.030	VLCL_TOP	24.58
Sample 87	0.628	49.89	4.61	0.031	VLCL_TOP	31.29
Sample 70	1.027	41.33	2.69	0.022	VLCL_TOP	36.42
Sample 86	0.825	51.60	3.70	0.026	VLCL_TOP	27.92
Sample 18	0.496	33.54	2.99	0.043	VLCL_TOP	25.33
Sample 44	0.705	41.47	1.92	0.017	VLCL_TOP	29.87
Sample 53	0.692	36.75	2.12	0.025	VLCL_TOP	30.91
Sample 130	0.256	49.18	3.41	0.038	VLCL_BOTOM	18.00
Sample 129	0.497	37.31	2.41	0.023	VLCL_BOTOM	30.43
Sample 10	0.359	44.54	3.45	0.042	VLCL_TOP	28.13
Sample 77	0.481	38.25	2.04	0.014	VLCL_TOP	27.96
Sample 106	0.360	39.94	2.58	0.026	VLCL_TOP	27.75
Sample 39	0.729	21.70	3.29	0.032	VLCL_BOTOM	29.45
Sample 100	0.148	26.72	0.66	0.012	ANLT	14.26
Sample 5	0.791	35.24	1.81	0.017	BX	34.32
Sample 48	0.305	11.69	1.06	0.012	BX	25.54
Sample 124	0.239	12.33	0.58	0.010	ANDE_UPPER	19.99
Sample 54	0.336	9.51	0.69	0.009	ANLT	29.23
Sample 91	0.185	9.10	1.08	0.006	ANLT	21.39
Sample 46	0.468	11.32	2.21	0.021	ANDE_UPPER	32.19
Sample 59	0.116	36.32	3.11	0.044	ANLT	16.44
Sample 3	0.851	35.86	2.78	0.048	VLCL_BOTOM	38.53
Sample 33	0.542	42.12	1.13	0.026	VLCL_BOTOM	30.84
Sample 58	0.715	38.35	2.65	0.042	VLCL_TOP	44.96
Sample 8	0.489	24.70	1.56	0.018	VLCL_TOP	30.34

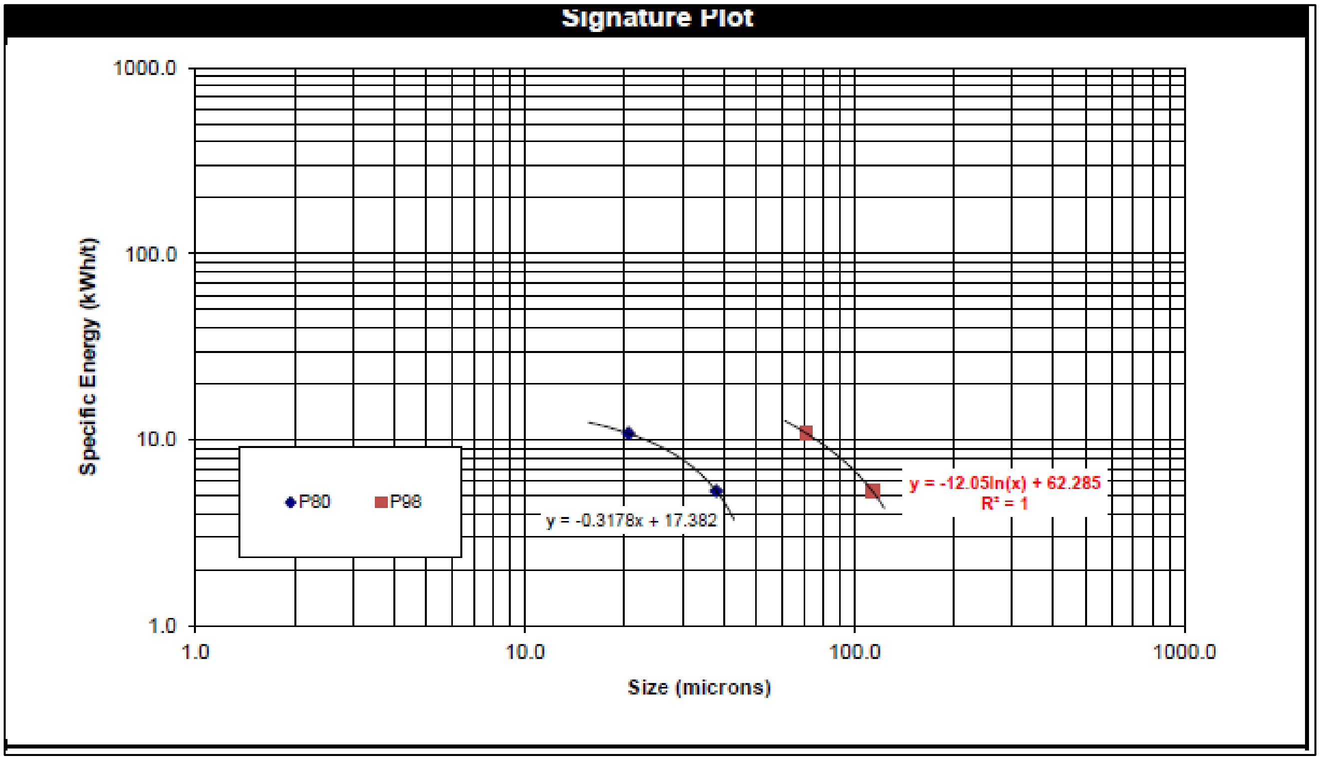
Source: SGS Minerals S.A, Chile, 2023.

13.1.5 Copper Regrind

Copper regrind test work has been carried out since 2011 when seawater was used for copper rougher concentrate obtention. During that time, vendor testing was completed with the objective to obtain the specific energy consumption value.

During 2023 specific IsaMill™ Grinding test work was completed with the objective to complete the signature plot for rougher copper concentrate, obtained from a representative Year 1-5 composite sample. The sample F_{80} was of about 68 µm and it was reground to target P_{80} of 34 µm.

Figure 13-4: Signature Plot for Rougher Copper Concentrate



Source: ALS Metallurgy Perth, Australia, 2023.

13.1.6 Copper Concentrate Thickening

13.1.6.1 Pre-2020 Concentrate Thickening Test Work

In 2014, tests were performed on a copper cleaner concentrate using synthetic seawater, identifying MF10 as the superior anionic flocculant for higher settling velocity and clearer overflow. Subsequent dynamic thickening and rheology tests analyzed feed rate effects on discharge concentration and overflow clarity, revealing better performance with lower feed rates despite reagent-induced foam, necessitating a de-aerator in the design. Recommendations for future test work included higher feed rates and the use of natural seawater without flocculants to achieve a 60% weight by weight solid concentration in the underflow. The 2015 pilot plant tests, utilizing seawater and contemporary flowsheets, aimed to refine thickener sizing and operating parameters. These tests, validated by low clay content, established the underflow density limits and necessary thickeners' torque, suggesting a repeat with desalinated water to confirm design criteria.

13.1.6.2 2020 to 2023 Concentrate Thickening Test Work

Settling and thickening test work for copper concentrate samples were completed in 2021 to further support the previous 2014 and 2015 testing campaigns. During the metallurgical piloting of 2020 to 2021, a single sample of copper concentrate was sent to Tailpro Consulting for dynamic thickening test work using equipment and procedures of Vietti Slurrytec for vendor-independent sizing of the industrial copper thickener. During 2023, additional settling and dynamic thickening testing was conducted on copper concentrate samples by three vendors with the intention of obtaining updated equipment sizing and corresponding thickener installation costs.

The 2023 design criteria considered that the copper concentrate thickener will operate with a nominal discharge density of 60% solids weight by weight, similar to the criteria considered as part of the 2014 project study.

Several vendor test programs were completed reporting that the copper concentrate exhibited an optimal specific feed rate of 0.24 t/h/m^2, with an upper value of 0.69 t/h/m^2 and a lower value of 0.15 t/h/m^2. Vietti Slurrytec's vendor-independent recommendation was 0.37 t/h/m^2 based on the 2020-2021 campaign sample and 0.25 t/h/m^2 based on the 2023 sample. The project adopted a thickening flux of 0.25 t/h/m^2 for the copper concentrate thickener. Vietti Slurrytec indicated that the residence time required is around 3 hours to achieve up to a maximum 90 Pa unsheared rheology for 65% solids weight by weight. The Maximum Operating Torque (MOT) and Normal Operating Torque (NOT) recommended by all parties are within the industry standard for the type of thickener specified as part of the 2023 production design criteria.

13.1.7 Copper Concentrate Filtration

13.1.7.1 Pre-2020 Concentrate Filtration Test Work

From 2011 to 2015 vendor filtration testing was performed on copper concentrate with the objective of determining the main filtration characteristics of the copper concentrate and the filter cake chloride content (from the use of direct seawater during the ore processing). The results are of little relevance to the current process, which considers desalinated water.

13.1.7.2 2020 to 2023 Concentrate Filtration Test Work

Samples were obtained from the 2023 Copper flotation pilot plant using desalinated seawater. Results confirmed pressure filters with filtration rate of 0.521 t/h/m^2 for design, based on Ausenco's recommendation.

13.1.8 Iron Concentration

13.1.8.1 Pre-2021 Iron Concentration Test Work

Davis Tube Test (DTT) and Low Intensity Magnetic Separators (LIMS) tests were conducted during this period using mainly direct seawater on composite and variability samples; LIMS was also used to recover magnetic iron for downstream testing, from the rougher flotation tailings during the 2015 flotation pilot plant using direct seawater (see Table 13-1, from year 2009 to year 2015). The DTT program on effective 206 samples when processing the ore with seawater, a decision was made to future use desalinated water for the process.

The change to desalinated water led to initial LIMS test work and to DTT variability programs for metallurgical response evaluation. The DTT program on 206 samples using desalinated seawater (see Table 13-1, year 2020) was completed at 80% less than 325 mesh or 44 µm (DTT local industry standard) and used as approximation to P80 design value of 40 µm. A range of iron head grades were tested, from 8.4% Fe to 61.9% Fe, with an average value of 30.1% Fe, from drill core representing the Santo Domingo and Iris Norte deposit. The general results demonstrated feasibility of achieving an iron concentrate target of 65% Fe, with an average mass recovery of 22.5%. This variability program was further complemented as described in Section 13.1.8.2.6.1 DTT Work.

In addition to the variability DTT program, approximately 5 tonnes of rougher flotation tailings from the 2020 desalinated water pilot test were processed in an industrial diameter (48 in) magnetic drum separator (see Table 13-1, year 2020). The study was conducted in two stages. The first stage consisted of a multivariable optimization study which allowed to find the optimum operating value for the following parameters evaluated:

- Feed flowrate,
- Feed solids percent,
- Equipment operating gap.

The second stage consisted of a process validation, using the optimum value selected for each parameter evaluated from stage one. From this second stage, process design criteria were established, and products were obtained for downstream testing.

The iron concentration processing consisted of a magnetic rougher stage, followed by regrind and classification by hydrocyclones at design P_{80} of 40 µm, followed by three cleaning stages.

The iron concentrate results of the validation stage are presented in Table 13-29.

Table 13-29: Results of 2020 Iron Concentration Pilot Plant at Validation Stage

Test	% Fe	% SiO_2	% CaO	% MgO	% P	% S	% Na_2O	% K_2O	% Mn	Cu ppm
2020 Iron Pilot Plant	64.94	4.60	0.82	0.48	0.013	0.008	0.180	0.130	0.069	70

Source: SGA Studiengesellschaft für Eisenerzaufbereitung GmbH & Co. KG, Germany, 2023.

The iron pilot plant achieved target iron concentrate grade of 64.94% Fe with an overall mass recovery of 15.7%. This concentrate was further used for additional operational units evaluation, including screening, hydroseparation and reverse flotation at bench scale (see Section 13.1.8.2.1 Composite Sample).

13.1.8.2 Iron Concentration Test Work (2021 to 2023)

13.1.8.2.1 Composite Sample

Exploratory bench scale test work including reverse flotation, screening and hydroseparation was completed mainly during 2021 on samples obtained from the iron concentration pilot plant (see Table 13-1, year 2021). The results indicated that reverse flotation has the potential to improve iron concentrate grade (from 65.1% Fe to 67.1% Fe) and significantly reduce silica content (from 4.7% SiO_2 to 3.0% SiO_2). Hydroseparator test work conducted on reground rougher concentrate showed some success, with iron concentrate grade improvement (from 52.9% Fe to 54.2% Fe) and silica content reduction (13.9% SiO_2 to 12.4% SiO_2). The high-frequency screening test work did not show promising results for the sample tested.

During 2022, an alternative LIMS study using bench L8 LIMS equipment and a representative Year 1-5 composite sample, tested increasing the cleaner magnetic stages from three - as per the design at the time, to eight cleaner magnetic stages. The results demonstrated that the iron concentrate grade improved through all stages tested and therefore the concept was later considered for design update. During this study, it was also considered a differentiation in magnetic strength of the cleaner circuit. The results of this test work indicated that difference in the iron concentration grade was achieved by using different magnetic strength in cleaner stages and therefore this concept was also considered for later design update. Finally, the test work considered reducing regrind particle size and although some improvement was reported to the iron concentrate grade, results were less favorable than the other parameters tested, and therefore this was not considered for design update. The results of the three-tests completed are provided in Table 13-30. Higher iron concentrate grade improvement from three cleaner stages to eight cleaner stages reported iron concentrate grade of 67.2% Fe (improvement from 65.6% Fe as measured by laboratory). Chemical assays as well as Loss of Ignition (LOI), specific weight and Blaine index completed on final concentrates are presented in Table 13-30.

Table 13-30: Final Iron Concentrate Specifications from L8 LIMS Test Work

Test	% Fe	% SiO_2	% CaO	% MgO	% P	% S	% Na_2O	% K_2O	% Mn	Cu ppm	%LOI	SG	Blaine
L8-1	66.94	3.13	0.63	0.34	0.008	0.003	0.100	0.086	0.061	150	-1,13	4,89	2960
L8-2	66.01	3.78	0.73	0.41	0.010	0.050	0.13	0.11	0.065	190	-0,83	4,84	2132
L8-3	67.22	2.91	0.59	0.32	0.006	<0.005	0.100	0.083	0.060	50	-1,21	4,90	2699

Source: SGA-Studiengesellschaft für Eisenerzaufbereitung GmbH & Co. KG, Germany, 2023.

Based on these and the previous results, additional magnetic separation cleaning stages, differentiated magnetic strength on cleaner stages and hydroseparation were considered for further test work. At this time, it was defined to do not include reverse flotation in further test work (potential challenges associated with water balance in the full-scale plant). Finer regrind was also considered for further test work for final evaluation for design.

13.1.8.2.2 Regrind Size Confirmatory Test Work

During 2023, an additional iron concentration flotation pilot plant campaign was completed using rougher flotation tails obtained from the copper flotation continuous pilot plant, representing Year 1-5, using an industrial diameter magnetic drum separator (48-inch) (see Table 13-1, year 2023). This pilot plant considered rougher magnetic separation, regrinding with classification via hydrocyclones to design P_{80} of 40 µm; a hydroseparation stage, and eight cleaning stages of magnetic separators in series configuration (pilot and bench LIMS). Desalinated seawater was used during the program. The iron concentrate grade and recovery results are shown in Table 13-31 and Table 13-32.

Table 13-31: Concentrate Chemical Assays, Year 1-5 Composite Sample at Design Regrind P_{80} of 40 µm

Test	% Fe	% SiO_2	% CaO	% MgO	% P	% S	% Na_2O	% K_2O	% Mn	Cu ppm
Design Regrind P_{80} of 40 µm	65.62	4.15	0.600	0.350	0.014	0.011	0.156	0.108	0.076	52

Source: San Lorenzo Ltda, Vallenar, Chile, 2023.

Table 13-32: Mass Recovery Results for 2023 Iron Concentration Pilot Plant at Design Regrind P_{80} of 40 µm

Stage	Overall Mass Recovery (%)	Mass Recovery by Stage (%)
Rougher	26.5	26.5
Hydroseparation	21.6	81.3
1st to 8th Cleaner	12.6	58.5
Global	12.6	-

Source: Aminpro Chile Spa, Santiago, Chile, 2023

Additional regrinding was then considered for evaluation of the final concentrate grade, providing the results presented in Table 13-33 and Table 13-34.

Table 13-33: Concentrate Chemical Assays, Year 1-5 Composite Sample at finer regrind (P_{80} of 30 µm)

Test	% Fe	% SiO_2	% CaO	% MgO	% P	% S	% Na_2O	% K2O	% Mn	Cu ppm
Two regrind (P_{80} of 30 µm)	66.07	3.77	0.660	0.390	0.011	0.022	0.16	0.099	0.076	52

Source: San Lorenzo Ltda, Vallenar, Chile, 2023.

Table 13-34: Mass Recovery Results Concentrate 2023 Iron Concentration Pilot Plant at finer regrind (P_{80} of 30 µm)

Stage	Overall Mass Recovery (%)	Mass Recovery by Stage (%)
Rougher	26.5	26.5
Hydroseparation	21.6	81.3
1st to 8th Cleaner	12.7	59.0
Global	12.7	-

Source: Aminpro Chile Spa, Santiago, Chile, 2023.

The additional regrind showed an iron concentrate grade improvement from 65.62% Fe to 66.07% Fe (0.45% Fe or 0.7% with respect to the first value) for the sample tested. Due to only 0.45% Fe improvement in concentrate grade and the significant additional energy to be required if finer regrind size was established as basis for the design, target regrind of 40 µm was maintained to be considered as base case.

13.1.8.2.3 Cleaner Magnetic Strength Confirmatory Test Work

Finally, test work was completed to evaluate the concept of differentiated magnetic strength through cleaner stages, considering two separated lines, one at 750 Gauss – mainly, and one at 500 Gauss. As per the current design, all tailings from each cleaner line reported to final tailings (denominated as Open circuit). Iron concentrate grade and recoveries are presented in Table 13-35 and Table 13-36.

Table 13-35: Concentrate Chemical Assays, Year 1-5 Composite Sample at Open Circuit

Test	% Fe	% SiO_2	% CaO	% MgO	% P	% S	% Na_2O	% K_2O	% Mn	Cu ppm
Cleaner at 500 Gauss	65.82	4.33	0.610	0.385	0.015	0.022	0.181	0.122	0.064	52
Cleaner at 750 Gauss	65.24	4.52	0.560	0.365	0.015	0.020	0.187	0.115	0.064	43

Source: San Lorenzo Ltda, Vallenar, Chile, 2023.

Table 13-36: Mass recovery Results to Concentrate for 2023 Test Work at Differentiated Cleaner Lines at Open Circuit

Stage	Cleaner at 500 Gauss	Cleaner at 750 Gauss		
Overall Mass Recovery (%)	Mass Recovery by Stage (%)	Overall Mass Recovery (%)	Mass Recovery by Stage (%)	
Rougher	20.2	20.2	20.2	20.2
Hydroseparation	16.9	83.9	16.9	83.9
1st to 8th Cleaner	11.9	70.3	13.0	76.5
Global	11.9	-	13.0	-

Source: Aminpro Chile Spa, Santiago, Chile, 2023.

The results confirm that differentiated cleaner lines by magnetic strength allow to differentiate the product for the same feed. For the sample tested, the iron concentrate grade differentiation was of 0.58% Fe or 0.9% with respect to the lower value (from 65.24% Fe to 65.82% Fe).

Mass recovery also varied by 1.1% mass recovery (13.0% versus 11.9%), being higher for the product with lower iron concentrate grade and lower for the product with higher iron concentrate grade corresponding to a more selective line (lower magnetic strength).

13.1.8.2.4 Evaluation of Cleaner Magnetic Tail Recirculation

Additional test work was completed considering recycle from all tailings from 500 Gauss cleaner line to the hydro-separator feeding the 750 Gauss cleaner line, with the purposes of evaluating mass recovery (denominated as Closed Circuit). The results of this test work are presented in Table 13-37 and Table 13-38:.

Table 13-37: Concentrate Chemical Assays, Year 1-5 Composite Sample at Closed Circuit

Test	% Fe	% SiO_2	% CaO	% MgO	% P	% S	% Na_2O	% K_2O	% Mn	Cu ppm
Cleaner at 500 Gauss	65.14	4.33	0.595	0.380	0.015	0.019	0.176	0.109	0.064	62
Cleaner at 750 Gauss	64.54	4.83	0.655	0.415	0.016	0.021	0.195	0.125	0.064	41

Source: San Lorenzo Ltda, Vallenar, Chile, 2023.

Table 13-38: Mass Recovery Results to Concentrate for 2023 Test Work at Differentiated Cleaner Lines at Closed Circuit

Stage	Cleaner at 500 Gauss	Cleaner at 750 Gauss		
Overall Mass Recovery (%)	**Mass Recovery by Stage (%)**	**Overall Mass Recovery (%)**	**Mass Recovery by Stage (%)**	
Rougher	20.2	20.2	20.2	20.2
Hydroseparation	16.9	83.9	20.2	99.8
1st to 8th Cleaner	11.9	70.3	17.2	85.1
Global	11.9	-	17.2	-

Source: Aminpro Chile Spa, Santiago, Chile, 2023.

The results confirmed that differentiated cleaner lines by magnetic strength allow to differentiate the product for the same feed reporting in this case a similar iron concentrate grade differentiation of 0.6%Fe or 0.9% with respect to the lower value (comparing 64.54% Fe to 65.14% Fe). Since in this case all tailings from low-strength cleaner tails were returned to the hydroseparator that feeds the high strength cleaner line, mass recovery increased more with respect to the open circuit test (from 13.0% to 17.2% therefore 4.2% in mass recovery).

Based on the results of this test work, open circuit was maintained as base case for later design, although eventual design lines may be considered later.

13.1.8.2.5 Pellet Quality Testing

Sample obtained from the pilot plant program (Table 13-1, Year 2023) was used to evaluate potential pellet quality produced from Santo Domingo iron concentrate. Several bench- and full-scale pot grate test were completed by the Natural Resources Research Institute (NRRI) from the University of Minnesota in United States, during early 2024. Results, presented in Table 13-39, indicate that the physical and metallurgical quality of the pellet are acceptable for blast furnace feed.

Table 13-39: Results of Santo Domingo Iron Pellet Quality

Sample	Average Compression Strength, lbf/p (ISO 4700)	Average Tumble Index, wt.% >1/4" (ISO 3271)	Porosity % (water displacement)	Reducibility Index (RI)%/min (ISO 4695)	'Reduction-Disintegration Index (RDI), wt% >6.3 mm (ISO 4696-1)	'Reduction-Disintegration Index (RDI), wt% <0.5 mm (ISO 4696-1)	Free Swelling Index % (ISO 4698)
NRRI minimum preferred value	>400 lbf/p	>94	>20	0.8	>85	<10	<20
Pellet (Y1-Y5 Comp)	598 lbf/p	97	28.7	0.94	93.9	3.4	16.2

Source: Natural Resources Research Institute (NRRI), United States, 2024.

13.1.8.2.6 Iron Ore Variability Testing

13.1.8.2.6.1 DTT Work

Additional magnetic iron test work was completed during 2021 to 2023 involving DTTs at target regrind P_{80} of 44 µm (325 mesh) on more than 1300 samples, covering different magnetic iron mineralized geological units through LOM, and including concentrate Fe grade and impurities analysis. These DTT were used to update the mass recovery algorithm, which is presented in Section 13.2 and for iron concentrate grade establishment (see Section 14).

In addition to this, exploratory DTT were completed at finer regrind (established as 80% less than 450 mesh or 32 µm) on 70 selected samples. The results indicate that for samples with poor performance (concentrate grade less than 62% Fe), there was a concentrate grade improvement with finer regrind, for all mineralized geological units tested. The degree of improvement varied depending on the geological unit and the original iron concentrate value achieved at the target regrind P_{80}.

13.1.8.2.6.2 LIMS Test Work

The design iron separation circuit was used for LIMS variability testing on 39 Cu rougher flotation tailings samples from the Cu flotation variability program. The tests considered a rougher magnetic concentration, via LIMS L8, followed by regrind to target P_{80} of 40 µm with screening classification – adapted system due to mass restriction – followed by hydro-separation and 8 cleaning stages via LIMS L8. The results of these tests demonstrated the Santo Domingo ore suitability to produce iron concentrate at a commercial product grade. The variability program also demonstrated the Santo Domingo ore suitability to produce iron concentrate at 67% Fe product for selected samples representing some areas of the deposit. Iron concentrate testing was developed on 38 samples.

Table 13-40: Iron Concentrate Grades from LIMS Test Work Program on Variability Samples

Composite Sample ID	Assay										
	Fe (%)	SiO_2 (%)	Al_2O_3 (%)	CaO (%)	MgO (%)	P (%)	S (%)	Na_2O (%)	K_2O (%)	Mn (%)	Cu (ppm)
134	64.32	5.70	1.22	1.40	0.495	0.025	0.025	0.136	0.136	0.094	89
127	63.94	6.00	1.20	1.40	0.540	0.015	0.049	0.13	0.119	0.094	148
65	65.98	5.04	1.12	0.370	0.360	0.012	0.032	0.197	0.211	0.066	62
49	63.26	4.12	0.800	0.720	0.415	0.011	0.025	0.121	0.122	0.075	68
20	67.79	3.66	0.574	0.355	0.155	0.008	0.018	0.028	0.043	0.048	81
55	69.00	2.13	0.534	0.345	0.150	0.004	0.030	0.028	0.056	0.068	111
74	66.73	3.73	0.770	0.475	0.315	0.008	0.021	0.065	0.103	0.074	107
131	68.17	3.13	0.632	0.555	0.165	0.006	0.021	0.034	0.068	0.072	64
123	64.92	3.62	0.778	1.01	0.410	0.008	0.022	0.042	0.148	0.076	128
138	66.13	3.81	1.16	1.32	0.320	0.010	0.022	0.120	0.134	0.084	86
70	65.37	3.69	0.818	0.465	0.365	0.008	0.030	0.127	0.108	0.076	177
57	66.88	4.15	0.792	0.720	0.345	0.010	0.031	0.048	0.093	0.112	61

Composite Sample ID	Assay										
	Fe (%)	SiO_2 (%)	Al_2O_3 (%)	CaO (%)	MgO (%)	P (%)	S (%)	Na_2O (%)	K_2O (%)	Mn (%)	Cu (ppm)
76	69.33	2.18	0.464	0.200	0.125	0.005	0.019	0.098	0.049	0.058	28
41	48.80	17.9	0.408	1.53	0.938	0.060	0.021	1.41	0.573	0.100	85
118	67.24	3.64	0.800	0.225	0.300	0.011	0.037	0.171	0.081	0.052	118
63	65.59	5.41	1.05	0.535	0.475	0.012	0.043	0.047	0.216	0.062	73
128	66.94	3.14	0.885	0.450	0.313	0.011	0.023	0.106	0.110	0.070	68
130	67.77	3.34	0.638	0.490	0.240	0.012	0.024	0.023	0.055	0.056	35
125	65.44	4.38	0.608	0.272	0.290	0.009	0.022	0.027	0.055	0.072	293
44	67.69	3.08	0.920	0.455	0.300	0.008	0.031	0.085	0.109	0.062	72
71	56.41	10.6	0.250	2.03	1.28	0.053	0.048	0.756	0.253	0.070	165
116	68.14	3.08	0.588	0.325	0.255	0.010	0.018	0.073	0.046	0.056	32
11	66.33	3.77	1.02	0.525	0.425	0.016	0.021	0.200	0.113	0.108	50
45	56.19	12.1	3.05	0.950	0.600	0.029	0.021	1.21	0.073	0.070	38
30	68.21	2.80	0.676	0.415	0.240	0.010	0.020	0.054	0.078	0.060	63
21	66.11	2.99	0.958	0.520	0.440	0.008	0.024	0.084	0.101	0.084	52
87	67.99	2.98	0.611	0.305	0.290	0.009	0.023	0.048	0.098	0.056	61
84	67.46	3.65	0.788	0.245	0.385	0.011	0.021	0.190	0.048	0.076	39
86	67.61	3.25	0.816	0.460	0.280	0.005	0.027	0.100	0.095	0.054	117
62	68.13	3.14	0.702	0.290	0.270	0.009	0.024	0.086	0.119	0.054	98
12	67.15	3.78	0.852	0.500	0.350	0.013	0.014	0.252	0.063	0.048	32
110	68.13	2.93	0.702	0.480	0.270	0.006	0.017	0.099	0.062	0.028	43
51	51.62	15.8	3.89	0.975	0.488	0.038	0.019	1.75	0.201	0.070	55
97	65.91	3.46	0.790	0.460	0.425	0.010	0.026	0.068	0.132	0.056	70
129	67.27	3.71	0.804	0.635	0.350	0.013	0.023	0.116	0.118	0.066	76
137	69.84	1.70	0.502	0.175	0.130	0.004	0.025	0.042	0.057	0.062	29
124	59.26	9.00	2.30	0.863	0.575	0.035	0.021	0.690	0.250	0.070	36
75	65.08	2.66	0.568	0.775	0.415	0.006	0.023	0.049	0.091	0.070	65

Source: San Lorenzo Ltda, Vallenar, Chile, 2023.

From the results obtained for the LIMS iron concentration variability program covering the life of mine mineralized geological units, the following conclusions were derived:

- The majority of the samples tested (87%) reported iron concentrate grades greater than 62% Fe,
- 76% of the samples reported iron concentrate grade greater than 65% Fe,
- 45% of the samples reported iron concentrate grade than 67% Fe,
- SiO_2 impurity appears as the most relevant impurity among the impurities evaluated for the samples tested.

Three of the five samples that yielded concentrate grades below 62% Fe iron concentrate grade belong to andesite lithology (upper and lower). One of the five belongs to Breccia lithology and the other belongs to volcaniclastic within Iris Norte deposit, similar to andesite lithology.

Considering potential complexity for these lithologies to obtain iron concentrate grade above 62% Fe during the operation, the following conclusions were derived:

- Andesite ore:
    - The more challenging geological units, in terms of the ability to produce commercial iron concentrate grades, appear to be andesite upper and lower. Over the life of mine, these units represent 10% of the ore feeding the plant (17% over the first 5 years).
    - Based on their representation in the mine plan, an-ore or concentrate blending strategy may be applicable during medium-short term mine planning. For example, blending andesite ore with other geological units such as volcaniclastic ore, which represent 68% of the ore feeding the plant LOM and 73% of the first 5 years of the mine plan, would significantly mitigate the challenges with the andesite ores.
    - Considering that it is possible to improve the iron concentration grade for the andesite units (upper and lower) by regrinding finer, an operational strategy may also be applied to treat andesite ore.
- Breccia ore:
    - The breccia unit represents 2% of the LOM ore feeding the plant, and 2% of the first 5 years of the mine plan.
    - The breccia unit also increased iron concentrate grade with finer regrind
- An ore or concentrate blending strategy, or an operational strategy are also possibilities for this unit.
- Volcaniclastic within Iris Norte:
    - Volcaniclastic rocks within Iris Norte reported iron concentration grades greater than commercial grade of 62% for the samples tested. However, one sample yielded grades lower than 62% Fe; the sample is located near the andesites rocks, which may be part of the reason for the low performance. Ore or concentrate blending, or operational strategies could also be applied for this type of ore.

A mass balance reconciliation was completed on each variability test for the iron variability program, resulting in the mass recovery values presented below. In general terms, higher mass recoveries related to high concentration grades.

Table 13-41: Overall Mass Recovery for Iron Concentration Variability Program

Parameter	Value											
Composite ID	11	12	20	21	30	41	44	45	49	51	55	57
Mass Recovery %	21.8	48.3	18.7	23.2	27.6	15.9	38.7	32.3	8.1	14.3	32.3	40.3

Source: Aminpro Chile Spa, Santiago, Chile, 2023.

Table 13-42: Overall Mass Recovery for Iron Concentration Variability Program (Continuation)

Parameter	Value											
Composite ID	62	63	65	70	71	74	75	76	84	86	87	97
Mass Recovery %	22.4	9.3	13.3	17.0	14.4	29.6	4.2	56.1	23.9	54.5	55.2	12.1

Source: Aminpro Chile Spa, Santiago, Chile, 2023.

Table 13-43: Overall Mass Recovery for Iron Concentration Variability Program (Continuation)

Parameter	Value													
Composite ID	110	116	118	123	124	125	127	128	129	130	131	134	137	138
Mass Recovery %	40.8	28.7	17.7	17.2	12.7	35.9	33.7	57.4	48.3	55.3	32.9	25.0	65.1	44.2

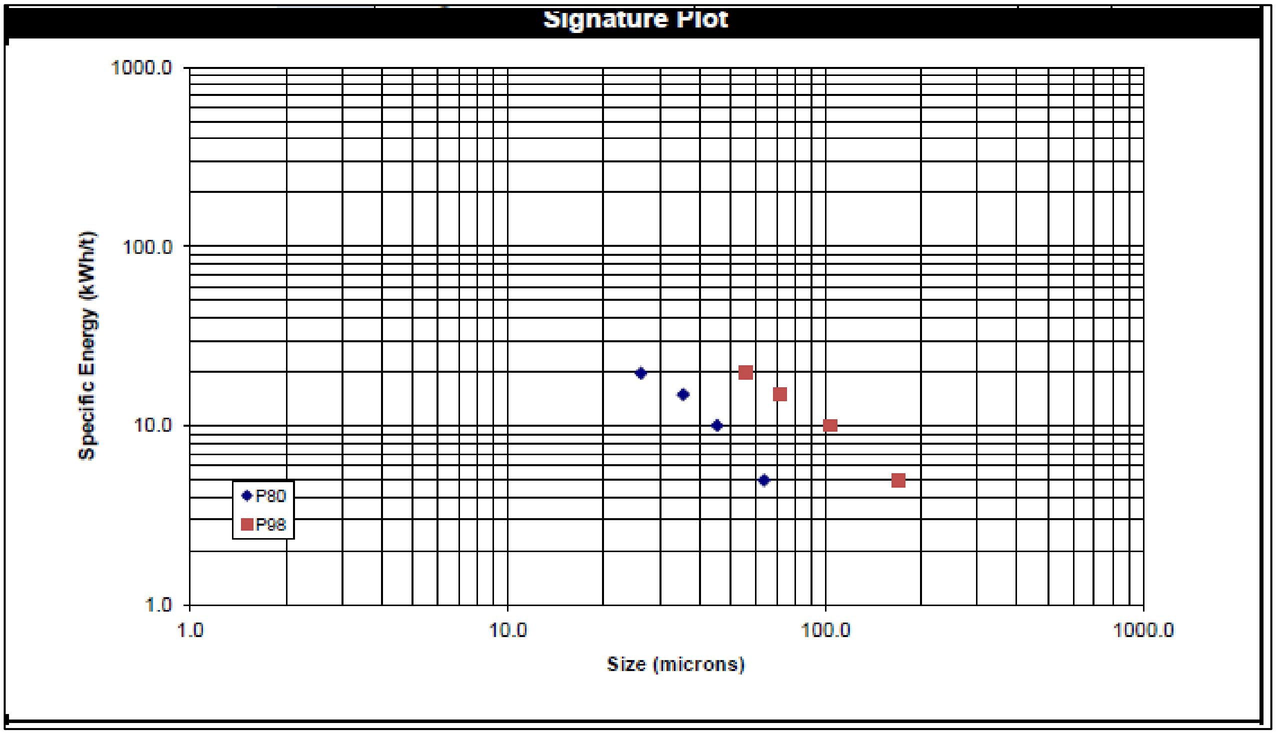
Source: Aminpro Chile Spa, Santiago, Chile, 2023.

13.1.9 Magnetite Regrind and Hydro-separation

Magnetite regrind test work has been carried out since 2011, when seawater was used for the magnetic rougher concentrate obtention. During that time, vendor testing was completed with the objective to obtain the specific energy requirements for magnetite regrinding.

During 2023, specific IsaMill^{TM} Grinding Test work was completed with the objective to complete the signature plot for rougher magnetic rougher concentrate was, obtained from the magnetic separation pilot plant for vendor regrind test work. From the test work completed, a specific energy consumption value of 11.4 kWh/t was considered for size reduction from a representative Year 1-5 composite sample. The sample F_{80} was about 142 µm to and it was reground to target P_{80} of 40 µm.

Figure 13-5: Signature Plot for Rougher Magnetite Concentrate



Source: ALS Metallurgy Perth, Australia, 2023.

A hydro-separation stage was tested during the 2020 iron concentration pilot program – using a separate sample for bench scale testing and incorporated within the pilot plant testing during the 2023 iron concentration pilot plant test work; reground magnetic rougher concentrate sample was obtained from this pilot plant and provided to a vendor for rise rate obtention. From the test work completed, a rise rate of 8.65 $m^3/h/m^2$ was selected for design purposes.

13.1.10 Magnetite Concentrate Thickening

During 2014, magnetite settling, and dynamic thickening test work was conducted by Outotec and Delkor. At the time of this previous study, a recommended thickening unit value of 0.68 $t/h/m^2$ was considered for the design of the magnetite concentrate thickener resulting in a single 36 m diameter high-rate thickener without a clarifier. This was in the mid-range of the recommendations made by Outotec and Delkor at the time.

During April 2021, a new magnetite concentrate sample was generated as part of a metallurgical pilot plant and made available to Tailpro Consulting for dynamic thickening test work using equipment and procedures of Vietti Slurrytec for subsequent industrial sizing of the magnetite thickener independent to a vendor. The same sample was also sent to Metso Outotec for dynamic thickening test work and subsequent industrial equipment sizing. A sample of magnetite concentrate was also tested and sized by Vietti Slurrytec in July 2021 as part of a series of tests conducted following the first stage of metallurgical piloting. During September 2021, an additional magnetite sample evaluating reverse flotation as part of the process circuit was generated and again tested for thickener sizing with Vietti Slurrytec. All dynamic thickening test work conducted by Tailpro Consulting and Vietti Slurrytec considered the use of magnetic flocculation to generate larger flocs under the influence of a magnetic field, characteristic of the final process design where this equipment will be installed prior to the thickener feed well entry. A demagnetization stage was also considered prior to underflow rheology measurements to simulate the incorporation of this equipment on the feed line to the concentrate handling and pipeline system. Synthetic flocculant is not considered as part of the magnetite thickener design; however, the magnetite concentrates are susceptible to flocculation as they are naturally coagulating, and this can be considered as part of the operation if overflow water is dirtier than usual.

During March to April 2023, Metso Outotec provided further testing on a sample of copper concentrate and a sample of magnetite concentrate. Magnetic flocculation was considered as part of this test work. All magnetite concentrate samples from 2021 onwards considered the use of desalinated water as part of the piloting and sample generation.

The 2023 design criteria considered that the magnetite thickener will operate with a nominal discharge density of 65% solids w/w and a design maximum of 70% solids w/w, similar to the criteria considered as part of the 2014 project study.

Metso Outotec concluded that for the magnetite concentrates tested from 2021 onwards that the optimal feed rate could be higher than the previous studies resulting in a reduction in thickener diameter whilst meeting the design underflow density. Vietti Slurrytec independently concluded the same for all three magnetite samples tested in 2021 and design considered a thickening unit value of 1.08 $t/h/m^2$ for the magnetite concentrate thickener resulting in a single 29 m diameter high-rate thickener without a clarifier. Vietti Slurrytec indicated that the residence time needed to achieve target underflow densities for the magnetite concentrates tested are typically less than 1 hour achieving up to 25 Pa unsheared rheology for 70% solids w/w. The Maximum Operating Torque (MOT) and Normal Operating Torque (NOT) recommended by vendors and independently are within the industry standard for the type of thickener in accordance with the 2023 production design criteria.

13.1.11 Magnetite Concentrate Filtration

13.1.11.1 Pre-2020 Concentrate Filtration Test Work

From 2011 to 2015 vendor filtration testing was performed on iron concentrate with the objective to determine the main filtration characteristics of the magnetic iron concentrate and filtrate cake chloride content from the use of direct seawater during the ore processing.

13.1.11.2 2020 to 2023 Concentrate Filtration Test Work

The current definition of the project is the use of desalinated seawater and therefore new samples were obtained for filtration vendor testing. These samples were obtained from the 2023 Magnetic separation pilot plant using desalinated seawater. Results confirmed pressure filters with filtration rate of 0.520 $t/h/m^2$ for design, based on Ausenco recommendation.

13.1.12 Tailings Thickening

13.1.12.1 Pre-2020 Tailings Test Work

Outotec and Delkor conducted bench-scale thickening test work on tailings samples from the Santo Domingo deposit using a 99 mm diameter thickener. The tailings test work results were used to evaluate the behaviour of the tailings under conditions of dynamic settling and to determine design values for the two-stage tailings thickening system. Two stages of thickening in series were selected due to the properties of thickened tailings and to avoid capital and operating costs associated with operating using non-thickened tailings.

Based on the test work results, the recommended tailings thickening conditions were for the first stage a High rate thickener with a thickening rate of 0.65 $t/h/m^2$, a solids percent of thickened tailings of 55% and a flocculant dose of 10 g/t tailings, while the recommended tailings thickening conditions for the second stage were a High density thickener with a thickening rate of 0.5 $t/h/m^2$, a solids percent of thickened tailings of 67% and a flocculant dose of 10 g/t tailings.

13.1.12.2 2020 to 2023 Tailings Test Work

During 2020 to 2021, additional tailings thickening test work was completed on various tailings samples following the previous testing campaign to validate a two-stage approach and identified that the tailings could be thickened and transported as a single stage solution utilizing modern day high density or paste thickening equipment. Consequently, a trade-off study was completed in 2022 and confirmed the feasibility of a single stage solution to reduce the complexity and risk of the tailings handling system for the project, but also identified significant upside to reduce capital and operational expenditure.

In 2023, subsequent settling and dynamic thickening testing were conducted by Tailpro Consulting and Vietti Slurrytec to verify the potential design parameters for both a two-stage and single-stage tailings thickening approach.

This test work was conducted both at a laboratory and semi-pilot stage to better understand the residence times needed and unsheared underflow densities to establish equipment torque requirements. Dynamic thickening testing was also conducted by Metso Outotec, again at both a laboratory and semi-pilot scale, to understand the vendor specified technology to dewater the tailings to design values. The 2023 tailings sample generated is representative of the final tailings and is a composite of the first 5 years of operation. This sample included the tailings from the copper flotation process, unlike the samples previously tested that only considered tailings from the magnetic concentration stage.

A two-stage thickening approach results in double flocculation which not only increases operational costs but also increases the unsheared yield stress of the tailings compared to a single-stage flocculation process.

Based on the tailings samples tested, the use of high density or paste thickening as a single stage would be feasible to obtain the target underflow densities utilizing drive units currently available in the market by two thickener vendors and operating at similar operations. As part of the most recent semi-pilot thickening test work, underflow unsheared yield stresses of 205 Pa were observed at 67% solids w/w for a single stage flocculation process and 257 Pa for the same density for a double flocculation process. A 50:50 mixture of Rheomax DR 1050 and Magnafloc 1011 supplied by

BASF was identified as the best performing flocculant type and has been used for copper concentrate tailings testing as part of previous campaigns.

Based on the more representative 2023 tailings sample tested, an independent thickener sizing exercise identified a thickening flux of 0.5 t/h/m^2 for the design of a single-stage tailings thickening system resulting in 3 x 48 m diameter high density thickeners. A Maximum Operating Torque (MOT) of 14.1 MNm was specified for each thickener to achieve the target design underflow density of 67% solids w/w.

As part of the 2023 tailings thickening test work program, a series of beach slope predictions were developed for the design values associated with deposition to the tailings storage facility (58%, 65%, and 67% solids w/w). These predictions considered a range of simultaneous operating independent discharges to understand the impact to the beach slope and were provided to the TSF designers (Knight Piésold) as part of the current design. The results were included in the TSF planning described in Section 18.5.

13.1.12.3 Tailings Variability

Throughout various tailings testing campaigns conducted so far, additional tests were developed to understand the variability of the project ore and its potential impact on the thickening, transport, and deposition of tailings. A total of 140 variability samples were generated from rougher flotation tests conducted at a laboratory scale to assess variability in sedimentation rates and overflow clarity. Additionally, 20 composite tailings samples from L8 magnetic separation tests were analyzed for dynamic thickening and rheological measurements. Of these composite samples, 60% were derived from ores expected to be processed in the initial five years of operation, as outlined in the 2023 mine plan, while the remaining 40% were from ores distributed across the proposed life of mine (LOM).

The analysis of these 140 samples and 20 composite materials revealed variabilities that could influence the efficiency of tailings thickening and their subsequent transport to the storage facility. Such variability is anticipated in mining operations, highlighting the necessity of thorough characterization to refine the mine plan. The objective was to minimize, where possible, substantial fluctuations in the tailings management system to ensure more efficient tailings handling. Overall, the 2023 composite tailings sample provided a representative average of the 20 composites analyzed during this variability stage.

The collected data was used to understand the variability in settling rates over the LOM, particularly to assess whether any ores would impact the proposed thickener sizing, specifically the design flux rate. Overflow clarity was also evaluated to ensure that the selected flocculant was optimal and to identify any samples with colloidal conditions that might cause issues in industrial-scale thickeners. This work increased confidence in the proposed thickener sizing for the project, as the main 2023 pilot sample, representing the first five years, was a good average representation and was not considered an outlier based on the 20 composite samples tested.

The results were incorporated into the final design to confirm the chosen flocculant, dosage, and final dimensions of the proposed tailings thickeners. Besides specific design modifications, the variability testing confirmed that the chosen design would be more reliable given the ore variability.

The 140 samples represent over 80% of the total final tailings. The 20 composite samples were generated from the 140 tailings samples, selected through detailed expert analysis. The copper tailings fraction was not included in these samples but was present in the 2023 pilot tailings.

The 20 composite samples were used to validate and fine-tune the final proposed thickener dimensions. No specific analysis was conducted for upper or lower bound ranges to assess CAPEX impact, as this was deemed too conservative. The 20 composite samples were tested at a benchtop scale due to mass availability. While dynamic rheology and settling characteristics were determined, projecting the residence time required to achieve the design underflow density remains challenging. This will be better addressed during semi-pilot tests with deeper bed conditions. Rheology data from the 140 static test variability samples was not used for thickener torque sizing; this was determined solely from the rheology associated with dynamic thickening tests, specifically the main 2023 pilot sample and the 20 composite samples.

13.1.13 Byproduct Cobalt

This section is included within the context of the cobalt opportunity detailed in Section 24.4 "Other Relevant Data and Information – Optimization Opportunities" & Information" and Section 25.16"Interpretations & Conclusions - Risks & Opportunities."

As noted, the Santo Domingo deposit is endowed with cobaltiferous pyrite. While the current study only considers the economic by-production of cobalt as a potential opportunity, considerable work has been done to evaluate this possible upside. The general processing route consists of three stages:

1. Recovery of cobaltiferous pyrite from the Santo Domingo tailings, via froth flotation
2. Oxidation of the pyrite, either via pyrometallurgical or hydrometallurgical techniques, to release the cobalt
3. Recovery and purification of the cobalt into a commercial product.

The following subsection details the results of this ongoing work.

13.1.13.1 Sample Description

The 2018 cobalt investigation used the Years 1-5 composite material remaining from the 2015 pilot testing program. This material had been stored in the core storage facility since late 2015 and had been crushed to -3/4" for comminution testing.

13.1.13.2 Cobalt Mineral Characterization

Two flotation separation tests were completed at Blue Coast in 2018 to produce separate copper and pyrite concentrates. The cobalt to pyrite ratio in the feed, tailings and the six individual flotation concentrates from each test were regressed to confirm the cobalt association with pyrite. In both tests the cobalt correlates closely with the inferred pyrite content of the individual samples as shown in Figure 13-6. For the two tests the implied cobalt concentration in pyrite is 0.73%.

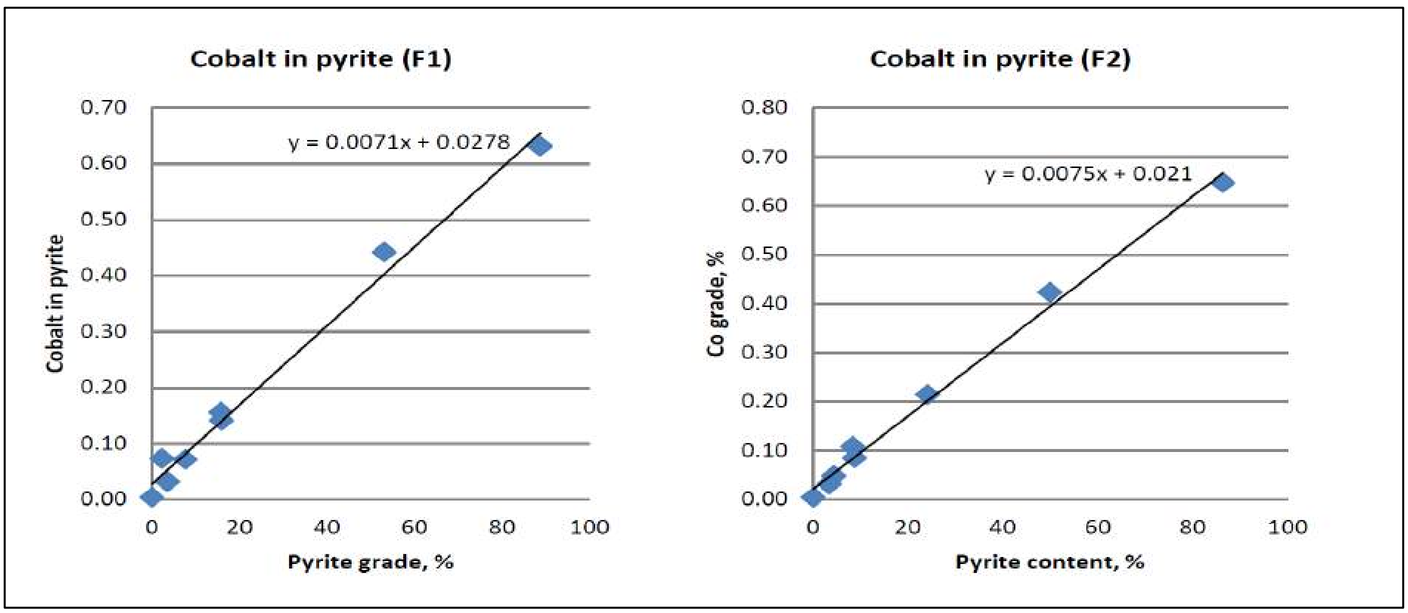
Samples of pyrite concentrate, and scavenger tailing produced in the laboratory during the 2018 cobalt investigations were submitted to AuTec in British Columbia for modal analysis and field emission gun–scanning electron microscope (FEG-SEM) imaging and Surface Science Western at the University of Western Ontario for dynamic secondary ion mass spectrometry (D-SIMS) depth profiling of individual pyrite particles. A total of 181 discrete pyrite grains from pyrite concentrate products and scavenger tailings were scanned; the average cobalt content of pyrite grains averaged 0.74%, closely matching the implied concentration from the regression of flotation products. Depth profiling individual pyrite particles does not suggest a concentration gradient away from the particle surface.

This work also found a wide variation in cobalt concentration across the sample set analyzed, ranging from 0% Co to 2.5% Co; 80% of the cobalt was contained in the 50% of the pyrite particles containing the highest cobalt content.

The modal analysis also found trace discrete cobalt sulphide minerals and cobalt present in hematite; however, these were not prevalent enough to quantify. The cobalt is not uniformly distributed in the pyrite; however, the cobalt is uniformly distributed in the grains containing cobalt, strongly suggesting a solid solution rather than surface coatings or discrete cobalt sulphide minerals locked with pyrite.

This work found that 83% of the cobalt in the samples analyzed was present in solid solution with pyrite. The remainder is likely in locked pyrite particles, in chalcopyrite and, in rare instances, in discrete cobalt sulphide minerals.

Figure 13-6: Co vs. Pyrite in Flotation Products



Source: Figure prepared by Blue Coast, 2018.

13.1.13.3 Pyrite Flotation

The Years 1-5 composite samples collected for the 2018 and the 2019 cobalt investigations were floated sequentially at Blue Coast to produce a copper rougher concentrate and a pyrite rougher concentrate. The purpose of this testing was to generate pyrite concentrate material for subsequent metallurgical processing.

Due to the aged nature of the samples, it was anticipated that the bulk rougher sulphide flotation would not perform in the copper cleaning circuit.

A total of 35 tests were conducted to produce approximately 25 kg of pyrite concentrate. Typical results from the sequential flotation tests are shown in Table 13-44. On average, 77% of the cobalt reported to the pyrite rougher concentrate and 12% to the copper rougher concentrate.

Table 13-44: 2019 Sequential Flotation Typical Results

Product	Weight		Assays (%)				%Distribution			
	g	%	Cu	Co	Fe	S	Cu	Co	Fe	S
Cu Rougher Conc. 1-4	378.0	3.79	15.4	0.10	30.7	19.0	92.9	12.0	3.5	28.2
PY Rougher Conc. 1-3	492.3	4.93	0.47	0.49	40.0	35.7	3.7	76.5	6.0	69.2
Rougher Tailings	9,106.3	91.3	0.02	0.00	32.7	0.07	3.4	11.5	90.5	2.6
Calculated Head	**9,976.6**	**100**	**0.63**	**0.03**	**32.9**	**2.55**	**100**	**100**	**100**	**100**

Cobalt deportment was analyzed for all the flotation testing program in 2019 as part of the refinement of the copper flotation circuit. In these tests, cobalt and inferred pyrite were traced through all the products generated. For the Years 1-5 composite, 90.2% of the cobalt and 97.5% of the pyrite reported to the cleaner tailings. The LCT cleaner tailings for the Years 1-5 composite averaged 0.268% Co and 22% S for the last three test cycles. This is considered to be representative of the cobalt/sulphur deportment to be expected over the mine life based on analysis of the drilling and block model databases.

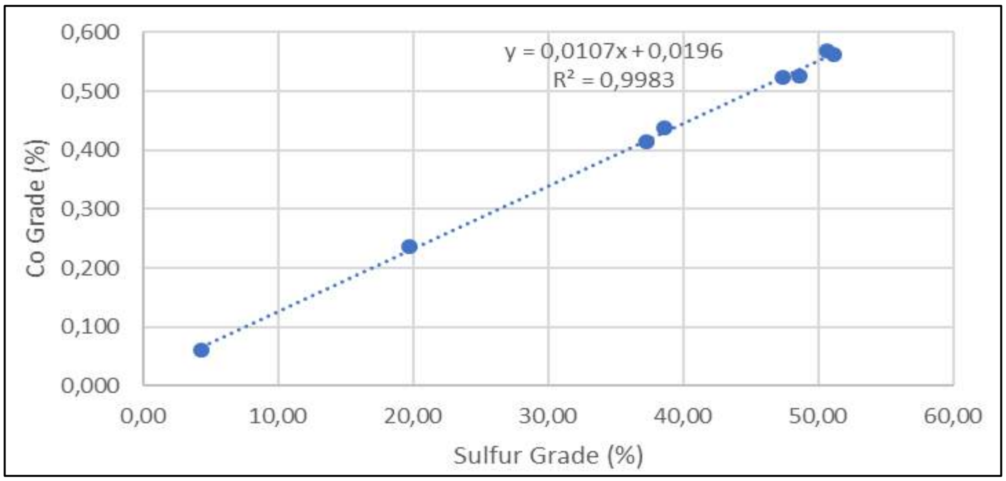
Subsequent investigation to test the amenability of re-flotation of the pyrite in the cleaner tailings by lowering the pH to 6–7 and PAX addition, produced pyrite concentrates with more than 47% S with 97% Co reporting to the concentrates, with proportional cobalt distributions to the pyrite. Insufficient material was available for complete testing of the pyrite cleaning process, but an assumption of 98% pyrite (and Co) recovery to a re-cleaned pyrite concentrate are considered reasonable based on the targeted concentration ratio from the cleaner tailings to the pyrite concentrate of approximately two (a typical copper cleaning circuit would target a ratio of 6 to 7). Two timed pyrite cleaning tests in the laboratory produced five sequential pyrite concentrates with consistent cobalt to sulphur ratios in the timed concentrates, suggesting a straightforward pyrite/gangue separation step. The regression of these products is shown in Figure 13-7.

Table 13-45: LCT Cobalt Testing Results

Sample	LCT Cleaner Tailings				Implied Co Grade in Pyrite (%)
	Grade		% Distribution		
	Co (%)	FeS_2 (%)	Co	FeS_2	
Years 1-5 composite	0.268	40.5	90.2	97.5	0.662
Year 1 composite	0.282	41.8	91.5	98.5	0.675
Year 2 composite	0.242	33.5	89.5	96.0	0.722
Year 3 composite	0.224	29.9	87.0	94.0	0.749
Year 4 composite	0.254	33.9	90.0	95.6	0.749
Year 5 composite	0.199	40.3	83.2	91.9	0.494
Avg. of composites	0.245	36.7	88.2	95.2	0.668

Source: Aminpro Chile Spa, Santiago, Chile.

Figure 13-7: Cobalt to Sulphur Regression for Timed Pyrite Flotation



Source: Figure prepared by Gregg Bush, based on Aminpro data, 2020.

13.1.13.4 Pyrite Leaching Test Work

An initial preliminary concentrate oxidation test work program consisting of one pressure leach test (POX) and one bacterial oxidation test (BiOX) indicated the potential for cobalt extraction through dissolution of the sulphide minerals. Therefore, three hydrometallurgical sulphide oxidation flowsheets were evaluated: Albion, BiOX and POX. A fourth flowsheet, consisting of bio-oxidation in a copper oxide heap leach facility, is currently under evaluation.

13.1.13.4.1 Albion Process

The Albion Process consists of ultra-fine grinding followed by atmospheric leaching at elevated temperature with oxygen sparging. Leach times are typically of the order of 48 hours. The process is currently applied commercially for the recovery of gold, copper and zinc.

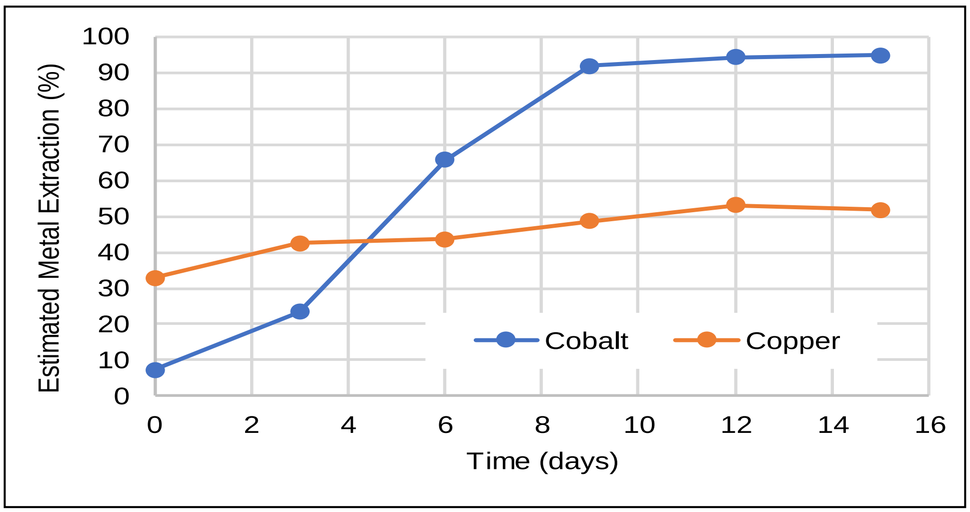
Two tests were conducted at KPM using Santo Domingo concentrate. The first test resulted in only 12% sulphide oxidation and only 17% of the cobalt going into solution. The second Albion test resulted in sulphur oxidation increased to 24% and overall cobalt extractions was 38%.

No further Albion Process tests were conducted.

13.1.13.4.2 Bio-oxidative (BiOX) Tank Leach

A batch BiOX vat test was conducted at SGS Lakefield using a 250 g sample of concentrate ground to a P_{80} of 25 µm. The sample was re-pulped to 15% solids and inoculated with bacteria and nutrient solution. The test was run for a total of 19 days. Sulphide oxidation was essentially complete after 10 days, as shown by the cobalt extraction in Figure 13-8.

Figure 13-8: BiOX Cobalt and Copper Extraction vs. Leach Time



Source: Figure prepared by SGS Lakefield, 2019.

Overall cobalt extraction exceeded 95% after 12 days, with residue assays measuring below the detection limit of 0.01% Co. However, after nine days of bioleaching the process had consumed 250 kg of calcium carbonate per tonne of test feed to maintain the pulp pH above 1.8; and 35 g/L of iron had leached into solution.

13.1.13.4.3 Pressure-oxidative (POX) Leach

In total, six 2 L batch pressure oxidation tests were conducted at SGS using Santo Domingo flotation concentrate. Initial testing used temperature conditions of 150°C and 225°C, both with 100 psi oxygen overpressure. At 225°C, a high degree of sulphide oxidation was achieved, and cobalt extraction reached 98%. In comparison, at 150°C less acid was produced, and the cobalt extraction was roughly half of that realized at 225°C (Table 13-46).

Table 13-46: POX Concentrate Leach Test Summary

Test ID	Unit	POX 1	POX 2	POX 3	POX 4	POX 5	POX 6
Feed P_{80}	µm	207	207	25	25	25	25
Temperature	°C	150	225	150	200	180	225
Retention time	h	3	3	3	3	3	2
Oxygen overpressure	kPa	689	689	689	689	689	689
Pulp density	%	15	15	15	15	15	10
Pre-acidification pH	—	2	2	1	2	2	2
Lignosol addition	g/L	0	0	1	0	0	0
Final free acid	g/L	36	67	43	56	63	59
Final Extraction							
Co	%	47	98	61	98	98	97
Cu	%	79	99	94	99	99	97
Fe	%	25	11	65	15	49	18
Mn	%	94	92	97	97	97	95
Mg	%	93	98	95	97	96	97
Ca	%	57	51	60	59	72	84
Al	%	33	54	40	57	52	63
Na	%	4	25	8	13	14	28
Zn	%	48	53	74	58	73	70

The disadvantage of the higher temperature conditions was an increase in acid generation that would result in higher neutralization costs downstream. Follow-up testing looked at re-grinding the concentrate from the P_{80} size of 207 µm to a P_{80} of 25 µm (POX 3 to POX 6), addition of Lignosol as a sulphur dispersant (POX 3,) and temperatures of 180°C (POX 4) and 200°C (POX 5).

The finer grind and the addition of Lignosol in POX3 slightly improved cobalt extraction over POX1, to 61%. In contrast, higher leach temperature in POX4 and POX5 achieved results closer to the POX6 result. Copper extraction by POX was as good or better than for cobalt, reaching 99% in some tests.

The POX work demonstrated that high cobalt extraction can be achieved, but under the conditions tested this was only possible with high oxidation of sulphide to sulphate. No opportunity was identified to produce elemental sulphur and reduce the final acid concentration.

13.1.13.4.4 Bio-oxidative Heap Leach

The bio-oxidative heap leach approach considers the recovery of the pyrite from the cleaner-scavenger tails via froth flotation to produce a cobaltiferous pyrite concentrate, which is then leached in dynamic copper leach pads to oxidize the pyrite and dissolve the cobalt. The cobalt is recovered from the solution by treating a bleed stream of copper SX raffinate through a continuous, counter-current ion exchange (CCIX) facility. This approach to by-production of cobalt confers the following benefits:

- It requires significantly lower capex compared with the pyrometallurgical approach to cobalt liberation from pyrite (oxidative roasting)
- It reduces acid consumption, as the pyrite is acid-forming on dissolution.
- The bio-oxidation of pyrite produces ferric ion and increases the temperature of the heap, both of which assist in primary copper sulphide leaching
- The pyrite concentrate contains some copper, which will also leach and report to cathode. This also provides for an additional degree of freedom when optimizing the concentrator grade-recovery curve for changing copper prices, gold prices, freight costs, concentrate treatment charges, and copper refining costs.
- The process removes acid-producing pyrite from the tailings impoundment facility and redirects it to a lined heap leach facility, reducing mine acid drainage potential.

Column test results and pilot plant testing of the ion-exchange recovery circuit indicate that this approach to cobalt recovery is likely to be technically and economically feasible for Santo Domingo.

Column leach tests were conducted at Aminpro in Santiago Chile, scaling up the simple bottle roll test work to columns. The inital test matrix was designed to determine the impact of several key parameters on the extraction of cobalt from pyrite. The parameters include:

- Bio-oxidation,
- Aeration,
- Temperature,
- Pyrite concentrate loading,
- Pyrite concentrate source (Mantoverde vs. Santo Domingo),
- Pyrite concentrate particle size,
- Raffinate type, synthetic vs actual raffinate (from Mantoverde commercial leach circuit).

Although the test program is not complete, initial unreconciled results from the columns show that over 90% of the cobalt can be extracted from the pyrite within the typical dynamic leach cycle that currently exists at Mantoverde. Further analysis of the results is required and will be available in future technical reports post the completion of the column program.

In parallel to the column tests conducted at Aminpro, a pilot program was undertaken at Mantoverde to simulate the conditions of the third stage of cobalt extraction. Capstone Copper have proposed a novel process for concentrating the leached cobalt using ion exchange. The process description is described below.

Feed to the pilot plant consisted of raw raffinate solutions, derived from the Mantoverde commercial heap operation, which were spiked with cobalt sulphate to simulate the modelled concentrations of cobalt reporting to the heap leach PLS in the future. The pilot circuit, shown in Figure 13-9, consisted of an iron cementation stage followed by filtration of the precipitate. Filtrates were pH adjusted and passed through a continuous counter current ion exchange unit (CCIX), developed by Puritech, filled with BPA resin to target the cobalt in solution. Loaded elements, including cobalt, nickel and iron, were stripped from the resin in stages to produce a cobalt rich PLS, nickel rich PLS and raffinate. Key variables tested in the program included:

- Feed grade with cobalt, nickel, total iron and ferrous to ferric ratio adjusted,
- Iron powder dosage,
- pH of CCIX feed,
- Cementation residence time and ORP target,
- Feed flow and strip flow rates,
- Acid concentration in strip,
- PLS recycle,
- CCIX indexing time.

Figure 13-9: Pilot Ion Exchange Circuit at Mantoverde for The Recovery of Synthetic Cobalt Solutions.

Source: Capstone Copper, 2024

Although testwork is ongoing to simulate the full range of parameters, the results suggest that cobalt can effectively be loaded and stripped from the resin given suitable conditions. A visual of the cobalt loading can be found in Figure 13-10.

Figure 13-10: Cobalt Loading onto BPA resin.

Source: Capstone Copper, 2024

Further work is required to determine the optimal conditions for cobalt loading while reducing deportment of deleterious elements the cobalt PLS. This work and the detailed results of the pilot program Will be published in future technical reports.

13.1.13.5 Pyrite Roasting and Calcine Leaching

A pyrometallurgical approach to sulphide oxidation was investigated in a series of roasting tests conducted at KPM. In total, seven roasting tests were carried out, summarized in Table 13-47. The residue from each test was subjected to acid leach testing to quantify the potential cobalt and copper extractions.

The first roasting test was carried out in a continuously fed, 2” fluid bed roaster. Residue collected from the test was split into test charges and leached at varying acid concentration for 24 hours. The results indicated that the soluble cobalt species leached quickly, within the first few hours of the test.

Subsequent tests evaluated batch fed equipment including a short rotary furnace and a rotary kiln under varying operating temperatures and oxygen additions. Optimum cobalt leach extraction was achieved with residues from tests conducted at a temperature of 660–680°C with SO_2 and O_2 at 10% (Leach Tests K and M).

In Leach Test L the roasting temperature of 800–840°C resulted in poor metal extraction. This is believed to be due to the formation of ferrites at the higher temperature. A POX leach was conducted on the residue under more aggressive leach conditions (250°C and 50 g/L acid) achieved cobalt and copper recoveries of 92% and 90%, respectively. This suggests the potential to operate at a higher roasting temperature followed by pressure leaching of the residue.

Table 13-47: Concentrate Roast/Leach Test Summary

Roasting Test work		Leaching Test work						
Furnace Type (Test ID)	Conditions	Leach Test ID	Leach Temp. (°C)	Time hrs	Acid Addition (kg/t feed)	% Extraction		
						Co	Cu	Fe
Fluid bed (R1)	600°C, continuous	A	95	24	358	92	95	38
Fluid bed (R1)	600°C, continuous	B	95	24	137	84	93	16
Fluid bed (R1)	600°C, continuous	F	95	4	204	82	84	14
Short rotary (R2)	600°C, low O_2	C	95	4	779	78	91	17
Short rotary (R2)	600°C, low O_2	E	95	4	96	77	88	22
Short rotary (R2)	600°C, low O_2	G	95	4	502	65	85	0
Short rotary (R3)	600°C, high O_2	D	95	4	150	71	88	14
Short rotary (R3)	600°C, high O_2	H	95	4	185	69	80	4
Rotary kiln (R4)	600–630°C, O_2, SO_2, N_2	J	95	4	44.9	79	78	0
Rotary kiln (R5)	660–680°C, O_2, SO_2, N_2	K	95	4	97.1	92	76	0
Rotary kiln (R6)	800–840°C, O_2, SO_2, N_2	L	95	4	110	6	22	0
Rotary kiln (R7)	660–680°C, O_2, SO_2, N_2	M	95	2	31.7	87	62	0

Leach results for cobalt were found to be essentially independent of leach time or acid addition. Maximum cobalt extraction was achieved after one hour of leaching and, in many cases, before the addition of acid. Higher acid additions and longer leach times did favour higher copper extraction, but also dissolved more iron.

13.1.13.6 Cobalt Solution Purification

Preliminary solution purification test work was conducted at SGS Lakefield to evaluate flowsheet options for downstream treatment. For this work a bulk pregnant leach solution (PLS) was generated using leach residue reject from the roasting test work. The solids were leached at a controlled pH of 3 for 2 hours and then filtered.

The PLS was neutralized with limestone in two stages. The primary stage, to pH of 3.2, achieved 95% precipitation of the contained iron and 25% precipitation of the contained aluminum. Cobalt and copper were found to remain in solution. The second stage precipitation to pH 4.0 precipitated virtually all the remaining iron and 96% of the remaining aluminum.

A sample of post-secondary neutralization PLS was used to evaluate copper precipitation (as CuS) using sodium hydrosulphide (NaHS). Laboratory-scale dosing of the solution with 110% stoichiometric addition of NaHS resulted in approximately 50% Cu recovery. The results indicate that higher NaHS addition is likely required to achieve a copper recovery typical of an operating plant (i.e. greater than 98%).

The final purification test was conducted using Caro's acid to precipitate manganese from the CuS precipitation raffinate. The feed was 140 ppm Mn and after 105 minutes of precipitation the filtered solution was 9.6 ppm Mn. The cobalt grade of the final solution was equal to the feed at 1,060 ppm, indicating minimal losses.

13.1.13.7 Cobalt Solvent Extraction

Based on the results obtained in the purification test work, the design criteria for head grade leach density and leach extraction, an estimate was made of solution tenors for the feed to a cobalt SX circuit. This information was provided to Solvay for circuit modelling.

13.1.13.8 Cobalt Variability

Sample selection was dictated largely by the availability of samples that could be used without impacting sample availability for the copper/iron process flowsheet development programs. The samples were limited to materials remaining from the 2015 pilot testing program conducted at ALS Santiago. A statistical analysis was conducted to establish the legitimacy of applying results from the limited sample set.

The statistical analysis suggests that there is little correlation between the copper concentration in the samples and the total sulphur concentration (pyrite content is independent of copper content). There is, however, a strong correlation between the sulphur content and the cobalt content in the drilling database for the sulphide zone.

13.1.14 Copper Oxide

Metallurgical test work was completed using copper oxide composite samples at bench bottle roll (30 samples) and leaching columns (6 columns) for the Santo Domingo ore.

Qemscan mineralogy from the composites used for the column test reported dominance of green copper oxides (chrysocolla, malachite, brochantite) over black copper oxides (CuMn silicates) and copper limonite. Gangue mineral corresponded to calcite and gypsum.

The overall average Cu soluble recovery for heap leaching via column testing obtained was 72.4% Cu soluble recovery (Cu head grade measured of 0.46% Cu with solubility ratio of 71%) and 48% Cu soluble recovery for ROM (Cu head grade measured of 0.51% Cu with solubility ratio of 63%).

13.2 Recovery Estimates

13.2.1 Santo Domingo Flotation Recovery Model Development

Santo Domingo flotation modelling was conducted using the Aminpro flotation testing and modelling method to develop the key flotation kinetic parameters: R_{max} (recovery at infinite time), and k (flotation rate constant). The model utilizes a simplified compartmental model that accounts for recovery by flotation from the slurry to the froth interface (Collection Zone), subsequent recovery to concentrate (Froth Zone), and the effect of recovery by entrainment see below:

- Collection Zone modelling utilizes kinetic parameters generated from laboratory scale testing assuming a First Order Plug Flow Reactor Rate Equation. Modelling of the continuous flotation process utilized the same kinetic parameters but applied to a First Order Continuous Stirred Tank Reactor (CSTR) Rate Equation.

- Froth Zone recovery is determined for each flotation stage as a function of the relative froth depth and the superficial gas velocity (J_g).
- Recovery by physical entrainment is estimated as a function of water recovery and the degree of entrainment which increases with finer particle sizes.
- Model calculations were conducted using mineral assays estimated based on comprehensive mineralogy studies, and an iterative procedure was applied to determine the overall metallurgical flowsheet balance.

13.2.2 Summary of Recovery Projection Procedures

The procedures utilized to develop the Santo Domingo flotation recovery equations are summarized below:

Testing - A series of bench scale batch flotation tests were designed and conducted for each major flotation stage operating under specific grind, and pH conditions (Node). The Santo Domingo Flowsheet the following Nodes were defined:

a) Node 1-Rougher Flotation
b) Node 2-Cleaner Flotation
c) Node 3 (Future)- Pyrite Flotation

Flotation testing includes both Full Kinetic Tests (FKT's) which are based on multiple assay-by-size-fraction data, and/or Simple Kinetic Tests (SKT's) based on the total product assays.

Extract Kinetic Parameters - Raw laboratory test assays were balanced, converted to mineral assays, and analysed to extract R_{max} and k values for each of the following key minerals or mineral groups as identified by modal mineralogy studies within each Flowsheet Node:

- Chalcopyrite ($CuFeS_2$),
- Bornite (Cu_5FeS_4),
- Chalcocite (Cu_2S),
- Covellite (CuS),
- Tennantite ($Cu_8As_2S_7$),
- Pyrite (FeS_2)c,
- Hematite (Fe_2O_3),
- Magnetite (Fe_3O_4),
- Fe Gangue (Other Iron bearing Gangue- primarily Goethite),
- Si Gangue (Silica Gangue-primarily SiO_2).

Develop Correlation Equations for Kinetic Parameters - Statistical analyses of the kinetic parameter data were conducted to develop correlation equations allowing projection of R_{max} and k values for each mineral within each Node as a function of key head assays and/or operating parameters. These correlation equations were input to the Santo Domingo Flotation Simulation Model. The following Table 13-48 presents a summary of the calculated flotation kinetic parameters for the key elements and minerals by Flowsheet Node.

Table 13-48: Summary of Key Kinetic Parameter

Rougher Flotation (Node 1)				
-	Au	Co	CuFe S_2	Fe S_2
R_{max}	66.38	99.25	93.69	93.99
K	2.30	1.19	1.91	1.84
Cleaner Flotation (Node 2)				
-	Au	Co	CuFe S_2	Fe S_2
R_{max}	87.31	37.60	99.01	29.08
K	2.66	0.16	3.18	1.11
Pyrite Flotation (Node 3)				
-	Au	Co	CuFe S_2	Fe S_2
R_{max}	82.59	94.95	62.08	96.77
K	2.59	2.79	0.59	2.15

Model Proposed Flowsheet - A series of 80 flotation circuit simulations were run using: the expected processing rate, flowsheet configuration, equipment type and size data, and the expected key operating parameters (grind/pH/%Solids/Reagent Dosage). Final model cases were defined by varying the following key head grade parameters:

- Total Cu (%Cu_T),
- Acid Soluble Cu (%CuSA),
- Cyanide Soluble Cu (%CuCN),
- Acid Soluble Cu Ratio ($CuSA_{ratio}$),
- CN Soluble Cu Ratio ($CuCN_{ratio}$),
- Total Cu/Total S Ratio (%Cu_T/%S_T),
- Cobalt ppm (Co),
- Au g/t (Au).

In all cases, simulations were run to optimize the concentrate Net Smelter Return (NSR) without considering the effect of by-product cobalt recovery.

Develop Correlation Equations To Predict Metallurgical Performance - The database of model projected metallurgical results was then correlated to the ore grade parameters available within the block model to generate performance projection equations presented in the following Table 13-49. The recovery Multiple Correlation Coefficients show that high degrees of correlation were achieved. It should be noted that minimum and maximum limits shown were determined by running model simulations using the average head grade parameters for each year of the preliminary mine-plan.

Table 13-49: Santo Domingo Metallurgical Projections- Regression Coefficients

Parameters		Co	Cu	Cu^2	Cu^3	Cu^4	Cu SA Ratio	Cu / S	Intercept	Multiple R	Min	Max
Definition of Parameters		Co head grade	Cu head grade				Acid Sol Cu/Total Cu	Total Cu/ Totl S				
Units		g/t	%									
Cu Rec	Coefficients	(0.011)	11.967	(13.470)	5.502	(0.920)	(68.714)	-	94.068	0.9947	-	95
Au Rec	Coefficients	-	49.959	(43.778)	-	-	-	41.219	44.979	0.974	20	80
Cu Conc %Cu	Coefficients	-	(1.701)	-	-	-	7.767	23.362	22.330	0.8143	20	38

Because acid soluble Cu (%CuSA) assays, and therefore CuSARatio values, were not available in the block models, the effect of oxidized mineralization was estimated using the weathering model to assign representative copper soluble ratios to each Oxidation Zone as follows:

- Mixed Zone- 0.2 or 20% (range 10%-30%)
- Sulphide Zone- 0.05 or 5% (range 0-10%)

13.2.2.1 Copper

During 2018 a global copper recovery algorithm was defined based on copper head grade, which was further refined considering the test work completed during 2019. The incorporation of the 140 variability samples allowed for an update of the copper recovery algorithm and for the incorporation of an algorithm to establish copper concentrate grade.

Cu head grade and soluble copper content in acid were identified as the most relevant variables for the copper recovery and concentrate grade algorithm. The algorithms established are as follow:

- Cu Recovery algorithm

$$Cu\ Rec = -0.011*Co+11.967*\%CuT-13.740*\%CuT^2+5.502*\%CuT^3-0.920*CuT^4-CuSAratio*68.714+94.068$$

- Cu concentrate grade algorithm

$$Cu\ Conc\ grade = -1.701*\%CuT+CuSAratio*7.767+(CuT/S)*23.362+22.330$$

The identification by mineralization zone according to the weathering model allowed to assign a representative copper soluble content value to each zone (0.2 or 20% for mixed zones and 0.05 or 5% for the sulfide zone). The algorithms then established are as follow:

- Cu Recovery and Cu concentrate grade algorithm for Mixed zone.

$$Cu\ Rec = -0.011*Co+11.967*\%CuT-13.740*\%CuT^2 +5.502*\%CuT^3-0.920*CuT^4-0.20*68.714 +94.068$$

$$Cu\ Conc\ grade = -1.701*\%CuT+0.2*7.767+(CuT/S)*23.362 + 22.330$$

- Cu Recovery and Cu concentrate grade algorithm for Sulphide zone.

$$Cu\ Rec = -0.011*Co+11.967*\%CuT-13.740*\%CuT^2 +5.502*\%CuT^3-0.920*CuT^4-0.05*68.714 +94.068$$

$$Cu\ Conc\ grade = -1.701*\%CuT+0.05*7.767+(CuT/S)*23.362 + 22.330$$

13.2.3 Gold

During 2018 a global gold recovery algorithm was defined based on copper head grade, which was further refined considering the test work completed during 2019.

The incorporation of the 140 variability samples also allowed for an update of the gold recovery algorithm.

Au and Cu head grade along with the soluble copper content were initially identified as the most relevant variables for the gold recovery algorithm. The initial algorithm established was as follow:

- Au Recovery algorithm

$$Au\ Rec = -142.598*Au+44.752*\%CuT-10.095*\%CuT^2- CuSAratio*46.680+51$$

Similarly, the identification by mineralization zone according to the weathering model allowed to assign a representative copper soluble content value to each zone (0.2 or 20% for mixed zones and 0.05 or 5% for the sulfide zone). The gold algorithm then established is as follow:

- Au Recovery algorithm for Mixed zone.

$$Au\ Rec = -142.598*Au+44.752*\%CuT-10.095*\%CuT^2-0.2*46.680+51$$

- Au Recovery algorithm for Sulfide zone.

$$Au\ Rec = -142.598*Au+44.752*\%CuT-10.095*\%CuT^2-0.05*46.680+51$$

While this algorithm was used for resources and reserves optimization, significant late-stage work was performed to improve the gold recovery model, and this resulted in a new equation, as shown below. Because the impact on the resources and reserves were deemed to be negligible, no changes were made the project resource and reserve statement as a result of the improved expected gold recovery.

- Au Recovery algorithm

$$Au\ Rec = 49.959*\%CuT - 43.778*\%CuT^2 + 41.219*(Cu/S) + 44.979$$

13.2.4 Iron

Magnetic susceptibility readings and DTT data are used to predict the mass recovery to the final concentrate. In this context, complementary DTT completed during 2021 allowed for the iron mass recovery algorithm update. The new regression used a total of 1,324 samples for the generation of an iron mass recovery model that have two different expressions and different cutoffs depending on:

- Deposit zone: Santo Domingo or Iris Norte.
- Cut-off grades by lithology

Santo Domingo DTT data contains 1,096 samples, resulting in the relationship:

$$MassRec = (-0.0054 * (magsus/1000)^2 + (1.3444*magsus/1000))$$

Iris Norte DTT data contains 228 samples, resulting in the relationship:

$$MassRec = (-0.0042 * (magsus/1000)^2 + (1.1936*magsus/1000))$$

Details on iron recovery algorithm application, iron concentrate grade and cut-off by lithology are described in 14.3.3.3.

13.3 Deleterious Elements

13.3.1 Copper Concentrate

Copper concentrate generated from a Year 1-5 composite sample at different test work levels such as bench and pilot scales were chemically analyzed (see Section 13.1.4). Also, copper concentrates generated from different mineralized geological units and yearly composites covering LOM were chemically analyzed (see Section 13.1.4).

From all test work completed it is concluded that arsenic values were low, the silica level is acceptable and heavy minerals such as bismuth, antimony and cadmium are low. In the QP's opinion, the levels of deleterious elements in the copper concentrate are such that no penalties are likely to be levied.

Table 13-50: Full Results of Final Cu concentrate from Composite Year 1-5 Extended 20-cycle LCT

Element	Unit	Final Cu Concentrate
Ag	g/t	21.6
Al	%	0.68
As	%	0.02
Au	g/t	2.92
Ba	ppm	109
Be	ppm	<0.2
Bi	ppm	<2

Element	Unit	Final Cu Concentrate
C_{tot}	%	0.58
Ca	%	0.77
Cd	ppm	8.0
Cl	%	< 0.01
Co	ppm	467
Cr	ppm	18
Cu	%	27.9
F	%	< 0.01
Fe	%	30.2
Ga	ppm	<20
Ge	ppm	<20
Hf	ppm	<20
Hg	ppb	824
In	ppm	<20
K	%	0.2
Li	%	9
Mg	%	0.39
Mn	ppm	390
Mo	ppm	68
Na	%	0.08
Nb	ppm	<10
Ni	ppm	31
P	%	0.269
Pb	ppm	236
Pd	g/t	<0.01
Pt	g/t	0.01
Rb	ppm	<20
Re	ppm	53.6
S_{tot}	%	28.9
Sb	ppm	46.6
Se	ppm	31
Si	%	1.99
Sn	ppm	34
Sr	ppm	24
Ta	ppm	21
Te	ppm	45
Ti	%	0.08
Tl	ppm	3
V	ppm	25.4
W	ppm	29
Zn	ppm	694
Zr	ppm	37

Source: Blue Coast Research, Canada, 2022.

13.3.2 Iron Concentrate

Iron concentrate generated from a Year 1-5 composite sample at different test work levels such as bench and pilot scales were chemically analyzed (see Section 13.1.7). Also, variability samples from different mineralized geological units covering LOM were chemically analyzed.

Iron concentration test work demonstrated feasibility for the Santo Domingo ore to achieve target 65% Fe but also demonstrated feasibility to achieve 67% Fe.

Impurities levels varied though the test work demonstrating capacity to achieve low phosphorus and sulfur value. Main impurities such as silica and aluminium reported medium to low level.

Table 13-51: Results from LIMS Test Work Program on Variability Samples

Composite Sample ID	Assay										
	Fe (%)	SiO_2 (%)	Al_2O_3 (%)	CaO (%)	MgO (%)	P (%)	S (%)	Na_2O (%)	K_2O (%)	Mn (%)	Cu (ppm)
134	64.32	5.70	1.22	1.40	0.495	0.025	0.025	0.136	0.136	0.094	89
127	63.94	6.00	1.20	1.40	0.540	0.015	0.049	0.13	0.119	0.094	148
65	65.98	5.04	1.12	0.370	0.360	0.012	0.032	0.197	0.211	0.066	62
49	63.26	4.12	0.800	0.720	0.415	0.011	0.025	0.121	0.122	0.075	68
20	67.79	3.66	0.574	0.355	0.155	0.008	0.018	0.028	0.043	0.048	81
55	69.00	2.13	0.534	0.345	0.150	0.004	0.030	0.028	0.056	0.068	111
74	66.73	3.73	0.770	0.475	0.315	0.008	0.021	0.065	0.103	0.074	107
131	68.17	3.13	0.632	0.555	0.165	0.006	0.021	0.034	0.068	0.072	64
123	64.92	3.62	0.778	1.01	0.410	0.008	0.022	0.042	0.148	0.076	128
138	66.13	3.81	1.16	1.32	0.320	0.010	0.022	0.120	0.134	0.084	86
70	65.37	3.69	0.818	0.465	0.365	0.008	0.030	0.127	0.108	0.076	177
57	66.88	4.15	0.792	0.720	0.345	0.010	0.031	0.048	0.093	0.112	61
76	69.33	2.18	0.464	0.200	0.125	0.005	0.019	0.098	0.049	0.058	28
118	67.24	3.64	0.800	0.225	0.300	0.011	0.037	0.171	0.081	0.052	118
63	65.59	5.41	1.05	0.535	0.475	0.012	0.043	0.047	0.216	0.062	73
128	66.94	3.14	0.885	0.450	0.313	0.011	0.023	0.106	0.110	0.070	68
130	67.77	3.34	0.638	0.490	0.240	0.012	0.024	0.023	0.055	0.056	35
125	65.44	4.38	0.608	0.272	0.290	0.009	0.022	0.027	0.055	0.072	293
44	67.69	3.08	0.920	0.455	0.300	0.008	0.031	0.085	0.109	0.062	72
116	68.14	3.08	0.588	0.325	0.255	0.010	0.018	0.073	0.046	0.056	32
11	66.33	3.77	1.02	0.525	0.425	0.016	0.021	0.200	0.113	0.108	50
30	68.21	2.80	0.676	0.415	0.240	0.010	0.020	0.054	0.078	0.060	63
21	66.11	2.99	0.958	0.520	0.440	0.008	0.024	0.084	0.101	0.084	52
87	67.99	2.98	0.611	0.305	0.290	0.009	0.023	0.048	0.098	0.056	61
84	67.46	3.65	0.788	0.245	0.385	0.011	0.021	0.190	0.048	0.076	39
86	67.61	3.25	0.816	0.460	0.280	0.005	0.027	0.100	0.095	0.054	117
62	68.13	3.14	0.702	0.290	0.270	0.009	0.024	0.086	0.119	0.054	98
12	67.15	3.78	0.852	0.500	0.350	0.013	0.014	0.252	0.063	0.048	32

Composite Sample ID	Assay										
	Fe (%)	SiO_2 (%)	Al_2O_3 (%)	CaO (%)	MgO (%)	P (%)	S (%)	Na_2O (%)	K_2O (%)	Mn (%)	Cu (ppm)
110	68.13	2.93	0.702	0.480	0.270	0.006	0.017	0.099	0.062	0.028	43
97	65.91	3.46	0.790	0.460	0.425	0.010	0.026	0.068	0.132	0.056	70
129	67.27	3.71	0.804	0.635	0.350	0.013	0.023	0.116	0.118	0.066	76
137	69.84	1.70	0.502	0.175	0.130	0.004	0.025	0.042	0.057	0.062	29
75	65.08	2.66	0.568	0.775	0.415	0.006	0.023	0.049	0.091	0.070	65

Source: San Lorenzo Ltda, Vallenar, Chile, 2023.

13.4 Conclusions and Recommendations

13.4.1 Conclusions

Additional metallurgical test work was completed for all main processing areas such as comminution, flotation, iron concentration and tailings, improving general representativeness of the Santo Domingo and Iris Norte orebodies. In detail:

- For the comminution circuit update, a total of 45 variability samples were added - this is about 15% increase in SMC compared to previous dataset. This data plus the previous data collected allowed to develop a geometallurgical model that indicated that an increase in throughput was possible during the first 4 years, which was further established within the updated mine plan.
- Flotation test work completed on volcaniclastic units (main units with respect to current reserves), allowed for an optimization of copper flotation reagents which allowed then for a successful pilot plant using conventional cells, reporting 26% Cu with global Cu recovery of 91.1%.
- The flotation test work also considered different mineralized geological units and yearly composites covering LOM which demonstrated the feasibility for the ore to produce saleable copper concentrate under the flotation conditions established – copper concentrate grade may also improve when pneumatic technology for cleaner stage is used, as per current flotation circuit design.
- For flotation, 140 variability samples were added to the original 28 variability samples tested with desalinated water during 2019. This allowed for the Cu and Au algorithm update and recognition of one relevant variable not previously identified, as it is the soluble copper content.
- From the 140 variability samples, it was also possible to establish a copper concentrate algorithm, which has been used to populate the block model.
- With respect to iron concentration, more than 1,300 DTT were added to the previous dataset of about 500 DTT which allowed to establish differentiated mass recovery algorithms – one for Santo Domingo and one for Iris Norte – Norte – and a methodology to estimate the iron concentrate grade within the block model.

- Also relative to iron concentration, 39 variability samples were tested following the current processing diagram, demonstrating the feasibility for the Santo Domingo ore to produce iron concentrate >65% Fe. This test also allowed to demonstrate the feasibility for the Santo Domingo ore to produce iron concentrate grade >67% Fe.
- The analysis of the samples with iron concentrate < 62%Fe and their respective geological units, indicated that ore/concentrate blending strategies or operational strategies (such as finer regrind) may be implemented to improve their iron concentrate grade during the plant operation.
- Tailings test work was completed using a representative sample containing tailings from copper and iron concentration circuit which allowed to establish a more cost-effective design with only one stage of tailings thickening. Also, first tailings variability samples were tested for FS Optimization allowing to identify potential risks and their handling strategies.
- Finally, independent and vendor testing was also completed on several areas such as flotation, iron concentration, regrind, concentrates thickening and filtration, as well as tailings thickening which contributed to a more robust design and thus reducing its associated risk.

13.4.2 Recommendations

Respect to Metallurgical Test work, the following recommendations are made:

- Completion of additional hardness physical characterization test work in order to improve geometallurgical modelling and throughput increasing, especially in areas where additional reserves were added.
- Completion of complementary flotation test work to evaluate the possibility to reduce lime addition, especially in (Cu rougher flotation); and for also evaluate flotation performance evaluation at coarser P_{80}, both for investment and operational cost reduction.
- Completion of additional flotation test work including Cu soluble variable measurement to increase flotation data for copper recovery algorithm, especially in areas where additional reserves were added and mixed zone.
- Completion of additional DTT to update the mass recovery and iron concentrate grade algorithms, especially in those areas where additional reserves were added.
- Conduct iron concentrate downstream testing for marketing requirements.
- Continue progressing with variability tailings study.
- Continue progressing with oxide metallurgical test work for Santo Domingo project.

13.5 Comments on Section 13

13.5.1 Copper, Gold, and Iron

Metallurgical test work including physical characterization, sulfide flotation, regrind, settling and filtration tests on the copper concentrate as well as iron concentration, regrind, and settling and filtration tests on the magnetite concentrate, along with settling and thickening test work completed on final tailings, was carried out to support the Feasibility study optimization.

The sulfide flotation test work included several flotation test work programs at bench and pilot scale, including flotation conditions optimization and main mineralized geological units and yearly composite samples testing. Some of the most relevant tests completed on the Year 1-5 composite sample were the extended 20-cycles LCT with a copper head grade sample of 0.52% Cu which reported a Cu circuit recovery of 91.3% with Cu concentrate grade of 28.3%; and the flotation pilot plant with a Cu head grade of 0.56% Cu and desalinated seawater which reported a Cu circuit recovery of 91.1% with Cu concentrate grade of 26.0% Cu. Also, a comprehensive flotation variability program was run on 140 (12 m-height) variability samples for copper and gold algorithm update, complemented with a flotation variability program using dilution cleaner test which reported an average Cu concentrate grade of 28% Cu. Assays completed on concentrate samples do not indicate relevant levels of deleterious elements in the copper concentrate. Gold recoveries and grades were confirmed through the different flotation programs developed.

The iron concentration test work also included several programs at bench and pilot scale; including the first magnetic separation pilot plant using an industrial diameter 48-inch LIMS magnetic separation drum and desalinated seawater using rougher flotation tails from flotation pilot program for a Year 1-5 composite sample - this pilot plant reported an iron concentrate grade of 64.94% Fe with overall mass recovery of 15.7% - several bench scale test work programs to identify alternatives for iron concentrate grade improvement such as reverse flotation, screening, hydroseparation, additional magnetic cleaning stages and finer regrind were developed; from all these programs, the selected conditions were established for a second magnetic separation pilot plant were the selected alternatives hydroseparation and additional cleaning stages were considered.

These programs were complemented with an iron concentration variability program using LIMS equipment, reporting an average iron concentrate grade of 65.2% but more significantly, confirming achievement of iron concentrate grades of 67% Fe and over. The results indicated that a significant number of samples (45% of the samples) reported iron concentrate grade over 67% Fe. Mass recoveries varied from 4.2% to 65.1% where – in general terms - high mass recovery was related to high iron concentrate grade.

Impurities elements vary through the variability samples tested and also confirmed suitability for Santo Domingo ore to achieve low impurities levels such as for Direct reduction category. Additional information on marketability is included in Section 19.

14 MINERAL RESOURCE ESTIMATES

14.1 Summary

Capstone retained SLR to complete an updated Mineral Resource estimate for Santo Domingo Sur, Iris, Iris Norte, and Estrellita areas of the Santo Domingo Project, situated in the Atacama Region, Chile. SLR updated the Estrellita (ES) model in 2022 and the combined Santo Domingo Sur – Iris – Iris Norte (SD-IN) model in 2023. Subsequently, in early 2024 Capstone used updated recovery curves for Cu, Au and Fe, updated metal prices and costs, and an updated NSR script to calculate updated NSR values within both SLR models. The effective date for the updated SD-IN and ES resources is 31 March 2024.

Capstone provided SLR with three new sets of geological and oxidation models for the Project. The first encloses Santo Domingo Sur and Iris, the second encloses Iris Norte, and the third encloses Estrellita.

Capstone also provided a new database, reconstructed from source logs and assay certificates. The work was internally verified as it was being completed.

Incorporating the new geological modelling and the rebuilt database, SLR constructed two new sub-blocked models. The first (SD-IN) incorporated Santo Domingo Sur, Iris, and Iris Norte into one large model. The second model (ES) was constructed for Estrellita. A summary of the nomenclature used is provided in Table 14-1.

Domaining strategies were revised based on the most important controls to mineralization in each area. Block grade estimates were generally performed in three passes in the order of 70 m, 140 m, and 300 m, using ordinary kriging. Both models used density weighting for Au, Co, Cu, Fe, S, and pyrite estimates, and employed Dynamic Anisotropy to vary search orientations, and used regression curves to estimate block density (Figure 13-1 to Figure 13-3) . Magnetic susceptibility (magsus), mass recovery (massrec), iron in concentrate (FeDTT), and density estimates were not density weighted. For SD-IN, SLR estimated iron mass recovery using regression formulae developed from Davis Tube tests. SLR then generated two regular models from the sub-blocked models to generate Whittle shells, classify the estimated material, and produce the final Mineral Resource estimates.

Table 14-1: Mineral Resource Nomenclature

Area	Area Code	Geological And Oxidation Models	Block Model
Santo Domingo Sur	1	SD	SD-IN
Iris	-	-	-
Iris Norte	2	IN	-
Iris Norte West outside Whittle area	4	-	-
Estrellita	3	ES	ES

Note: 1. Area Code 2 and 4 will be merged to Area Code 2 in later models.

Table 14-2: Santo Domingo Project Mineral Resource Estimates as of 31 March 2024

Category	Tonnage	Grade				Contained Metal		
	(Mt)	NSR ($/t)	Cu (%)	Fe (%)	Au (g/t)	Cu (kt)	Fe (Mt)	Au (koz)
Measured								
Santo Domingo	134	46	0.51	26.9	0.07	679	36	293
Total Measured	**134**	**46**	**0.51**	**26.9**	**0.07**	679	**36**	**293**
Indicated								
Santo Domingo	298	34	0.26	25.7	0.04	786	77	362
Iris Norte	74	28	0.14	24.0	0.02	106	18	43
Santo Domingo + Iris Norte	372	33	0.24	25.4	0.03	892	95	405
Estrellita	41	24	0.32	-	0.03	133	-	44
Total Indicated	**413**	**32**	**0.25**	**N/A**	**0.03**	1,025	**95**	**449**
Total Measured + Indicated	**547**	**35**	**0.31**	**N/A**	**0.04**	1,704	**131**	**742**
Inferred								
Santo Domingo	173	28	0.20	22.2	0.03	349	38	157
Iris Norte	30	28	0.12	23.9	0.01	35	7	14
Santo Domingo + Iris Norte	203	28	0.19	22.5	0.03	384	46	171
Estrellita	27	25	0.34	-	0.03	93	-	29
Total Inferred	**230**	**28**	**0.21**	**N/A**	**0.03**	477	**46**	**200**

Notes: **1.** Mineral Resources are classified according to CIM (2014) definitions. **2**. Mineral Resources are reported inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. **4.** Mineral Resources for the Santo Domingo, Iris Norte and Estrellita deposits have an effective date of 31 March 2024. **5.** Only mixed and primary material were recognized in the NSR calculation; oxides were excluded. **6.** The average Iron grade for the Project Total Indicated, Total Measured plus Indicated, and Total Inferred Resources cannot be calculated because Estrellita does not contain iron resources**. 7.** Notes specific to the Mineral Resources for the Santo Domingo and Iris Norte deposits: **a.** Mineral Resources for SD include Iris. **b.** Mineral Resources are reported using a net smelter return (NSR) cut-off value of US$9.85/t. NSR is calculated using average long-term prices of US$4.10/lb Cu, US$1,600/oz Au, and Fe prices that depend on the expected grade of the Fe concentrate (US$94.75/dmt or $129.77/dmt or $140.26/dmt Fe concentrate). **c.** Only copper, gold, and iron were recognized in the NSR calculation; cobalt and sulphur were excluded. **d.** Mineral Resources are constrained by preliminary pit shells derived using a Lerchs–Grossmann algorithm and the following assumptions: pit slopes 36.3°- 47.9°; mining cost is calculated using a function that depends on where the material comes from (Santo Domingo or Iris Norte) and its destination (dumps, plant or stock), processing cost based on Fe concentrate routing code (including general and administrative (G&A) costs); processing recovery are calculated based in the recovery curves for copper and gold. Iron recoveries are calculated based on magnetic susceptibility. Metal prices used are those listed on Note 5b**. 8.** Notes specific to the Mineral Resources for the Estrellita deposit: **a.** Mineral Resources are reported using an NSR cut-off value of US$9.63/t. NSR is calculated using average long-term prices of US$4.10/lb Cu, and US$1,600/oz Au. **b**. Only copper, and gold were considered in the NSR calculation; iron, cobalt and sulphur were excluded. **c.** Mineral Resources are constrained by preliminary pit shells generated using a Lerchs–Grossmann algorithm and the following assumptions: pit slopes 43º; mining cost of US$1.55/t, processing cost of US$9.46/t (including G&A cost); processing recovery are calculated based in the recovery curves for copper and gold, and metal prices of US$4.10/lb Cu, and US$1,600/oz Au. **9.** Rounding as required by reporting standards may result in apparent summation differences. **10.** Tonnage measurements are in metric units. Copper and iron are reported as percentages (%) and gold as grams per tonne (g/t).

Mineral Resources for the Property (Table 14-2), including the SD-IN and ES areas, have been prepared with an effective date of 31 March 2024. Only mixed and primary (sulphide) materials were considered in the resource estimation both SD-IN and ES. Oxides are not considered part of the resource at this stage of the project. The estimates for SD-IN are based on Whittle shell optimization at an NSR cut-off value of US$ 9.85 per tonne. The Mineral Resource estimates for ES are based on Whittle shell optimization at an NSR cut-off value of US$9.63 per tonne. Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves (CIM (2014) definitions) were used for Mineral Resource classification.

A significant proportion of the deposit gets its value almost solely from iron, so it is not appropriate to calculate a Copper Equivalent (CuEq) value for that kind of material. Because of that, Capstone decided to stop reporting Copper equivalent values for the Santo Domingo Project, and average NSR values (in US$/t) are reported instead.

Mineral resources that are not mineral reserves do not have demonstrated economic viability.

The QP is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the Mineral Resource Estimate.

14.2 Resource Database

As described in Section 10, Capstone has reconstructed the entire diamond drill hole database from source information into a new acQuire database, checking and verifying all the records as they were compiled.

The cut-off date for assay data was October 7, 2021, for SD-IN and March 31, 2008, for ES. The assay database contained 79,105 assay intervals. The Mineral Resource database contains 65,936 non-zero values for copper; 66,168 non-zero values for iron, cobalt and sulphur; and 39,166 non-zero values for gold. Most sampled intervals are two metres in length for RC holes and one metre to two metres for diamond drill holes. The database contains 27,549 samples with non-zero magsus values. Unlogged and/or unsampled intervals were set to 0.0001 and then flagged as 'use' or 'ignore'.

Cobalt and sulphur are recent additions to the resource modelling, having been included in 2018 and 2020, respectively. Cobalt initially lacked a complete range of QA/QC data, however, additional QA/QC work for cobalt was carried out.

The assay QA/QC work for sulphur was reviewed in February 2020 (SLR, 2020), and concluded that:

- The sampling and assaying for sulphur has been conducted in a reasonable fashion using industry-standard techniques and a certified commercial laboratory.
- An acceptable level of assay QA/QC samples were collected, and this sampling had found no issues with the assaying for sulphur at Santo Domingo Sur.
- The drill hole database for sulphur is reasonably free from error and suitable for use in Mineral Resource estimation.
- There are a large number of samples at Estrellita and Iris Norte in which sulphur concentrations are low or near the detection limit (DL). Many of these samples, especially at Estrellita, are coincident with high copper values, which is not congruent with the current understanding of the mineralization at Santo Domingo Sur.
- Statistical analyses indicated that top cuts are generally not required at Santo Domingo Sur.

As a result of the study, Capstone reviewed the mineralogy at SD-IN to develop a better understanding of sulphur in the deposits, amended the treatment of over-limit sulphur values to include re-assays in order to determine the true grade of these samples, and included sulphur in the drill program protocols for assay QA/QC review and remediation.

Magnetic susceptibility was measured on 25,664 coarse reject samples and 2,078 pulp duplicate samples at regular intervals using three different instruments and is used to calculate mass recovery in composites. The instruments were calibrated at regular intervals, differential readings were standardized to one instrument's output, and results were subjected to review and quality control.

A summary table of the Mineral Resource database is provided in Table 14-3.

Table 14-3: Summary of Drill hole data used in Resource Estimates

Variable Name	Count	Min	Max	Mean	CV
Au (g/t)	79,105	0.0001	4.710	0.018	2.67
Co (ppm)	79,105	0.0001	4,820.0	97.3	1.75
Cu (%)	79,101	0.0001	8.360	0.147	2.19
Density (calculated t/m^3)	78,562	2.0900	4.806	2.870	0.10
Fe (%)	79,105	0.0001	68.300	12.157	1.00
Pyrite (calculated %)	79,105	0.0000	33.149	1.079	1.99
S (%)	79,098	0.0001	20.200	0.702	1.83
Magsus (SI units)	27,685	1	223,375	11,721	1.52

In the QP's opinion, the overall quality of the database is sufficient to support the estimation of Mineral Resources.

14.2.1 Excluded Holes

After discussions with Capstone, SLR excluded various twinned, geotechnical, hydrological, condemnation, and water exploration holes as listed in Table 14-4 and shown in red in Figure 14-2 from the Mineral Resource estimate.

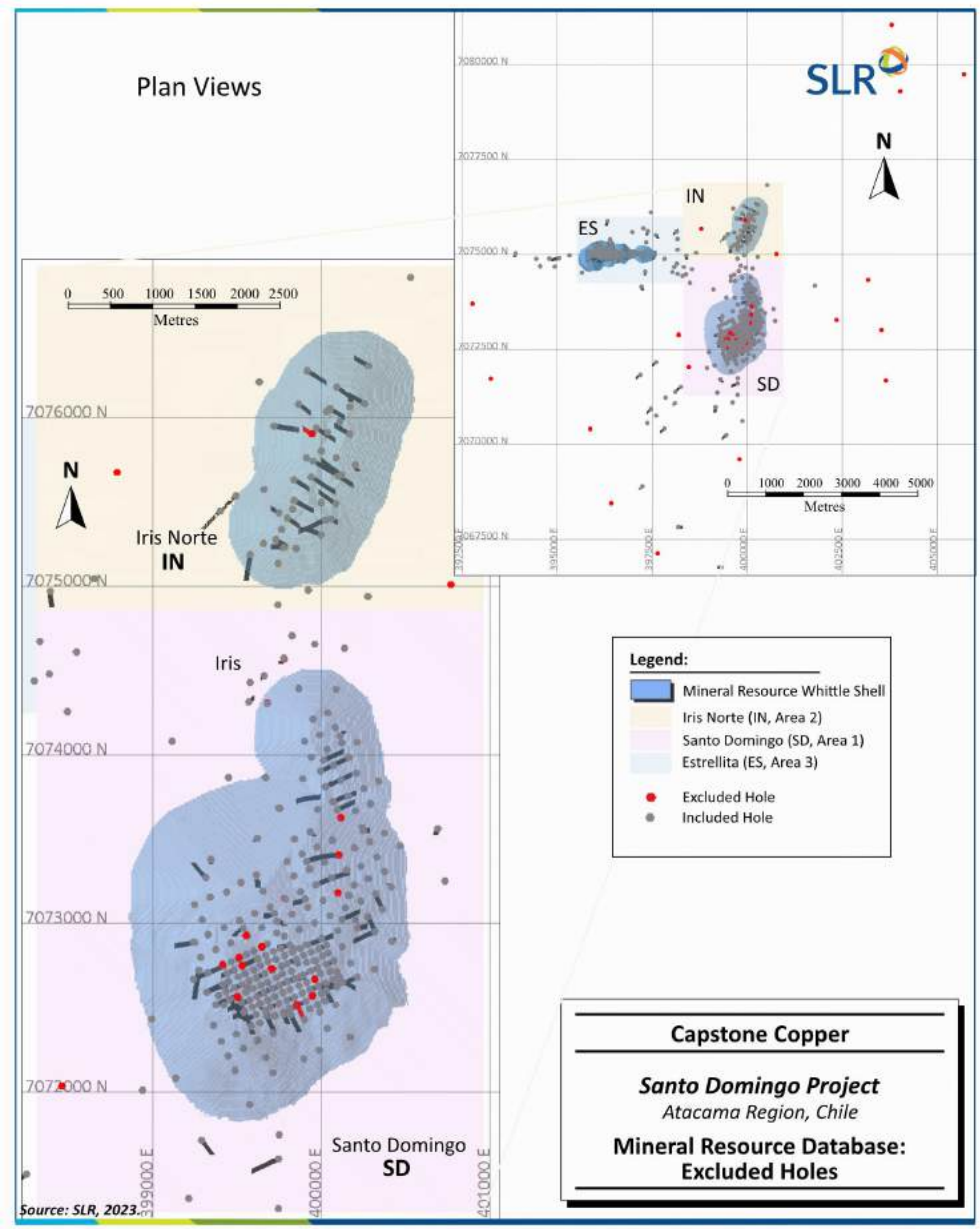
Table 14-4: Holes Excluded from the Mineral Resource Estimate

HoleID	Drill Hole Type	Purpose
4a3-06-107	RC	Exploration / Resource Definition
4a3-06-108	RC	Exploration / Resource Definition
4a3-06-130	RC	Exploration / Resource Definition
4a3-06-136	RC	Exploration / Resource Definition
4a3-06-139	RC	Exploration / Resource Definition
4a3-06-150	RC	Exploration / Resource Definition
4a3-07-262	RC	Exploration / Resource Definition
4a3-10-077DD-B	DDH	Exploration / Resource Definition
4a3-10-093-B	RC	Exploration / Resource Definition
4a3-10-099-B	RC	Exploration / Resource Definition
4a3-11-414DD	RC-DDH	Metallurgy / Resource definition
4a3-14-463DD	DDH	Metallurgy
4a3-14-464DD	DDH	Metallurgy
4a3-14-465DD	DDH	Metallurgy
4a3-14-466DD	DDH	Metallurgy
4a3-14-467DD	DDH	Metallurgy
4a3-14-468DD	DDH	Metallurgy
4a3-14-469DD	DDH	Metallurgy
4a3-15-474DD	DDH	Metallurgy
4a3-19-476DD	DDH	Metallurgy
4a3-19-477DD	DDH	Metallurgy
4a3-19-478DD	DDH	Metallurgy
4a3-19-479DD	DDH	Metallurgy

HoleID	Drill Hole Type	Purpose
4a3-19-480DD	DDH	Metallurgy
4a3-19-481DD	DDH	Metallurgy
4a3-19-483DD	DDH	Metallurgy
4a3-19-486DD	DDH	Metallurgy
4a3-19-487DD	DDH	Metallurgy
4a3-21-508DD	RC/DDH	Geomet samples
4a3-21-509DD	RC/DDH	Geomet samples
4a3-21-511DD	RC/DDH	Geomet samples
4a3-21-515DD	RC/DDH	Geomet samples
4a3-21-528DD	DDH	Geomet samples
4a3-21-528DD-B	DDH	Geomet samples
4a3-21-534DD	DDH	Geomet samples
4a3-21-535DD	DDH	Geomet samples
4a3-21-537DD	DDH	Geomet samples
4a3-21-537DD-B	DDH	Geomet samples
BH-IN-05	DDH	Geotechnical
BH-MP-01	DDH	Geotechnical
BH-MP-02	DDH	Geotechnical
BH-MP-03	DDH	Geotechnical
BH-MP-04	DDH	Geotechnical
BH-MP-05	DDH	Geotechnical
BH-TSF-01	DDH	Geotechnical
BH-TSF-01H	DDH	Hydrologic
BH-TSF-02H	DDH	Hydrologic
BH-TSF-03H	DDH	Hydrologic
BH-TSF-04H-B	DDH	Hydrologic
CDH-01	RC	Condemnation
CDH-03	RC	Condemnation
CDH-04	RC	Condemnation
CDH-05	RC	Condemnation
CDH-06	RC	Condemnation
CDH-07	RC	Condemnation
CDH-11	RC	Condemnation
CDH-12	RC	Condemnation
CDH-13	RC	Condemnation
CDH-14	RC	Condemnation
CDH-15	RC	Condemnation
CDH-16	RC	Condemnation
CDH-17	RC	Condemnation
CDH-18	RC	Condemnation
H-07-001	RC	Water Exploration
H-07-002	RC	Water Exploration
H-07-003	RC	Water Exploration
H-07-004	RC	Water Exploration
H-08-005	RC	Water Exploration
H-08-006	RC	Water Exploration
H-08-007	RC	Water Exploration
H-08-008	RC	Water Exploration
H-08-009	RC	Water Exploration
H-08-010	RC	Water Exploration

HoleID	Drill Hole Type	Purpose
KP12-PB01	RC	Hydrologic
KP12-SG01	RC	Hydrologic
KP12-SG03	RC	Hydrologic
KP12-SG04	RC	Hydrologic
KP12-SG07	RC	Hydrologic
KP12-SG08	RC	Hydrologic
KP12-SG09	RC	Hydrologic
KP12-SH01	RC	Hydrologic
KP12-SH02	RC	Hydrologic
KP12-SH03	RC	Hydrologic
KP12-SH04	RC	Hydrologic
KP12-SH05	RC	Hydrologic
KP12-SH06	RC	Hydrologic
PB1	DDH	Hydrologic
PT-01	RC	Geotechnical
PT-02	RC	Geotechnical
PT-03	RC	Geotechnical
PT-04	RC	Geotechnical
PT-06	RC	Geotechnical
PT-07	RC	Geotechnical
PT-08	RC	Geotechnical
PT-09	RC	Geotechnical
PT-10	RC	Geotechnical
PT-12	RC	Geotechnical
PT-13	RC	Geotechnical
PT-14	RC	Geotechnical
PT-15	RC	Geotechnical
PT-16	RC	Geotechnical
PT-17	RC	Geotechnical
PT-18	RC	Geotechnical
PT-19	RC	Geotechnical
PT-20	RC	Geotechnical
PT-21	RC	Geotechnical
PT-22	RC	Geotechnical
PT-23A	RC	Geotechnical
PT-26	RC	Geotechnical

Figure 14-1: Mineral Resource Database: Excluded Holes



Source: Figure prepared by Capstone Copper, 2024. Modified from SLR, 2023.

14.2.2 Pyrite Calculation

Pyrite is not included in the Mineral Resources but is estimated for internal purposes and for exploration potential in saleable acid. Pyrite calculations were reviewed in a September 2021 presentation by Capstone (Quiñones and Jeraldo, 2021). The premise is to assume that copper is exclusively present in chalcopyrite, based on mineralogical knowledge of Santo Domingo. Chalcopyrite is then calculated based on copper grade and stoichiometric copper content in this species. As a second step, the sulphur content in chalcopyrite is deducted from the total sulphur. Finally, the pyrite content is determined, based on its stoichiometric sulphur content and the remaining sulphur.

To determine the pyrite concentration in any given sample, Capstone is continuing to use the pyrite calculation determined for the cobalt pre-feasibility study (PFS), as follows:

$$Pyrite(\%) = (ST(\%) - (CuT / Cu\ in\ cpy) * S\ in\ cpy) / S\ in\ py$$

Where:

- ST = total sulphur
- CuT = total copper
- Cu in cpy = stoichiometric copper content in chalcopyrite = 0.3463
- S in cpy = stoichiometric sulphur content in chalcopyrite = 0.3494
- S in py = stoichiometric sulphur content in pyrite = 0.5345

The formula assumes that all sulphur in the hypogene zone occurs as chalcopyrite and/or pyrite, and that all copper in the hypogene zone occurs as chalcopyrite.

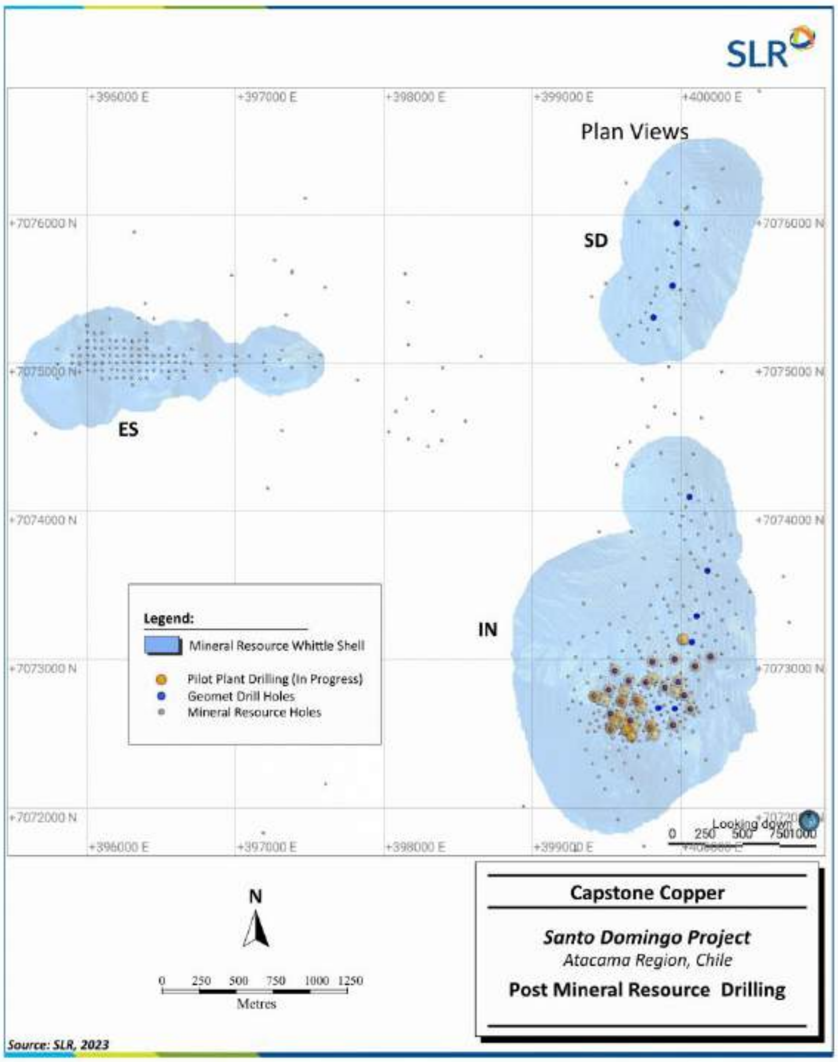
14.2.3 Post Mineral Resource Estimate Drilling

SLR investigated drilling after the Mineral Resource database cut-off date (October 7, 2021) in order to determine what effect newer holes might have on the Mineral Resource. Capstone provided Excel file exports from the master MS Access database including extant collars and surveys as of June 6th, 2023.

Previous to the Mineral Resource database cut-off date, Capstone had drilled 34 geometallurgical holes which had pending assays as of the Mineral Resource cut-off date. These holes were excluded from the Mineral Resource. The June 2023 database contains returned assays. To check for possible material changes to the Mineral Resource estimate, SLR imported the new database to Vulcan and flagged output straight composite files for lithology, class, and the Whittle pit volume. Then SLR performed comparative statistics on the 2021 database vs the 2023 database within classes 1-3 inside the Whittle pit volume. Averages for volcaniclastic and breccia Cu, Fe, Au, and S, inside the classified Whittle pit volume, were the same for Cu and Au. For volcaniclastic and breccia Fe and S, average grades with a difference of less than one percent between the two databases, occasionally ranging up to five percent in lith 205 at IN, which contains few samples relative to the Project. In general, secondary rock types (andesites and lithic tuffs) showed increases in Cu, Fe, and S average grades ranging from +0% up to +5%. Given these results, the QP does not expect that the geometallurgical drill results would significantly or materially affect the Mineral Resource estimate.

Since the Mineral Resource database cut-off date, Capstone has been engaged in a sampling program to run a pilot plant. A total of 40 holes have been drilled in the central volume of the Santo Domingo Whittle shell volume (Figure 14-2), mainly in Measured Mineral Resource and Indicated Mineral Resource material. Samples have not yet been returned for this program. Since the pilot plant sampling program at Santo Domingo is essentially twin drilling in the most densely drilled part of the Mineral Resource volume, the QP does not anticipate that the returned assays would result in a material change to the Mineral Resource.

Figure 14-2: Post Mineral Resource Drilling



Source: Figure prepared by Capstone Copper, 2024. Modified from SLR, 2023.

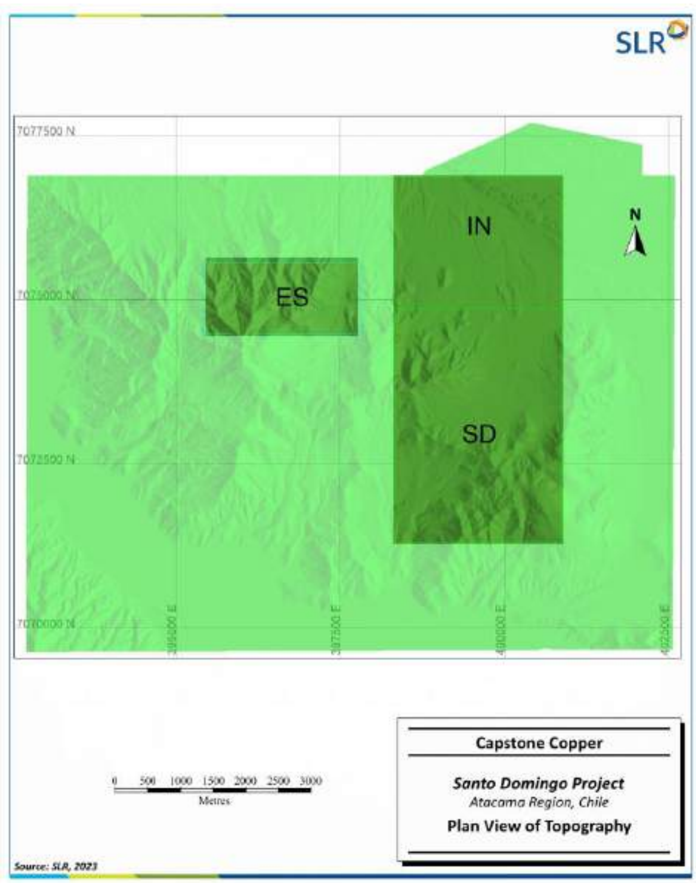
14.3 Santo Domingo and Iris Norte Mineral Resource Estimate

14.3.1 Topography

In 2012, Capstone commissioned a Light Detection and Ranging (LiDAR) and aerial photo survey of Santo Domingo and Iris Norte via Fugro fixed wing aircraft (Fugro, 2012). The topographic and photogrammetric survey was carried out with a LiDAR or Airborne Laser System.

SLR reviewed the data and found that it showed a marked improvement in resolution over the drilled areas. SLR found that the LiDAR data did not cover the extended block model area and merged the new data with the old topography data set to cover the entire model. SLR then validated the result against the previous topography to ensure that there were no unexpected errors, repaired minor artifacts on the margins resulting from the merge operation, and generated two surfaces to cover the SD-IN and ES block model extents (Figure 14-3). The QP is of the opinion that the resultant topography is of sufficient quality to support the Mineral Resource estimate.

Figure 14-3: Plan View of Topography



Source: Figure prepared by SLR, 2023.

14.3.2 Geological Interpretation

In late 2020, Capstone performed extensive reviews of the geological and geochemical information for the Project and revised the geological and oxidation models for both Santo Domingo and Iris Norte based on the consideration that the lithological units were continuous across the three previously modelled areas (Area 1, Area 2, and Area 4) of the deposit.

Two separate geological models were built in Leapfrog software (Leapfrog) using a simplified lithology table, a detailed lithology table, copper grades and magsus, revised drill hole database, and the core library. Dyke and diorite orientations and extents were also revised.

Capstone also constructed a lithogeochemical database to obtain a better understanding of the character and continuity of the modelled units. The lithogeochemistry model, which used immobile elements for the discrimination of lithological units, was compared with visual logs and the previous RPA geological model from 2012 (Rennie, 2014). Scandium (Sc), vanadium (V), phosphorus (P), aluminum (Al) and titanium (Ti), along with calcium (Ca) and iron (Fe) were used to construct plots for lithological identification. Eight different chemical units were classified (Daroch, 2020).

The results showed general agreement on the main lithological contacts previously defined by RPA, however, various changes were implemented. Although the basic stratigraphy remains the same, the work resulted in a more complex model with a greater degree of confidence attributable to each material type. The new model shows greater simplicity, continuity, and adherence to extant drill data (see Figure 14-4 and Figure 14-5). Descriptions of the major modelled lithological units are taken from the 2020 modelling report (Chávez Cortés, 2020) as follows:

Overburden groups together all types of quaternary deposits and unconsolidated rocks deposited at the top of the stratigraphic column.

Upper Andesite mainly groups rocks mapped as andesites in the area, primarily composed of amygdaloidal andesites, amygdaloidal porphyric andesites, and basaltic andesites and breccias, as well as small intercepts of tuffs, crystalline tuffs, magnetite mantos, etc., which are discontinuous in the unit. These rocks have a very representative lithogeochemical signature. At IN, this volcanic rock package has an intercalation of volcaniclastic and volcanic-sedimentary rocks that make the transition between this unit and the Lapilli Andesite Tuffs geological unit more gradual.

Lapilli Andesite Tuffs are composed of different types of volcanic and volcaniclastic rocks in the Upper Andesite that are characterized by low or no copper mineralization. This unit also has a lithogeochemical signature.

Upper Volcaniclastics host the bulk of the mineralization in the deposit and mainly consists of fine tuffs with clastic texture, crystal tuffs, tuffs with carbonates, magnetite, and specularite mantos (with tuff protoliths). As with the previous units, lithogeochemistry helps define the unit limits. This unit has the highest concentrations of copper and iron in the area, and grades clearly delineate the contact.

Lower Volcaniclastics are made up of lithic lapilli tuffs and crystal tuffs, within the volcaniclastic rock package, characterized by the absence of, or low, copper and iron mineralization. They are recognizable in the area as they have a clear lithogeochemical signature. This unit had not been modelled in the previous geological model and presents good continuity throughout the area.

Carbonates are found surrounding and interbedded with volcaniclastic units, creating an irregular boundary. The unit includes limestones, carbonate sediments, calcareous tuffs, and occasional rocks with skarn characteristics. Lithogeochemistry helps define the unit, as well as low copper grades and the absence of iron. Some intercalated and remnant volcaniclastic material at and proximal to limestone boundaries may present limited economic opportunity.

Lower Andesite shows similar characteristics to the Upper Andesite and consists of the same types of rocks defined in the Lith column of the database (amygdaloidal andesites, amygdaloidal porphyric andesites, basaltic andesites, and breccias). These rocks also have a lithogeochemical signature that allows better definition of the upper contact of the unit against volcaniclastic and carbonate units. As with the Upper Andesite, low copper grade also helps define the contact.

Diorite and Dyke intrusives were defined using a combination of variables from the database. First, intercepts mapped as intrusive in the simplified lithology table (Lith_simple) were grouped by trend, generating four dykes with different trends and a diorite that crosses the entire model. Then the dyke and diorite intercepts were filtered for similar characteristics in the detailed mapping (Lith in the database) and compared with their lithogeochemical signatures to correlate the drill sections that had the greatest amount of common characteristics. Thicker intercepts and those that were barren or very low grade were favoured. Very low magsus values helped define the contacts. These units are readily identifiable in diamond drill holes but are confused with andesites in reverse circulation drill holes. To provide better continuity, some intercepts with andesite signatures and similar geochemical characteristics as dykes have been interpreted as dykes.

Breccia is limited to a very small area to the south of the model and is mainly made up of rocks mapped as fault breccias. Breccia does not have a lithogeochemical signature that allows it to be clearly differentiated. High copper grades within and around the Breccia do not differentiate it from the rock units around it. Breccia is defined by lithological parameters in a limited area where the rock is very strongly fractured, and the protoliths are not clear because the Breccia is a mixture of its surrounding rock types. The modelling of this unit was partially based on review of the diamond drill core photographs.

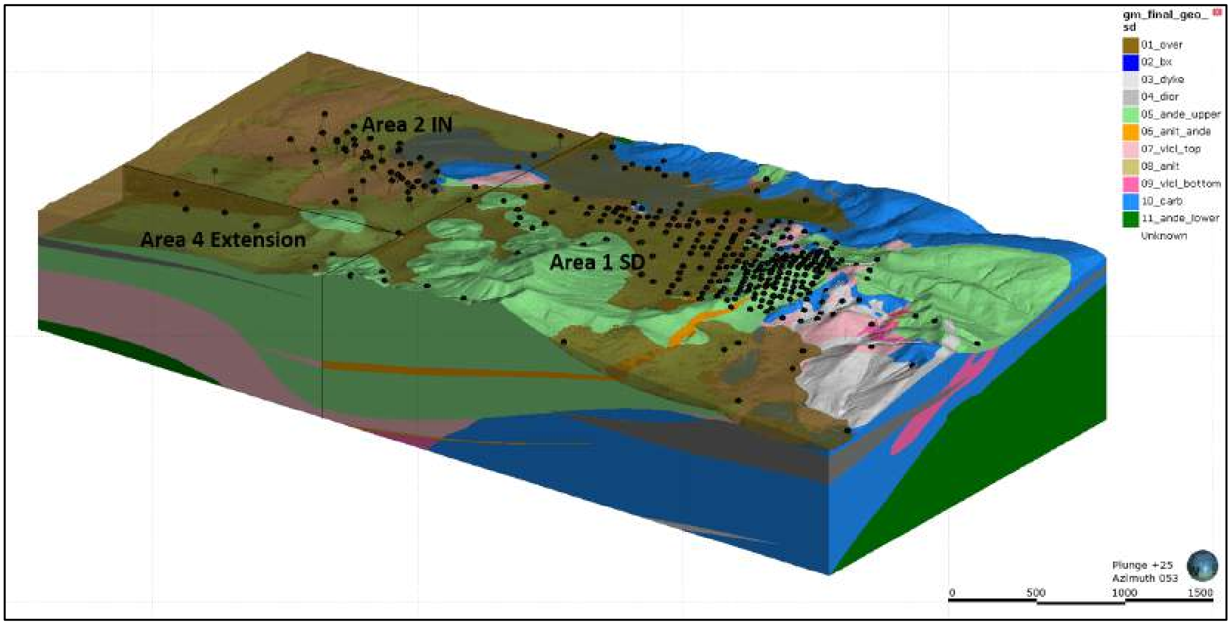
The lithology and redox models are being continuously reviewed and improved by Capstone. For the 2024 Mineral Resource estimate, the compiled SD-IN geological model consists of the SD model (Area 1), which encloses the Santo Domingo Sur and Iris areas of the previous Mineral Resource model; the smaller IN model (Area 2); and a third geological model built to fill the block model volume with lithological codes in the model's northwest quadrant (Area 4 – Estrellita is Area 3), away from the estimated material at SD and IN. During the development of the 2024 Mineral Resource model, Capstone updated the Area 2 and 4 solids to a unified IN geological model, which will be updated in future estimates. Capstone aims to produce one geological model for the entire SD-IN volume in future models.

Rock types and associated codes of the resultant Leapfrog wireframes are listed in Table 14-5. The QP is of the opinion that the new geological models represent a marked improvement to the previous model and are appropriate to support the Mineral Resource estimate.

Table 14-5: SD and IN Lithology Model

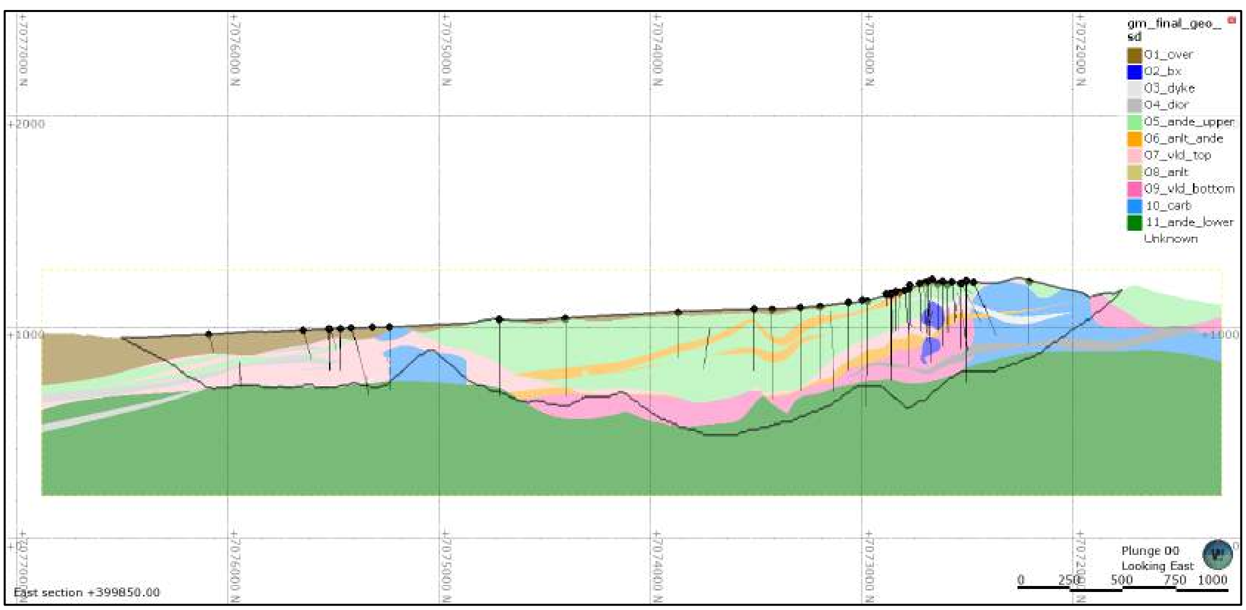
Lithology Unit	Area	Area Name	Vulcan Triangulation	Domain	Domain Code
Overburden	1	SD	lith_sd_01_over.00t	d_lith	101
Breccia	1	SD	lith_sd_02_bx.00t	d_lith	102
Dyke	1	SD	lith_sd_03_dyke.00t	d_lith	103
Diorite	1	SD	lith_sd_04_dior.00t	d_lith	104
Upper Andesite	1	SD	lith_sd_05_ande_upper.00t	d_lith	105
Lapilli Andesite Tuff	1	SD	lith_sd_06_anlt_ande.00t	d_lith	106
Upper Volcaniclastic	1	SD	lith_sd_07_vlcl_top.00t	d_lith	107
Lapilli Tuff	1	SD	lith_sd_08_anlt.00t	d_lith	108
Lower Volcaniclastic	1	SD	lith_sd_09_vlcl_bottom.00t	d_lith	109
Carbonate	1	SD	lith_sd_10_carb.00t	d_lith	110
Lower Andesite	1	SD	lith_sd_11_ande_lower.00t	d_lith	111
Overburden	2	IN	lith_in_01_over.00t	d_lith	201
Dyke	2	IN	lith_in_03_dyke.00t	d_lith	203
Diorite	2	IN	lith_in_04_dior.00t	d_lith	204
Upper Andesite	2	IN	lith_in_05_ande_upper.00t	d_lith	205
Volcaniclastic	2	IN	lith_in_07_vlcl.00t	d_lith	207
Carbonate	2	IN	lith_in_10_carb.00t	d_lith	210
Lower Andesite	2	IN	lith_in_11_ande_lower.00t	d_lith	211
Overburden	4	IN	newblock_01_over.00t	d_lith	401
Diorite	4	IN	newblock_04_dior.00t	d_lith	404
Upper Andesite	4	IN	newblock_05_ande_upper.00t	d_lith	405
Volcaniclastic	4	IN	newblock_07_vlcl.00t	d_lith	407
Lower Andesite	4	IN	newblock_11_ande_lower.00t	d_lith	411

Figure 14-4: SD and IN Geology: Oblique View Looking Northeast



Source: Figure prepared by SLR, 2023.

Figure 14-5: SD and IN Geology: Section 399,850E Looking East



Source: Figure prepared by Capstone Copper, 2024.

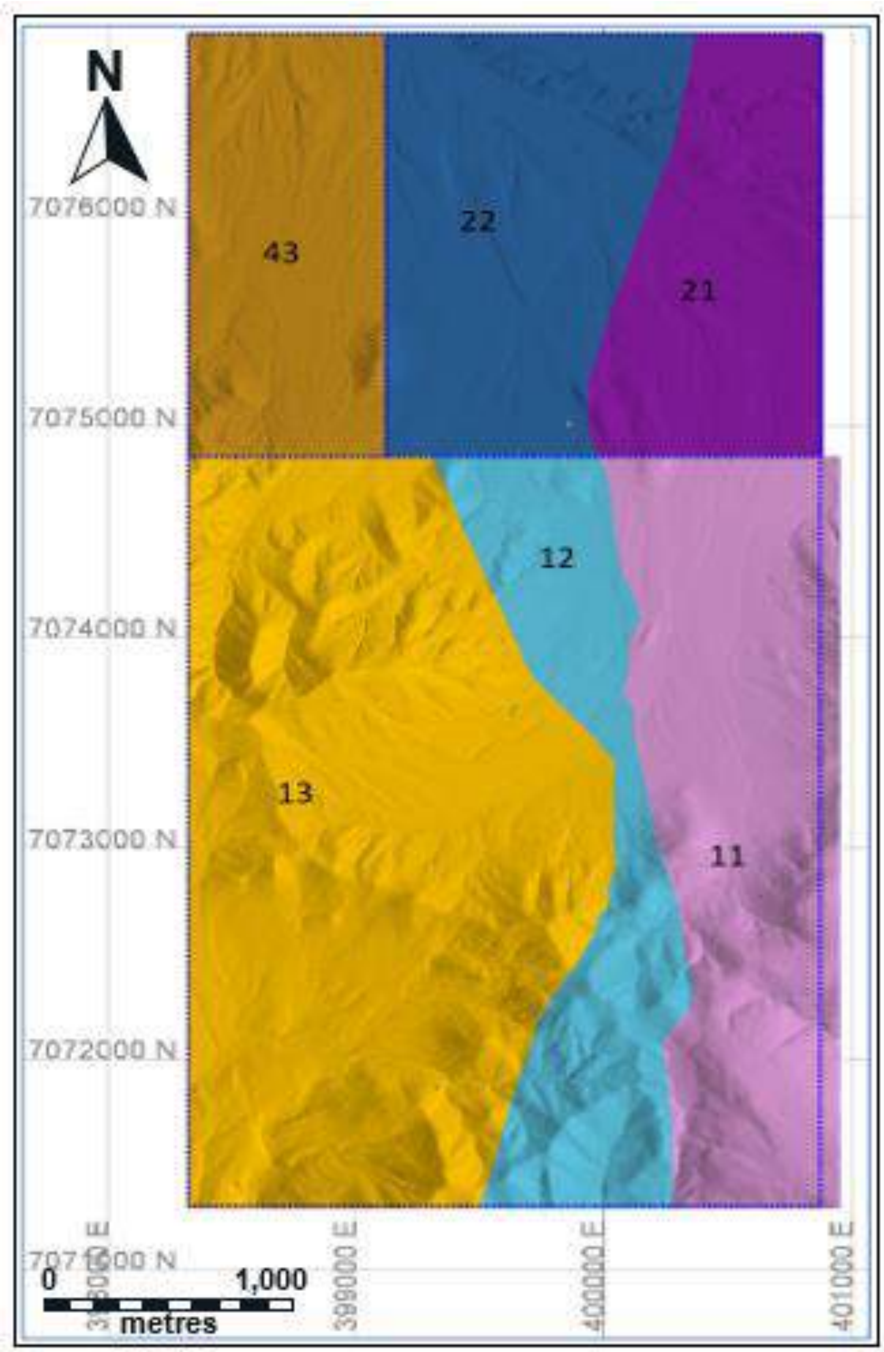
14.3.2.1 Structure

For the purposes of the mineral resource, Capstone used simplified structural fault blocks to model the structural geology of the SD-IN volume. Unlike the previous models, where many faults were used to explain the contacts between the various geological units, the new geological model includes only two major faults. This simplification became possible due to the conclusion that limestone is intercalated with the other lithologies. The two major faults were created from drill hole information. Both faults are high angle normal faults (between 70° and 80° to the west) and pass through the model with a predominant north-south orientation.

In Iris Norte (Area 2), only one main structure was generated that crosses the entire model. The fault was modelled from drilling data and faults previously interpreted in the area and correlates with the easternmost normal high angle fault (approximately 75° to the west) extension located in Santo Domingo with only a few metres of discontinuity between the geological models.

SLR noted that, overall, there is little apparent displacement across fault boundaries at SD-IN. SLR therefore chose to interpolate across the boundaries to make the size and complexity of the new estimate more manageable. Fault blocks were coded in the model with a two-digit code representing a concatenation of d_area and d_fb domain names and are shown in Figure 14-6.

Figure 14-6: SD and IN Fault Blocks (Plan View)



Source: Figure prepared by SLR, 2023.

14.3.2.2 Oxidation

Wireframe models were created to enclose oxidized material, which has been demonstrated to yield much lower metallurgical recoveries than unoxidized mineralization. Oxidation boundaries are still somewhat uncertain due to a lack of complete data for leachable copper. The primary criterion for defining the base of the oxidized zone is the presence of significant quantities of leachable copper or strong oxidation, noted in the logs. For the 2024 Mineral Resource estimate, the oxide model was not used to constrain grade interpolation as SLR determined that lithology was a much stronger control on mineralization. However, oxidation model solids were used to flag material within the resource volume, so that this material could be excluded from the Mineral Resource estimate or otherwise subjected to differential estimation parameters or economic criteria.

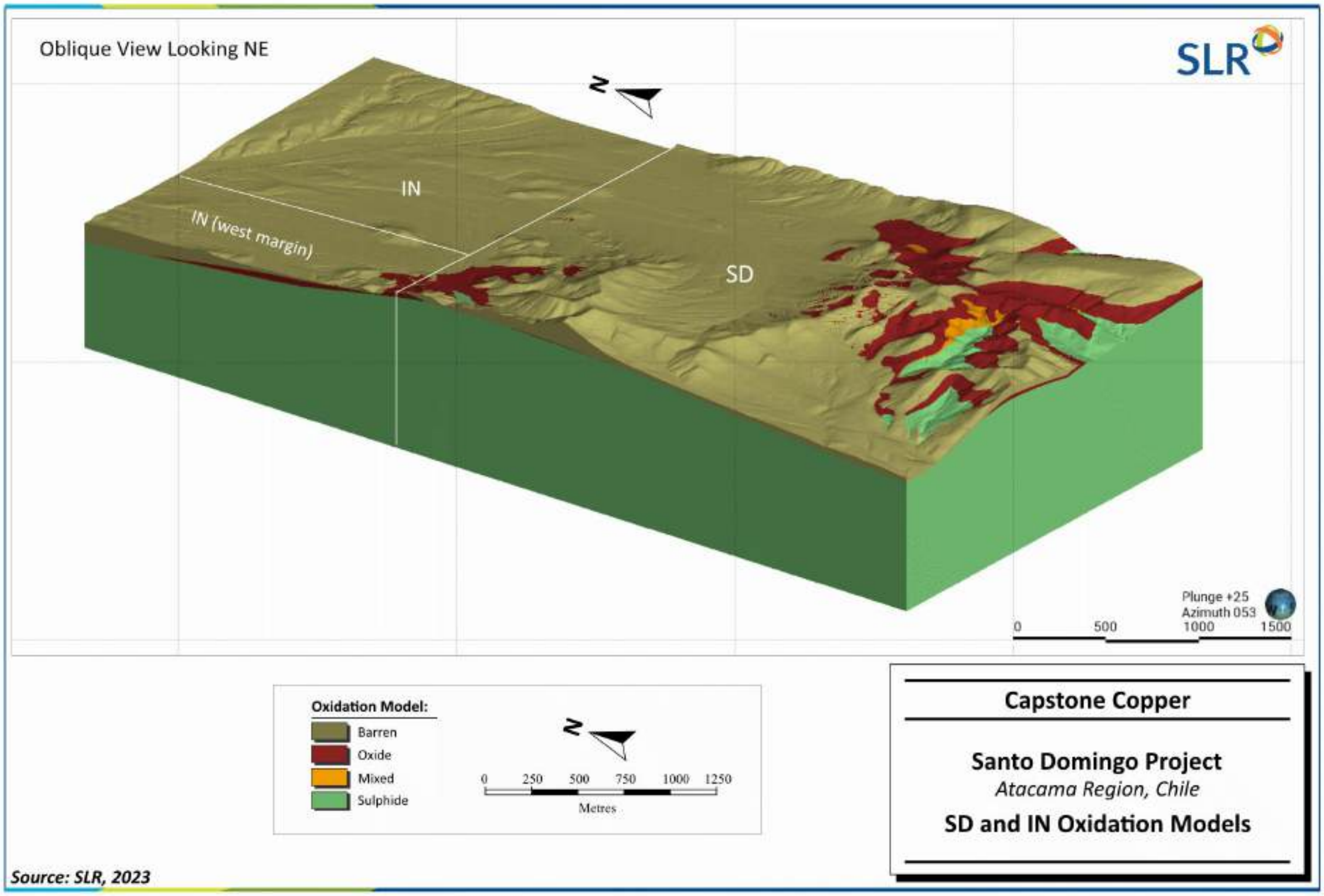
Since the oxide model interpretation for Iris Norte also did not cover the northwest quadrant of a unified model space (similar to rock type), Capstone filled the sparsely drilled northwest quadrant with additional solids for oxidation state in order to cover the block model extent. The oxidation model codes are presented in Table 14-6 and the models are shown in Figure 14-7.

The QP is of the opinion that the improved geological and oxidation models are a substantial improvement over previous models and are of sufficient quality to support a Mineral Resource estimate. The three separate geological models show good continuity at the margins where they about each other. The QP notes that creating a joint geological model for the three areas would result in an extraordinarily large Leapfrog project, but nonetheless recommends that the geological model be unified into one areal extent.

Table 14-6: SD and IN Oxidation Model

Code	Description	Triangulation
1	Barren	redox_1_barren.00t
2	Oxide	redox_2_cuox.00t
3	Mixed	redox_3_mixture.00t
4	Sulphide	redox_4_sulphides.00t

Figure 14-7: SD-IN Oxidation Model



Source: Figure prepared by SLR, 2023.

14.3.3 Resource Assays

SLR performed exploratory data analysis (EDA) for each estimation domain based on the lithological domains. This included univariate statistics, histograms, cumulative probability plots, box plots to compare geology domain statistics, visual review of assay grades against lithology and oxidation solids and contact plots to investigate grade profiles between estimation domains and to determine the extent of sample sharing across the geology contacts within the rock type domains.

SLR generated box plots for each element split on lithology, magsus, oxidation, fault block, and a litho-oxidation combination, and determined that domaining based on lithology most closely represented spatial changes in grade. Hard boundaries were determined for each of the estimation variables, except for the Breccia, which is a mixture of all host rock protoliths. A summary of uncomposited assay statistics within each lithological unit is provided in Table 14-7.

In general, coefficients of variation (CV) are low to moderate (0.5 to 2.0) for gold, cobalt, copper, and iron, except in Carbonate which reflects the intercalation of rock types at its margins.

The QP is of the opinion that the uniformity and density of the drill coverage is suitable to support a Mineral Resource Estimate.

Table 14-7: SD-IN Resource Assays: Summary Statistics

Element	Units	Area	Lith	Lith_Txt	Count	Uncapped			
						Min	Max	Mean	CV
Au	g/t	SD	102	Breccia	636	0.003	0.360	0.060	0.78
Au	g/t	SD	105	ANDE Upper	6,823	0.003	2.380	0.060	1.14
Au	g/t	SD	106	ANLT ANDE	696	0.003	0.390	0.020	1.64
Au	g/t	SD	107	VLCL Upper	6,303	0.003	4.710	0.080	1.33
Au	g/t	SD	108	ANLT	2,018	0.003	0.660	0.030	1.73
Au	g/t	SD	109	VLCL Lower	6,823	0.003	2.380	0.060	1.14
Au	g/t	SD	110	CARB	1,406	0.003	2.940	0.020	4.76
Au	g/t	SD	111	ANDE Lower	2,677	0.003	0.450	0.020	1.38
Au	g/t	IN	205	ANDE Upper	440	0.005	0.210	0.010	2.10
Au	g/t	IN	207	VLCL	1,277	0.003	0.980	0.040	1.35
Au	g/t	IN	210	CARB	137	0.003	0.060	0.010	0.72
Au	g/t	IN	211	ANDE Lower	181	0.005	0.300	0.010	2.61
Co	ppm	SD	105	ANDE Upper	8,456	1.0	2,860.0	261.6	0.93
Co	ppm	SD	106	ANLT ANDE	1,537	1.0	1,015.0	53.4	1.41
Co	ppm	SD	107	VLCL Upper	4,824	1.0	4,820.0	299.7	0.97
Co	ppm	SD	108	ANLT	2,942	1.0	2,940.0	101.3	1.63
Co	ppm	SD	109	VLCL Lower	8,456	1.0	2,860.0	261.6	0.93
Co	ppm	SD	111	ANDE Lower	4,585	1.0	3,080.0	82.8	1.26
Co	ppm	IN	205	ANDE Upper	666	1.0	1,180.0	65.2	1.60
Co	ppm	IN	207	VLCL	2,963	0.8	2,730.0	170.4	1.24
Co	ppm	IN	211	ANDE Lower	222	1.0	696.0	66.6	1.19
Cu	%	SD	102	Breccia	825	0.000	3.710	0.340	1.08
Cu	%	SD	105	ANDE Upper	12,875	0.000	3.020	0.080	2.00
Cu	%	SD	106	ANLT ANDE	1,572	0.000	2.150	0.080	2.21
Cu	%	SD	107	VLCL Upper	7,861	0.000	6.380	0.470	1.13
Cu	%	SD	108	ANLT	2,955	0.000	5.380	0.190	1.86
Cu	%	SD	109	VLCL Lower	8,490	0.000	5.340	0.330	1.39
Cu	%	SD	110	CARB	3,560	0.000	3.590	0.080	2.47
Cu	%	SD	111	ANDE Lower	4,589	0.000	4.460	0.080	2.41
Cu	%	IN	205	ANDE Upper	665	0.000	1.720	0.070	2.02
Cu	%	IN	207	VLCL	2,952	0.000	3.100	0.120	2.22
Cu	%	IN	210	CARB	305	0.001	1.270	0.030	3.30
Cu	%	IN	211	ANDE Lower	220	0.000	2.090	0.060	2.87
Fe	%	SD	102	Breccia	812	2.390	55.100	21.070	0.54

Element	Units	Area	Lith	Lith_Txt	Count	Uncapped			
						Min	Max	Mean	CV
Fe	%	SD	105	ANDE Upper	12,692	0.010	55.800	11.360	0.45
Fe	%	SD	106	ANLT ANDE	1,537	2.950	52.800	10.910	0.4
Fe	%	SD	107	VLCL Upper	7,824	0.460	65.100	30.980	0.43
Fe	%	SD	108	ANLT	2,942	0.490	68.300	15.350	0.74
Fe	%	SD	109	VLCL Lower	8,456	0.440	65.300	25.790	0.51
Fe	%	SD	110	CARB	3,543	0.010	50.100	5.110	1.27
Fe	%	SD	111	ANDE Lower	4,585	1.160	61.700	13.900	0.54
Fe	%	IN	205	ANDE Upper	666	0.010	45.200	9.010	0.53
Fe	%	IN	207	VLCL	2,963	0.010	62.500	21.010	0.68
Fe	%	IN	210	CARB	305	0.450	10.350	3.190	0.62
Fe	%	IN	211	ANDE Lower	222	2.020	23.100	9.760	0.29
Magsus	SI	SD	102	Breccia	808	8	68,948	8,268	1.26
Magsus	SI	SD	105	ANDE Upper	5,908	3	101,625	6,551	1.11
Magsus	SI	SD	106	ANLT ANDE	746	87	68,908	4,796	1.14
Magsus	SI	SD	107	VLCL Upper	6,616	2	150,240	15,576	1.43
Magsus	SI	SD	108	ANLT	2,072	1	165,537	8,365	2.14
Magsus	SI	SD	109	VLCL Lower	6,550	1	223,375	15,372	1.38
Magsus	SI	SD	110	CARB	1,027	2	60,527	352	7.35
Magsus	SI	SD	111	ANDE Lower	2,228	14	110,000	15,226	0.9
Magsus	SI	IN	205	ANDE Upper	202	1	62,768	3,283	2.75
Magsus	SI	IN	207	VLCL	1,892	2	126,867	19,532	1.06
Magsus	SI	IN	210	CARB	166	7	2,960	197	2.76
Magsus	SI	IN	211	ANDE Lower	55	697	14,505	6,411	0.54
S	%	SD	102	Breccia	812	0.005	10.000	1.160	1.12
S	%	SD	105	ANDE Upper	12,692	0.005	10.000	0.300	0.30
S	%	SD	106	ANLT ANDE	1,537	0.005	10.100	0.220	3.22
S	%	SD	107	VLCL Upper	7,821	0.005	20.200	1.960	0.92
S	%	SD	108	ANLT	2,940	0.005	10.000	0.910	1.37
S	%	SD	109	VLCL Lower	8,452	0.005	12.850	2.320	0.77
S	%	SD	110	CARB	3,543	0.005	9.230	0.790	1.05
S	%	SD	111	ANDE Lower	4,585	0.005	6.850	0.460	1.50
S	%	IN	205	ANDE Upper	666	0.005	5.360	0.220	2.82
S	%	IN	207	VLCL	2,963	0.005	13.700	1.500	1.23
S	%	IN	210	CARB	305	0.005	5.310	0.590	1.13
S	%	IN	211	ANDE Lower	222	0.010	4.170	0.520	1.36

Note: ANDE– Andesite (Upper and Lower); ANLT ANDE – Andesitic lithic tuff; VLCL – Volcaniclastic (Upper and Lower), CARB – Carbonate.

14.3.3.1 Unsampled Intervals

SLR reviewed unsampled intervals both in Leapfrog and in the database, compiling lists of holes and intervals to exclude from the Mineral Resource estimate. Initially, all unsampled intervals were set to 0.0001 for each element. Three different types of unsampled data were then identified and set to ignore using a flag field.

1. Long unsampled intervals from collars into oxide material and Upper Andesite, which were generally ignored.
2. Unsampled intervals which were well supported by adjacent sampling, which were ignored.
3. 10 cm to 30 cm unsampled intervals which were sent for density analyses, which were ignored.

SLR built scripts to write flags to the composite databases to treat these intervals appropriately. Separate exclusion lists were built for gold, copper, cobalt, and iron. An example table of copper intervals excluded, of the types (1) and (2) above, is provided in Table 14-8.

Table 14-8: Unsampled Interval Exclusions: Cu Example

HoleID	From (m)	To (m)	Notes
4a3-05-049	0.00	200.00	unsampled top with good part sampled below
4a3-06-076DD-B	0.00	199.80	ignore unsampled ANDE top
4a3-06-078DD	0.00	50.00	unsampled top in ANDE
4a3-06-079DD	0.00	168.00	ignore unsampled top in ANDE
4a3-06-080DD	0.00	100.00	ignore unsampled top
4a3-06-095	12.00	70.00	ignore unsampled ANDE
4a3-06-097	0.00	96.00	ignore unsampled ANDE
4a3-06-104	0.00	26.00	ignore upper ANDE
4a3-06-113	0.00	52.00	ignore completely unsampled
4a3-06-115	72.00	138.00	neighbouring holes show grade in ANDE
4a3-06-124	0.00	62.00	nullifies VLCL material
4a3-06-130	24.00	208.00	ignore unsampled in ANDE
4a3-06-133	0.00	140.00	ignore unsampled ANDE
4a3-06-140	0.00	70.00	ignore unsampled ANDE
4a3-06-140	240.00	254.00	ignore unsampled ANDE; ok to use low grade from 06-161DD
4a3-06-142	0.00	80.00	ignore unsampled in ANDE
4a3-06-144	30.00	80.00	ignore. Next door hole is low grade 06-127
4a3-06-147	0.00	100.00	ignore unsampled in ANDE
4a3-06-149	0.00	120.00	ignore unsampled in ANDE
4a3-06-217	0.00	52.00	ignore unsampled top in ANDE
4a3-06-218	0.00	90.00	ignore unsampled top in ANDE twinned too
4a3-07-254	0.00	74.00	ignore unsampled top in ANDE
4a3-07-258	0.00	156.00	ignore unsampled ANDE ANLT
4a3-07-261	0.00	80.00	ignore unsampled top in ANDE
4a3-07-262-B	0.00	278.00	ignore unsampled ANDE
4a3-07-266	0.00	90.00	ignore unsampled ANDE
4a3-07-267	0.00	122.00	ignore unsampled ANDE
4a3-07-269	0.00	12.00	ignore unsampled ANDE
4a3-07-271	0.00	44.00	ignore unsampled ANDE

HoleID	From (m)	To (m)	Notes
4a3-09-370DD	0.00	152.00	ignore unsampled top in ANDE
4a3-09-372DD	0.00	154.00	ignore unsampled top in ANDE
4a3-10-218-B	0.00	2.00	ignore unsampled ANDE
4a3-10-376DD	84.00	138.00	ignore: surrounded by good grade
4a3-10-379DD	0.00	10.00	ignore: right next to good grade in ANDE
4a3-11-400DD	70.00	106.00	ignore as unsampled relative to neighbouring holes
4a3-11-411DD	0.00	162.00	ignore unsampled top in ANDE
4a3-11-413DD	0.00	144.00	ignore unsampled top in ANDE
4a3-11-414DD	0.00	156.00	ignore unsampled top in ANDE
4a3-11-420DD	0.00	154.00	ignore unsampled top in ANDE
4a3-11-422DD	0.00	140.00	ignore unsampled top in ANDE
4a3-11-423DD	0.00	140.00	ignore unsampled top in ANDE
4a3-11-424DD	0.00	110.00	ignore unsampled top in ANDE
4a3-11-426DD	0.00	150.00	ignore unsampled top in ANDE
4a3-11-427DD	0.00	102.00	ignore unsampled top in ANDE
4a3-12-434DD	0.00	75.75	ignore unsampled top
4a3-12-448DD	0.00	138.00	ignore unsampled top in ANDE
4a3-12-457DD	0.00	150.00	ignore unsampled top in ANDE
4a3-14-463DD	0.00	144.00	ignore unsampled top in ANDE
4a3-14-464DD	0.00	164.00	ignore unsampled top in ANDE
4a3-14-465DD	0.00	152.00	ignore unsampled top in ANDE
4a3-14-466DD	0.00	161.80	ignore unsampled top in ANDE
4a3-14-467DD	0.00	139.80	ignore unsampled top
4a3-14-467DD	196.74	247.95	ignore as neighbouring holes have decent assays in ANLT
4a3-14-468DD	0.00	32.00	unsampled top in ANDE
4a3-14-469BDD	0.00	201.20	most of hole is unsampled: remove?
4a3-14-470DD	0.00	102.00	unsampled top in ANDE
4a3-14-471DD	0.00	82.00	unsampled top in ANDE
4a3-14-472DD	0.00	66.00	unsampled top in ANDE
4a3-15-473DD	0.00	99.85	ignore unsampled ANDE
4a3-15-474DD	0.00	158.00	ignore unsampled top in ANDE
4a3-15-475DD	0.00	91.90	ignore unsampled top in ANDE
4a3-19-476DD	0.00	162.00	ignore unsampled top in ANDE
4a3-19-477DD	0.00	158.00	ignore unsampled top in ANDE
4a3-19-478DD	0.00	152.00	ignore unsampled top in ANDE
4a3-19-479DD	0.00	164.00	ignore unsampled top in ANDE
4a3-19-480DD	0.00	144.00	ignore unsampled top in ANDE
4a3-19-481DD	0.00	92.00	ignore unsampled top in ANDE

HoleID	From (m)	To (m)	Notes
4a3-19-482DD	0.00	32.00	unsampled top in ANDE
4a3-19-483DD	0.00	140.00	unsampled top in ANDE
4a3-19-484DD	0.00	66.00	unsampled top in ANDE
4a3-19-485DD	0.00	90.00	unsampled top in ANDE
4a3-19-486DD	0.00	82.00	unsampled top
4a3-19-487DD	0.00	102.00	ignore as in ANDE and bx
4a3-19-488DD	0.00	100.00	ignore unsampled ANDE
4a3-21-489DD	0.00	300.00	ignore unsampled hole
4a3-21-490DD	0.00	200.00	ignore unsampled ANDE
4a3-21-492DD	0.00	350.00	ignore unsampled ANDE
4a3-21-505DD	0.00	220.00	ignore unsampled ANDE
GD-5	0.00	155.00	ignore unsampled in ANDE
KP12-SH05	0.00	102.00	ignore entire hole not sampled

14.3.3.2 Twinned Holes

SLR reviewed twinned holes in Leapfrog and in the drill database and excluded redundant holes or portions of redundant holes to simplify the grade estimate and avoid unwanted artifacts. Unsampled "stems" and twins were excluded from the Mineral Resource database. Adjacent twinned hole intervals with non-overlapping vertical coverage were merged into one alternative holeID used for the purpose of grade interpolation.

Excluded twin holes and intervals are listed in Table 14-9.

SLR performed analyses on three twinned holes (Figure 14-8) and determined that there are three types of drill holes in the same location:

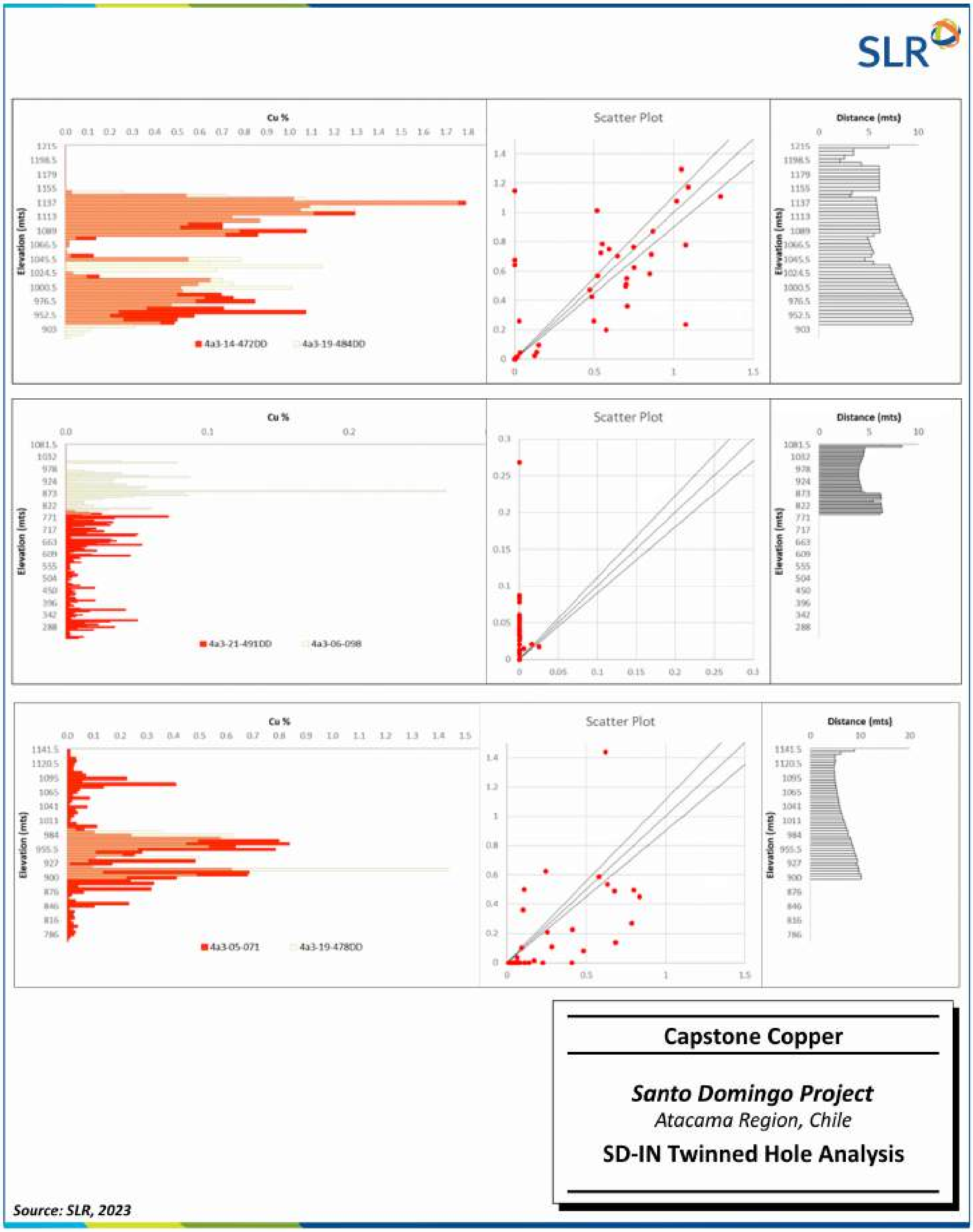
1. Twinned
2. Re-drilled
3. Partially sampled in high grade intervals

Early reviews showed that the drill holes located 0 to 10 m from each other locally had high variability for copper. Other elements showed less variability. The QP recommends further analysis of the twinned holes.

Table 14-9: Twinned Holes and Intervals Excluded from the Mineral Resource Estimate

HOLE ID	FROM	TO
4a3-07-262-B	entire drillhole	
4a3-06-080DD	0	360
4a3-14-472DD	entire drillhole	
4a3-19-487DD	entire drillhole	
4a3-14-469DD	entire drillhole	
4a3-19-485DD	entire drillhole	
4a3-14-468DD	entire drillhole	
4a3-19-486DD	entire drillhole	
4a3-06-106	entire drillhole	
4a3-10-077DD-B	entire drillhole	
4a3-19-483DD	entire drillhole	
4a3-14-467DD	entire drillhole	
4a3-14-463DD	entire drillhole	
4a3-19-480DD	entire drillhole	
4a3-15-474DD	entire drillhole	
4a3-15-475DD	0	91.26
4a3-11-414DD	entire drillhole	
4a3-19-477DD	entire drillhole	
4a3-14-465DD	entire drillhole	
4a3-19-478DD	entire drillhole	
4a3-19-479DD	entire drillhole	
4a3-14-466DD	entire drillhole	
4a3-19-476DD	entire drillhole	
4a3-10-093-B	entire drillhole	
4a3-19-481DD	entire drillhole	
4a3-10-099-B	entire drillhole	
4a3-10-218-B	82.3	350
4a3-06-218	0	84.36
4a3-07-259	300	350
4a3-21-489DD	0	300
4a3-21-492DD	0	350.67
4a3-21-490DD	0	197.72
4a3-06-139	entire drillhole	
4a3-21-491DD	0	274.36
4a3-06-130	entire drillhole	
4a3-06-150	entire drillhole	
4a3-14-464DD	entire drillhole	
4a3-19-481DD	entire drillhole	
4a3-19-479DD	entire drillhole	

Figure 14-8: SD-IN Twinned Hole Analysis



Source: Figure prepared by SLR, 2023.

For this model update, Capstone geologists have created a new magsus model using a grade shell in Leapfrog based on samples composited to 20 m length and using a threshold grade of 15,000 SI units. The magsus model represents a proxy for mass recovery and magnetite content.

SLR interpolated magsus into the Mineral Resource block model from regular measurements in drill core. SLR reviewed the magsus model against the interpolation results and found that changes in Cu and Fe grades, and magsus values, correlated well with the magsus solid. SLR thus concluded that it was not necessary to double the number of grade estimation domains by splitting them on the magsus model, though the model remains a useful guide to adhere to interpolation results.

14.3.3.3 Iron Mass Recovery Estimate

The iron mineralization at SD-IN is dominantly magnetite, which can be recovered by low intensity magnetic separation (LIMS) methods, and hematite, which generally cannot be recovered by LIMS. Consequently, the assays for total iron collected to date do not provide a basis for estimation of the recoverable iron component.

In 2008, a bulk sample was collected by blending drill cuttings from a number of SD and IN drill holes. This sample was subject to bulk flotation to remove the sulphide components, and the tailings from this process were subject to iron recovery testing. The results of the test work indicated that LIMS would produce a good quality magnetite iron concentrate.

Davis Tube test (DTT) work was performed to first determine if a saleable iron concentrate could be produced and to calibrate the expected mass recovery to magnetic susceptibility. A very strong linear relationship was found to exist between the magnetic susceptibility and both Satmagan and DTT mass recovery readings. Capstone subsequently initiated an expanded testing program in order to confirm the observed relationship and develop a reliable regression line equation for relating magnetic susceptibility to mass recovery for the previous Mineral Resource estimate.

Magnetic susceptibility readings were used to estimate the proportion of the mass of each block that could be recovered by LIMS methods. SLR performed a review of the available DTT work to incorporate new data into the regression analysis for massrec, as well as discarding 11 duplicate samples and one with no magsus data. Approximately 5% of the samples returned 0.0 DTT and showed correspondingly low magsus values. There is good coverage of the SD and IN pit areas with the DTT dataset. SLR's work resulted in two updated curves for SD and IN, which closely resemble those produced for the previous Mineral Resource (Figure 14-9). The curves were derived by sorting the data in order and then binning in groups of ten and using the mean of each bin.

For this model update, SLR noted that at higher magsus values the curves trend downward and may be negatively biased with respect to mass recovery. SLR adjusted the curves so that they reached the maximum value, and then retained that value. The new adjusted curves still utilize the formulae shown below which reach a sill beyond which the massrec remains constant (Figure 14-10).

SD DT data contains 1,096 samples, resulting in the relationship:

- If magsus is less than 124,500:

$$Massrec = (-0.0054 \times (magsus/1000)^{2} + (1.3444 \times magsus/1000))$$

- otherwise:

$$Massrec = 83.67645$$

IN DT data contains 228 samples, resulting in the relationship:

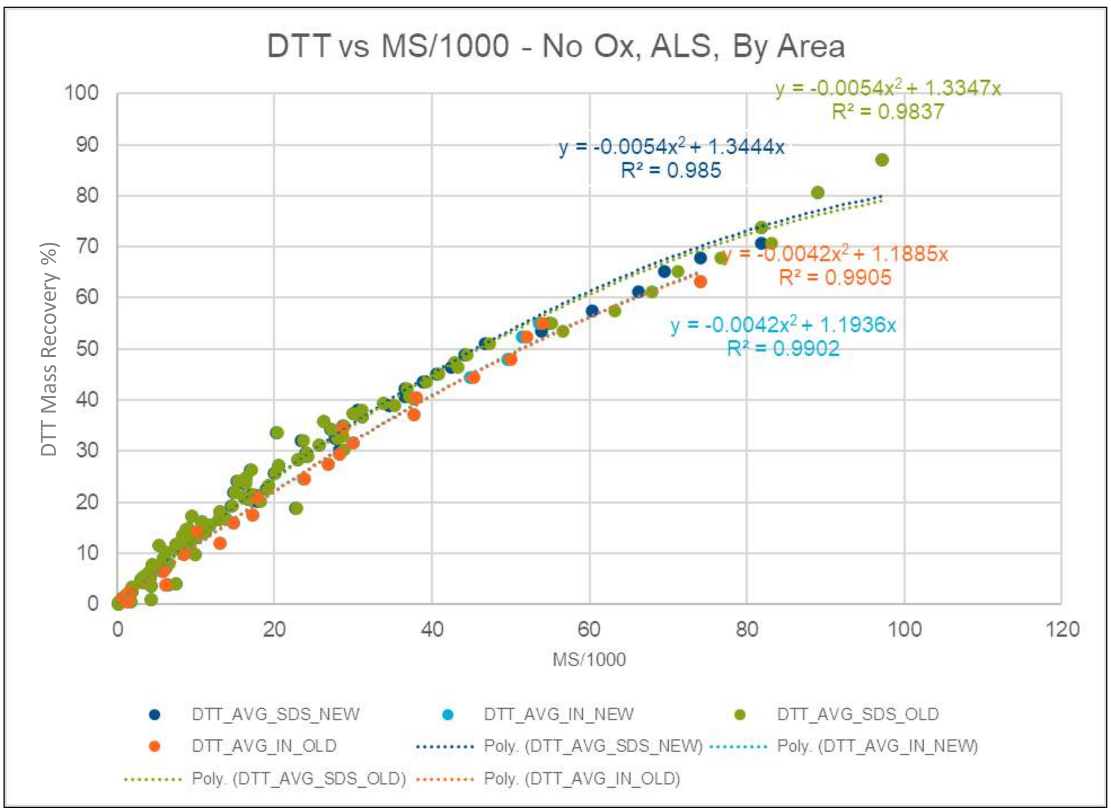
- If magsus is less than 142,150:

$$Massrec = (-0.0042 \times (magsus/1000)^{2} + (1.1936 \times magsus/1000))$$

- otherwise:

$$Massrec = 81.99070$$

Figure 14-9: Davis Tube Test Regression Analysis for SD and IN



Source: Figure prepared by SLR, 2023.

Figure 14-10: Mass Recovery Final Regression Curves

Source: Figure prepared by SLR, 2023.

Sample statistics for massrec are shown in Table 14-11.

Table 14-10: SD-IN Sample Statistics: MassRec

Element	Units	Domain	Count	Min	Max	Mean	CV
MassRec	%	102	709	0.011	67.0	10.1	1.17
MassRec	%	105	5,637	0.004	80.9	8.3	1.00
MassRec	%	106	677	0.085	67.0	6.3	1.00
MassRec	%	107	6,109	0.003	83.7	17.0	1.23
MassRec	%	108	1,764	0.001	83.4	8.0	1.93
MassRec	%	109	6,276	0.003	83.7	16.9	1.19
MassRec	%	110	907	0.003	17.8	0.2	6.00
MassRec	%	111	2,078	0.019	82.5	18.1	0.75
MassRec	%	205	202	0.001	58.4	3.5	2.57
MassRec	%	207	1,892	0.002	83.8	19.9	0.97
MassRec	%	210	166	0.008	3.5	0.2	2.76
MassRec	%	211	55	0.830	16.4	7.4	0.53

Capstone reviewed massrec relative to magsus measurements and determined that a magsus cut-off grade of 18,500 magsus units should set mass recovery to zero in lapilli tuffs, upper and lower andesite material, and carbonate (lith codes 105, 106, 108, 110, 111, 205, 210, 211), and that a cut-off grade of 1,500 magsus units should set mass recovery to zero in breccias and volcaniclastics (lith codes 102, 107, 109, and 207). These mass recoveries were set in a post-estimation block model script.

14.3.3.4 Iron Grade in Concentrate

For this model update, Capstone reviewed the DTT data, carried out exploratory geometallurgical data analysis (EDA) to identify the variables that impact the expected Fe Concentrate grade (FeDTT) obtained in the Davis tube test. Based on the EDA, a machine learning exercise was performed to model the FeDTT based on 15 separate lithogeochemical factors, including other variables from the Davis Tube test results, Fe grade, rock type, magnetic susceptibility, and other elements such as Si, Al, and Na. Three regression models were generated, grouped by geological unit. Once FeDTT was populated in the output drillhole data, Capstone found that the distribution of returned FeDTT values was similar to the laboratory FeDTT.

SLR obtained the database of 27,530 FeDTT samples modelled from 1,275 DT training/evaluation tests with the DTT and other valid factors present to complete the modelled values. FeDTT was estimated in the sub-blocked model without density weighting using five separate estimation passes. Values were very lightly capped. This enabled the SLR to utilize the variable FeDTT block grades in NSR value calculations instead of imposing a fixed 65% grade per the previous model. Sample statistics for FeDTT are shown in Table 14-11.

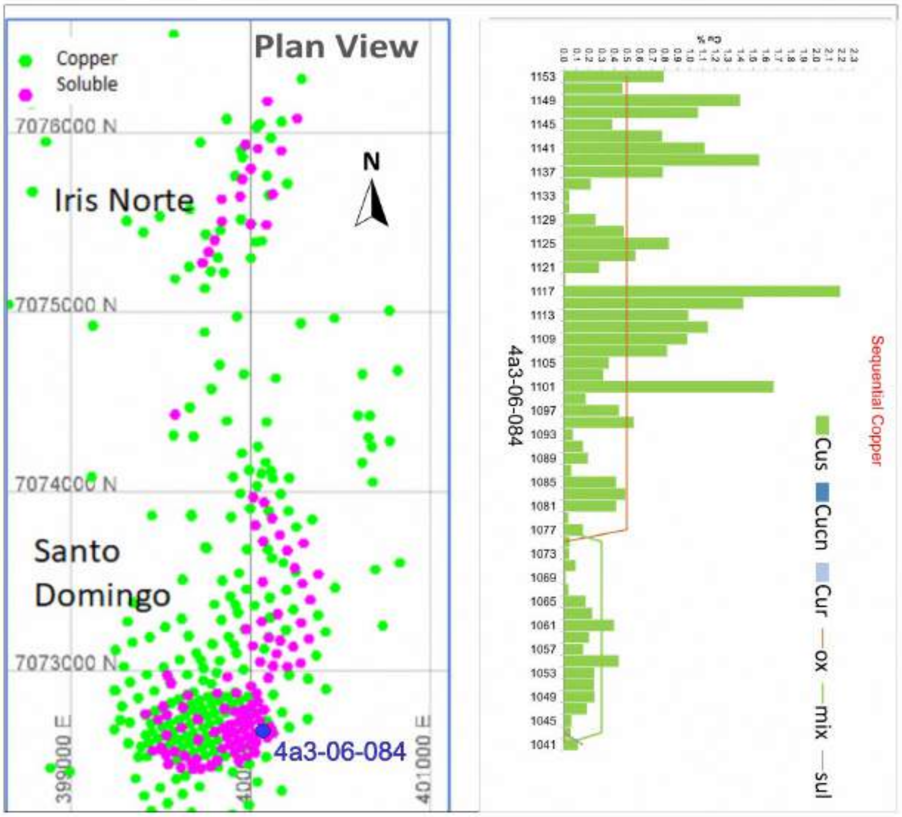
Table 14-11: SD-IN Sample Statistics: FeDTT

Element	Units	Domain	Count	Min	Max	Mean	CV
FeDTT	%	102	622	32.03	68.20	58.82	0.11
FeDTT	%	105	5,142	26.72	68.76	50.95	0.14
FeDTT	%	106	411	31.30	67.25	52.31	0.16
FeDTT	%	107	5,093	32.23	71.92	64.05	0.07
FeDTT	%	108	1,903	28.00	68.95	61.19	0.08
FeDTT	%	109	6,497	36.40	70.75	63.36	0.07
FeDTT	%	110	1	60.07	60.07	60.07	0.16
FeDTT	%	111	2,498	30.68	68.78	54.44	0.15
FeDTT	%	205	96	37.23	67.94	48.81	0.19
FeDTT	%	207	1,616	50.90	71.17	65.51	0.06
FeDTT	%	211	55	43.05	66.89	57.90	0.11

14.3.3.5 Sequential Copper

The soluble copper dataset available is limited, and in some areas scattered and discontinuous. Approximately 8% of the samples at SD-IN have soluble copper information. SLR reviewed sequential copper analyses in several holes, as shown in the example in Figure 14-11. it was noted that the oxidation boundaries contain some uncertainty, because the sequential copper dataset contains less complete, irregular coverage - approximately 8% of the samples at SD-IN have soluble copper information - compared to the total copper assays. There are few available soluble copper assays to define a mixed domain. The mixed domain has a low soluble copper/total copper ratio, which means that the residual copper (chalcopyrite) values are high and that the mixed domain may contain a significant amount of primary and secondary sulphide mineralization.

Figure 14-11: SD-IN Sequential Cu



Model Performance

Model	Composit	#	Meters	Cut	CuSol
Barren	Low grade Cut<0.1	13	26	0.046	0.010
CuOx		243	485	0.051	0.018
Mixture		110	216	0.051	0.011
Sulfides		403	799	0.047	0.005
Barren	CuS/Cut <0.15	1	2	0.150	0.016
	CuS/Cut 0.15-0.3	15	30	0.254	0.060
	CuS/Cut 0.3-0.5	2	4	0.317	0.147
	CuS/Cut >0.5	7	14	0.701	0.561
CuOx	CuS/Cut <0.15	35	69	0.200	0.022
	CuS/Cut 0.15-0.3	113	219	0.438	0.105
	CuS/Cut 0.3-0.5	433	859	0.386	0.152
	CuS/Cut >0.5	672	1340	0.737	0.541
Mixture	CuS/Cut <0.15	283	543	0.630	0.067
	CuS/Cut 0.15-0.3	432	838	0.521	0.110
	CuS/Cut 0.3-0.5	79	149	0.389	0.143
	CuS/Cut >0.5	12	20	0.600	0.407
Sulfides	CuS/Cut <0.15	1169	2314	0.670	0.038
	CuS/Cut 0.15-0.3	107	215	0.535	0.105
	CuS/Cut 0.3-0.5	30	59	0.485	0.183
	CuS/Cut >0.5	5	10	0.680	0.467

	Sondajes #	Sondajes Meters	Sample #
Santo Domingo	112	8212	4165
Iris Norte	16	439	219

SLR

Capstone Copper

Santo Domingo Project

Atacama Region, Chile

SD-IN Sequential Cu:
4a3-06-084

Source: SLR, 2023

Source: Figure prepared by SLR, 2023.

14.3.4 Treatment of High-Grade Assays

SLR performed EDA in X10 Geo software, grouping assays by lithology, oxidation, fault block, magsus, and other domains. SLR found that lithology produced the greatest differences in sample populations. Visual review of the statistical analyses confirmed that capping should be performed on a lithological basis. SLR entered capped copper, iron, gold, pyrite, and sulphur into new capped fields in the Mineral Resource database.

14.3.4.1 Capping Levels

SLR performed capping on a per lithology basis on separate SD and IN populations. Capping results are summarized in Table 14-12 and an example of capping analysis for Cu is provided in Figure 14-12. SLR applied light capping to most elements, with very little metal loss, as they showed long continuous tails in each rock type.

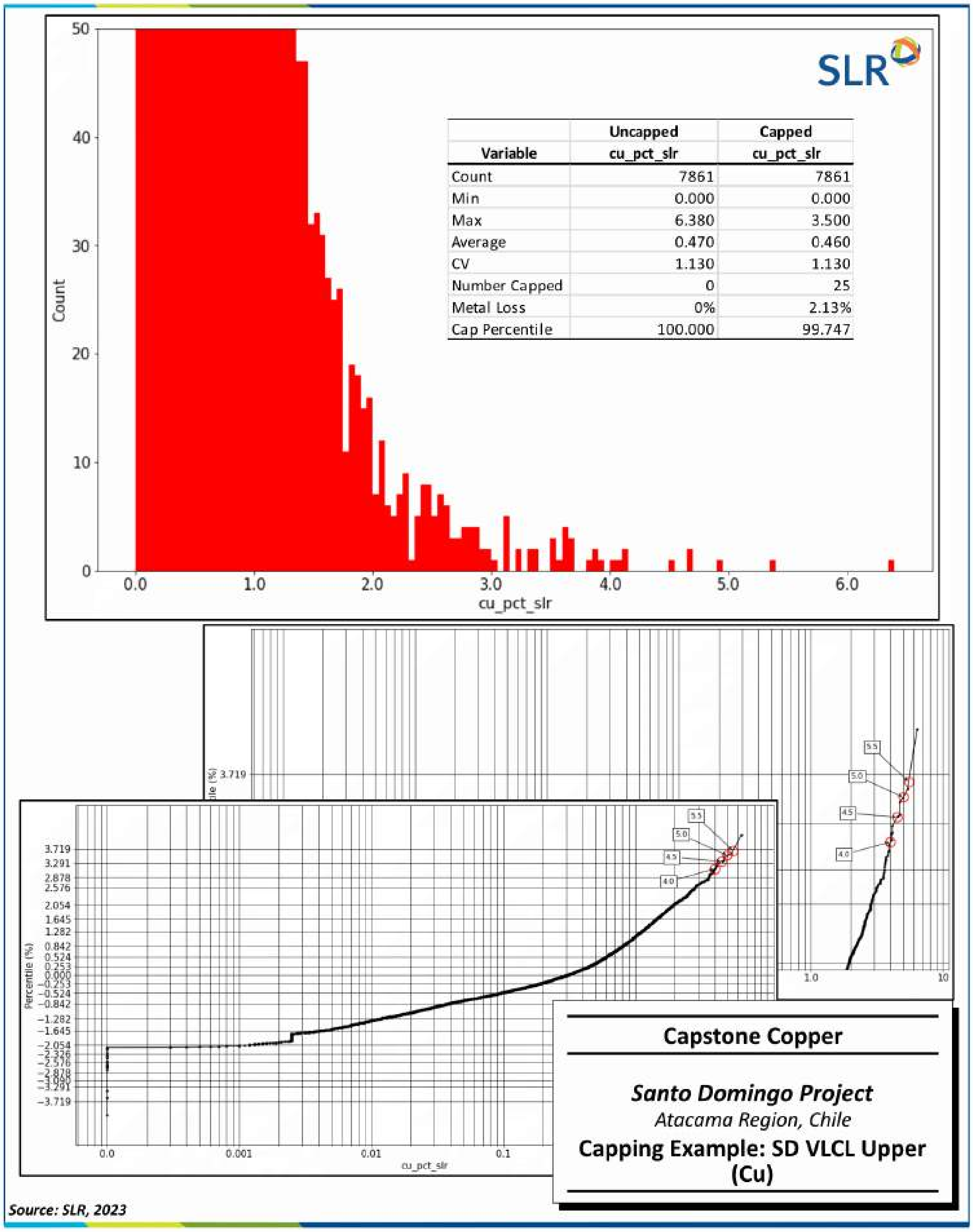
Table 14-12: SD-IN Capping Summary: Metals

Element	Units	Area	Lith	Lith_Txt	Count	Uncapped				Capped			Capped Count	Metal Loss (%)	Cap Percentile
						Min	Max	Mean	CV	Max	Mean	CV			
Au	g/t	SD	102	Breccia	636	0.003	0.360	0.060	0.78	0.250	0.060	7.50	3	0.0	99.47
Au	g/t	SD	105	ANDE Upper	6,823	0.003	2.380	0.060	1.14	0.700	0.060	1.09	1	0.0	99.98
Au	g/t	SD	106	ANLT ANDE	696	0.003	0.390	0.020	1.64	0.250	0.020	1.41	3	0.0	99.51
Au	g/t	SD	107	VLCL Upper	6,303	0.003	4.710	0.080	1.33	0.700	0.080	0.98	9	0.0	99.86
Au	g/t	SD	108	ANLT	2,018	0.003	0.660	0.030	1.73	0.500	0.030	1.71	1	0.0	99.95
Au	g/t	SD	109	VLCL Lower	6,823	0.003	2.380	0.060	1.14	0.700	0.060	1.09	1	0.0	99.98
Au	g/t	SD	110	CARB	1,406	0.003	2.940	0.020	4.76	0.500	0.020	2.51	3	0.0	99.76
Au	g/t	SD	111	ANDE Lower	2,677	0.003	0.450	0.020	1.38	0.350	0.020	1.32	2	0.0	99.90
Au	g/t	IN	205	ANDE Upper	440	0.005	0.210	0.010	2.10	0.100	0.010	1.70	5	0.0	98.77
Au	g/t	IN	207	VLCL	1,277	0.003	0.980	0.040	1.35	0.300	0.040	1.14	6	0.0	99.53
Au	g/t	IN	210	CARB	137	0.003	0.060	0.010	0.72	0.030	0.010	0.46	2	0.0	98.29
Au	g/t	IN	211	ANDE Lower	181	0.005	0.300	0.010	2.61	0.100	0.010	1.64	3	0.0	98.53
Co	ppm	SD	105	ANDE Upper	8,456	1.0	2,860.0	261.6	0.93	2,100.0	261.4	0.92	3	0.1	99.95
Co	ppm	SD	106	ANLT ANDE	1,537	1.0	1,015.0	53.4	1.41	600.0	52.8	1.31	6	1.3	99.62
Co	ppm	SD	107	VLCL Upper	4,824	1.0	4,820.0	299.7	0.97	2,000.0	266.1	0.94	9	0.3	99.86
Co	ppm	SD	108	ANLT	2,942	1.0	2,940.0	101.3	1.63	1,500.0	99.7	1.46	6	1.6	99.77
Co	ppm	SD	109	VLCL Lower	8,456	1.0	2,860.0	261.6	0.93	2,100.0	261.3	0.92	3	0.1	99.95
Co	ppm	SD	111	ANDE Lower	4,585	1.0	3,080.0	82.8	1.26	1,500.0	82.7	1.25	1	0.1	99.99
Co	ppm	IN	205	ANDE Upper	666	1.0	1,180.0	65.2	1.60	700.0	64.2	1.50	4	1.5	99.32
Co	ppm	IN	207	VLCL	2,963	0.8	2,730.0	170.4	1.24	1,500.0	169.2	1.19	9	0.7	99.70
Co	ppm	IN	211	ANDE Lower	222	1.0	696.0	66.6	1.19	400.0	65.3	1.08	1	2.0	99.35
Cu	%	SD	102	Breccia	825	0.000	3.710	0.340	1.08	2.000	0.340	1.01	4	0.0	99.44
Cu	%	SD	105	ANDE Upper	12,875	0.000	3.020	0.080	2.00	2.000	0.080	1.95	12	0.0	99.93
Cu	%	SD	106	ANLT ANDE	1,572	0.000	2.150	0.080	2.21	1.500	0.080	2.07	4	0.0	99.52
Cu	%	SD	107	VLCL Upper	7,861	0.000	6.380	0.470	1.13	3.500	0.460	1.13	25	2.1	99.75
Cu	%	SD	108	ANLT	2,955	0.000	5.380	0.190	1.86	3.500	0.190	1.82	3	0.0	99.95

Element	Units	Area	Lith	Lith_Txt	Count	Uncapped				Capped			Capped Count	Metal Loss (%)	Cap Percentile
						Min	Max	Mean	CV	Max	Mean	CV			
Cu	%	SD	109	VLCL Lower	8,490	0.000	5.340	0.330	1.39	3.500	0.330	1.37	8	0.0	99.90
Cu	%	SD	110	CARB	3,560	0.000	3.590	0.080	2.47	2.000	0.080	2.36	3	0.0	99.93
Cu	%	SD	111	ANDE Lower	4,589	0.000	4.460	0.080	2.41	2.500	0.080	2.05	5	0.0	99.87
Cu	%	IN	205	ANDE Upper	665	0.000	1.720	0.070	2.02	1.000	0.060	2.02	3	14.3	99.49
Cu	%	IN	207	VLCL	2,952	0.000	3.100	0.120	2.22	2.300	0.120	2.15	4	0.0	99.85
Cu	%	IN	210	CARB	305	0.001	1.270	0.030	3.30	0.300	0.030	1.91	5	0.0	98.24
Cu	%	IN	211	ANDE Lower	220	0.000	2.090	0.060	2.87	1.000	0.050	2.34	2	16.7	99.23
Fe	%	SD	102	Breccia	812	2.390	55.100	21.070	0.54	55.000	21.070	0.54	1	0.0	99.97
Fe	%	SD	105	ANDE Upper	12,692	0.010	55.800	11.360	0.45	55.000	11.360	0.45	1	0.0	99.99
Fe	%	SD	106	ANLT ANDE	1,537	2.950	52.800	10.910	0.4	40.000	10.880	0.38	4	0.3	99.68
Fe	%	SD	107	VLCL Upper	7,824	0.460	65.100	30.980	0.43	65.000	30.980	0.43	1	0.0	99.99
Fe	%	SD	108	ANLT	2,942	0.490	68.300	15.350	0.74	65.000	15.350	0.74	4	0.0	99.88
Fe	%	SD	109	VLCL Lower	8,456	0.440	65.300	25.790	0.51	65.000	25.790	0.51	1	0.0	99.98
Fe	%	SD	110	CARB	3,543	0.010	50.100	5.110	1.27	50.000	5.100	1.27	5	0.2	99.86
Fe	%	SD	111	ANDE Lower	4,585	1.160	61.700	13.900	0.54	55.000	13.890	0.54	9	0.1	99.77
Fe	%	IN	205	ANDE Upper	666	0.010	45.200	9.010	0.53	40.000	9.000	0.52	3	0.1	99.58
Fe	%	IN	207	VLCL	2,963	0.010	62.500	21.010	0.68	60.000	21.010	0.68	5	0.0	99.83
Fe	%	IN	210	CARB	305	0.450	10.350	3.190	0.62	9.000	3.180	0.61	5	0.3	98.28
Fe	%	IN	211	ANDE Lower	222	2.020	23.100	9.760	0.29	16.000	9.700	0.27	6	0.61	97.41
Magsus	SI	SD	102	Breccia	808	8	68,948	8,268	1.26	45,000	8,101	1.19	18	2.02	98.43
Magsus	SI	SD	105	ANDE Upper	5,908	3	101,625	6,551	1.11	75,000	6,541	1.1	6	0.15	99.90
Magsus	SI	SD	106	ANLT ANDE	746	87	68,908	4,796	1.14	35,000	4,725	1.03	3	1.47	99.56
Magsus	SI	SD	107	VLCL Upper	6,616	2	150,240	15,576	1.43	140,000	15,570	1.43	6	0.04	99.91
Magsus	SI	SD	108	ANLT	2,072	1	165,537	8,365	2.14	120,000	8,311	2.1	4	0.64	99.80
Magsus	SI	SD	109	VLCL Lower	6,550	1	223,375	15,372	1.38	130,000	15,352	1.37	5	0.13	99.93
Magsus	SI	SD	110	CARB	1,027	2	60,527	352	7.35	10,000	234	4.84	15	33.35	98.93
Magsus	SI	SD	111	ANDE Lower	2,228	14	110,000	15,226	0.9	100,000	15,219	0.89	2	0.05	99.88
Magsus	SI	IN	205	ANDE Upper	202	1	62,768	3,283	2.75	50,000	3,183	2.65	3	3.04	98.57

Element	Units	Area	Lith	Lith_Txt	Count	Uncapped				Capped			Capped Count	Metal Loss (%)	Cap Percentile
						Min	Max	Mean	CV	Max	Mean	CV			
Magsus	SI	IN	207	VLCL	1,892	2	126,867	19,532	1.06	100,000	19,505	1.06	3	0.14	99.82
Magsus	SI	IN	210	CARB	166	7	2,960	197	2.76	2,200	185	2.64	5	6.54	96.99
Magsus	SI	IN	211	ANDE Lower	55	697	14,505	6,411	0.54	12,000	6,307	0.52	3	1.63	93.96
S	%	SD	102	Breccia	812	0.005	10.000	1.160	1.12	6.000	1.140	1.04	8	1.72	99.00
S	%	SD	105	ANDE Upper	12,692	0.005	10.000	0.300	0.30	7.000	0.290	2.17	15	3.33	99.88
S	%	SD	106	ANLT ANDE	1,537	0.005	10.100	0.220	3.22	5.000	0.200	3.01	9	9.09	99.35
S	%	SD	107	VLCL Upper	7,821	0.005	20.200	1.960	0.92	15.000	1.960	0.92	1	0	99.98
S	%	SD	108	ANLT	2,940	0.005	10.000	0.910	1.37	7.000	0.900	1.30	23	1.1	99.26
S	%	SD	109	VLCL Lower	8,452	0.005	12.850	2.320	0.77	12.850	2.320	0.77	0	0	NA
S	%	SD	110	CARB	3,543	0.005	9.230	0.790	1.05	5.000	0.780	0.99	14	1.27	99.60
S	%	SD	111	ANDE Lower	4,585	0.005	6.850	0.460	1.50	5.000	0.460	1.46	11	0	99.78
S	%	IN	205	ANDE Upper	666	0.005	5.360	0.220	2.82	4.000	0.220	2.64	4	0	99.38
S	%	IN	207	VLCL	2,963	0.005	13.700	1.500	1.23	9.000	1.500	1.22	6	0	99.79
S	%	IN	210	CARB	305	0.005	5.310	0.590	1.13	2.500	0.580	0.98	2	1.69	99.18
S	%	IN	211	ANDE Lower	222	0.010	4.170	0.520	1.36	3.000	0.500	1.28	3	3.85	98.45

Figure 14-12: **Capping Example: SD VLCL Upper (Cu)**



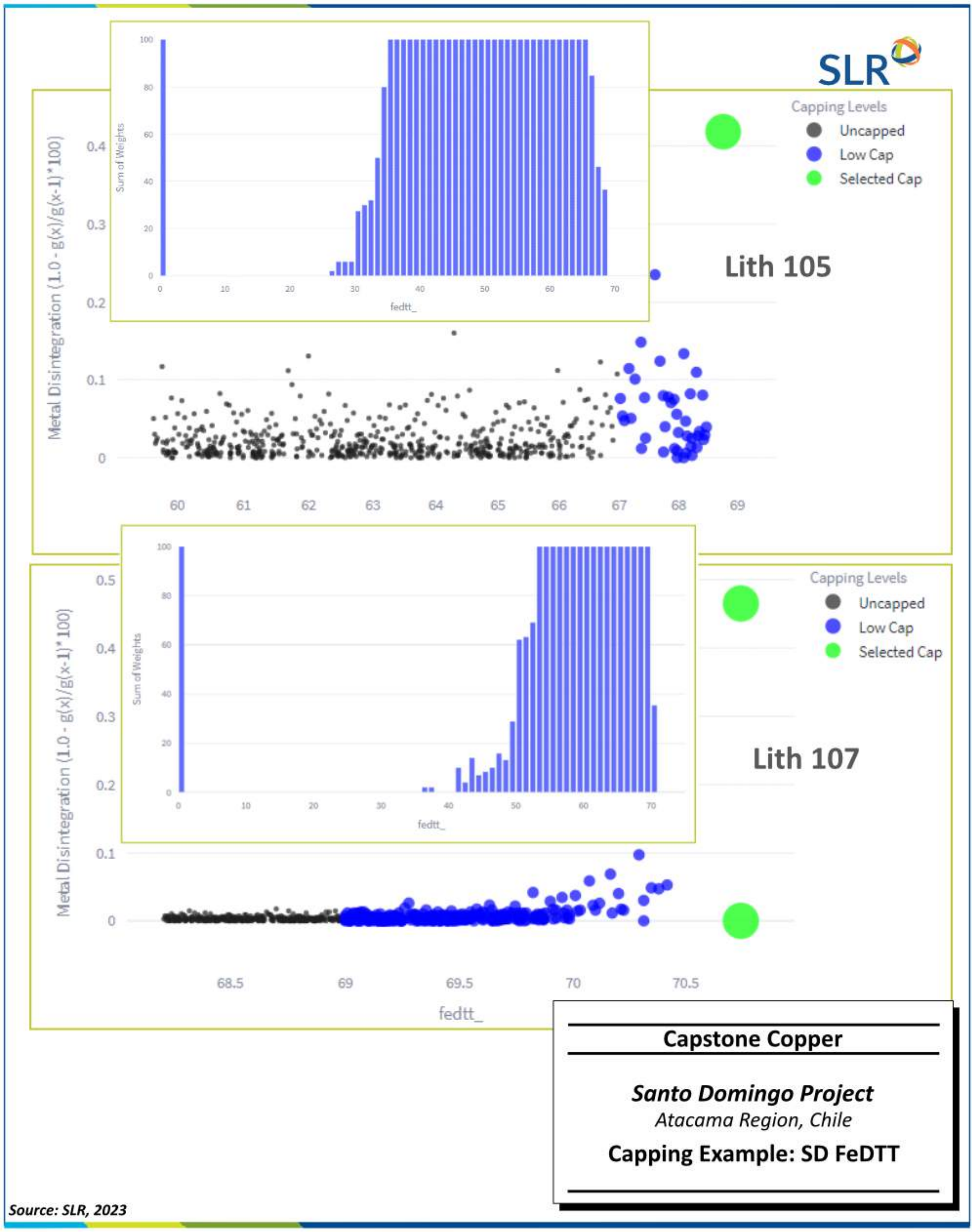
Source: Figure prepared by SLR, 2023.

For FeDTT model data, SLR examined the sample distribution of the FeDTT samples and noted that histograms and cumulative distribution plots showed very continuous grades at the high end of the spectrum. SLR then performed very light capping resulting in effectively zero 'metal loss', capping only a few values based on disintegration plots (Table 14-13 and Figure 14-13).

Table 14-13: SD-IN Capping Summary: FeDTT

Domain	Count	Min	Min Capped	Max	Max Capped	Mean	Mean Capped	Mean Diff	Mean Diff (%)	CV	CV Capped	CV Diff (%)
102	622	32.03	32.03	68.20	67.00	58.82	58.82	0.00	0.00	0.11	0.11	0.00
105	5,142	26.72	26.72	68.76	68.50	50.95	50.95	0.00	0.00	0.14	0.14	0.00
106	411	31.30	31.30	67.25	67.00	52.31	52.31	0.00	0.00	0.16	0.16	0.00
107	5,093	32.23	32.23	71.92	70.50	64.05	64.05	0.00	0.00	0.07	0.07	0.00
108	1,903	28.00	28.00	68.95	68.00	61.19	61.19	0.00	0.00	0.08	0.08	0.00
109	6,497	36.40	36.40	70.75	70.50	63.36	63.36	0.00	0.00	0.07	0.07	0.00
110	1	60.07	60.00	60.07	60.00	60.07	60.00	-0.07	-0.12	0.16	0.16	0.00
111	2,498	30.68	30.68	68.78	68.50	54.44	54.44	0.00	0.00	0.15	0.15	0.00
205	96	37.23	37.23	67.94	67.50	48.81	48.81	0.00	0.00	0.19	0.19	0.00
207	1,616	50.90	50.90	71.17	70.50	65.51	65.50	-0.01	-0.02	0.06	0.06	0.00
211	55	43.05	43.05	66.89	66.00	57.90	57.88	-0.02	-0.03	0.11	0.11	0.00

Figure 14-13: Capping Example: SD FeDTT



Source: Figure prepared by SLR, 2023.

14.3.4.2 High Grade Restriction

High yield restrictions (HYR) were set as indicators on specific hole intervals (Table 14-14) in the composite files, using an HYR_USE field set in a database script. Then the indicator was set in pass three of the grade estimate to 70 m x 70 m x 10 m for pass 3 in all rock types.

Table 14-14: Local High Yield Restrictions

Hole ID	Condition
4a3-05-069	FROM >= 340.0
4a3-10-397DD	FROM >= 340.0
4a3-06-079DD	FROM >= 451.0
4a3-09-370DD	FROM >= 400.0
4a3-10-387DD	FROM >= 410.0
4a3-10-077DD-B	FROM >= 360.0
4a3-06-077DD	FROM >= 360.0
4a3-21-502DD	FROM >= 276.0 and TO <= 332
4a3-06-123	FROM >= 340.0
4a3-06-124	FROM >= 340.0
4a3-21-503DD	FROM >= 360.0
4a2-07-011	entire hole
GD-6	entire hole
4a3-06-080DD	FROM >= 407.0
CDH-02	d_lith = 105

14.3.5 Compositing

Assays in the Mineral Resource database were composited at two metre intervals from the top of the drill hole, breaking on lithologic boundaries. SLR chose a two-metre length for the following reasons:

- Two metres represents the modal majority sample length for the Mineral Resource database, requiring the least breakage at sample boundaries.
- Six metre composites would have been too long given the five-metre block height.
- Five metre composites would make dilution at composite stage where two metre samples at the margins are merged with low grade samples.
- Two metre lengths allow for more samples to be selected than four metre samples at the discretization stage of grade estimation, while still remaining inside the block.
- Two metre samples produced a higher resolution sub-blocked model which adheres better to the actual sample grades.

All composites are broken on lithology solids and then flagged for lithology and oxidation. A summary of capped compositing statistics by lithology code is provided in Table 14-15.

Table 14-15: Composite Statistics Summary

Element	Units	Lith Code	Count	Min	Max	Mean	CV
Au	g/t	102	677	0.0001	0.25	0.04	1.08
Au	g/t	105	12,901	0.0001	0.70	0.01	2.48
Au	g/t	106	1,425	0.0001	0.25	0.01	2.52
Au	g/t	107	6,278	0.0001	0.70	0.07	1.19
Au	g/t	108	2,405	0.0001	0.39	0.02	1.98
Au	g/t	109	7,085	0.0001	0.66	0.05	1.37
Au	g/t	110	3,768	0.0001	0.70	0.01	4.53
Au	g/t	111	4,501	0.0001	0.35	0.01	2.33
Au	g/t	205	812	0.0001	0.21	0.01	2.21
Au	g/t	207	3,098	0.0001	0.70	0.01	2.50
Au	g/t	210	464	0.0001	0.06	0.00	2.33
Au	g/t	211	222	0.0001	0.30	0.01	2.02
Co	g/t	102	677	5.0000	650.00	103.51	1.05
Co	g/t	105	12,901	0.0001	2,100.00	65.55	1.67
Co	g/t	106	1,425	0.0001	600.00	49.84	1.30
Co	g/t	107	6,278	0.0001	2,000.00	260.63	0.95
Co	g/t	108	2,405	0.0001	1,500.00	89.56	1.53
Co	g/t	109	7,085	0.0001	2,100.00	261.46	0.91
Co	g/t	110	3,768	0.0001	2,000.00	21.67	2.60
Co	g/t	111	4,501	0.0001	1,080.00	71.12	1.38
Co	g/t	205	812	0.0001	1,180.00	52.87	1.85
Co	g/t	207	3,098	0.0001	2,000.00	161.12	1.29
Co	g/t	210	464	0.0001	78.00	10.19	1.17
Co	g/t	211	222	1.0000	696.00	66.62	1.19
Cu	%	102	677	0.0015	2.00	0.34	0.99
Cu	%	105	12,901	0.0000	2.00	0.09	1.87
Cu	%	106	1,425	0.0001	1.50	0.08	1.92
Cu	%	107	6,278	0.0001	4.50	0.48	1.12
Cu	%	108	2,405	0.0001	3.50	0.16	1.73
Cu	%	109	7,085	0.0001	3.50	0.33	1.36
Cu	%	110	3,768	0.0001	2.00	0.06	2.45
Cu	%	111	4,501	0.0000	2.50	0.07	2.18
Cu	%	205	812	0.0001	1.72	0.05	2.46
Cu	%	207	3,098	0.0001	3.10	0.12	2.21

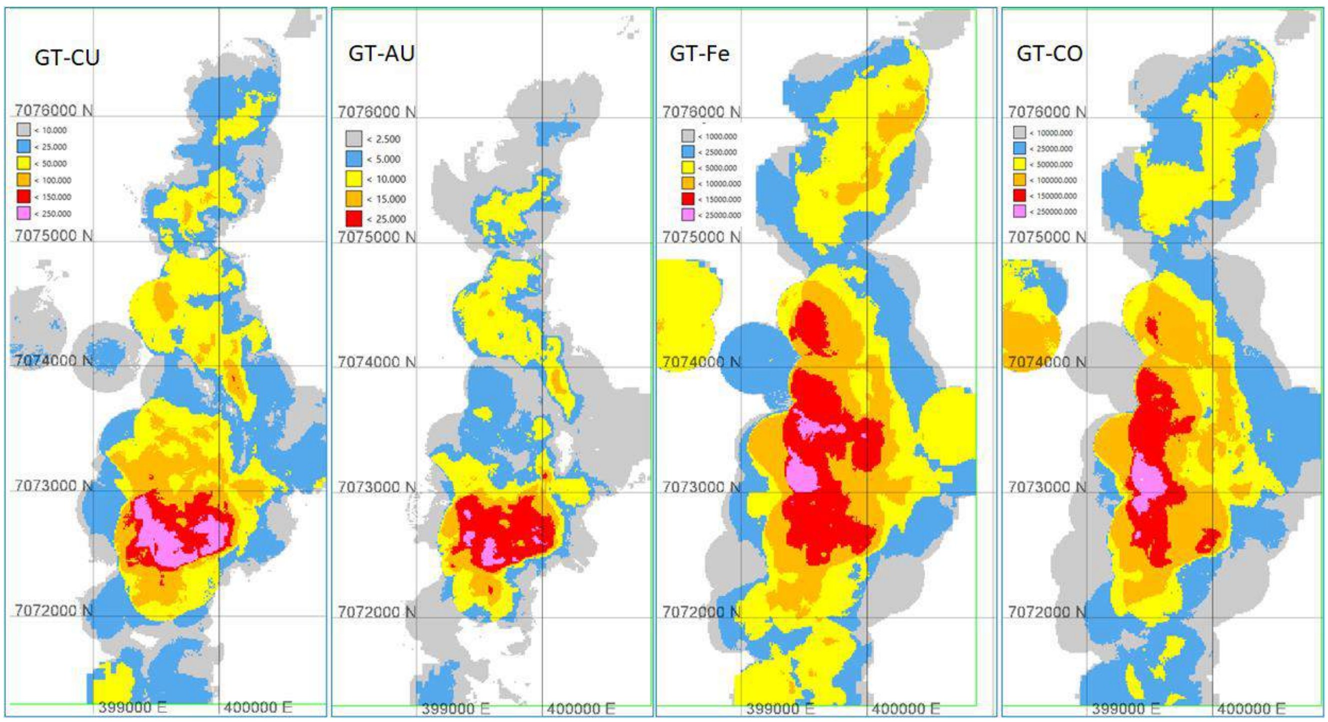
Element	Units	Lith Code	Count	Min	Max	Mean	CV
Cu	%	210	464	0.0001	1.27	0.02	3.66
Cu	%	211	222	0.0001	2.09	0.06	2.90
Fe	%	102	677	2.3900	54.40	20.40	0.54
Fe	%	105	12,901	0.0001	55.00	10.47	0.56
Fe	%	106	1,425	0.0001	40.00	10.29	0.45
Fe	%	107	6,278	0.0001	65.00	30.48	0.45
Fe	%	108	2,405	0.0001	61.20	13.58	0.71
Fe	%	109	7,085	0.0001	65.00	25.69	0.51
Fe	%	110	3,768	0.0001	50.00	3.27	1.50
Fe	%	111	4,501	0.0001	55.00	12.05	0.68
Fe	%	205	812	0.0001	45.20	7.26	0.78
Fe	%	207	3,098	0.0001	62.50	19.77	0.75
Fe	%	210	464	0.0001	10.35	2.09	1.05
Fe	%	211	222	2.0200	23.10	9.76	0.29
Py	%	102	677	0.0002	17.84	1.55	1.44
Py	%	105	12,901	0.0000	18.33	0.39	2.75
Py	%	106	1,425	0.0000	18.83	0.29	4.51
Py	%	107	6,278	0.0002	25.84	2.83	1.07
Py	%	108	2,405	0.0001	18.68	1.21	1.65
Py	%	109	7,085	0.0001	23.74	3.75	0.83
Py	%	110	3,768	0.0000	12.95	1.03	1.16
Py	%	111	4,501	0.0000	12.71	0.59	1.79
Py	%	205	812	0.0002	9.52	0.30	3.40
Py	%	207	3,098	0.0000	25.58	2.50	1.33
Py	%	210	464	0.0002	9.92	0.71	1.59
Py	%	211	222	0.0002	7.79	0.86	1.37
S	%	102	677	0.0050	10.00	1.13	1.16
S	%	105	12,901	0.0001	10.00	0.27	2.31
S	%	106	1,425	0.0001	10.10	0.20	3.72
S	%	107	6,278	0.0001	15.00	1.95	0.92
S	%	108	2,405	0.0001	10.00	0.81	1.46
S	%	109	7,085	0.0001	12.85	2.34	0.76
S	%	110	3,768	0.0001	8.31	0.60	1.13
S	%	111	4,501	0.0001	6.85	0.38	1.64
S	%	205	812	0.0001	5.36	0.18	3.13
S	%	207	3,098	0.0001	13.70	1.42	1.29
S	%	210	464	0.0001	5.31	0.39	1.57
S	%	211	222	0.0100	4.17	0.52	1.37

14.3.6 Trend Analysis

14.3.6.1 Grade Contouring

SLR generated several grade interpolants for copper, iron, gold, and cobalt in Leapfrog, comparing them to the lithology, structure, oxidation, and magsus models. SLR determined that lithology is the most important control on mineralization. SLR examined the relationship between iron and copper using interpolants in Leapfrog and determined that correlation between them is weak, though mineralized lithologies (107, 109) contain strong concentrations of both Fe and Cu. A block model investigation using grade-thickness (GT) maps showed that there is strong correlation in copper and gold, and iron and cobalt pairs (Figure 14-14). The QP notes that the work showed the overwhelming controls on mineralization to be lithology, and recommends further work on correlation between metals, including correlation matrices split by both area and lithology to determine local changes in elemental dependencies.

Figure 14-14: Block GT Correlations: Cu-Au-and Fe-Co



Source: Figure prepared by SLR, 2023.

14.3.6.2 Dynamic Anisotropy

Dynamic Anisotropy (DA) was used to orient grade interpolation search ellipsoids according to the orientation of the mineralization at each point in space, which coincides well with rock type boundaries at SD-IN. SLR performed this technique in Vulcan by generating a set of hanging wall, footwall, and centre surfaces in Leapfrog and then exporting them to Vulcan surfaces. A DA specification file was generated to use appropriate surfaces for grade estimation in each rock type. DA angles were visually validated in Vulcan cross-sections and long sections.

14.3.6.3 Variography

SLR used Vulcan's visual data analyser software to generate variograms for copper, iron, gold, cobalt, and pyrite accumulation, as well as for density, massrec, FeDTT, and magnetic susceptibility. In general, variograms were well structured with sills ranging from 120 m to 150 m. Nuggets generally occurred at 0.1 to 0.25. Variograms were generated on a per lithology basis at SD-IN and then applied to the DA grade estimation files for each lithological domain. Variograms for IN were less well defined than those for SD. Lith 205 and 211 (IN Upper and Lower Andesite) were combined due to a scarcity of data. A summary of the variograms is presented in Table 14-16 for density weighted metal accumulations and in Table 14-17 for density, massrec and magsus. Figure 14-15 provides examples of variograms for copper and iron accumulations.

Table 14-16: Summary of Accumulation Variogram Parameters

Element	Lith	Nugget	Rotn XY (Deg)	Rotn XZ (Deg)	Rotn YZ (Deg)	Structure 1					Structure 2					Structure 3				
						Type	Diff Sill	Major Range	Semi Major Range	Minor Range	Type	Diff Sill	Major Range	Semi Major Range	Minor Range	Type	Diff Sill	Major Range	Semi Major Range	Minor Range
au*sg	102	0.20	305	-15	0	Sph	0.55	25	25	15	Sph	0.25	120	120	50	-	-	-	-	-
au*sg	105	0.30	310	-21	0	Sph	0.42	55	12	12	Sph	0.28	110	110	75	-	-	-	-	-
au*sg	106	0.20	305	-15	0	Sph	0.45	70	55	7	Sph	0.35	70	55	50	-	-	-	-	-
au*sg	107	0.25	310	-21	0	Sph	0.35	10	10	5	Sph	0.25	50	50	70	Sph	0.15	150	175	70
au*sg	108	0.30	310	-21	0	Sph	0.40	60	15	10	Sph	0.18	200	15	35	Sph	0.125	200	100	35
au*sg	109	0.25	310	-21	0	Sph	0.35	15	20	10	Sph	0.20	15	20	80	Sph	0.20	270	200	80
au*sg	110	0.35	310	-21	0	Sph	0.40	50	40	7	Sph	0.25	50	40	55	-	-	-	-	-
au*sg	111	0.20	310	-21	0	Sph	0.55	45	45	5	Sph	0.25	45	45	60	-	-	-	-	-
au*sg	205	0.30	310	-5	0	Sph	0.50	60	40	7	Sph	0.20	180	110	60	-	-	-	-	-
au*sg	207	0.20	305	-5	0	Sph	0.60	25	5	5	Sph	0.20	90	60	70	-	-	-	-	-
au*sg	210	0.10	310	-5	0	Sph	0.58	25	125	5	Sph	0.32	240	125	110	-	-	-	-	-
au*sg	205_211	0.30	310	-5	0	Sph	0.50	20	20	5	Sph	0.20	50	20	70	-	-	-	-	-
co*sg	102	0.25	310	-21	0	Sph	0.37	10	70	20	Sph	0.38	30	70	55	-	-	-	-	-
co*sg	105	0.25	310	-21	0	Sph	0.50	10	50	10	Sph	0.25	80	50	100	-	-	-	-	-
co*sg	106	0.30	305	-21	0	Sph	0.26	170	40	5	Sph	0.44	170	90	35	-	-	-	-	-
co*sg	107	0.15	310	-21	0	Sph	0.45	10	25	5	Sph	0.25	10	25	35	Sph	0.15	160	150	35
co*sg	108	0.25	305	-15	0	Sph	0.42	30	20	5	Sph	0.33	90	60	40					
co*sg	109	0.25	310	-21	0	Sph	0.42	20	25	10	Sph	0.33	140	70	50	-	-	-	-	-
co*sg	110	0.30	310	-21	0	Sph	0.35	60	20	5	Sph	0.35	60	45	80	-	-	-	-	-
co*sg	111	0.15	310	-21	0	Sph	0.20	35	40	7	Sph	0.65	35	40	310	-	-	-	-	-
co*sg	205	0.20	310	-5	0	Sph	0.48	50	25	10	Sph	0.32	50	25	50	-	-	-	-	-
co*sg	207	0.20	310	-5	0	Sph	0.30	30	50	10	Sph	0.50	30	50	40	-	-	-	-	-
co*sg	210	0.15	310	-5	0	Sph	0.85	50	50	40	Sph	-	-	-	-	-	-	-	-	-
co*sg	205_211	0.20	310	-5	0	Sph	0.55	50	30	10	Sph	0.25	130	80	90	-	-	-	-	-

Element	Lith	Nugget	Rotn XY (Deg)	Rotn XZ (Deg)	Rotn YZ (Deg)	Structure 1					Structure 2					Structure 3				
						Type	Diff Sill	Major Range	Semi Major Range	Minor Range	Type	Diff Sill	Major Range	Semi Major Range	Minor Range	Type	Diff Sill	Major Range	Semi Major Range	Minor Range
cu*sg	102	0.15	310	-21	0	Sph	0.47	10	30	8	Sph	0.38	50	30	40	-	-	-	-	-
cu*sg	105	0.25	310	-21	0	Sph	0.42	60	15	12	Sph	0.33	135	120	150	-	-	-	-	-
cu*sg	106	0.20	305	-15	0	Sph	0.55	90	50	10	Sph	0.25	90	50	80	-	-	-	-	-
cu*sg	107	0.20	310	-21	0	Sph	0.38	15	10	6	Exp	0.42	120	70	150	-	-	-	-	-
cu*sg	108	0.25	310	-21	0	Sph	0.44	15	15	5	Sph	0.31	120	90	35	-	-	-	-	-
cu*sg	109	0.30	310	-21	0	Sph	0.25	15	12	7	Sph	0.25	15	45	90	Sph	0.20	300	200	90
cu*sg	110	0.18	310	-21	0	Sph	0.48	50	15	10	Sph	0.34	50	80	70	-	-	-	-	-
cu*sg	111	0.18	310	-21	0	Sph	0.54	25	20	5	Sph	0.28	110	20	40	-	-	-	-	-
cu*sg	205_211	0.30	310	-5	0	Sph	0.55	100	40	5	Sph	0.15	100	40	80	-	-	-	-	-
cu*sg	205	0.20	310	-5	0	Sph	0.60	20	40	8	Sph	0.20	150	40	100	-	-	-	-	-
cu*sg	207	0.20	310	-5	0	Exp	0.55	40	50	7	Exp	0.25	100	80	80	-	-	-	-	-
cu*sg	210	0.40	310	-5	0	Sph	0.35	100	90	5	Sph	0.25	100	90	45	-	-	-	-	-
fe*sg	102	0.10	310	-21	0	Sph	0.50	60	50	15	Sph	0.40	60	120	70	-	-	-	-	-
fe*sg	105	0.10	310	-21	0	Sph	0.30	20	30	10	Sph	0.35	90	30	100	Sph	0.25	140	130	100
fe*sg	106	0.05	305	-15	0	Sph	0.32	80	45	5	Sph	0.63	80	45	110	-	-	-	-	-
fe*sg	107	0.15	310	-21	0	Sph	0.45	25	25	7	Sph	0.40	400	350	95	-	-	-	-	-
fe*sg	108	0.10	305	-15	0	Sph	0.40	30	12	5	Sph	0.10	135	12	40	Sph	0.40	135	100	40
fe*sg	109	0.15	310	-21	0	Sph	0.55	25	20	12	Sph	0.30	150	110	110	-	-	-	-	-
fe*sg	110	0.08	310	-21	0	Sph	0.40	30	15	12	Sph	0.52	30	60	100	-	-	-	-	-
fe*sg	111	0.05	310	-21	0	Exp	0.60	50	120	60	Sph	0.35	150	120	60	-	-	-	-	-
fe*sg	205	0.15	310	-5	0	Sph	0.45	70	40	7	Sph	0.40	150	40	250	-	-	-	-	-
fe*sg	207	0.20	305	-15	0	Sph	0.47	15	30	10	Exp	0.33	110	110	50	-	-	-	-	-
fe*sg	210	0.10	310	-5	0	Sph	0.35	25	20	20	Sph	0.55	50	50	100	-	-	-	-	-
fe*sg	205_211	0.15	310	-5	0	Sph	0.45	35	25	10	Sph	0.40	130	25	250	-	-	-	-	-
s*sg	102	0.20	310	-21	0	-	0.30	20	10	12	-	0.50	20	50	60	-	-	-	-	-
s*sg	105	0.10	310	-21	0	-	0.50	15	28	10	-	0.40	110	50	90	-	-	-	-	-
s*sg	106	0.15	310	-21	0	-	0.20	100	30	7	-	0.65	100	30	75	-	-	-	-	-

Element	Lith	Nugget	Rotn XY (Deg)	Rotn XZ (Deg)	Rotn YZ (Deg)	Structure 1					Structure 2					Structure 3				
						Type	Diff Sill	Major Range	Semi Major Range	Minor Range	Type	Diff Sill	Major Range	Semi Major Range	Minor Range	Type	Diff Sill	Major Range	Semi Major Range	Minor Range
s*sg	107	0.20	310	-21	0	-	0.40	10	10	10	-	0.40	130	170	90	-	-	-	-	-
s*sg	108	0.30	310	-21	0	-	0.31	25	12	5	-	0.39	135	60	35	-	-	-	-	-
s*sg	109	0.20	310	-21	0	-	0.50	15	10	10	-	0.30	90	90	80	-	-	-	-	-
s*sg	110	0.10	310	-21	0	-	0.35	40	40	7	-	0.55	40	40	90	-	-	-	-	-
s*sg	111	0.15	310	-21	0	-	0.25	30	35	5	-	0.60	30	35	40	-	-	-	-	-
s*sg	205	0.20	310	-5	0	-	0.60	125	30	12	-	0.20	125	140	35	-	-	-	-	-
s*sg	207	0.15	310	-5	0	-	0.46	20	20	10	-	0.39	220	275	85	-	-	-	-	-
s*sg	210	0.05	310	-5	0	-	0.30	140	140	10	-	0.65	250	140	220	-	-	-	-	-
s*sg	205_211	0.20	310	-5	0	-	0.60	40	20	12	-	0.20	150	180	80	-	-	-	-	-

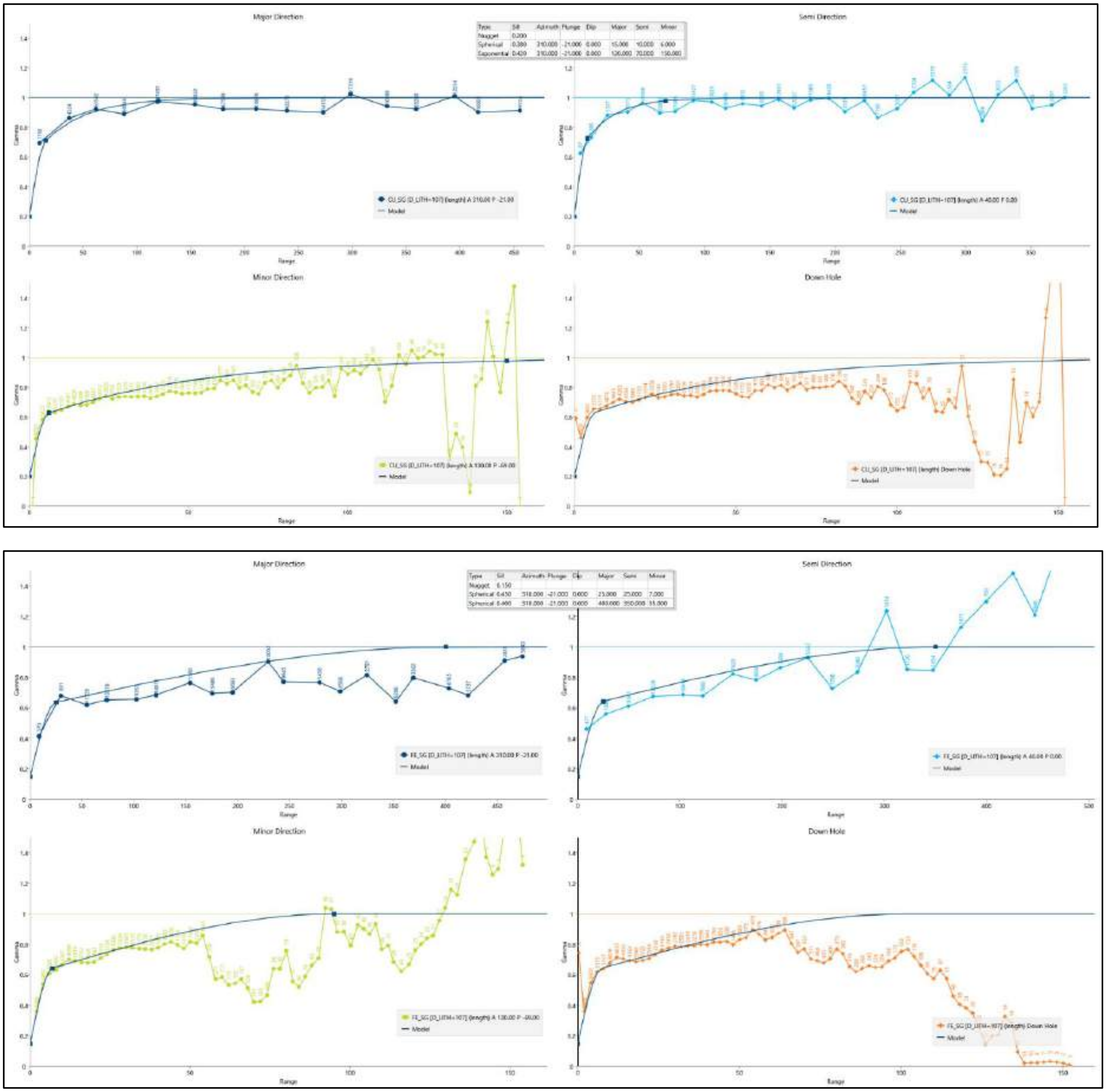
Table 14-17: Summary of Variogram Parameters (without accumulation)

Element	Lith	Nugget	Rotn XY (Deg)	Rotn XZ (Deg)	Rotn YZ (Deg)	Structure 1					Structure 2					Structure 3				
						Type	Diff Sill	Major Range	Semi Major Range	Minor Range	Type	Diff Sill	Major Range	Semi Major Range	Minor Range	Type	Diff Sill	Major Range	Semi Major Range	Minor Range
Density	102	0.15	310	-21	0	Sph	0.20	120	70	10	Sph	0.65	120	70	120	-	-	-	-	-
Density	105	0.05	305	-5	0	Sph	0.40	20	20	10	Sph	0.55	500	50	100	-	-	-	-	-
Density	106	0.10	315	-21	0	Sph	0.25	100	220	5	Sph	0.65	350	220	35	-	-	-	-	-
Density	107	0.10	310	-21	0	Sph	0.45	20	15	5	Sph	0.45	320	350	90	-	-	-	-	-
Density	108	0.05	315	-21	0	Sph	0.55	80	10	5	Sph	0.40	80	100	90	-	-	-	-	-
Density	109	0.10	310	-21	0	Sph	0.45	15	10	8	Sph	0.45	120	70	45	-	-	-	-	-
Density	110	0.20	310	-21	0	Sph	0.40	70	350	12	Sph	0.40	70	350	80	-	-	-	-	-
Density	111	0.00	310	-21	0	Sph	0.20	140	90	5	Sph	0.80	140	90	55	-	-	-	-	-
Density	205_211	0.10	310	-5	0	Sph	0.90	150	60	10	-	-	-	-	-	-	-	-	-	-
Density	205	0.15	310	-5	0	Sph	0.85	100	45	8	-	-	-	-	-	-	-	-	-	-
Density	207	0.10	305	-15	0	Sph	0.67	25	25	10	Sph	0.23	90	75	50	-	-	-	-	-
Density	210	0.10	310	-5	0	Sph	0.90	300	170	6	-	-	-	-	-	-	-	-	-	-

Element	Lith	Nugget	Rotn XY (Deg)	Rotn XZ (Deg)	Rotn YZ (Deg)	Structure 1					Structure 2					Structure 3				
						Type	Diff Sill	Major Range	Semi Major Range	Minor Range	Type	Diff Sill	Major Range	Semi Major Range	Minor Range	Type	Diff Sill	Major Range	Semi Major Range	Minor Range
MasRec	102	0.25	310	-21	0	Sph	0.40	10	30	10	Sph	0.35	110	60	40	-	-	-	-	-
MasRec	105	0.18	310	-21	0	Sph	0.25	25	20	10	Sph	0.35	25	20	70	Sph	0.22	240	140	70
MasRec	106	0.28	310	-15	0	Sph	0.15	80	100	10	Sph	0.57	80	100	80	-	-	-	-	-
MasRec	107	0.10	310	-21	0	Sph	0.60	40	20	10	Sph	0.30	340	140	120	-	-	-	-	-
MasRec	108	0.20	310	-21	0	Sph	0.58	10	10	5	Sph	0.22	150	85	40	-	-	-	-	-
MasRec	109	0.20	310	-21	0	Sph	0.40	30	20	10	Sph	0.40	260	90	110	-	-	-	-	-
MasRec	110	0.85	310	-21	0	Exp	0.15	40	40	12	-	-	-	-	-	-	-	-	-	-
MasRec	111	0.10	310	-21	0	Exp	0.90	50	50	60	-	-	-	-	-	-	-	-	-	-
MasRec	205	0.10	310	-5	0	Exp	0.90	60	40	5	-	-	-	-	-	-	-	-	-	-
MasRec	207	0.15	310	-5	0	Sph	0.60	25	15	10	Sph	0.25	170	160	55	-	-	-	-	-
MasRec	210	0.01	310	-5	0	Sph	0.99	50	50	15	-	-	-	-	-	-	-	-	-	-
MasRec	211	0.01	310	-5	0	Exp	0.99	50	30	7	-	-	-	-	-	-	-	-	-	-
FeDTT	102	0.30	310	-21	0	Sph	0.39	23	24	77	Sph	0.31	83	55	100					
FeDTT	105	0.25	305	-15	0	Sph	0.20	54	40	18	Sph	0.55	161	178	111					
FeDTT	106	0.30	310	-21	0	Sph	0.08	49	37	59	Sph	0.62	73	68	88					
FeDTT	107	0.20	310	-21	0	Sph	0.26	35	17	75	Sph	0.54	323	307	187					
FeDTT	108	0.30	310	-21	0	Sph	0.26	30	10	17	Sph	0.44	84	45	93					
FeDTT	109	0.20	310	-21	0	Sph	0.21	29	21	32	Sph	0.59	198	180	152					
FeDTT	110	0.25	310	-21	0	Sph	0.41	75	69	71	Sph	0.34	110	110	110					
FeDTT	111	0.20	310	-21	0	Sph	0.12	43	25	29	Sph	0.68	206	215	107					
FeDTT	205	0.15	310	-21	0	Sph	0.32	41	25	18	Sph	0.53	103	100	107					
FeDTT	207	0.18	310	-21	0	Sph	0.41	34	32	78	Sph	0.41	126	162	129					
FeDTT	210	0.15	310	-21	0	Sph	0.36	38	19	18	Sph	0.49	97	94	105					
FeDTT	211	0.15	310	-21	0	Sph	0.32	41	25	18	Sph	0.53	103	100	107					
Magsus	102	0.250	310	-21	0	Sph	0.40	10	30	10	-	-	-	-	-	-	-	-		
Magsus	105	0.180	310	-21	0	Sph	0.25	25	20	10	Sph	0.350	25.00	20.00	70.00	Sph	0.220	240.00	140.00	70.00
Magsus	106	0.280	305	-15	0	Sph	0.15	80	100	10	Sph	0.570	80.00	100.00	80.00	-	-	-	-	-
Magsus	107	0.100	310	-21	0	Sph	0.60	40	20	10	Sph	0.300	340.00	140.00	120.00	-	-	-	-	-
Magsus	108	0.200	310	-21	0	Sph	0.58	10	10	5	Sph	0.220	150.00	85.00	40.00	-	-	-	-	-
Magsus	109	0.200	310	-21	0	Sph	0.40	30	20	10	Sph	0.400	260.00	90.00	110.00	-	-	-	-	-

Element	Lith	Nugget	Rotn XY (Deg)	Rotn XZ (Deg)	Rotn YZ (Deg)	Structure 1					Structure 2					Structure 3				
						Type	Diff Sill	Major Range	Semi Major Range	Minor Range	Type	Diff Sill	Major Range	Semi Major Range	Minor Range	Type	Diff Sill	Major Range	Semi Major Range	Minor Range
Magsus	110	0.850	310	-21	0	Exp	0.150	40.00	40.00	12.00	-	-	-	-	-	-	-	-	-	-
Magsus	111	0.100	310	-21	0	Exp	0.900	50.00	50.00	60.00	-	-	-	-	-	-	-	-	-	-
Magsus	205_211	0.010	310	-5	0	Exp	0.990	50.00	30.00	7.00	-	-	-	-	-	-	-	-	-	-
Magsus	207	0.150	310	-5	0	Sph	0.600	25.00	5.00	10.00	Sph	0.250	170.00	160.00	55.00	-	-	-	-	-
Magsus	210	0.010	310	-5	0	Sph	0.990	50.00	50.00	15.00	-	-	-	-	-	-	-	-	-	-

Figure 14-15: Example Variogram, SD Volcaniclastic Unit, Cu and Fe Accumulations



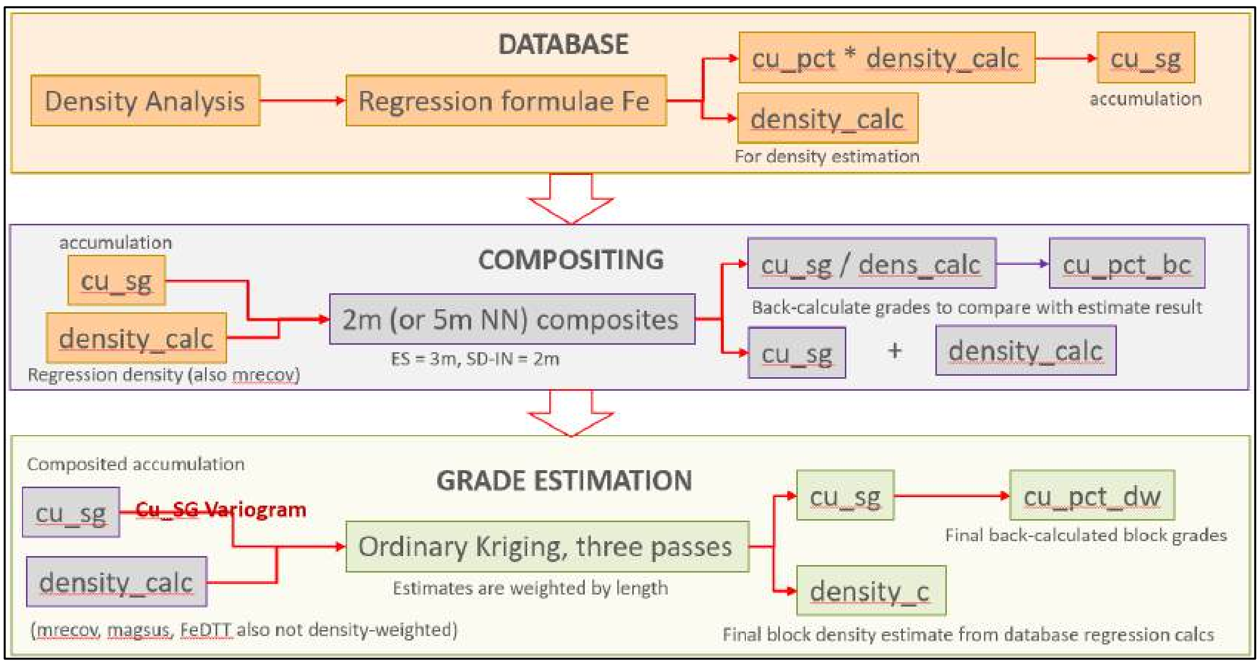
Source: Figure prepared by SLR, 2023.

14.3.7 Search Strategy and Grade Interpolation Parameters

SLR's review of grade continuity and extents for all elements indicated that grade estimation should optimally be done on a per lithology basis using hard boundaries for all lithological domains except Breccia. To ensure continuity between SD and IN, soft boundaries were set up between equivalent lithologies in each area. SLR used Vulcan's DA functionality to match the orientation of the search ellipsoids with that of the lithological units being estimated. DA surfaces were generated as form interpolants in Leapfrog based on manual digitization of grade trends spanning the geological model.

SLR performed density weighted grade estimates for copper, iron, gold, cobalt, pyrite, and sulphur using the methodology illustrated in Figure 14-16. Initially, accumulations of the elements multiplied by density were generated in the sample database. The accumulations were then composited and estimated. Finally, grades were back calculated in the block model after interpolation of the accumulations.

Figure 14-16: Density Weighted Grade Interpolation Methodology: Cu Example



Source: Figure prepared by SLR, 2023. Grade estimation was performed in three passes at 70 m x 70 m x 10 m, 140 m x 140 m x 20 m, and 280 m x 280 m x 40 m in accordance with the variography results (Table 14-18 and Table 14-19).

Passes 1, 2, and 3 required a minimum of three holes, two holes, and two holes with a maximum of four holes, three holes, and three holes, respectively, with up to three two-metre samples allowed per hole. Two additional passes were added to FeDTT estimations to cover the more sparse data distribution in the model space.

A soft boundary was applied between Breccia (102) and all other mineralized lithologies. A soft boundary was also applied for equivalent lithologies between SD and IN so that grades would not change abruptly between the adjoining volumes.

Table 14-18: Summary of Search Parameters: SD-IN - Metal Estimates (Density Weighted) 2023

Parameter	Units	D.Lith	Pass 1	Pass 2	Pass 3	Pass 4 (FeDTT)	Pass 5 (FeDTT)
Search Major	m	102	70	140	280	70 (SD)	420 (SD)
Search Major	m	105, 106, 107, 108, 109, 110, 111, 205, 207, 210, 211	70	140	280	55 (IN)	350 (IN)
Search Semi-Major	m	102	70	140	280	70 (SD)	420 (SD)
Search Semi-Major	m	105, 106, 107, 108, 109, 110, 111, 205, 207, 210, 211	70	140	280	55 (IN)	350 (IN)
Search Minor	m	All	10	20	40	10	60 (SD) & 50 (IN)
Discretization Steps X	Count	All	5	5	5	5	5
Discretization Steps Y	Count	All	5	5	5	5	5
Discretization Steps Z	Count	All	5	5	5	5	5
Min Samples Per Estimate	Count	All	9	6	3	3	3
Max Samples Per Estimate	Count	All	12	9	9	9	9
Min Samples Per DDH	Count	All	3	3	3	3	3
Max Samples Per DDH	Count	All	3	3	3	3	3
Min DDH per Estimate	Count	All	3	2	2	1	2
Max DDH per Estimate	Count	All	4	3	3	3	3

Table 14-19: Massrec, FeDTT and Density Estimates (Unweighted)

Parameter	Units	D_Lith	Pass 1	Pass 2	Pass 3
Search Major	m	102	70	140	280
Search Major	m	105, 107, 205, 207, 210, 211	70	140	280
Search Major	m	106, 108, 109, 110, 111	70	140	280
Search Semi-Major	m	102	70	140	280
Search Semi-Major	m	105, 107, 205, 207, 210, 211	70	140	280
Search Semi-Major	m	106, 108, 109, 110, 111	70	140	280
Search Minor	m	105, 107, 205, 207, 210, 211	10	20	40
Search Minor	m	106, 108, 109, 110, 111	10	20	40
Discretization Steps X	Count	All	5	5	5
Discretization Steps Y	Count	All	5	5	5
Discretization Steps Z	Count	All	5	5	5
Min Samples Per Estimate	Count	All	3	3	3
Max Samples Per Estimate	Count	All	3	3	3
Min Samples Per DDH	Count	All	3	2	2
Max Samples Per DDH	Count	All	4	3	3

14.3.8 Bulk Density Regression Analyses and Application in the Block Model

The Capstone density database, provided in October 2021 to SLR, includes 7,039 samples, with 5,463 deemed usable for the Mineral Resource estimate. Due to uneven coverage of these samples across the resource, especially in oxide material, Capstone applied regression formulae based on iron content to estimate densities to produce density curves for each ore type. For more details, refer to Section 13.1.3.

14.3.9 Block Models

SLR initially constructed a sub-blocked model with 6 m x 6 m x 5 m parent blocks and 3 m x 3 m x 2.5 m sub-blocks to estimate all variables using density weighting and DA, and then regularized the model to a final 12 m x 12 m x 15 m model for Whittle analyses and final reporting. In the sub-blocked model, the approximately three metre resolution assisted in accurately defining barren dyke material in the grade estimation before reblocking. The sub-blocked model was split on lithology. The block model parameters for sub-blocked and regular models are listed in Table 14-20 through Table 14-23, respectively.

The new sub-blocked model framework intended to use a compressed model to make the model size as small as possible. However, throughout the course of model development, SLR noted that model size began to increase past 40 GB. At that point, SLR determined that a change to a much larger parent block size would require too much time in re-running, validation, and other changes relative to the processing time involved for the model. SLR deleted temporary and non-essential variables in a Geology copy of the sub-blocked model for visual review and validation.

Table 14-20: Sub-Blocked Model Specifications

Block Model Origin and Orientation	
Model name:	bm_sdin_rsc
Format:	extended
Structure:	non-regular
Compressed:	no
Smooth:	no
Number of blocks	75498542
Number of variables:	205
Number of schemas:	2
Origin:	398325.000000 7071300.000000 199.000000
Bearing/Dip/Plunge:	90.000000 0.000000 0.000000
Created on	Thu Mar 30 08:33:29 2023
Last modified on	Thu Mar 30 08:33:29 2023
Block Dimensions	
Schema <parent>	
Offset minimum:	0.000000 0.000000 0.000000
maximum:	2544.000000 5568.000000 1110.000000
Blocks minimum:	6.000000 6.000000 5.000000
maximum:	6.000000 6.000000 5.000000
No of blocks:	424 928 222
Schema <sub-block>	
Offset minimum:	0.000000 0.000000 0.000000
maximum:	2544.000000 5568.000000 1110.000000
Blocks minimum:	3.000000 3.000000 2.500000
maximum:	6.000000 6.000000 5.000000
No of blocks:	848 1856 444

Table 14-21: Sub-Blocked Model: Key Variables

Variables	Default	Type	Description
rock_0_1	-9	double	topography % (below topo)
d_lith	-9	integer	flagged from solids
d_class_eng	-9	integer	flagged from solids exported from regular model
d_area	-9	integer	flagged from solids
d_magsus	-9	double	flagged from solids
d_redox	-9	integer	flagged from solids
density_c	-9	double	Density gr/cm³
tonnes_block	-9	double	Tonnes
magsus_g	-9	double	magsus estimated
mrecov_pct	-9	double	mass recovery estimated (No COG (Cut Off Grade) and No constraints by lithology)
massrec	-9	double	mass recovery (after 1,500/18,500 COG constraints)
cu_sg	-9	double	Copper accumulation - estimated
fe_sg	-9	double	Iron accumulation - estimated
au_sg	-9	double	Gold accumulation - estimated
co_sg	-9	double	Cobalt accumulation - estimated
s_sg	-9	double	Sulphur accumulation - estimated
py_sg	-9	double	Pyrite accumulation – estimated
cu_pct_dw	-9	double	Copper % (official)
fe_pct_dw	-9	double	Iron % (official)
au_ppm_dw	-9	double	Gold g/t (official)
co_ppm_dw	-9	double	Cobalt ppm (official)
s_pct_dw	-9	double	Sulphur % (official)
py_pct_dw	-9	double	Pyrite % (official)
fedtt_pct	-9	double	raw Fe in Conc Grade
fe_inconc_grade_final -9	-9	double	Final Fe in Conc after routing upgrades
au_sg_5nn	-9	-	double nn_sg with 5 m comp
co_sg_5nn	-9	-	double nn_sg with 5 m comp
cu_sg_5nn	-9	-	double nn_sg with 5 m comp
fe_sg_5nn	-9	-	double nn_sg with 5 m comp
s_sg_5nn	-9	-	double nn_sg with 5 m comp
py_sg_5nn	-9	-	double nn_sg with 5 m comp
da_bearing	0	double	dynamic anisotropy
da_plunge	0	double	dynamic anisotropy
da_dip	0	double	dynamic anisotropy
da_dist_maj	0	double	dynamic anisotropy
da_dist_semi	0	double	dynamic anisotropy
da_dist_min	0	double	dynamic anisotropy

Table 14-22: Regularised Model: Specifications

Block Model Origin and Orientation	
Model name:	bm_sdin_rsc_reg12x12x12m_eng
Format:	extended
Structure:	regular
Compressed:	no
Smooth:	no
Number of blocks	7279232
Number of variables:	38

Block Model Origin and Orientation	
Number of schemas:	1
Origin:	398325.000000 7071300.000000 199.000000
Bearing/Dip/Plunge:	90.000000 0.000000 0.000000
Created on	Thu Apr 20 20:15:02 2023
Last modified on	Thu Apr 20 20:15:02 2023
Block Dimensions	
Schema <parent>	
Offset minimum:	0.000000 0.000000 0.000000
maximum:	2544.000000 5568.000000 1110.000000
Blocks minimum:	12.000000 12.000000 15.000000
maximum:	12.000000 12.000000 15.000000
No of blocks:	212 464 74

Table 14-23: Regularised Model: Key Variables

Variables	Default	Type	Description
rock_0_1	0	double	topography % (below topo)
d_lith	0	integer	Lithology
d_class	0	integer	Resource classification
d_potential	0	integer	Internal class by redox and area
d_area	0	integer	Area
d_magsus	0	double	majority d_magsus
d_redox	0	integer	majority d_redox
d_slope	0	integer	majority d_slope
density_c	0	double	Density gr/cm³
tonnes_block	0	double	Tonnes
cu_pct_dw	0	double	Copper % (official)
fe_pct_dw	0	double	Iron % (official)
au_ppm_dw	0	double	Gold g/t (official)
co_ppm_dw	0	double	Cobalt ppm (official)
s_pct_dw	0	double	Sulphur % (official)
py_pct_dw	0	double	Pyrite % (official)
magsus_g	0	double	magsus
rmcu2	0	double	Estimated Cu recovery (%)
rmau2	0	double	Estimated Au recovery (%)
Ley_au_conc_m	0	double	Estimated Au grade in Cu con (in g/t)
aupay	0	double	Estimated proportion of payable Au in Cu con (as %)
fedtt_pct	0	double	Estimated raw Fe grade in Conc Grade (%)
fe_dtt_m³	0	double	Estimated Final Fe grade in Conc after routing upgrades (%)
massrec	0	double	Estimated raw Fe mass recovery (%)
rpfe3	0	double	Estimated Final Fe mass recovery (%)
conc_cu3	0	Double	Estimated mass of Cu Conc (tonnes)
conc_fe3	0	Double	Estimated mass of Fe Conc (tonnes)
proc3	0	Double	Estimated Final processing cost in US$/t
nsr3	0	double	NSR (official incl Cu, Fe and Au) in US$/t

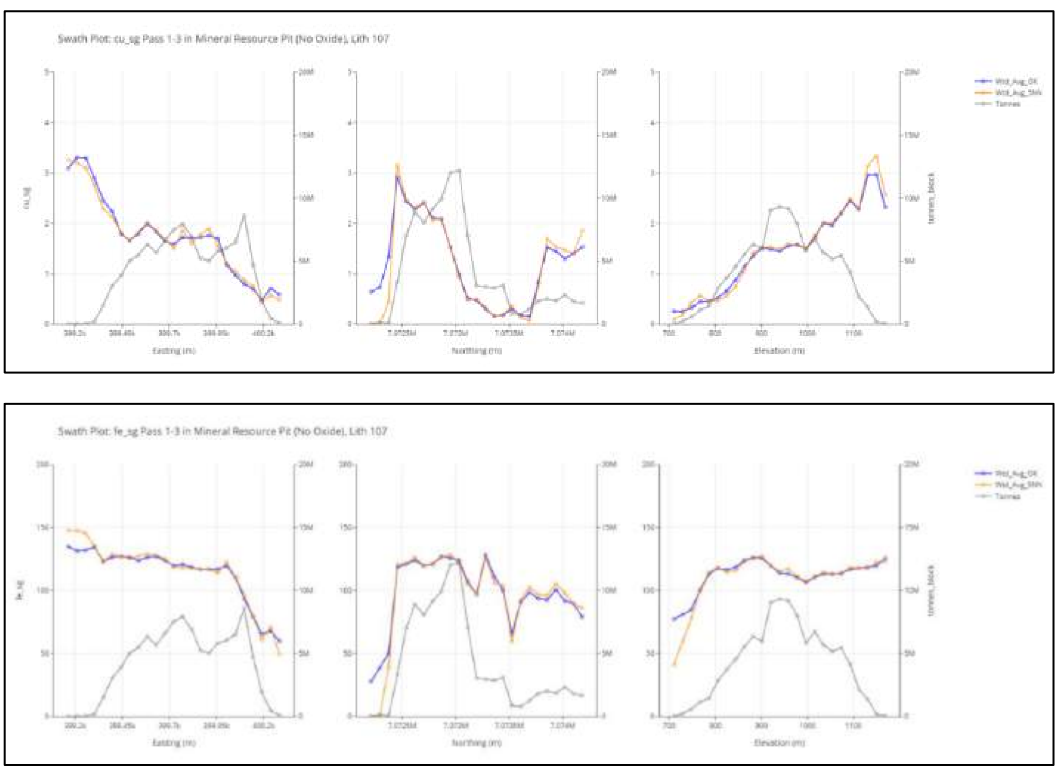
14.3.10 Block Model Validation

SLR validated the block models using summary statistical comparisons to composites, nearest neighbour (NN) estimates, and non-density weighting (DW) ordinary kriged (OK) estimates to compare with the DW results. DA trends were visually validated in Vulcan against input surfaces and drilled grades. NSR and CuEq block model calculations were validated both in the original Excel COG work sheets and in Python-adapted versions of the Vulcan scripts used to calculate them. Massrec cut-offs by lithology were manually checked in Vulcan. Regular model block grade values were validated using accumulations in the sub-blocked model (*_xt), which were then back calculated and compared with the original regularized grades. Block grade and composite grade statistics were compared by rock type within estimated classified material in the Whittle shell (Table 14-24).

SLR found that visual checks of composite and drilled grades against block grades showed very good spatial control with little smearing, especially in areas of denser drilling. Local HYR was applied to control occasional over-extrapolation. Swath plots versus NN estimates were overall very well correlated. Example swath plots are shown in Figure 14-17, and example visual validations are shown in Figure 14-18 through Figure 14-20.

In the QP's opinion, the validation results show that the grade estimates are reasonable and adequate for their use in resource estimation and mine planning.

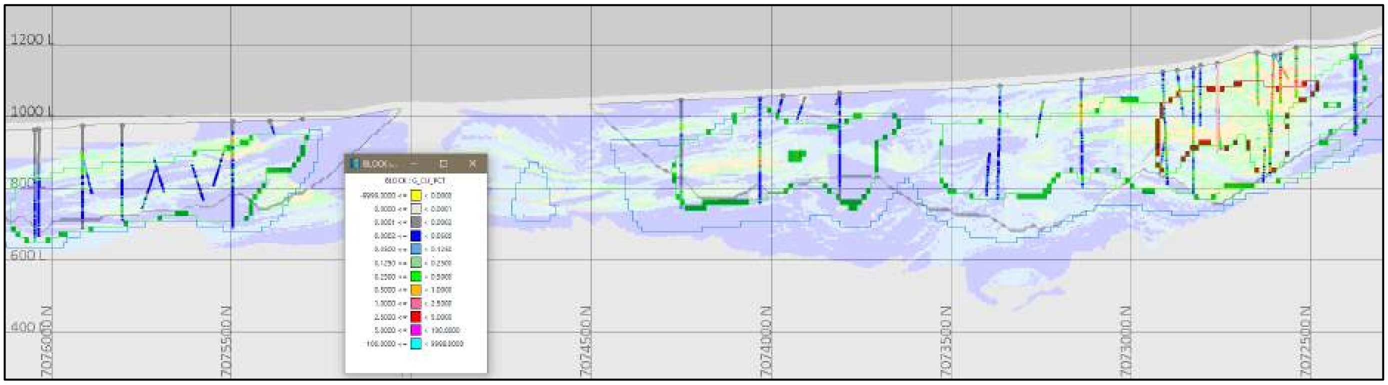
Figure 14-17: Example Swath Plots: SD Cu_SG and Fe_SG, Class 1 and 2 Material



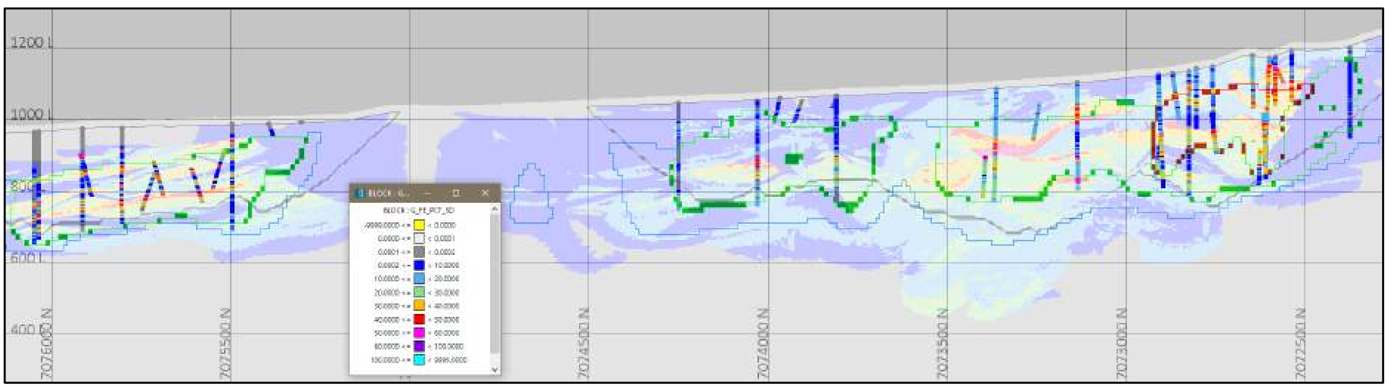
Source: Figure prepared by SLR, 2023.

Figure 14-18: Example Validation Cross Sections: 400,025E, SD+IN

Cu (%)



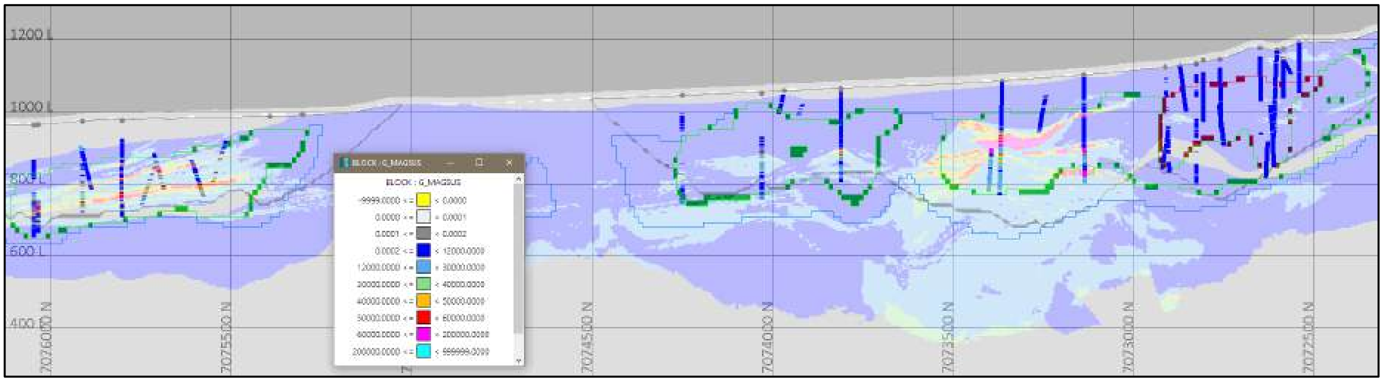
Fe (%)



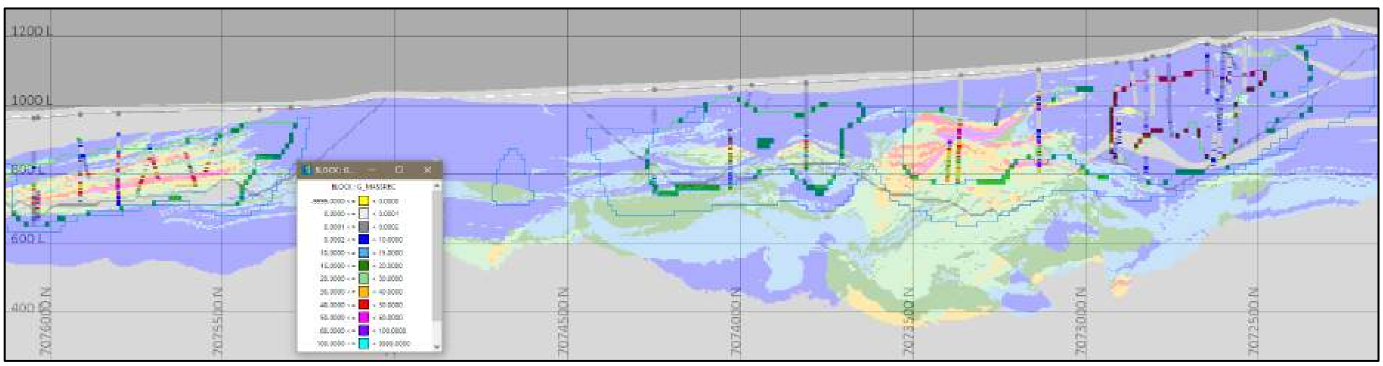
Source: Figure prepared by SLR, 2023. Section view is looking east

Figure 14-19: Example Validation Cross Sections II: 400,025E, SD+IN

Magsus (%)

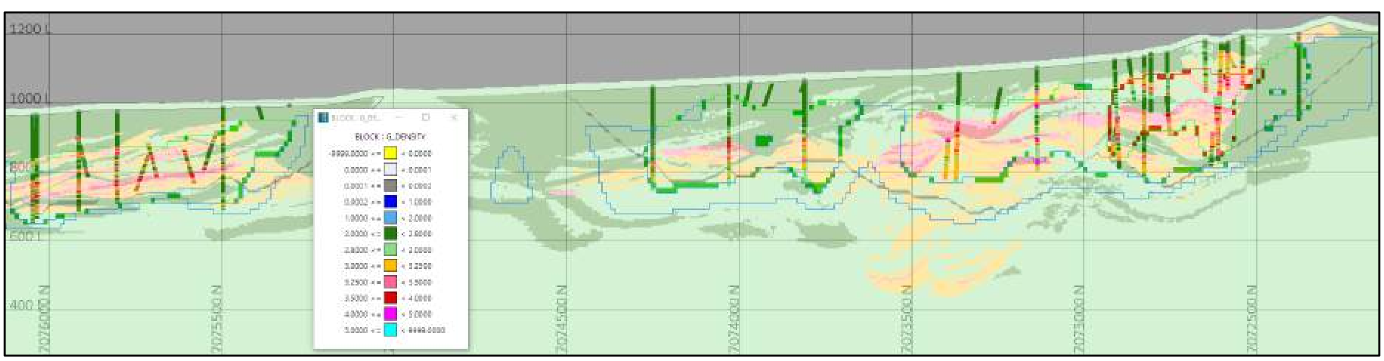


Massrec (%)



Source: Figure prepared by SLR, 2023. Section view is looking east.

Figure 14-20: Example Cross Section, SD Density, 400,025E



Source: Figure prepared by SLR, 2023. Section view is looking east.

Table 14-24: SD-IN Block and Composite Statistics Inside Mineral Resource Pit Shell, Class 1 and 2

Element	Lith Code	Count Comps	Count Blocks	Min Comps	Min Blocks	Max Comps	Max Blocks	Mean Comps	Mean Blocks	Mean Diff	Mean Diff Pct	CV Comps	CV Blocks	CV Diff Pct
Au (g/t)	102	545	26,214	0.000	0.000	0.250	0.187	0.040	0.050	0.010	20.000	1.04	0.66	-57.58
Au (g/t)	105	7,019	965,012	0.000	0.000	0.700	0.322	0.010	0.010	0.000	0.000	2.31	1.41	-63.83
Au (g/t)	106	559	145,711	0.000	0.000	0.250	0.150	0.010	0.010	0.000	0.000	2.79	1.65	-69.09
Au (g/t)	107	4,848	356,735	0.000	0.000	0.700	0.564	0.070	0.060	-0.010	-16.667	1.09	0.76	-43.42
Au (g/t)	108	2,325	215,390	0.000	0.000	0.389	0.204	0.020	0.020	0.000	0.000	1.97	1.15	-71.30
Au (g/t)	109	6,014	722,502	0.000	0.000	0.660	0.324	0.050	0.040	-0.010	-25.000	1.25	0.85	-47.06
Au (g/t)	111	2,036	238,526	0.000	0.000	0.350	0.246	0.010	0.010	0.000	0.000	2.52	1.08	-133.33
Au (g/t)	205	193	42,904	0.000	0.000	0.159	0.102	0.010	0.010	0.000	0.000	2.18	1.41	-54.61
Au (g/t)	207	2,028	259,453	0.000	0.000	0.700	0.247	0.010	0.020	0.010	50.000	2.55	1.26	-102.38
Au (g/t)	211	11	956	0.000	0.003	0.062	0.068	0.020	0.010	-0.010	-100.000	1.03	0.70	-47.14
Co (ppm)	102	545	26,214	8.00	0.00	650.00	537.62	103.95	132.62	28.67	21.62	1.02	0.70	-45.71
Co (ppm)	105	7,019	965,012	0.00	0.00	1350.00	911.13	59.80	55.33	-4.47	-8.08	1.41	0.91	-54.95
Co (ppm)	106	559	145,711	0.00	0.00	600.00	425.61	55.31	47.67	-7.64	-16.03	1.46	0.79	-84.81
Co (ppm)	107	4,848	356,735	0.00	0.00	2000.00	1551.46	270.10	269.19	-0.91	-0.34	0.88	0.54	-62.96
Co (ppm)	108	2,325	215,390	0.00	0.00	1500.00	971.90	89.61	93.93	4.32	4.60	1.52	0.83	-83.13
Co (ppm)	109	6,014	722,502	0.00	0.00	2100.00	1569.75	271.46	259.28	-12.18	-4.70	0.88	0.57	-54.39
Co (ppm)	111	2,036	238,526	0.00	0.00	1080.00	824.37	73.43	76.35	2.92	3.82	1.40	0.77	-81.82
Co (ppm)	205	193	42,904	0.00	0.00	764.00	576.86	62.58	76.59	14.01	18.29	1.84	0.75	-145.33
Co (ppm)	207	2,028	259,453	0.00	0.00	2000.00	1557.19	178.56	198.30	19.74	9.95	1.22	0.65	-87.69
Co (ppm)	211	11	956	7.00	20.24	105.00	110.14	41.82	60.20	18.38	30.53	0.71	0.27	-162.96
Cu (%)	102	545	26,214	0.002	0.000	2.000	1.438	0.350	0.310	-0.040	-12.903	0.96	0.80	-20.00
Cu (%)	105	7,019	965,012	0.000	0.000	2.000	1.330	0.090	0.070	-0.020	-28.571	1.67	1.02	-63.73
Cu (%)	106	559	145,711	0.000	0.000	1.500	1.024	0.090	0.080	-0.010	-12.500	1.87	1.31	-42.75
Cu (%)	107	4,848	356,735	0.000	0.000	4.500	2.902	0.510	0.430	-0.080	-18.605	1.04	0.75	-38.67
Cu (%)	108	2,325	215,390	0.000	0.000	3.500	1.378	0.170	0.140	-0.030	-21.429	1.71	0.99	-72.73
Cu (%)	109	6,014	722,502	0.000	0.000	3.500	2.431	0.380	0.270	-0.110	-40.741	1.25	0.91	-37.36
Cu (%)	111	2,036	238,526	0.000	0.000	2.500	1.599	0.070	0.080	0.010	12.500	2.06	0.93	-121.51
Cu (%)	205	193	42,904	0.000	0.000	1.715	0.847	0.070	0.090	0.020	22.222	2.11	1.20	-75.83
Cu (%)	207	2,028	259,453	0.000	0.000	3.100	1.763	0.120	0.120	0.000	0.000	2.14	1.16	-84.48
Cu (%)	211	11	956	0.006	0.000	0.276	0.465	0.090	0.080	-0.010	-12.500	0.98	0.83	-18.07
Fe (%)	102	545	26,214	2.39	0.00	54.40	51.86	21.60	21.19	-0.41	-1.93	0.52	0.42	-23.81
Fe (%)	105	7,019	965,012	0.00	0.00	54.30	48.15	10.68	10.80	0.12	1.11	0.47	0.40	-17.50
Fe (%)	106	559	145,711	0.00	0.00	40.00	33.54	10.45	10.37	-0.08	-0.77	0.38	0.29	-31.03
Fe (%)	107	4,848	356,735	0.00	0.00	65.00	61.03	32.20	31.66	-0.54	-1.71	0.40	0.29	-37.93
Fe (%)	108	2,325	215,390	0.00	0.00	61.20	48.18	13.56	13.86	0.30	2.16	0.71	0.45	-57.78

Element	Lith Code	Count Comps	Count Blocks	Min Comps	Min Blocks	Max Comps	Max Blocks	Mean Comps	Mean Blocks	Mean Diff	Mean Diff Pct	CV Comps	CV Blocks	CV Diff Pct
Fe (%)	109	6,014	722,502	0.00	0.00	65.00	60.13	26.72	25.34	-1.38	-5.45	0.49	0.34	-44.12
Fe (%)	111	2,036	238,526	0.00	0.00	55.00	53.67	14.36	15.44	1.08	6.99	0.66	0.47	-40.43
Fe (%)	205	193	42,904	0.00	0.00	45.20	36.26	8.93	10.14	1.21	11.93	0.76	0.49	-55.10
Fe (%)	207	2,028	259,453	0.00	0.79	62.50	56.09	21.85	23.44	1.59	6.78	0.66	0.39	-69.23
Fe (%)	211	11	956	4.78	7.04	13.70	16.66	9.13	9.91	0.78	7.87	0.29	0.20	-45.00
Py (%)	102	545	26,214	0.00	0.00	17.84	12.66	1.65	1.80	0.15	8.33	1.37	0.76	-80.26
Py (%)	105	7,019	965,012	-0.24	0.00	18.33	9.29	0.38	0.28	-0.10	-35.71	2.65	1.34	-97.76
Py (%)	106	559	145,711	0.00	0.00	13.43	8.89	0.25	0.20	-0.05	-25.00	3.64	1.98	-83.84
Py (%)	107	4,848	356,735	0.00	0.00	25.84	15.85	3.31	3.30	-0.01	-0.30	0.93	0.56	-66.07
Py (%)	108	2,325	215,390	0.00	0.00	18.68	13.97	1.20	1.28	0.08	6.25	1.67	0.86	-94.19
Py (%)	109	6,014	722,502	0.00	0.00	23.74	16.84	3.81	3.88	0.07	1.80	0.80	0.50	-60.00
Py (%)	111	2,036	238,526	0.00	0.00	12.71	9.89	0.59	0.64	0.05	7.81	1.87	1.08	-73.15
Py (%)	205	193	42,904	0.00	0.00	9.52	7.42	0.64	0.72	0.08	11.11	2.45	1.24	-97.58
Py (%)	207	2,028	259,453	0.00	0.00	25.58	12.76	3.31	3.64	0.33	9.07	1.06	0.65	-63.08
Py (%)	211	11	956	0.07	0.14	2.73	4.74	0.70	1.31	0.61	46.56	1.15	0.62	-85.48
S (%)	102	545	26,214	0.01	0.00	10.00	7.46	1.20	1.30	0.10	7.69	1.10	0.62	-77.42
S (%)	105	7,019	965,012	0.00	0.00	10.00	7.15	0.28	0.21	-0.07	-33.33	2.15	1.21	-77.69
S (%)	106	559	145,711	0.00	0.00	7.59	4.34	0.20	0.17	-0.03	-17.65	2.92	1.71	-70.76
S (%)	107	4,848	356,735	0.00	0.00	15.00	8.50	2.28	2.18	-0.10	-4.59	0.79	0.52	-51.92
S (%)	108	2,325	215,390	0.00	0.00	10.00	7.58	0.81	0.83	0.02	2.41	1.48	0.78	-89.74
S (%)	109	6,014	722,502	0.00	0.00	12.85	9.22	2.42	2.35	-0.07	-2.98	0.73	0.47	-55.32
S (%)	111	2,036	238,526	0.00	0.00	6.85	5.13	0.39	0.42	0.03	7.14	1.71	0.96	-78.13
S (%)	205	193	42,904	0.00	0.00	5.36	4.42	0.40	0.45	0.05	11.11	2.22	1.18	-88.14
S (%)	207	2,028	259,453	0.00	0.01	13.70	7.11	1.87	2.06	0.19	9.22	1.02	0.63	-61.90
S (%)	211	11	956	0.07	0.14	1.74	2.55	0.47	0.79	0.32	40.51	1.04	0.53	-96.23

14.3.11 Classification

Definitions for resource categories used in this report are consistent with those defined by CIM (2014) and adopted by NI 43-101. In the CIM classification, a Mineral Resource is defined as "a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction". Mineral Resources are classified into Measured, Indicated, and Inferred categories. Economic value (using NSR calculations) was only assigned to blocks within the mixed and primary zones (Section 14.3.12). Thus, mineral resources at SD-IN only include material from within those two zones. Oxides are excluded from the resources at SD-IN.

SLR used a scripting methodology to set block categories in the 12 m x 12 m x 15 m regularized block model by interpolating a three-hole distance variable into the regular block model. An average three-hole drill spacing for each block was then calculated from the average distance in a script, and the block categories one to three were set based on drill spacing distances. SLR then reviewed the initial results and performed an iterative subsequent clean-up which also excluded holes by indexed name.

In general accord with the variogram results, the classification results are nominally 70 m and 50 m three-hole drill spacing for Measured material for SD and IN, respectively. For Indicated material, SLR chose a 130 m and 100 m for andesite material. Indicated Mineral Resources at IN were set to a three-hole spacing of 115 m, with andesite set to 100 m (Figure 14-21). At the outboard edges of the drilling, radii are applied which limit Mineral Resource class extrapolation to nominally half the two-hole distance from the last hole for Measured, Indicated, and Inferred, respectively.

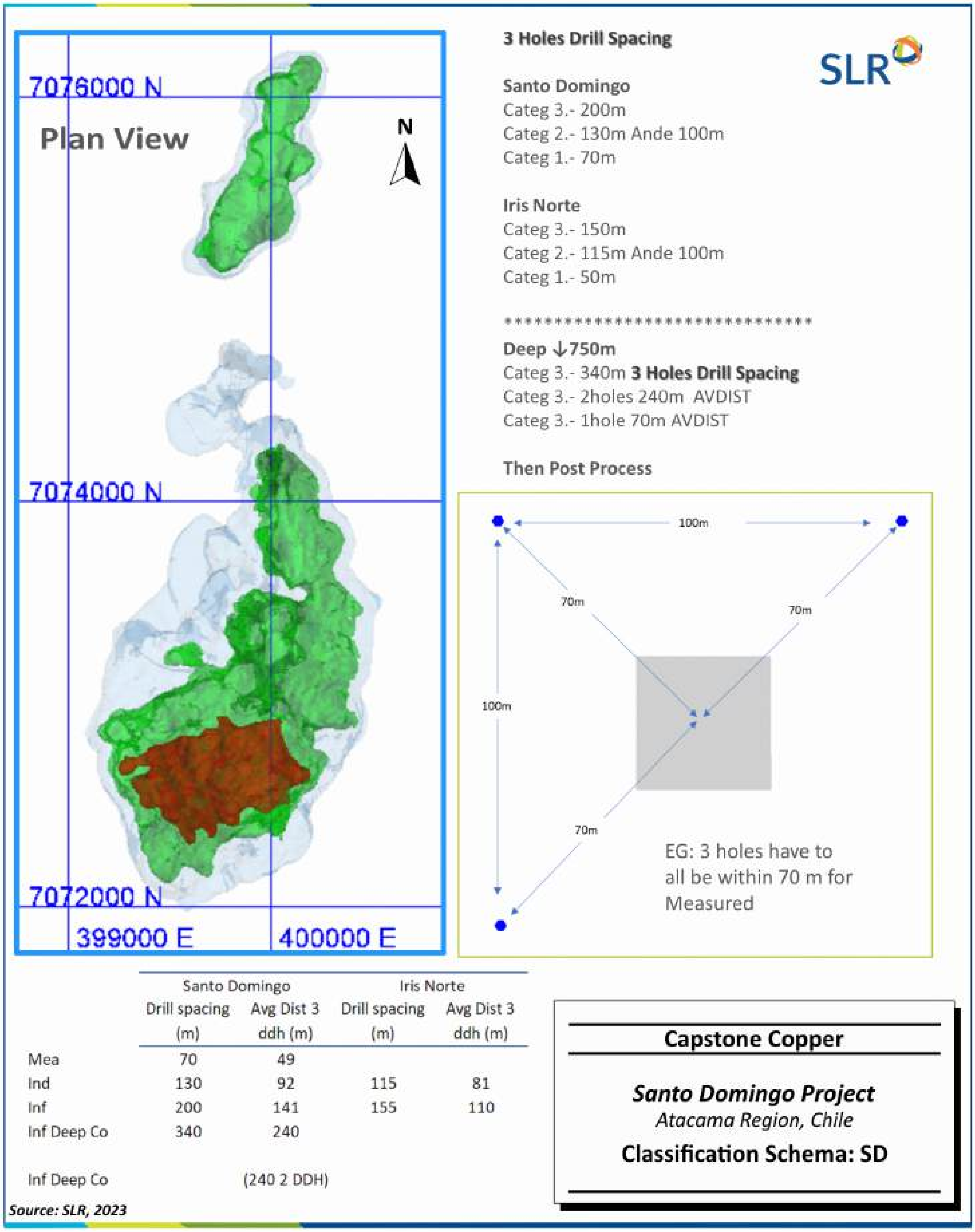
Special considerations were made for a deep, thick and continuous volume of material with persistently high cobalt and copper in Lith 109 at the bottom of the SD model, supported by holes 4a3-21-496DD, 4a3-21-489DD, 4a3-21-490DD, 4a3-21-491DD, and 4a3-21-492DD (Figure 14-22). The drilling in this volume is nominally 190 m to 250 m apart, while mineralization is in the order of 200 m thick. SLR built a solid volume around the holes which reclassified blocks with up to 340 m thee-hole spacing to Inferred Mineral Resource. SLR recommends further drilling in and around this volume to support the classification and potentially expand the mineralized tonnage.

Vulcan's grade shell algorithm was then used to generate solids to flag the regularized model for Mineral Resource classification. A vertical cross section of the classification schema at SD is shown in Figure 14-23. The new classification is more rules-based and requires less manual construction. It tends to add more Indicated Mineral Resource material and extends Inferred Mineral Resource material between the two previous SD and IN Whittle shells.

SLR notes that classification could be upgraded from Inferred Mineral Resource in several pockets around the main SD orebody (Figure 14-24), as well as in peripheral areas outboard of regular drill patterns.

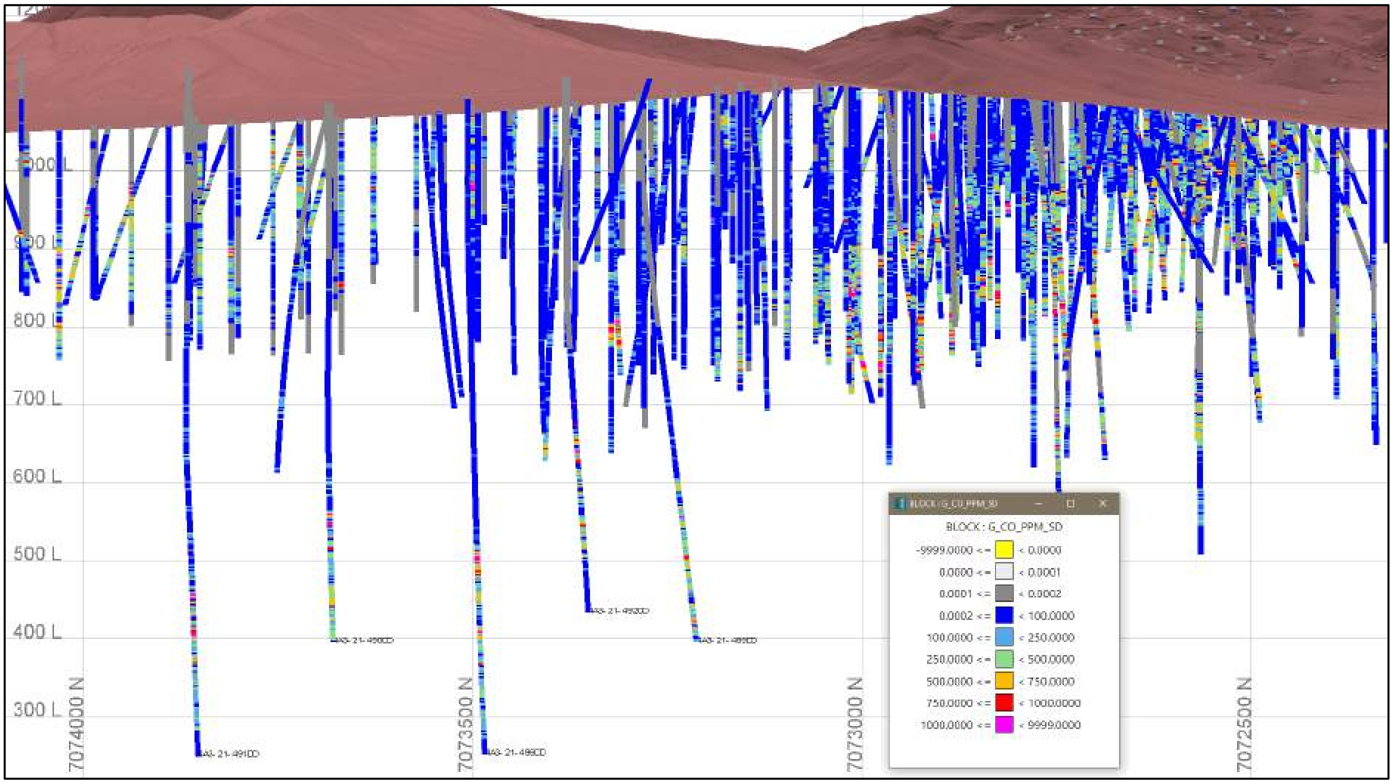
The QP considers that the methodology used for resource classification at SD and IN is appropriate for the type of deposit and the spatial distribution of the drill hole data.

Figure 14-21: Classification Schema: SD



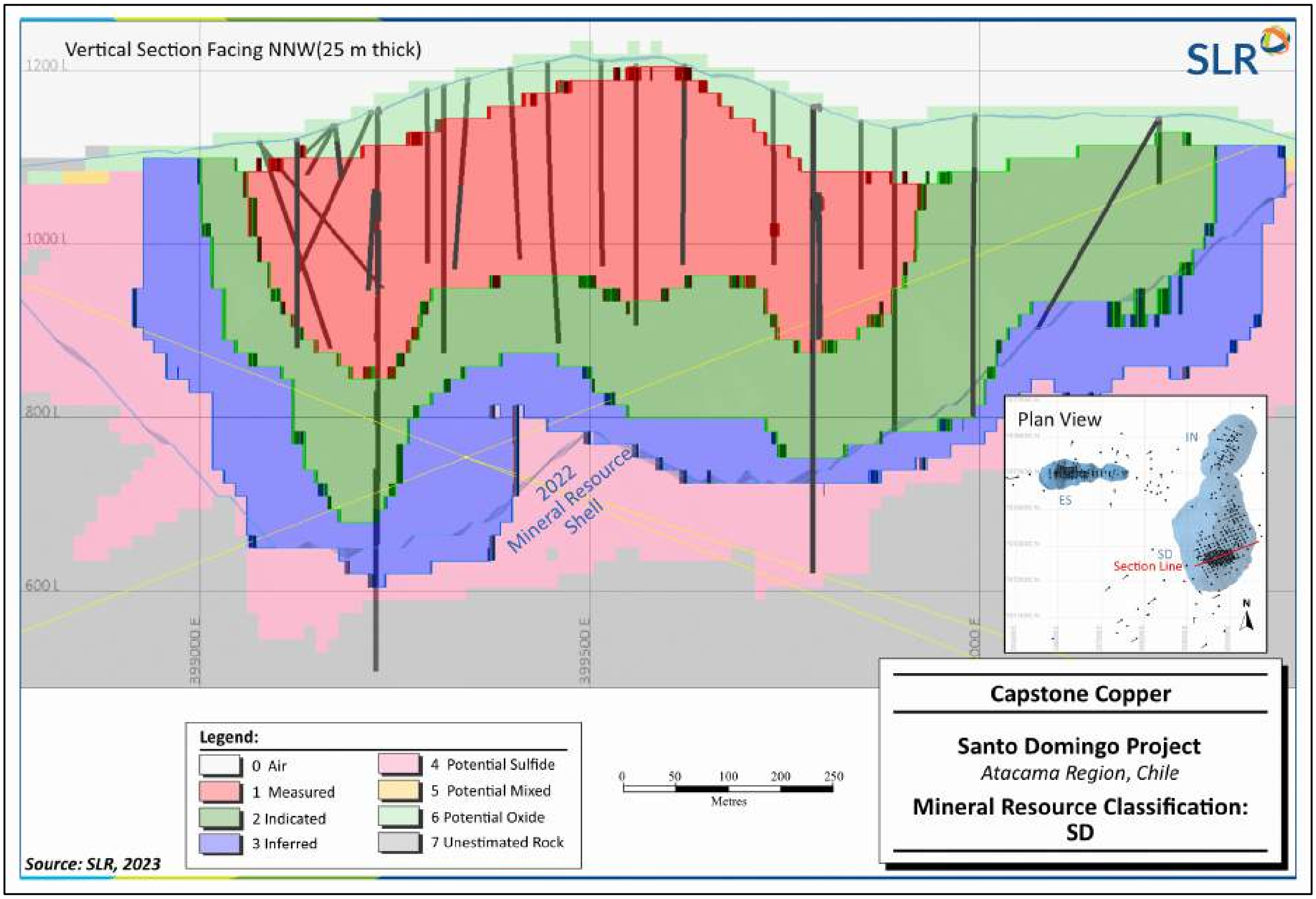
Source: Figure prepared by SLR, 2023.

Figure 14-22: SD: Deep Cobalt Rich Drilling (Thick Section Facing East)



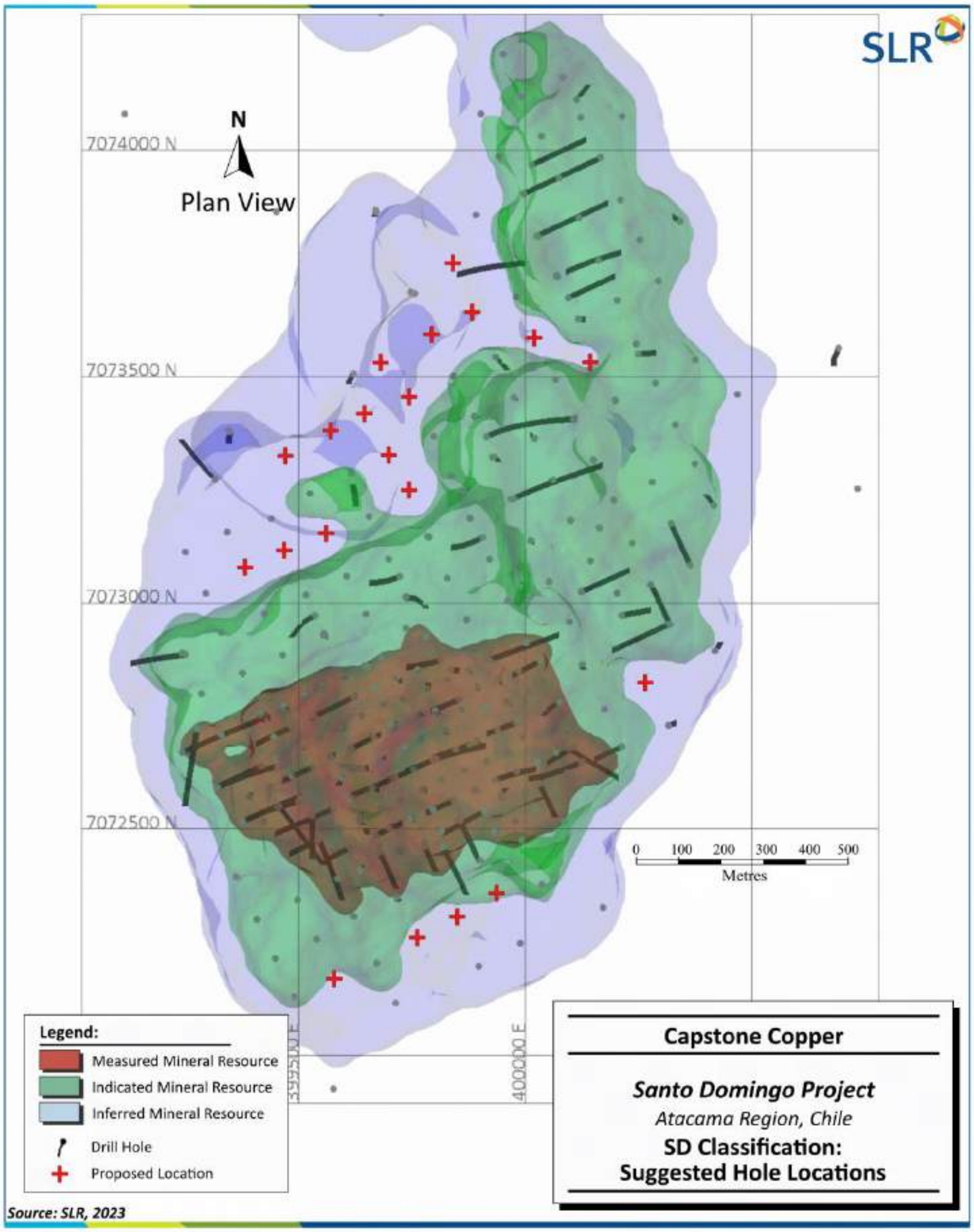
Source: Figure prepared by SLR, 2023.

Figure 14-23: Mineral Resource Classification: SD



Source: Figure prepared by SLR, 2023.

Figure 14-24: SD Classification: Suggested Hole Locations



Source: Figure prepared by SLR, 2023.

14.3.12 Net Smelter Return Cut-off Value and Whittle Parameters

SD and IN are polymetallic deposits in which revenue will be generated from the sale of copper-sulphide and iron-oxide concentrates. In addition, the copper concentrate contains some payable quantities of gold credits. At both SD and IN only mixed and primary (sulphide) materials were considered in the estimation of the resources. Oxides are not considered part of the resource at this stage of the project. The Mineral Resources are reported utilizing an NSR, based on metallurgical test results and current long term metal pricing supplied by Capstone. The NSR ($/t) was assigned to each block by combining the contribution from copper, gold, and iron. Cobalt and Sulphur were excluded from the NSR determination.

Table 14-25 summarizes the parameters used to calculate the NSR estimate. Metal prices used by Capstone for revenue estimates are $4.10/lb for copper, $1,600/oz for gold, and variable prices per dmt CFR Chile for iron.

Capstone performed internal studies resulting from the upgraded modelled variable FeDTT which led to a fairly sophisticated schema to determine routing codes for three different mill processes associated with different anticipated upgrades to iron concentrate material, anticipated Fe price, NSR value, and processing cost for each block. The changes required by Capstone for the 2024 NSR value and processing cost calculations include:

- Four new routing codes for Fe concentrate, based on the new modelled FeDTT values.
- New Cu price of US$4.10 per pound, and three new Fe prices which reflect the expected Fe grades attained in the final concentrate.
- Two new processing costs for Fe concentrate types.
- Mass recovery factored according to the process methodology.

NSR calculations are only run in the regularized model. This is to preserve the integrity between routing codes, metal prices, and mass recovery factors where reblocking may result in inconsistent final values.

14.3.12.1 NSR Cut-off Value

The NSR cut-off value incorporates all operating costs except mining, using the operating costs provided by Capstone (Table 14-25). When calculating the average NSR value, the costs for the iron ores were weighed by the proportion of blocks that pay for iron processing. The resulting internal (or mill) cut-off value is US$9.85/t.

Table 14-25: Net Smelter Return Parameters – SD-IN

Parameters	Units	Value	Note
Metal Price			
Copper	US$/lb	4.10	-
Gold	US$/oz	1,600	-
Iron (CFR China; 62-65%/65-67%/>67% Fe)	US$/dmt	94.75 – 129.77 - 140.26	-
Recovery to Concentrate			
Copper (RecCu)	%	-0.011*Co+11.967*Cu-13.47*Cu^2+5.502×Cu^3-0.92*Cu^4-68.714*0.2+94.068	Mixed Zone
		-0.011*Co+11.967*Cu-13.47*Cu^2+5.502×Cu^3-0.92*Cu^4-68.714*0.05+94.068	Sulphide zone
Gold (RecAu)	g/ton	-142.598*Au+44.752*Cu-10.095*Cu^2-46.68*0.2+51	Mixed Zone
		-142.598*Au+44.752*Cu-10.095*Cu^2-46.68*0.05+52	Sulphide zone
Mass Recovery for Magnetite Concentrate	%	Variable, based on magnetic susceptibility	-
Copper Concentrate Grade			
Copper (GCCu)	%	-1.701*Cu+7.767*0.2+(Cu/S)*23.362+22.33	Mixed Zone
		-1.701*Cu+7.767*0.05+(Cu/S)*23.362+22.34	Sulphide zone
Gold (GCAu) (Note: Au and CuT correspond to the grades of gold and copper in the Block model)	g/ton	(Rec Au∗Au)/((Rec Cu∗%CuT)/GCCu)	Mixed Zone
		(Rec Au∗Au)/((Rec Cu∗%CuT)/GCCu)	Sulphide zone
Moisture Content	%	8	-
Magnetite Concentrate Grade			
Iron Grade	%Fe	Variable	-
Moisture Content	%	8	-
Smelter Payables			
Copper Pay Factor	%	96.5	1
Gold Pay Factor	%	90-96.5	2
Off-Site Costs			
Copper Conc. Transport Cost	$/wmt Cu	54.60	3
Magnetite Conc. Transport Cost	$/wmt Fe	23.69	4
Cu Concentrate Treatment	$/dmt conc.	80.0	-
Copper Refining Charge	$/lb Cu	0.08	-
Gold Refining Charge	$/oz Au	5.0	-
Royalties	%	2.0	-

Notes: **1.** Excludes Cu unit deduction at 1%. Excludes transport losses. Cu concentrate considered clean, per Capstone guidance. **2.** Excludes Au deduction. Depends on the gold grade in the concentrate. **3.** No penalties applied; concentrate considered clean per Capstone; excludes insurance cost referenced at 0.05%. **4.** No penalties applied for deleterious elements (S, P, SiO2, etc.) grades above rejection limits. Excludes ocean freight (price basis is CFR Chile, which assumes the purchaser is based in Chile). **5.** No penalties applied (early-stage work). Excludes insurance cost referenced at 0.05%.

14.3.12.2 Pit Optimization Parameters

Pit optimization was performed based on the NSR of each block in the resource model. The open pit cut-off value utilized for the SD-IN Mineral Resource estimation was 9.85 US$/t feed.

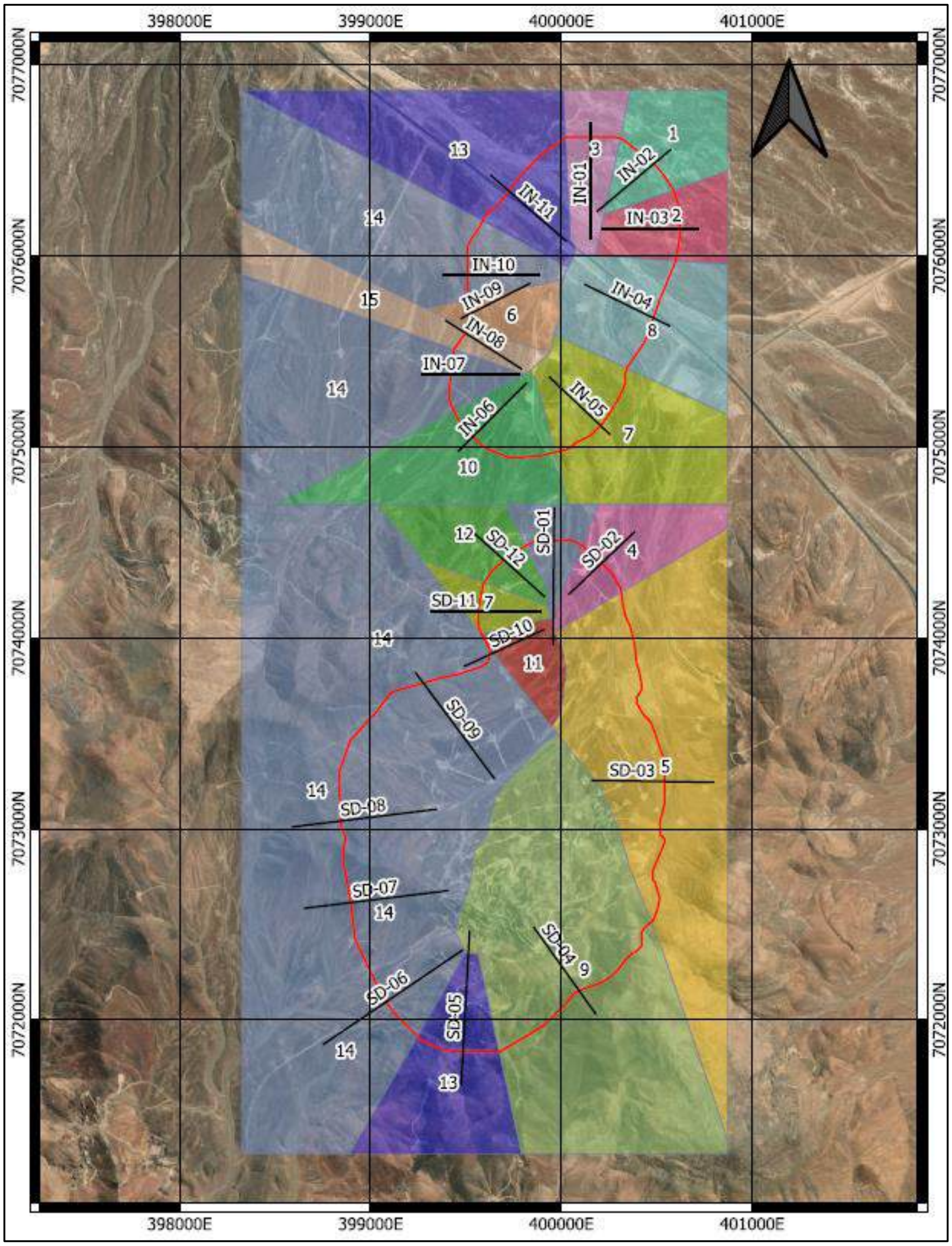
The pit optimization was carried out in Whittle software on the regularized NSR resource model exported from the Vulcan mining software. The Whittle model with a block size of 12 m x 12 m x 15 m contains material types (oxide, mixed, and sulphide mineralization and barren rock). In-situ NSR is the net value of metals contained in a concentrate produced from an ore block after smelting and refining. The optimizer uses the LG algorithm to determine the economic pit limits based on process costs, tailings, treatment, transportation, recoveries, payability, etc. (Table 14-26) and estimates for pit slope angles.

Pit overall slope angles (OSA) vary by structural domains from 36° to 48° as per geotechnical recommendations. Pit slope angles are summarized in Table 14-26 and illustrated in Figure 14-25.

Table 14-26: Pit Slope Angles, Santo Domingo, and Iris Norte

Sector	Units	Overall slope angle
1	Degree	36.3
2	Degree	38.6
3	Degree	40.3
4	Degree	45
5	Degree	40
6	Degree	45.6
7	Degree	39.2
8	Degree	46.2
9	Degree	45.3
10	Degree	41.8
11	Degree	45.6
12	Degree	46.5
13	Degree	47.3
14	Degree	47.9
15	Degree	43.2

Figure 14-25: Geotechnical Sectors, SD+IN



Source: Figure prepared by Capstone, 2024.

14.3.13 SD-IN Mineral Resources

14.3.13.1 2024 SD-IN Mineral Resources

CIM (2014) definitions were used for Mineral Resource classification. The Mineral Resource estimates for SD-IN with an effective date of 31 March 2024, are shown in Table 14-27. They are based on Whittle shell optimization at an NSR cut-off value of US$ 9.85 per tonne. An example cross section of the deposit showing the 2024 Resource pit shell is shown on Figure 14-26.

A significant proportion of the deposit gets its value almost solely from Iron, so it is not appropriate to calculate a Copper Equivalent value for that kind of material. Because of that, Capstone decided to stop reporting Copper Equivalent values for the Santo Domingo project, and average NSR values (in US$/t) are reported instead.

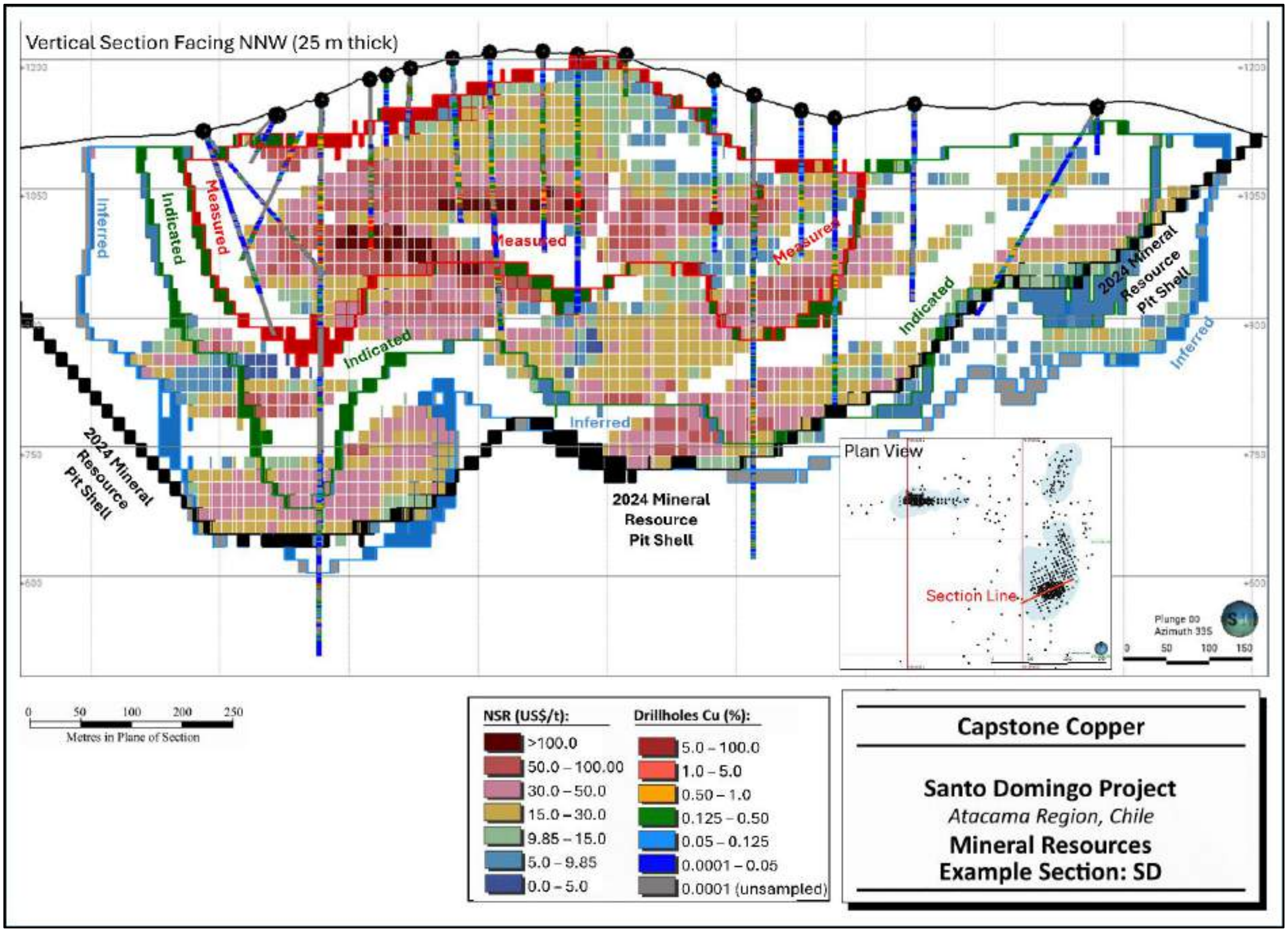
The QP is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the Mineral Resource estimate.

Table 14-27: Summary of Mineral Resources: Santo Domingo and Iris Norte – 31 March 2024

Category	Tonnage (Mt)	Grade				Contained Metal		
		NSR ($/t)	Cu (%)	Fe (%)	Au (g/t)	Cu (kt)	Fe (Mt)	Au (koz)
Measured SD	134	46	0.51	26.9	0.07	679	36	293
Total Measured	**134**	**46**	**0.51**	**26.9**	**0.07**	679	**36**	**293**
Indicated SD	298	34	0.26	25.7	0.04	786	77	362
Indicated IN	74	28	0.14	24.0	0.02	106	18	43
Total Indicated	**372**	**33**	**0.24**	**25.4**	**0.03**	892	**95**	**405**
Total Measured + Indicated	**506**	**36**	**0.31**	**25.8**	**0.04**	1,571	**131**	**698**
Inferred SD	173	28	0.20	22.2	0.03	349	38	157
Inferred IN	30	28	0.12	23.9	0.01	35	7	14
Total Inferred	**203**	**28**	**0.19**	**22.5**	**0.03**	384	**46**	**171**

Notes: **1.** Mineral Resources are classified according to CIM (2014) definitions. **2.** Mineral Resources are reported inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. **4.** Mineral Resources for the Santo Domingo (SD) and Iris Norte (IN) deposits have an effective date of 31 March 2024. **5**. Mineral Resources for the Santo Domingo and Iris Norte deposits are reported using a net smelter return (NSR) cut-off value of US$9.85/t. NSR is calculated using average long-term prices of US$4.10/lb Cu, US$1,600/oz Au, and Fe prices that depend on the expected grade of the Fe concentrate (US$94.75/dmt or $129.77/dmt or $140.26/dmt Fe concentrate). **6**. Mineral Resources for SD include Iris. **7**. Only copper, gold, and iron were recognized in the NSR calculation; cobalt and sulphur were excluded. **8.** Only mixed and primary material were recognized in the NSR calculation; oxides were excluded. **9**. Mineral Resources are constrained by preliminary pit shells derived using a Lerchs–Grossmann algorithm and the following assumptions: pit slopes 36.3°- 47.9°; mining cost is calculated using a function that depends on where the material comes from (Santo Domingo or Iris Norte) and its destination (dumps, plant or stock), processing cost based on Fe concentrate routing code (including general and administrative (G&A) costs); processing recovery are calculated based in the recovery curves for copper and gold. Iron recoveries are calculated based on magnetic susceptibility. Metal prices used are those listed on Note 5. **10.** Rounding as required by reporting standards may result in apparent summation differences. **11.** Tonnage measurements are in metric units. Copper and iron are reported as percentages (%) and gold as grams per tonne (g/t).

Figure 14-26: Example Mineral Resource Cross Section: SD



Source: Figure prepared by Capstone, 2024.

14.3.13.2 Sensitivity of Resources to cut-off

Table 14-28 shows the sensitivity of the total reported resources at SD-IN to changes in the NSR cut-off used, for a range of cut-offs from 20% lower to 50% higher than the one used in reporting (US$9.85/t, highlighted in gray in the table). All those tabulations are within the resource pit constructed using a US$9.85/t cut-off.

At higher cut-offs, the increase in grade does not compensate for the decrease in tonnage, yielding lower contained metal. On the other hand, a cut-off increase of 50% only causes a decrease of 16% in the ore tonnes and ~6-10% in the contained metal. This shows that the resources at SD-IN are not very sensitive to the cut-off used. This is a reflection of the fact that the average NSR value of the deposit is ~2.5-3 times the NSR cut-off.

Because of the relatively small changes even at the highest cut-offs, the QP considers that all those tabulations fulfill the requirements for Reasonable Prospects for Eventual Economic Extraction.

Table 14-28: SD-IN Sensitivity of Resources to Cut-Off

NSR Cut-off ($/t)	Tonnage (Mt)	Grade				Contained Metal		
		NSR ($/t)	Cu (%)	Fe (%)	Au (g/t)	Cu (kt)	Fe (Mt)	Au (koz)
7.88	766	32	0.26	24.1	0.04	2,015	184	892
8.87	736	33	0.27	24.5	0.04	1,985	180	881
9.85	709	34	0.28	24.8	0.04	1,955	176	869
10.84	685	35	0.28	25.2	0.04	1,928	173	859
11.82	660	36	0.29	25.5	0.04	1,900	169	847
12.81	637	37	0.29	25.9	0.04	1,873	165	836
14.78	597	38	0.31	26.4	0.04	1,823	158	814
Relative Difference (%)								
NSR Cut-off ($/t)	Tonnage (Mt)	Grade				Contained Metal		
		NSR ($/t)	Cu (%)	Fe (%)	Au (g/t)	Cu (kt)	Fe (Mt)	Au (koz)
7.88	8%	-6%	-5	-3	-5%	3%	5%	3%
8.87	4%	-3%	-2	-2	-2%	2%	2%	1%
9.85	0%	0%	0	0	0%	0%	0%	0%
10.84	-3%	2%	2	1	2%	-1%	-2%	-1%
11.82	-7%	5%	4	3	5%	-3%	-4%	-3%
12.81	-10%	8%	7	4	7%	-4%	-6%	-4%
14.78	-16%	12%	11	6	11%	-7%	-10%	-6%

14.3.13.3 Cobalt and Sulphur Concentrations

The QP notes that the material containing the Mineral Resource also contains significant concentrations of cobalt and sulphur. These concentrations are shown in Table 14-29 for informational purposes only and are not included in the Mineral Resources.

Table 14-29: Cobalt and Sulphur Concentrations in Classified Material

Category	Tonnage (Mt)	Grade	
		Co (ppm)	S (%)
Measured SD	134	233	2.04
Total Measured	**134**	**233**	**2.04**
Indicated SD	298	238	1.98
Indicated IN	74	200	2.15
Total Indicated	**372**	**230**	**2.01**
Total Measured + Indicated	**506**	**231**	**2.02**
Inferred SD	173	203	1.74
Inferred IN	30	218	2.17
Total Inferred	**203**	**205**	**1.80**

Notes: **1.** Cobalt and sulphur are not included in the Mineral Resources or NSR value calculations. **2.** Grades are listed in the table for informational purposes.

14.3.13.4 Comparison with the SD-IN 2020 Mineral Resource

The QP compared the 2024 SD-IN Mineral Resource estimate with the 2020 Mineral Resources , shown in Table 14-30 and Table 14-31. The QP notes that updates, and improvements to the Mineral Resource estimate, resulted in Measured and Indicated Resources with overall higher Cu and Au grades, lower Fe grades and a slight increase in tonnage. The 2024 SD-IN Mineral Resource estimate has an additional 12% contained copper and 15% contained gold in Measured and Indicated Mineral Resources, four times more copper and gold and three times more contained iron in Inferred Mineral Resources compared to 2020.

The 2024 Mineral Resource estimate incorporates substantial improvements and changes to nearly every component of the modelling process. As such, comparison with the previous model is generally only valid in a global sense. Changes from the 2020 Mineral Resources include:

- Updated NSR calculations are based on revised metal price assumptions to reflect 2024 pricing, and updated processing costs, recovery factors, routing codes, FeDTT estimations, factored mass recovery, and payable metal assumptions based on the most up to date metallurgical test work.
- Resource reporting based on NSR cut-off; use of copper equivalent values was dropped;
- Completely recompiled and validated Mineral Resource database to increase confidence;
- Updated density database with additional samples, discarded lower quality subset;
- New geological interpretation and model based on geochemical analyses of rock types;
- New oxide model;
- New LiDAR topography;
- Discarded old domaining system in favour of domaining by lithology;

- New and more sophisticated massrec formulae with stricter lower constraints;
- Updated, more sophisticated regression formulae for density estimate;
- Iron in concentrate modelling based on machine learning methodology;
- DA to vary estimation direction with mineralised trends;
- Density weighted estimation to better reflect iron content.

Table 14-30: SD-IN Comparison with Previous Mineral Resource Estimate

Category	2024 Mineral Resource Estimate								2020 Mineral Resource Estimate							
	Tonnage (Mt)	Grade				Contained Metal			Tonnage (Mt)	Grade				Contained Metal		
		NSR ($/t)	Cu (%)	Fe (%)	Au (g/t)	Cu (kt)	Fe (Mt)	Au (koz)		CuEq (%)	Cu (%)	Fe (%)	Au (g/t)	Cu (kt)	Fe (Mt)	Au (koz)
Measured SD	134	46	0.51	26.9	0.07	679	36	293	66	0.81	0.61	30.9	0.08	403	20	172
Total Measured	**134**	**46**	**0.51**	**26.9**	**0.07**	679	**36**	**293**	**66**	**0.81**	**0.61**	**30.9**	**0.08**	403	**20**	**172**
Indicated SD	298	34	0.26	25.7	0.04	786	77	362	327	0.51	0.27	26.4	0.04	890	86	399
Indicated IN	74	28	0.14	24.0	0.02	106	18	43	89	0.44	0.12	26.7	0.01	107	24	40
Total Indicated	**372**	**33**	**0.24**	**25.4**	**0.03**	892	**95**	**405**	**416**	**0.50**	**0.24**	**26.4**	**0.03**	997	**110**	**439**
Total Measured + Indicated	**506**	**36**	**0.31**	**25.8**	**0.04**	1,571	**131**	**698**	**482**	**0.54**	**0.29**	**27.1**	**0.04**	1,399	**130**	**611**
Inferred SD	173	28	0.20	22.2	0.03	349	38	157	28	0.40	0.22	23.3	0.03	60	7	29
Inferred IN	30	28	0.12	23.9	0.01	35	7	14	14	0.45	0.09	28.1	0.01	13	4	4
Total Inferred	**203**	**28**	**0.19**	**22.5**	**0.03**	384	**46**	**171**	**42**	**0.42**	**0.18**	**25.0**	**0.02**	76	**11**	**32**

Notes: **1.** Mineral Resources are classified according to CIM (2014) definitions. **2.** Mineral Resources are reported inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. **3.** Qualified Persons February 13, 2020, Mineral Resources, the Qualified Person for the estimates is Mr. David Rennie, P. Eng., an associate of SLR Consulting (Canada) Ltd. (formerly Roscoe Postle Associates Inc.). **4.** Metal Prices and COG/COV: **a.** February 13, 2020, Mineral Resources are reported using a cut-off grade of 0.125% CuEq. CuEq grades are calculated using average long-term prices of US$3.50/lb Cu, US$1,300/oz Au and US$99/dmt Fe conc. Only copper, gold, and iron were recognized in the CuEq calculation, cobalt was excluded. **b.** 31 March 2024, Mineral Resources are using an NSR cut-off value of US$9.85/t. NSR is calculated using average long-term prices of US$4.10/lb Cu, US$1,600/oz Au, and Fe prices that depend on the expected grade of the Fe concentrate (US$94.75/dmt or $129.77/dmt or $140.26/dmt Fe concentrate). Only copper, gold, and iron were recognized in the NSR calculation; cobalt and sulphur were excluded. **5**. Whittle Pit Shell Parameters: **a.** February 13, 2020 Mineral Resources are constrained by preliminary pit shells derived using a Lerchs–Grossmann algorithm and the following assumptions: pit slopes averaging 45º; mining cost of US$1.90/t, processing cost of US$7.27/t (including G&A cost); processing recovery of 89% copper and 79% gold, iron recoveries are calculated based om magnetic susceptibility; and metal prices of US$3.50/lb Cu, US$1,300/oz Au and US$99/dmt iron concentrate. **b.** 31 March 2024 Mineral Resources are constrained by preliminary pit shells derived using a Lerchs–Grossmann algorithm and the following assumptions: pit slopes 36.3°- 47.9°; mining cost is calculated using a function that depends on where the material comes from (Santo Domingo or Iris Norte) and its destination (dumps, plant or stock), processing cost based on Fe concentrate routing code (including general and administrative (G&A) costs); processing recovery are calculated based in the recovery curves for copper and gold. Iron recoveries are calculated based on magnetic susceptibility. Metal prices used are those listed on Note 4b. **6.** Rounding as required by reporting standards may result in apparent summation differences. **7.** Tonnage measurements are in metric units. Copper and iron are reported as percentages, and gold as grams per tonne.

Table 14-31: SD-IN Comparison with Previous Mineral Resource Estimate: Relative Differences

Category	Tonnage (Mt)	Grade			Contained Metal		
		Cu (%)	Fe (%)	Au (g/t)	Cu (kt)	Fe (Mt)	Au (koz)
Measured SD	103%	-17%	-13%	-15%	69%	80%	70%
Total Measured	**103%**	**-17%**	**-13%**	**-15%**	**69%**	**80%**	**70%**
Indicated SD	-9%	-2%	-2%	-6%	-12%	-11%	-9%
Indicated IN	-17%	20%	-10%	83%	0%	-26%	9%
Total Indicated	**-10%**	**0%**	**-4%**	**13%**	**-11%**	**-14%**	**-8%**
Total Measured + Indicated	**5%**	**7%**	**-5%**	**7%**	**12%**	**0%**	**14%**
Inferred SD	518%	-8%	-5%	-6%	478%	450%	441%
Inferred IN	116%	30%	-15%	44%	180%	81%	250%
Total Inferred	**384%**	**5%**	**-10%**	**31%**	**407%**	**316%**	**434%**

Note: Relative Difference calculated as: relative difference = 100%*(value2024 – value2020)/value2020

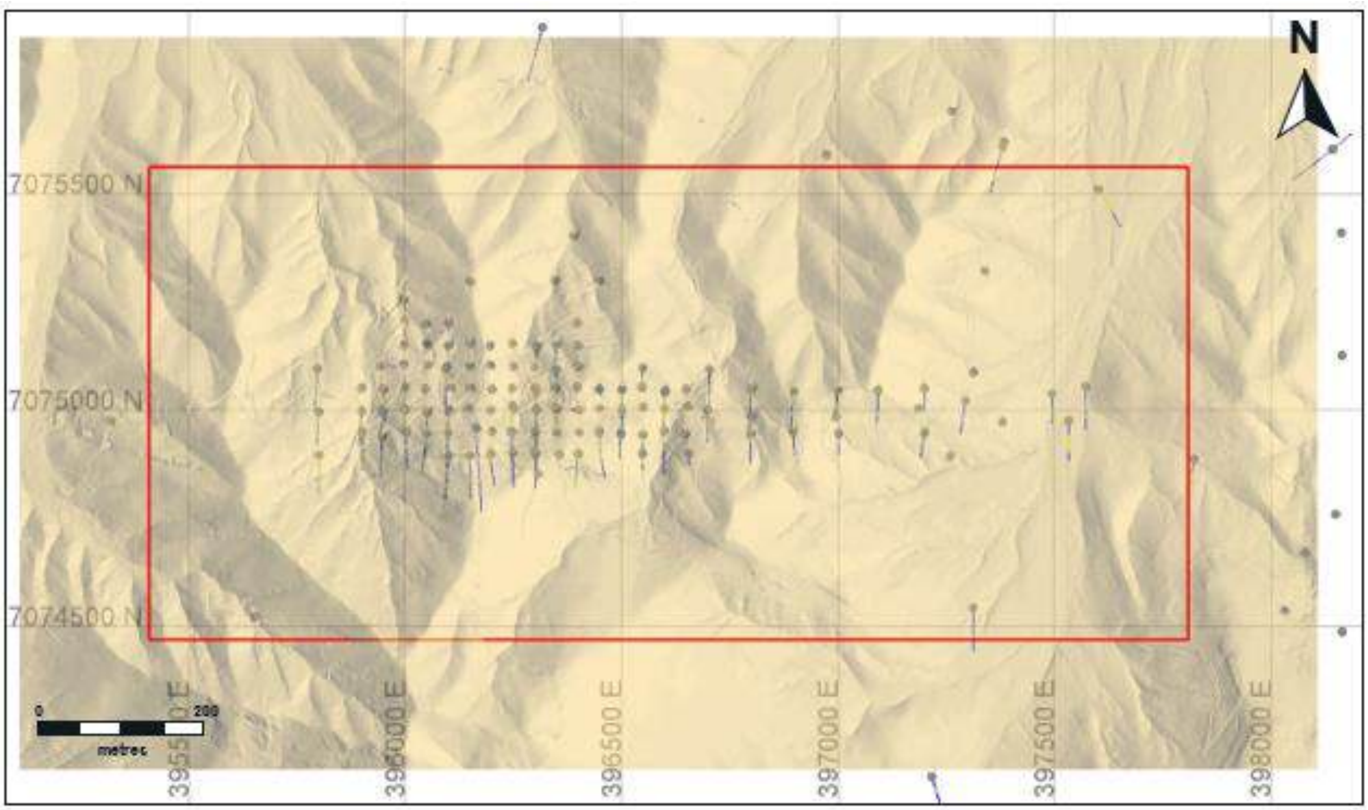
14.4 Estrellita Mineral Resource Estimate

14.4.1 Topography

In 2023, Capstone performed aerophotogrammetric raster and LiDAR surveys on surface at Estrellita. Six new surface survey monuments were established in the ES area. An unmanned aerial vehicle (DJI MATRICE 300) was used for raster photogrammetry and LiDAR. The QP recommends incorporating the new surface surveys into the next Mineral Resource update.

SLR clipped new topographic bounds from the regional model to cover the updated Mineral Resource model extents, including additional area for pit optimization. The topography for ES is shown in Figure 14-27.

Figure 14-27: Estrellita Topography

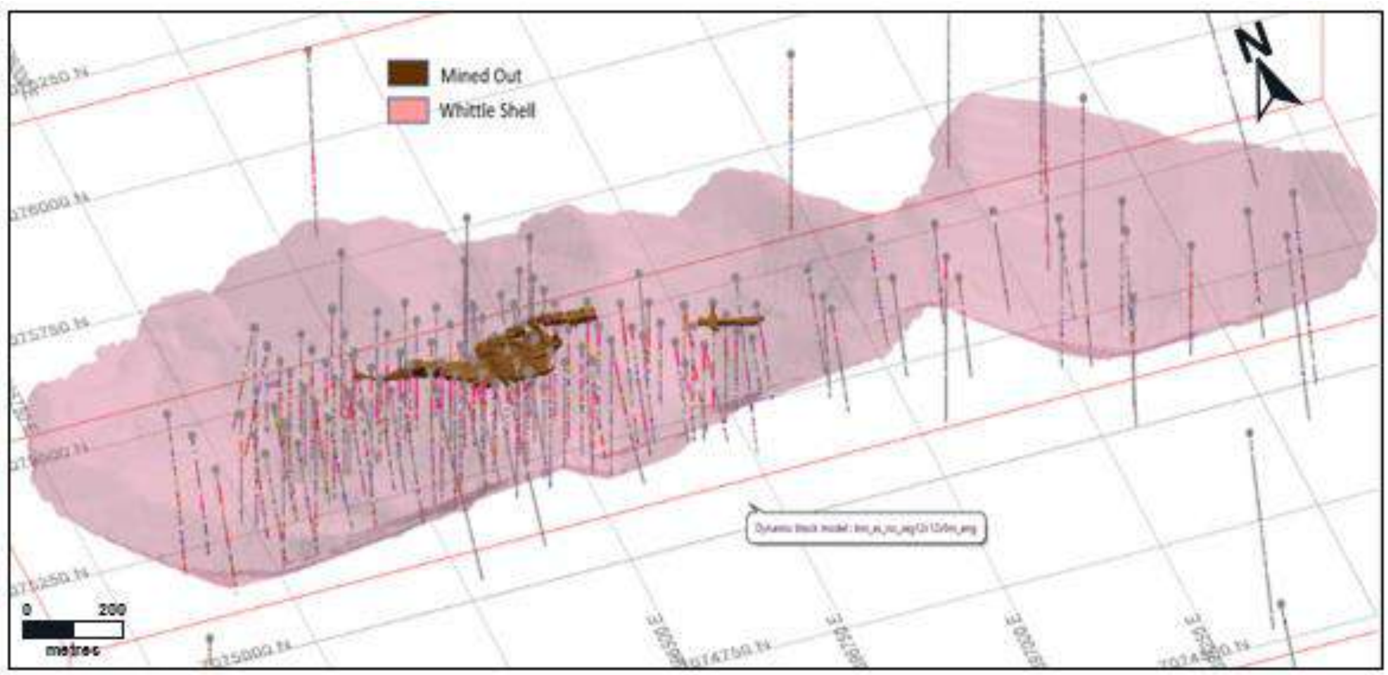


Source: Figure prepared by SLR, 2023.

14.4.2 Underground Workings

There is a small amount of artisanal underground mining at Estrellita which has been removed from the ES block model using previously constructed wireframe solids of the mined workings, as illustrated in Figure 14-28. SLR removed the mined volumes for Whittle analysis by flagging the relevant blocks with a variable (mined_out_0_1) and then recalculated the tonnes in each block with the mined proportion removed (tonnes_block).

Figure 14-28: ES Artisanal Mining Volume, Oblique View



Source: Figure prepared by SLR, 2023.

Acting upon recommendations from the site visit, Capstone updated the initial underground mined out solids with new surveys in early 2023. The underground surveys were performed with mobile LiDAR scans with the ZEB HORIZON laser scanner. These new surveys will be incorporated into the next iteration of the ES Mineral Resource model along with the new topography.

14.4.3 Geological Interpretation

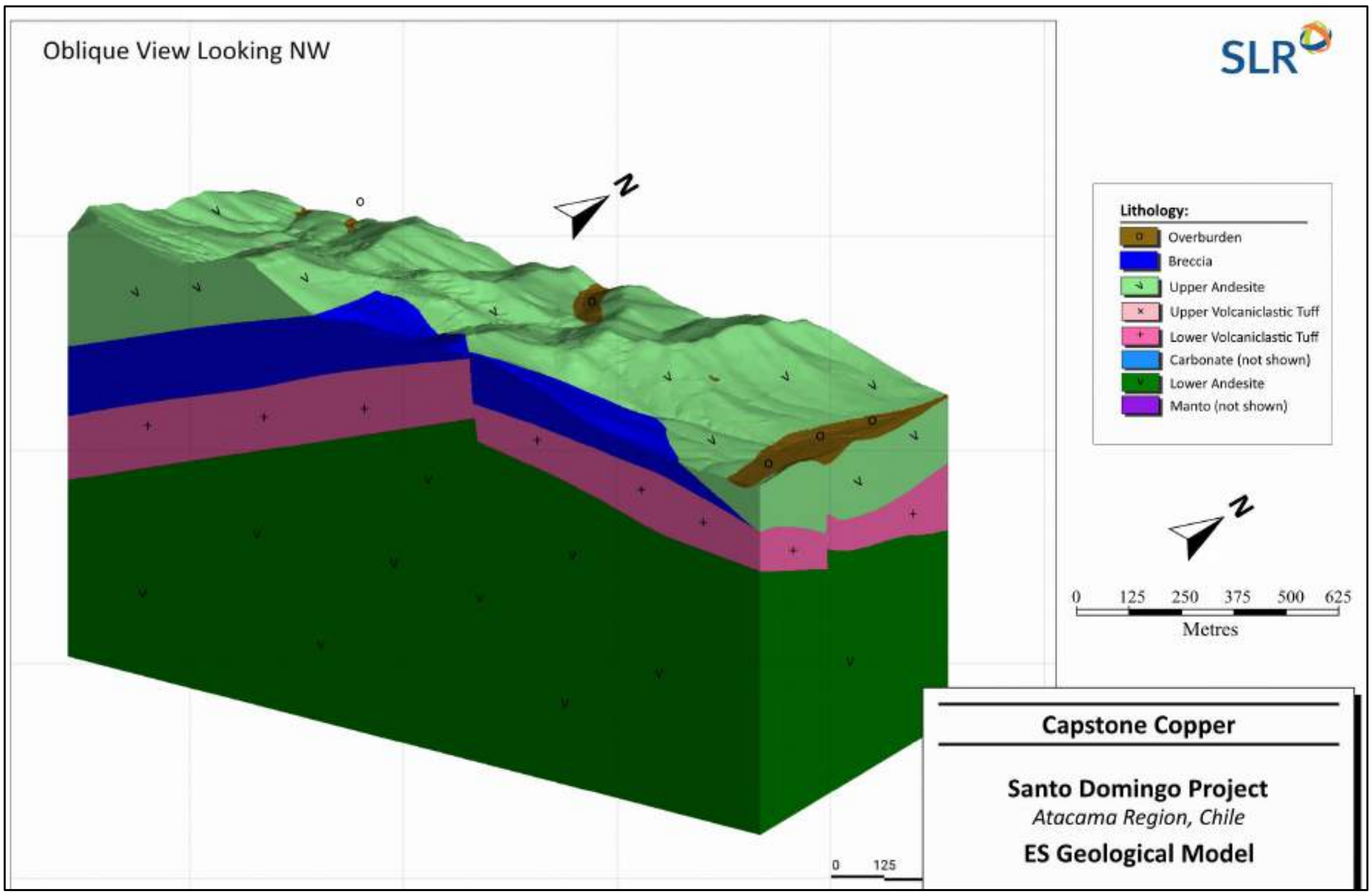
Capstone staff produced a new geological interpretation for the Project in late 2020, as described in Section 14.3.1. A new geological model for ES (Area 3) was developed based on this updated interpretation. An oblique view of the new geological model is shown in Figure 14-32. Table 14-32 summarizes the geological codes used for modelling.

On completion of their Mineral Resource estimation work, SLR noted that pit shell optimization produced shells that went beyond the extents of the geological and oxidation models; SLR recommends that Capstone increases the extents of these models to ensure that any future pit shells are suitably enclosed.

Table 14-32: ES Geological Model Codes

Lithology	Area	Vulcan Triangulation	Domain	D_LITH
Overburden	3	lith_es_301_over.00t	d_lith	301
Breccia	3	lith_es_302_ande_bx.00t	d_lith	302
Upper Andesite	3	lith_es_305_ande_upp.00t	d_lith	305
Upper VLCL Tuff	3	lith_es_307_tuff_upp.00t	d_lith	307
Lower VLCL Tuff	3	lith_es_309_tuff_low.00t	d_lith	309
Carbonate	3	lith_es_310_carb.00t	d_lith	310
Lower Andesite	3	lith_es_311_ande_low.00t	d_lith	311
Manto	3	lith_es_312_manto_2.00t	d_lith	312

Figure 14-29: Estrellita Geological Model – Oblique View

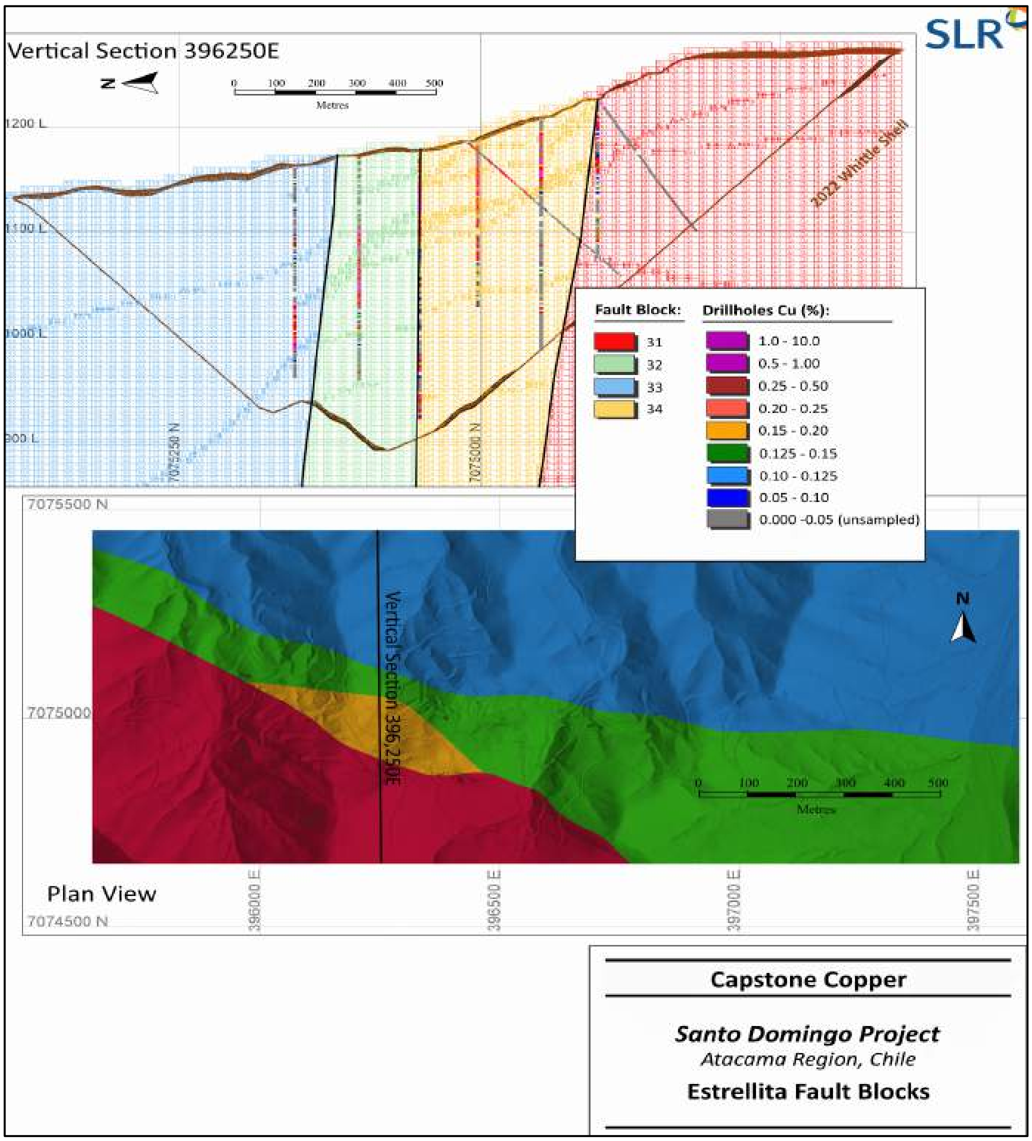


Source: Figure prepared by SLR, 2023.

14.4.3.1 Structure

A new structural model was generated as part of the geological modelling in Leapfrog, resulting in four fault blocks split by three sub-parallel, steeply dipping normal faults, as illustrated in Figure 14-30. The two outboard faults tend to show abrupt changes in grade whereas the fault at centre tends to show only minor displacement, if any. Block codes were assigned as a concatenation of area and fault block.

Figure 14-30: Estrellita Fault Blocks



Source: Figure prepared by SLR, 2023.

14.4.3.2 Oxidation Model

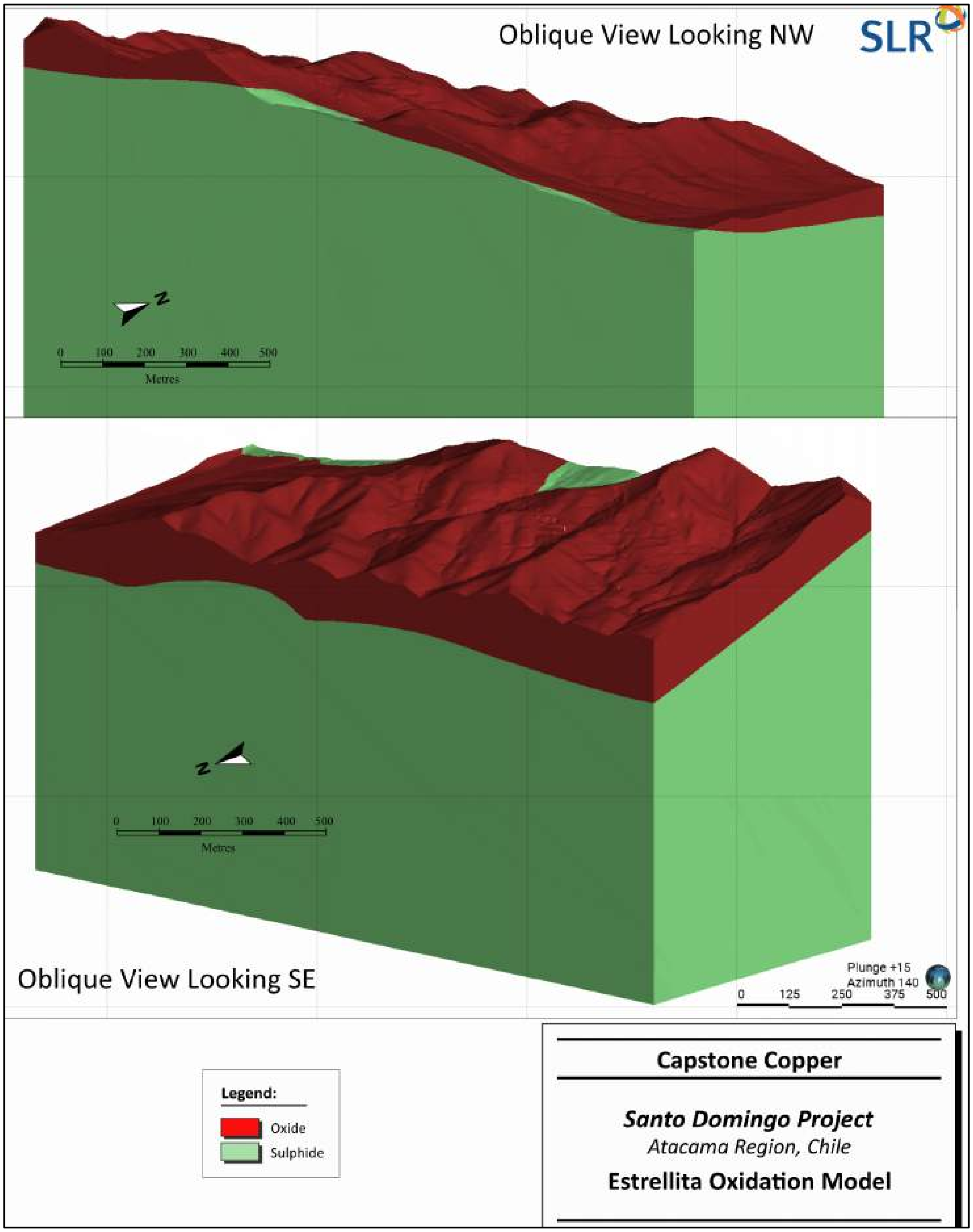
Capstone generated a new oxidation model in Leapfrog for 2022. Wireframe models were created to enclose oxidized material, which is demonstrated to yield much lower metallurgical recoveries than the un-oxidized mineralization, and sulphide material. It was considered that there was insufficient data to accurately model mixed material. Oxidation boundaries are still somewhat uncertain due to incomplete data for leachable copper. The primary criterion for defining the base of the oxidized zone is the presence of significant quantities of leachable copper or log notes stating strong oxidation. The ES model codes for oxidation are listed in Table 14-33 and shown in Figure 14-31.

Oxidation model solids were used to flag material within the resource volume, so that this material could be excluded from the Mineral Resource estimate or otherwise subjected to differential treatment or calculation criteria.

Table 14-33: ES Oxidation Model Codes

d_redox Mineralization		
D_REDOX	Description	Triangulation
32	Oxide	redox_es_CuOx.00t
34	Sulphide	redox_es_Sulphides.00t

Figure 14-31: Estrellita Oxidation Model



Source: Figure prepared by SLR, 2023.

14.4.4 Resource Assays

The 2024 Estrellita database contains 130 drill holes comprising approximately 25 km of drilling that have been completed at Estrellita from 2004 to 2008. The previous 2020 Mineral Resource model for Estrellita was built in 2007. A total of 53 holes totalling 9,500 m were added to the 2024 Mineral Resource estimate database relative to the 2007 model. This includes 41 drill holes that were drilled after the 2007 resource model and 12 peripheral holes that were not used in 2007 (Figure 14-32).

Assay statistics for the ES Mineral Resource database are provided in Table 14-34.

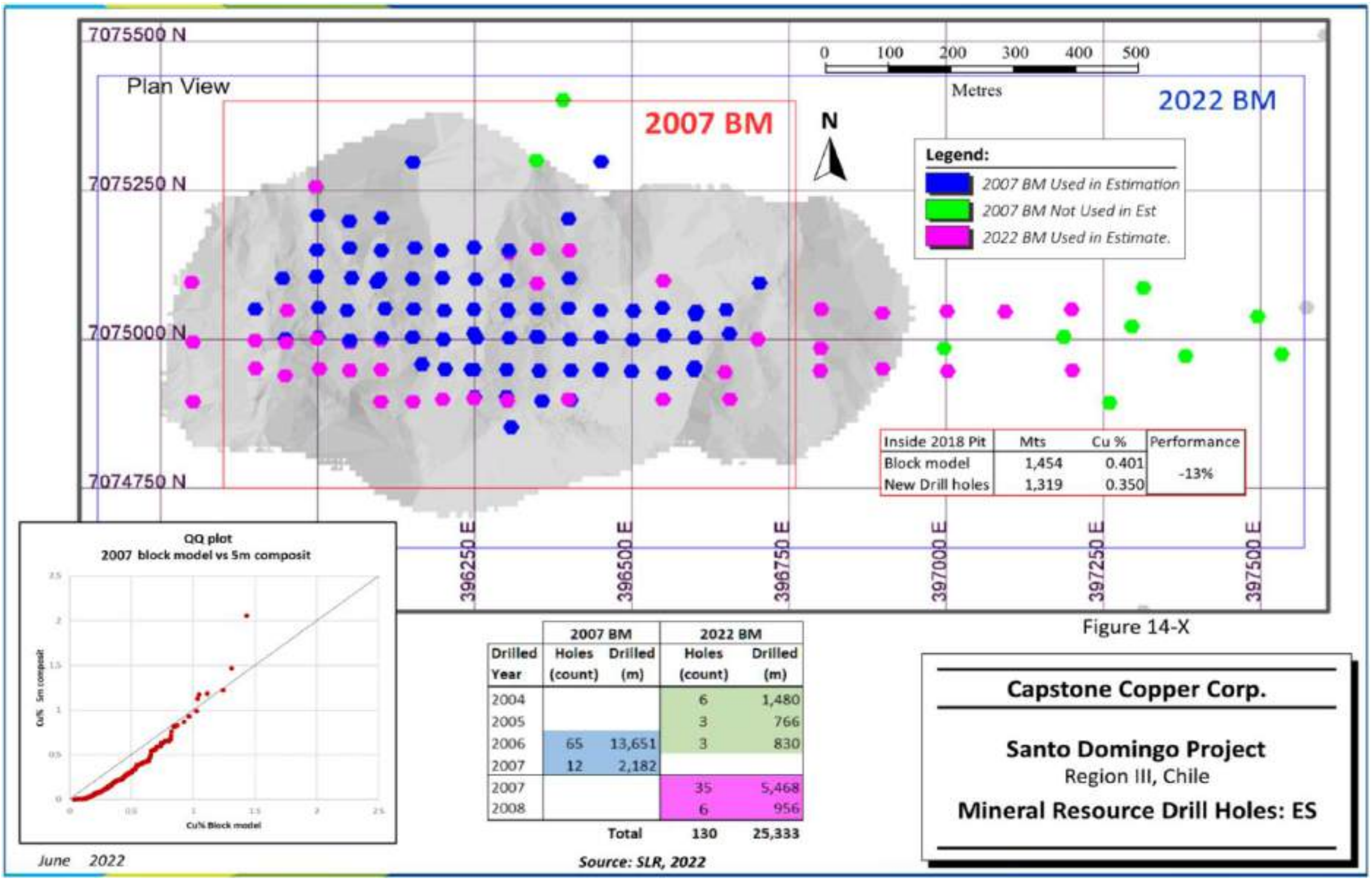
SLR compared the updated Mineral Resource database to that used for the 2020 Mineral Resource estimate and found that the additional holes tend to have lower grades than nearby drill holes used in the previous estimate (Figure 14-32). Inside the 2020 modelled pit shell, which used a 0.125% Cu cut-off grade, inclusion of these additional drill holes results in a potential reduction in average grade of approximately 13%.

Table 14-34: ES Resource Assay Statistics

Element	Units	Lith Code	Count	Min	Max	Mean	CV
Au	g/t	301	20	0.0001	0.017	0.000	6.47
Au	g/t	302	3,956	0.0001	1.100	0.015	2.57
Au	g/t	305	7,333	0.0001	0.979	0.017	2.03
Au	g/t	307	445	0.0001	0.590	0.069	1.11
Au	g/t	309	2,211	0.0001	0.250	0.013	1.86
Au	g/t	310	20	0.0001	0.212	0.013	3.67
Au	g/t	311	234	0.0001	0.142	0.011	1.68
Au	g/t	312	160	0.0001	0.569	0.128	0.74
Co	ppm	301	20	0.0001	58.0	3.5	3.44
Co	ppm	302	3,956	0.0001	1,350.0	73.2	1.29
Co	ppm	305	7,333	0.0001	2,570.0	74.3	1.43
Co	ppm	307	445	0.0001	3,290.0	213.9	1.37
Co	ppm	309	2,211	0.0001	1,665.0	134.9	1.42
Co	ppm	310	20	0.0001	44.0	9.2	1.32
Co	ppm	311	234	7.0000	604.0	69.7	1.13
Co	ppm	312	160	0.0001	1,290.0	429.5	0.71
Cu	%	301	20	0.0001	0.047	0.004	3.34
Cu	%	302	3,956	0.0001	5.930	0.149	1.92
Cu	%	305	7,333	0.0001	7.640	0.179	1.81
Cu	%	307	445	0.0001	6.430	0.750	1.16
Cu	%	309	2,211	0.0001	2.170	0.117	2.00
Cu	%	310	20	0.0001	1.060	0.069	3.52
Cu	%	311	234	0.0009	1.500	0.094	1.87
Cu	%	312	160	0.0001	8.360	1.199	0.85
Fe	%	301	20	0.0001	8.42	0.69	3.40
Fe	%	302	3,956	0.0001	50.10	10.47	0.39

Element	Units	Lith Code	Count	Min	Max	Mean	CV
Fe	%	305	7,333	0.0001	50.10	10.49	0.43
Fe	%	307	445	0.0001	46.30	16.27	0.63
Fe	%	309	2,211	0.0001	50.10	9.05	0.73
Fe	%	310	20	0.0001	11.40	2.36	1.39
Fe	%	311	234	5.4100	32.30	9.80	0.30
Fe	%	312	160	0.0001	50.10	27.61	0.41
S	%	301	20	0.0001	0.29	0.01	4.56
S	%	302	3,956	0.0001	10.00	0.28	1.99
S	%	305	7,333	0.0001	6.95	0.20	2.18
S	%	307	445	0.0001	4.41	0.30	2.22
S	%	309	2,211	0.0001	7.51	0.76	1.27
S	%	310	20	0.0001	1.42	0.10	2.90
S	%	311	234	0.0100	2.68	0.28	1.46
S	%	312	160	0.0001	7.40	1.92	0.82
Py (calculated)	%	301	20	0.0002	0.49	0.01	7.88
Py (calculated)	%	302	3,956	0.0001	16.82	0.27	2.54
Py (calculated)	%	305	7,333	0.0001	9.08	0.18	2.81
Py (calculated)	%	307	445	0.0002	6.41	0.24	3.55
Py (calculated)	%	309	2,211	0.0001	12.72	1.21	1.39
Py (calculated)	%	310	20	0.0002	2.31	0.17	2.96
Py (calculated)	%	311	234	0.0002	3.31	0.35	1.65
Py (calculated)	%	312	160	0.0002	9.08	1.88	1.07
Density (calculated)	g/cm^3	301	20	2.75	2.81	2.76	0.01
Density (calculated)	g/cm^3	302	3,956	2.72	4.81	2.84	0.04
Density (calculated)	g/cm^3	305	7,333	2.72	4.78	2.85	0.04
Density (calculated)	g/cm^3	307	445	2.75	4.29	3.03	0.09
Density (calculated)	g/cm^3	309	2,211	2.72	4.81	2.83	0.09
Density (calculated)	g/cm^3	310	20	2.75	2.85	2.77	0.01
Density (calculated)	g/cm^3	311	234	2.75	3.61	2.82	0.03
Density (calculated)	g/cm^3	312	160	2.75	4.81	3.46	0.15

Figure 14-32: Summary of Drill Holes Used in Mineral Resource Estimates: Estrellita



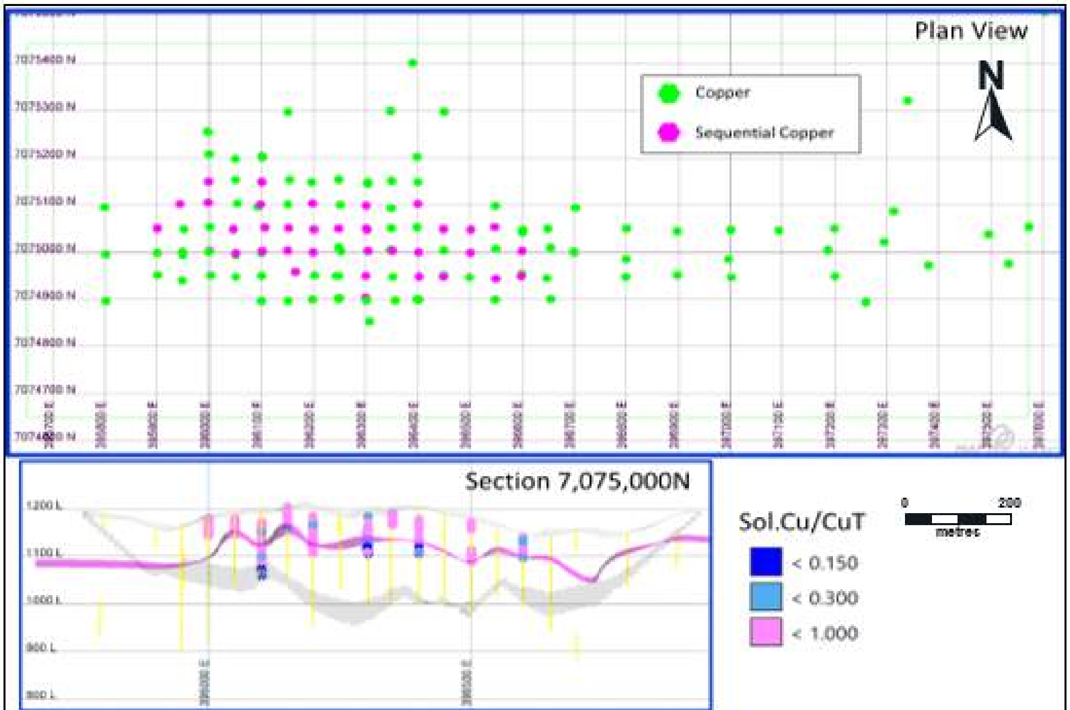
Source: Figure prepared by SLR, 2023.

14.4.4.1 Sequential Copper Analyses

SLR analyzed the sequential copper samples inside the 2020 pit shell (Figure 14-33). There are 1,341 samples from 36 drill holes spanning a total of 2,296.5 m (approximately 9% of the total metreage). Review indicates that the copper oxide mineralization in the sequential copper data, defined as acid soluble copper (CuSA) / total copper (CuT) > 0.5, contains approximately 64% acid soluble copper mineralization, approximately 14% cyanide soluble mineralization, and approximately 22% non-leachable mineralization. To de-risk the oxide material, the QP recommends that Capstone gather and incorporate additional metallurgical and sequential copper data.

Sequential copper data show that the oxide contact could be on the order of five to fifteen metres shallower than interpreted in the geological model. Below the oxide contact, the mineralization has a high secondary enrichment contribution. Improving the mineralogical domaining could further the understanding of the metallurgical characterization of the orebody. Figure 14-34 shows an example drill hole where oxide material is likely shallower than interpreted presently.

Figure 14-33: Summary of Drill Holes with Copper and Sequential Copper Analysis: Estrellita



Sequential Copper

	Metes	Cut	CuCn	CuS	CuR
Oxide	1,544	0.47	0.07	0.30	0.10
		Ratio	14%	64%	22%
Sulfides	752	0.34	0.14	0.09	0.11
		Ratio	40%	27%	33%

Sequential Copper (Cut > 0.12 %)

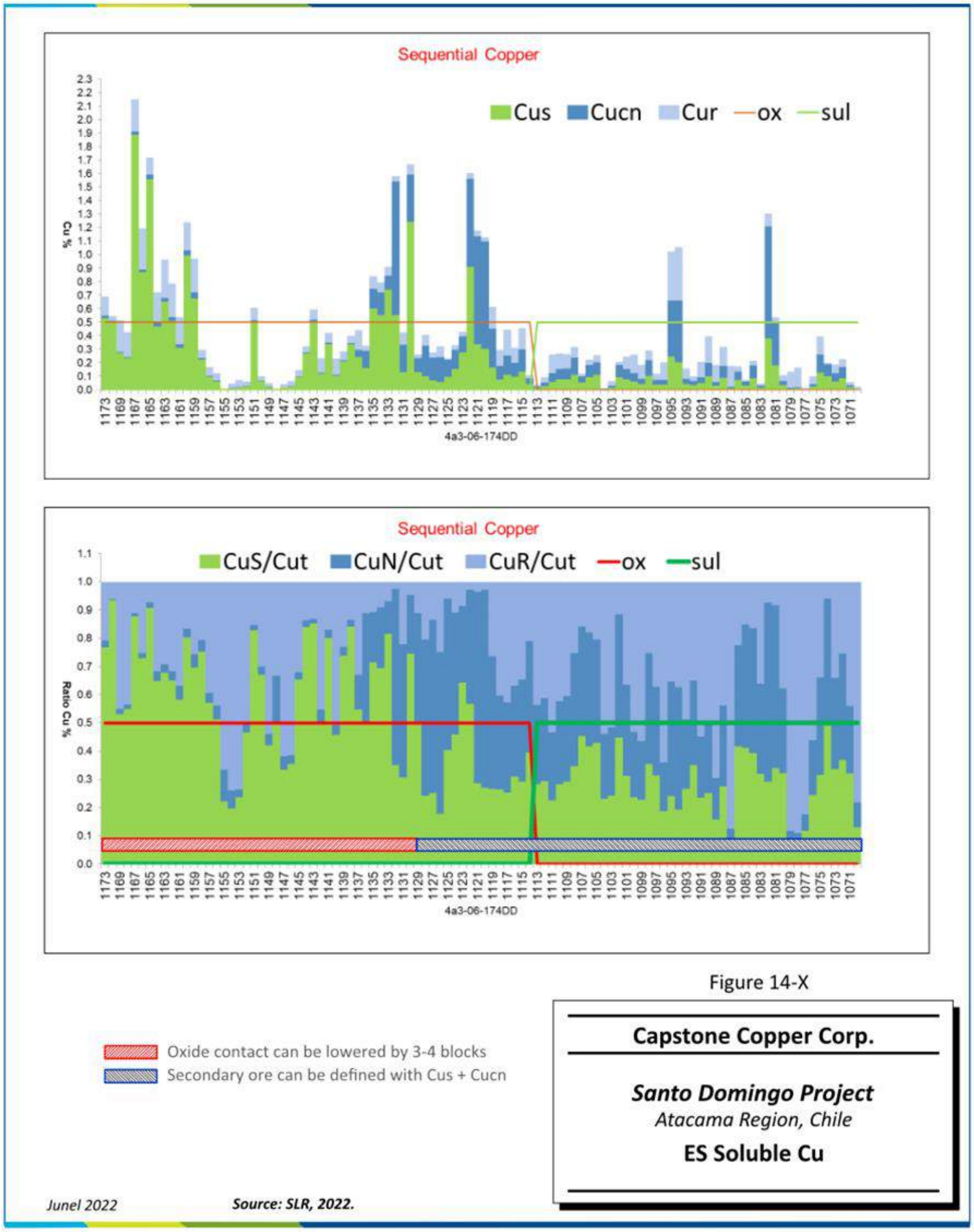
	Metes	Cut	CuCn	CuS	CuR
Oxide	1,206	0.59	0.08	0.38	0.12
		Ratio	14%	65%	21%
Sulfides	501	0.49	0.20	0.13	0.16
		Ratio	41%	27%	32%

Capstone Copper Corp.

Santo Domingo Project
Atacama Region, Chile
ES Soluble Cu inside 2018 Pit Shell

Source: Figure prepared by SLR, 2023.

Figure 14-34: Soluble Copper Example: Estrellita



Source: Figure prepared by SLR, 2023.

14.4.4.2 Unsampled Intervals

SLR applied the same procedures for unsampled intervals as described above for SD-IN. There were no additional special cases with differential treatment.

14.4.5 Treatment of High-Grade Assays

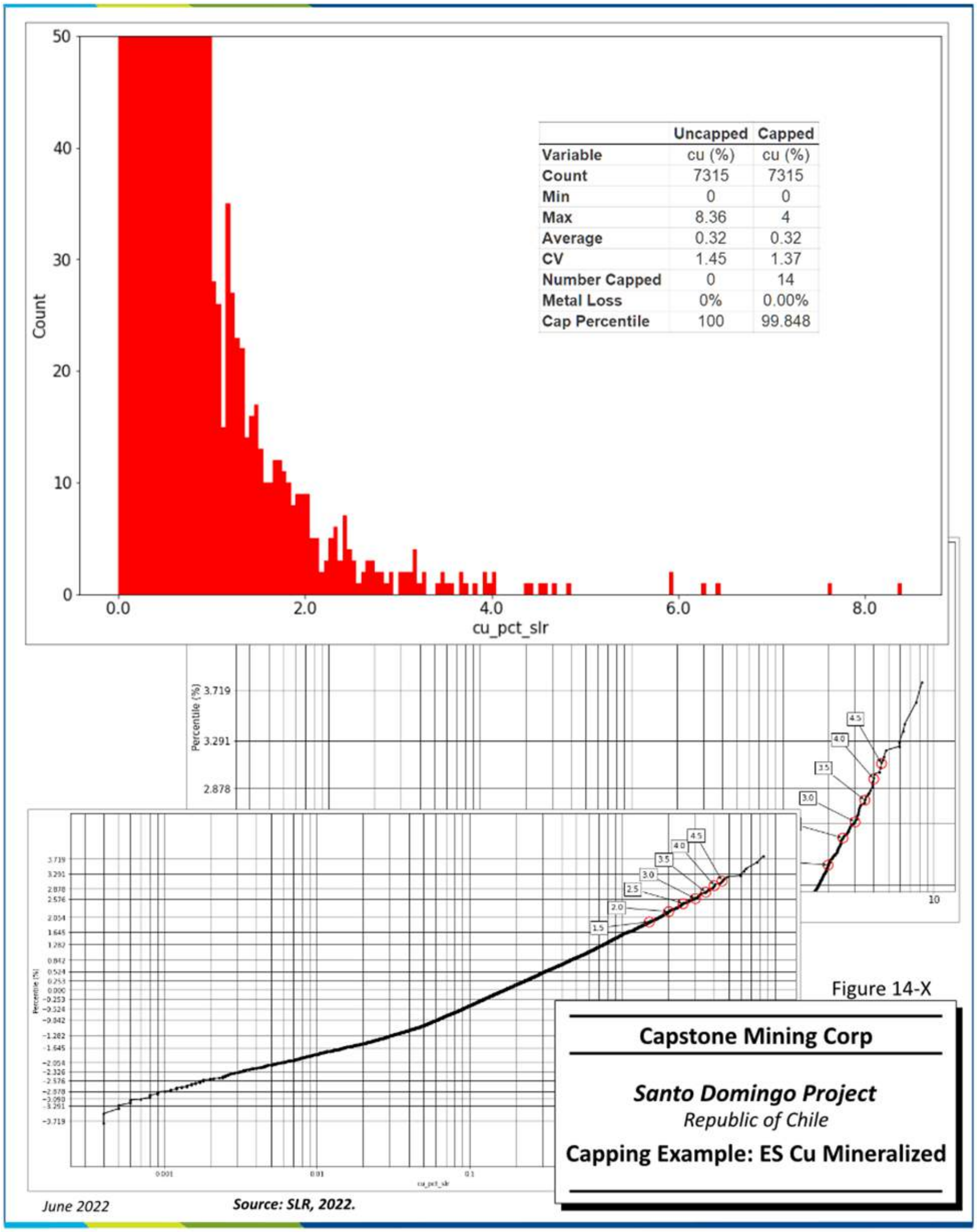
14.4.5.1 Capping Levels

Given that vertical changes in grade with lithology are not as strong as at SD-IN, and the drill hole dataset is markedly smaller, SLR chose to aggregate the capping domains according to whether they were inside or outside of the mineralized domain generated in Leapfrog. A summary of the capping for Au, Co, and Cu is presented in Table 14-35, and an example for the capping of Cu is shown on Figure 14-35. S and Fe were left uncapped. Cumulative plots showed long, continuous tails both inside and outside the mineralized volume. CV values were low to moderate inside the mineralized volume, but markedly higher outside of it, where mineralization is more sporadic. As with SD-IN, light capping levels were applied in general to control high grade assays, resulting in little metal loss.

Table 14-35: ES Capping Summary

Element	Units	Mineralized Volume	Count	Uncapped				Capped			Capped Count	Metal Loss (%)	Cap Percentile
				Min	Max	Mean	CV	Max	Mean	CV			
Au	g/t	Inside	5,401	0.005	1.100	0.040	1.35	0.450	0.040	1.21	10	0.0	99.82
Au	g/t	Outside	3,370	0.005	0.910	0.010	2.74	0.250	0.010	2.02	6	0.0	99.83
Co	ppm	Inside	7,316	1.0	3,290.0	126.2	1.28	1,250.0	125.6	1.23	14	0.5	99.83
Co	ppm	Outside	1,537	1.0	1,660.0	59.9	1.77	1,100.0	59.6	1.71	12	0.6	99.87
Cu	%	Inside	7,315	0.000	8.360	0.320	1.45	4.000	0.320	1.37	14	0.0	99.85
Cu	%	Outside	8,728	0.000	2.570	0.050	2.12	2.000	0.050	2.09	1	0.0	99.99

Figure 14-35: Capping Analysis Example: ES Cu



Source: Figure prepared by SLR, 2023.

14.4.5.2 High Grade Restriction

To control local extrapolation, SLR used HYR via a flagged field in the database which set searches for all grades for a few selected holes in mineralized fault block 31 to 60 m x 60 m x 10 m.

14.4.6 Compositing

Compositing was completed at three metre intervals from the top of the drill hole, breaking on lithologic boundaries and a flag field for omitting unsampled intervals from the Mineral Resource estimate (RSC_USE). SLR chose a three-metre composite length for ES to represent half of the 6 m block height and to better reflect the smaller scale and greater variability of the grade data relative to SD. A summary of the composited data by estimation domain is presented in Table 14-36 and by lithology in Table 14-37.

Table 14-36: ES Composite Statistics by Estimation Domain

Element	Units	Domain	Fault Block	Mineralization	Count	Min	Max	Mean	CV
au	g/t	fb31_minz0	31	0	854	0.0001	0.1414	0.0034	3.72
au	g/t	fb31_minz31	31	31	1046	0.0001	0.5223	0.0276	1.44
au	g/t	fb3234_minz0	3234	0	1865	0.0001	0.1265	0.0042	2.35
au	g/t	fb3234_minz31	3234	31	2805	0.0001	0.6847	0.0347	1.3
au	g/t	fb33_minz0	33	0	1541	0.0001	0.25	0.0037	2.4
au	g/t	fb33_minz31	33	31	771	0.0001	0.6862	0.0256	1.49
co	ppm	fb31_minz0	31	0	854	0.0001	726.3835	43.8392	1.44
co	ppm	fb31_minz31	31	31	1046	0.0001	1440	117.2529	1.08
co	ppm	fb3234_minz0	3234	0	1865	0.0001	1100	62.4531	1.64
co	ppm	fb3234_minz31	3234	31	2805	0.0001	1599.7205	126.6615	1.17
co	ppm	fb33_minz0	33	0	1541	0.0001	1100	45.4077	1.54
co	ppm	fb33_minz31	33	31	771	0.0001	1264.8158	124.9446	1.17
cu	%	fb31_minz0	31	0	854	0.0001	1.0215	0.0424	2.21
cu	%	fb31_minz31	31	31	1046	0.0001	3.4067	0.2844	1.3
cu	%	fb3234_minz0	3234	0	1865	0.0001	0.8607	0.0421	1.63
cu	%	fb3234_minz31	3234	31	2805	0.0001	5.2429	0.3464	1.23
cu	%	fb33_minz0	33	0	1541	0.0001	0.5213	0.038	1.23
cu	%	fb33_minz31	33	31	771	0.0001	7.5964	0.2509	1.48
density	g/cm^3	fb31_minz0	31	0	854	2.7155	3.265	2.7886	0.02
density	g/cm^3	fb31_minz31	31	31	1046	2.7155	4.2473	2.8886	0.05
density	g/cm^3	fb3234_minz0	3234	0	1865	2.7155	3.2198	2.7898	0.01
density	g/cm^3	fb3234_minz31	3234	31	2805	2.7155	4.7834	2.9059	0.06
density	g/cm^3	fb33_minz0	33	0	1541	2.7155	4.4165	2.8026	0.03
density	g/cm^3	fb33_minz31	33	31	771	2.7184	4.8137	2.9267	0.1

Element	Units	Domain	Fault Block	Mineralization	Count	Min	Max	Mean	CV
fe	%	fb31_minz0	31	0	854	0.0001	23.6769	7.8895	0.31
fe	%	fb31_minz31	31	31	1046	0.0001	44.2691	12.4468	0.38
fe	%	fb3234_minz0	3234	0	1865	0.0001	22.7045	7.789	0.33
fe	%	fb3234_minz31	3234	31	2805	0.0001	49.728	13.0695	0.42
fe	%	fb33_minz0	33	0	1541	0.0001	45	8.2016	0.41
fe	%	fb33_minz31	33	31	771	0.0001	50.1	12.9608	0.55
py	%	fb31_minz0	31	0	854	0.0001	5.9282	0.1974	3.2
py	%	fb31_minz31	31	31	1046	0.0002	8.5041	0.4235	1.88
py	%	fb3234_minz0	3234	0	1865	0.0001	9.314	0.4279	2.32
py	%	fb3234_minz31	3234	31	2805	0.0002	9.8414	0.4771	1.91
py	%	fb33_minz0	33	0	1541	0.0001	11.2211	0.1618	3.94
py	%	fb33_minz31	33	31	771	0.0001	9.3415	0.5826	1.82
s	%	fb31_minz0	31	0	854	0.0001	3.2059	0.1331	2.66
s	%	fb31_minz31	31	31	1046	0.0001	7.0833	0.4148	1.54
s	%	fb3234_minz0	3234	0	1865	0.0001	5	0.2574	2.05
s	%	fb3234_minz31	3234	31	2805	0.0001	6.8171	0.4671	1.42
s	%	fb33_minz0	33	0	1541	0.0001	6	0.1114	3.03
s	%	fb33_minz31	33	31	771	0.0001	6.0256	0.5032	1.52

Table 14-37: ES Composite Statistics by Lithology

Element	Units	Lith Code	Count	Weight	Min	Max	Mean	CV
Au	g/t	301	14	36.08	0.0001	0.017	0.000	6.56
Au	g/t	302	2,369	6864	0.0001	0.612	0.015	2.15
Au	g/t	305	4,626	13448	0.0001	0.686	0.017	1.78
Au	g/t	307	260	656.9	0.0001	0.468	0.069	0.94
Au	g/t	309	1,357	3918	0.0001	0.174	0.013	1.63
Au	g/t	310	14	38.57	0.0001	0.141	0.013	2.97
Au	g/t	311	150	420.9	0.0001	0.128	0.011	1.51
Au	g/t	312	92	224.7	0.0001	0.395	0.128	0.65
Co	ppm	301	14	36.08	0.0001	58.0	3.5	3.48
Co	ppm	302	2,369	6864	0.0001	1,030.0	73.2	1.12
Co	ppm	305	4,626	13448	0.0001	1,599.7	74.3	1.29
Co	ppm	307	260	656.9	0.0001	1,533.3	213.9	1.17
Co	ppm	309	1,357	3918	0.0001	1,264.8	134.9	1.27
Co	ppm	310	14	38.57	0.0001	37.0	9.2	1.26
Co	ppm	311	150	420.9	9.6221	481.6	69.7	1.00
Co	ppm	312	92	224.7	0.0001	1,228.4	429.5	0.65

Element	Units	Lith Code	Count	Weight	Min	Max	Mean	CV
Cu	%	301	14	36.08	0.0001	0.047	0.004	3.38
Cu	%	302	2,369	6864	0.0001	2.729	0.149	1.62
Cu	%	305	4,626	13448	0.0001	7.596	0.179	1.62
Cu	%	307	260	656.9	0.0001	4.533	0.750	1.00
Cu	%	309	1,357	3918	0.0001	1.816	0.117	1.77
Cu	%	310	14	38.57	0.0001	0.712	0.069	2.87
Cu	%	311	150	420.9	0.0012	1.319	0.094	1.76
Cu	%	312	92	224.7	0.0001	5.243	1.199	0.75
Fe	%	301	14	36.08	0.0001	8.33	0.69	3.45
Fe	%	302	2,369	6864	0.0001	41.16	10.47	0.35
Fe	%	305	4,626	13448	0.0001	44.20	10.49	0.41
Fe	%	307	260	656.9	0.0001	44.27	16.27	0.60
Fe	%	309	1,357	3918	0.0001	50.10	9.05	0.70
Fe	%	310	14	38.57	0.0001	8.19	2.36	1.19
Fe	%	311	150	420.9	5.4100	22.74	9.80	0.27
Fe	%	312	92	224.7	0.0001	49.73	27.61	0.37
S	%	301	14	36.08	0.0001	0.29	0.01	4.61
S	%	302	2,369	6864	0.0001	7.08	0.28	1.75
S	%	305	4,626	13448	0.0001	6.03	0.20	2.00
S	%	307	260	656.9	0.0001	3.91	0.30	2.03
S	%	309	1,357	3918	0.0001	6.00	0.76	1.13
S	%	310	14	38.57	0.0001	0.94	0.10	2.44
S	%	311	150	420.9	0.0100	2.33	0.28	1.33
S	%	312	92	224.7	0.0001	6.25	1.92	0.74
Py (calculated)	%	301	14	36.08	0.0002	0.49	0.01	7.96
Py (calculated)	%	302	2,369	6864	0.0001	9.80	0.27	2.19
Py (calculated)	%	305	4,626	13448	0.0001	7.09	0.18	2.55
Py (calculated)	%	307	260	656.9	0.0002	5.39	0.24	2.99
Py (calculated)	%	309	1,357	3918	0.0002	11.22	1.21	1.23
Py (calculated)	%	310	14	38.57	0.0002	1.54	0.17	2.50
Py (calculated)	%	311	150	420.9	0.0002	2.97	0.35	1.49
Py (calculated)	%	312	92	224.7	0.0002	6.77	1.88	0.97
Density (calculated)	g/cm^3	301	14	36.08	2.75	2.81	2.76	0.01
Density (calculated)	g/cm^3	302	2,369	6864	2.72	4.25	2.84	0.04
Density (calculated)	g/cm^3	305	4,626	13448	2.72	4.36	2.85	0.03
Density (calculated)	g/cm^3	307	260	656.9	2.75	4.17	3.03	0.08
Density (calculated)	g/cm^3	309	1,357	3918	2.72	4.81	2.83	0.08
Density (calculated)	g/cm^3	310	14	38.57	2.75	2.82	2.77	0.01

Element	Units	Lith Code	Count	Weight	Min	Max	Mean	CV
Density (calculated)	g/cm^3	311	150	420.9	2.75	3.21	2.82	0.02
Density (calculated)	g/cm^3	312	92	224.7	2.75	4.78	3.46	0.14

14.4.7 Trend Analysis

SLR examined the metal assays in section against the geological model and generated box plots against lithology, structure, and oxidation. Mineralization generally trends parallel to the lithological units, but the vertical variation with rock type appears to be less strong than at SD-IN. Grade changes more abruptly against major fault blocks than lithologies, and there is lateral variation about a mineralized envelope generated in Leapfrog. SLR thus choses to rely on the Dynamic Anisotropy to orient grade estimation searches along lithological orientations, and then use a combination of fault blocks and the mineralization envelope to partition the estimation domains. Little variation was observed between the central fault blocks 32 and 34, so they were treated as one domain for the purposes of Mineral Resource estimation.

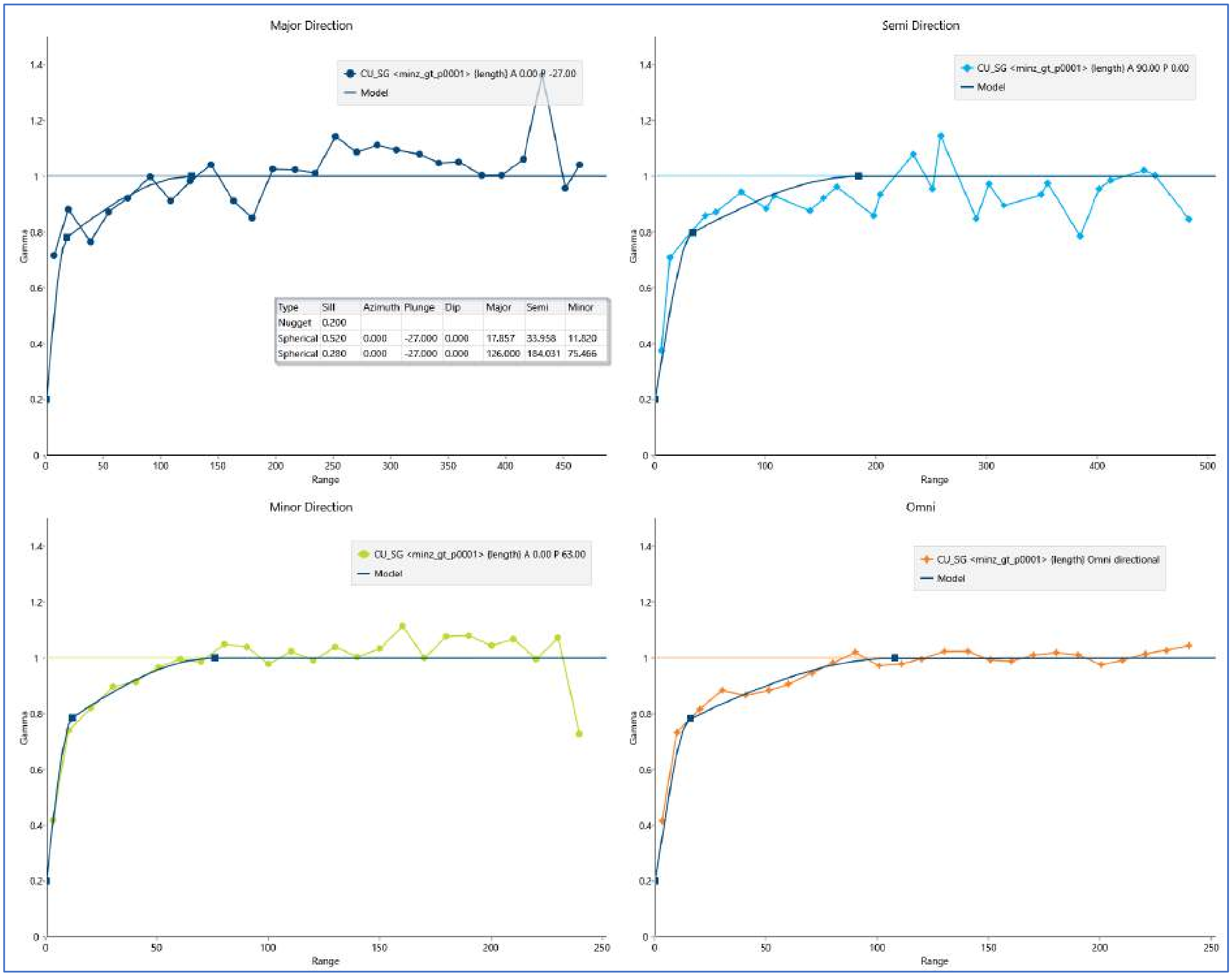
14.4.7.1 Variography

SLR generated variograms for copper, iron, gold, cobalt, and pyrite accumulations, density and magnetic susceptibility in Vulcan's visual data analyser software. In general, variograms show similar behaviour to IN, but with shorter ranges on the order of 50 m and 120 m for the first and second structures, respectively. Nuggets generally occurred at 0.15 to 0.25. Variograms were generated for each metal inside the mineralized envelope, and then applied to the DA grade estimation files for each fault block domain. A summary of the variograms for ES is presented in Table 14-38 and an example variogram for Cu is shown in Figure 14-36.

Table 14-38: ES Variogram Parameters

Accumulation	Nugget	No. of Struct	Struct 1 Diff Sill	Major Struct 1 Range (m)	Semi Major Struct 1 Range (m)	Minor Struct 1 Range (m)	Struct 1 Rotn Alpha (Deg)	Struct 1 Rotn Zeta (Deg)	Struct 1 Rotn Beta (Deg)	Struct 2 Diff Sill	Major Struct 2 Range (m)	Semi Major Struct 2 Range (m)	Minor Struct 2 Range (m)	Struct 2 Rotn Alpha (Deg)	Struct 2 Rotn Zeta (Deg)	Struct 2 Rotn Beta (Deg)
Au	0.250	2	0.417	16.11	45.07	23.45	0	-27	0	0.333	130.62	110.00	82.45	0	-27	0
Co	0.200	2	0.480	18.37	40.44	23.35	0	-27	0	0.320	117.84	125.04	81.30	0	-27	0
Cu	0.200	2	0.426	17.86	47.10	23.35	0	-27	0	0.374	126.10	184.03	75.47	0	-27	0
Fe	0.150	2	0.438	20.73	58.06	35.49	0	-27	0	0.412	127.47	159.38	86.15	0	-27	0
Py	0.150	2	0.481	18.70	22.10	22.62	0	-27	0	0.369	119.48	110.00	104.26	0	-27	0
Density	0.200	2	0.455	21.21	26.35	20.67	0	-27	0	0.345	123.88	110.48	61.14	0	-27	0
S	0.150	2	0.441	17.36	51.90	18.25	0	-27	0	0.409	128.52	149.80	83.12	0	-27	0

Figure 14-36: ES Variogram Example: Cu Accumulations

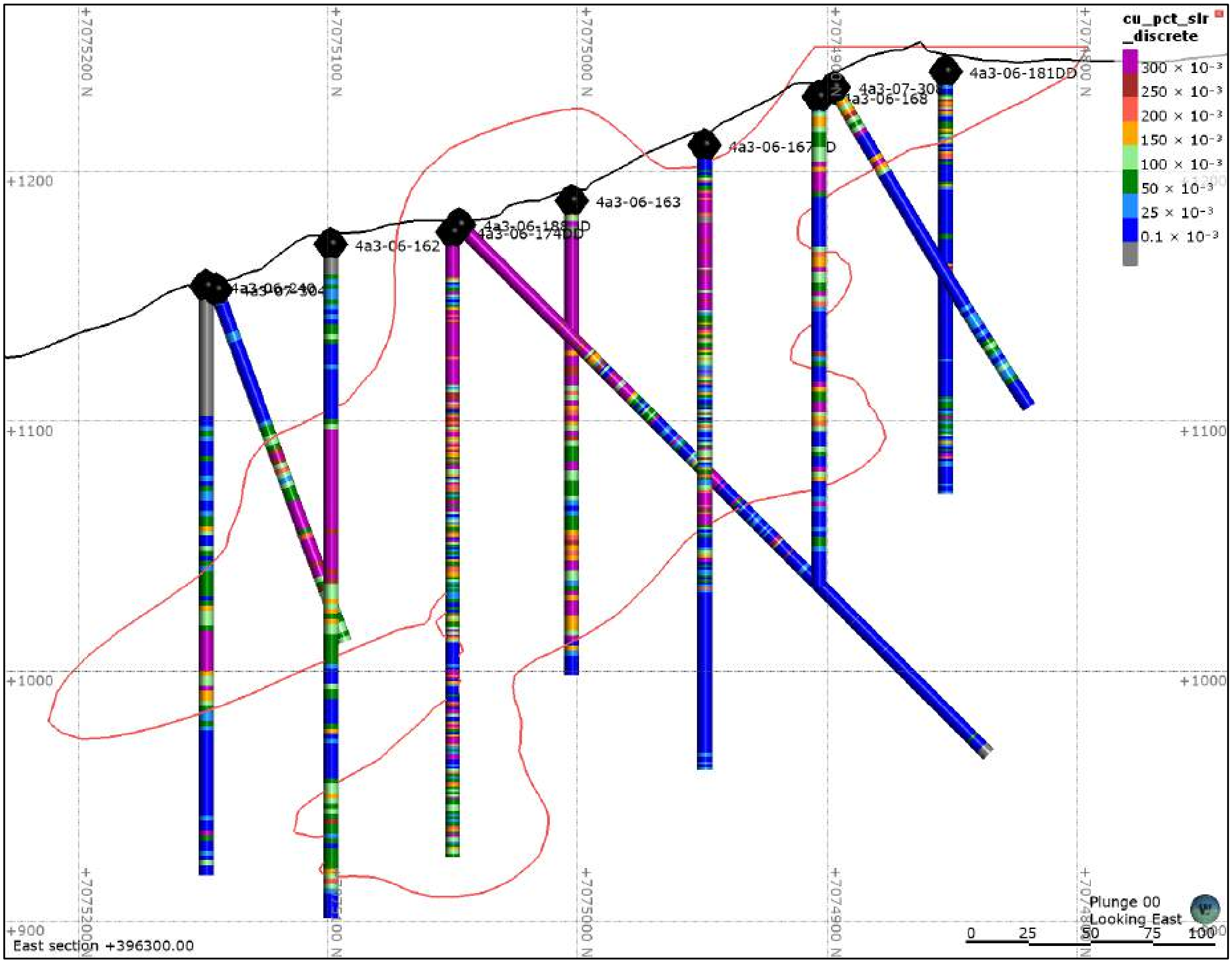


Source: Figure prepared by SLR, 2023.

14.4.7.2 Grade Contouring

SLR generated several grade interpolants for copper, iron, gold, and cobalt in Leapfrog software, comparing them to the lithology, structure, oxidation, and magsus models. SLR determined that lithology is less important for controlling mineralization at ES than it is at SD and decided to use a combination of the mineralized envelope and the fault block solids as structural domains to create estimation domains, while ensuring that DA interpolation approximated vertical changes in grade with lithology. A soft boundary of 10 m was applied at the mineralization envelope boundary. The mineralization envelope is illustrated in Figure 14-37.

Figure 14-37: ES Mineralization Envelope



Source: Figure prepared by SLR, 2023.

14.4.8 Search Strategy and Grade Interpolation Parameters

SLR performed density weighted grade estimates for copper, iron, gold, cobalt, pyrite, and sulphur using the methodology illustrated in Figure 14-16. Initially, accumulations of the elements multiplied by density were generated in the sample database. The accumulations were then composited and estimated. Finally, grades were back calculated in the block model after interpolation of the accumulations. Iron is not currently deemed to be of sufficient economic strength to be part of the ES Mineral Resource as most of the iron occurs in the form of specular hematite. There are no magsus samples at ES, and therefore no mass recovery estimate was performed.

Grade estimation was performed in three passes at 60 m x 60 m x 10 m, 120 m x 120m x 20 m, and 280 m x 280 m x 40 m in accordance with the results of the variography results. SLR notes that the variograms indicated slightly shorter search distances than those used at SD-IN. Passes 1 and 2 required a minimum of two holes per estimate, with a maximum of four holes, with up to two 3 m composites allowed per hole. The largest search, Pass 3, required a minimum of one hole and two composites. A summary of grade estimation search parameters is provided in Table 14-39.

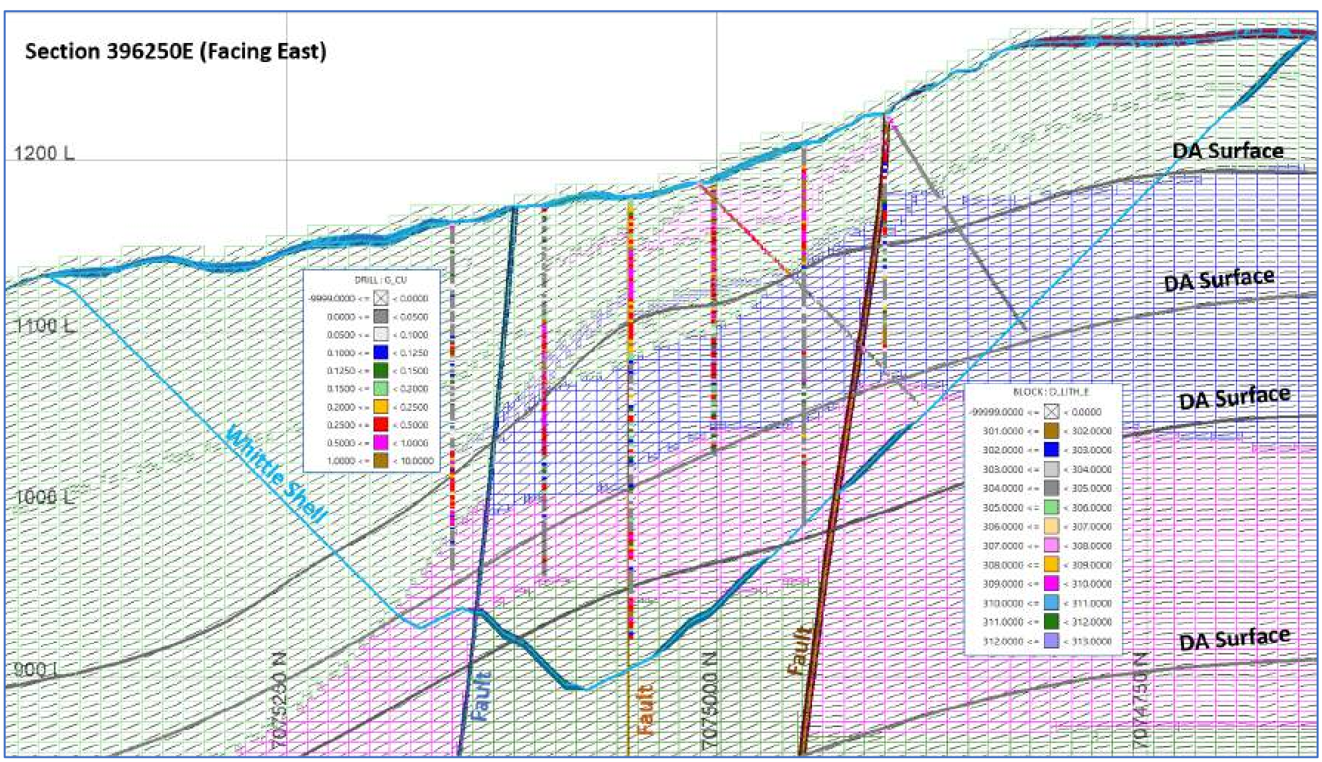
SLR used Vulcan's DA functionality to orient the search ellipsoids according to the orientation of lithological units being estimated. DA angles were generated from several form interpolant surfaces generated in Leapfrog software, generally using two surfaces to populate blocks for each lithology. The DA angles for ES are illustrated in Figure 14-38.

Since there is little apparent displacement on the two central fault blocks, 32 and 34, they were treated as one domain for the purposes of grade interpolation.

To control local extrapolation, SLR used HYR via a flagged field in the database which set searches for all grades for a few selected holes in mineralized fault block 31 to 60 m x 60 m x 10 m.

The QP reviewed the interpolation methodology used, and results obtained by SLR and consider them adequate for the deposit.

Figure 14-38: Vertical Section Showing Dynamic Anisotropy Angles: ES



Source: Figure prepared by SLR, 2023.

Table 14-39: Summary of Search Parameters: ES

Metal Estimates (Density Weighted)						
Parameter	**Units**	**Fault Block**	**Mineralized Envelope**	**Pass 1**	**Pass 2**	**Pass 3**
Search Major	m	All	0, 31	60	120	280
Search Semi-Major	m	All	0, 31	60	120	280
Search Minor	m	All	0, 31	10	20	40
Discretization Steps X	Count	All	0, 31	5	5	5
Discretization Steps Y	Count	All	0, 31	5	5	5
Discretization Steps Z	Count	All	0, 31	5	5	5
Min Samples Per Estimate	Count	31, 32, 33	0, 31	4	4	2
Min Samples Per Estimate	Count	34	31	2	2	2
Max Samples Per Estimate	Count	31, 33, 34	0, 31	8	8	8
Max Samples Per Estimate	Count	32	31	8	6	4
Max Samples Per DDH	Count	All	0, 31	2	2	2
Min DDH per Estimate	Count	All	0, 31	2	2	1
Max DDH per Estimate	Count	31, 33, 34	0, 31	4	4	4
Max DDH per Estimate	Count	32	31	4	3	2
Search Major	m	All	All	70	140	280
Search Semi-Major	m	All	All	70	140	280
Search Minor	m	All	All	10	20	40
Discretization Steps X	Count	All	All	5	5	5
Discretization Steps Y	Count	All	All	5	5	5
Discretization Steps Z	Count	All	All	5	5	5
Min Samples Per Estimate	Count	All	All	2	2	2
Max Samples Per Estimate	Count	All	All	8	8	8
Max Samples Per DDH	Count	All	All	2	2	2
Min DDH per Estimate	Count	All	All	2	2	1
Max DDH per Estimate	Count	All	All	4	4	4

14.4.9 Bulk Density

14.4.9.1 Density Regression

The Estrellita density database provided to SLR in October 2021 contains 784 density measurements, which were taken from a small selection of holes. For the 2024 model, SLR updated the density regression curves for ES separately for copper oxide and sulphide material, in keeping with practice at SD-IN, as follows (Figure 14-39):

$$Density_{Oxide}=2.7533-(0.0008 \times Fe(\%))+(0.0007 \times Fe(\%)^2)$$

$$Density_{Sulphide}=2.7155-(0.0018 \times Fe(\%))+(0.0008 \times Fe(\%)^2)$$

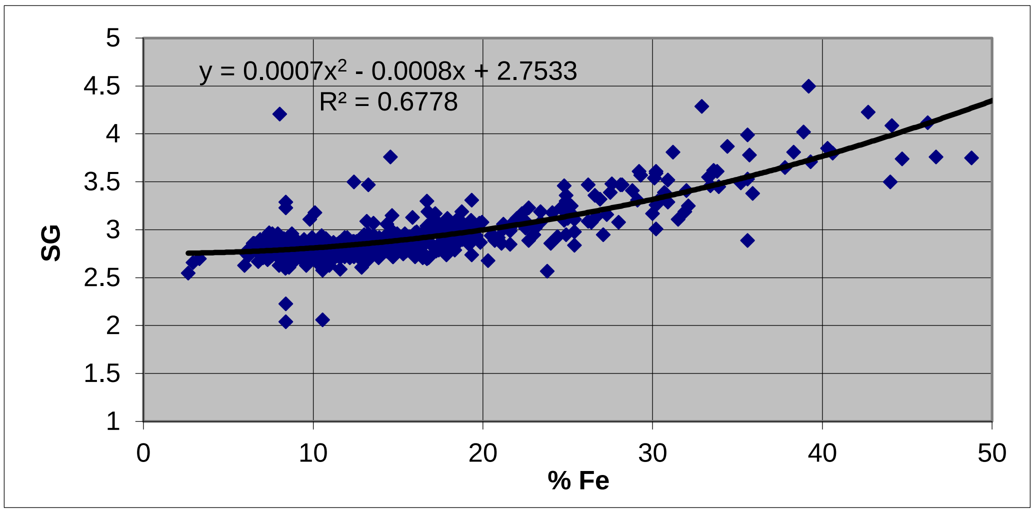
Densities were calculated into flagged drill hole data, with a default of 2.80 where there were no iron assays.

SLR notes that the copper oxide material at ES includes local volumes of sulphide material, and that there is a longer tail of mineralized material in the oxide samples, which may be affecting the overall density of material in the oxide domain. The QP recommends that Capstone perform further review of sequential copper data to refine the oxide boundary, and perhaps model a mixed zone.

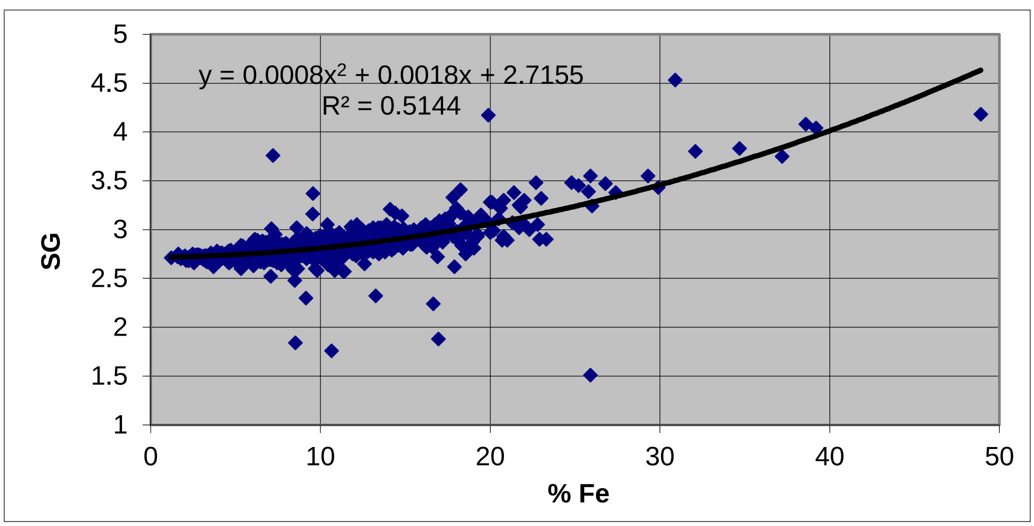
The density regression curves are presented in Figure 14-42.

Figure 14-39: Estrellita Density Regression Curves

Oxide



Sulphide



Source: Figure prepared by SLR, 2023.

14.4.10 Block Models

The block size for the 2024 ES model was changed from the previously used 10 m x 10 m x 5 m dimensions to 12 m x 12 m x 6 m. SLR initially constructed a sub-blocked model with 12 m x 12 m x 6 m parent with 3 m x 3 m x 3 m sub-blocks to estimate all variables using density weighting and DA, and then regularized the model to a final 12 m x 12 m x 6 m model for Whittle analyses and final reporting. The sub-blocked model was split on lithology, oxidation, and fault block wireframes.

The block model parameters for the sub-blocked and regular models are provided in Table 14-40 through Table 14-43, respectively. The block model extents reflect an expansion to cover the resultant Whittle pit shells for 2024.

Table 14-40: ES Sub-Blocked Model Parameters

Block Model Details	
Model name:	ES_Final\bm_es_rsc
History list:	bm_es_rsc02Jul2022.bhst
Format:	extended
Structure:	non-regular
Compressed:	no
Smooth:	no
Number of blocks:	2168919
Number of variables:	165
Number of schemas:	2
Origin:	395410.000000 7074470.000000 748.000000
Bearing/Dip/Plunge:	90.000000 0.000000 0.000000
Created on:	Sat Jul 2 17:04:56 2022
Last modified on:	Sat Jul 2 17:04:56 2022
Block Dimensions	
Schema <Parent>	
Offset minimum:	0.000000 0.000000 0.000000
maximum:	2400.000000 1092.000000 576.000000
Blocks minimum:	12.000000 12.000000 6.000000
maximum:	12.000000 12.000000 6.000000
No of blocks:	200 91 96
Schema <Sub>	
Offset minimum:	0.000000 0.000000 0.000000
maximum:	2400.000000 1092.000000 576.000000
Blocks minimum:	3.000000 3.000000 3.000000
maximum:	12.000000 12.000000 6.000000
No of blocks:	800 364 192

Table 14-41: Block Model Key Variables

Variables	Default	Type	Description
py_pct_calc	-99.0	double	bm_pyrite.bcf
magsus_g_nn	-99.0	double	-
magsus_g	-99.0	double	-
mrecov_pct_nn	-99.0	double	-
mrecov_pct	-99.0	double	-
density_c_nn	-99.0	double	-
density_c	-99.0	double	estimated from density composites per domain
density_bm_calc	-99.0	double	bm_set_density.bcf
da_bearing	0	double	Variable for DA angles
da_plunge	0	double	Variable for DA angles
da_dip	0	double	Variable for DA angles
da_dist_maj	0	double	Variable for DA angles
da_dist_semi	0	double	Variable for DA angles
da_dist_min	0	double	Variable for DA angles
d_lith	-99.0	integer	1 to 11 integers for both SD and IN
d_redox	-99.0	integer	32:oxide 34:sulphide
d_magsus	-99.0	integer	1=inside
d_minz	-99.0	integer	31 = ES Cu envelope
d_buffer	-99.0	integer	150-175m buffer around bluesky_b and ddh traces.
d_fb	-99.0	integer	31=fb1 32=fb2 33=fb3 34=fb4
d_class	0	integer	-
d_dhspc	0	integer	Category to generate Mineral Resource / Mineral Reserve
d_potential	0	integer	Category to generate Mineral Resource / Mineral Reserve
d_area	3	integer	1=SD 2=IN 3=ES
d_air	-99.0	integer	1=air
tonnes_block	0.000	double	volume * density_c
tonnes_cu	0.000	double	cu_pct * tonnes_block
au_sg	-99.0	double	accumulation
co_sg	-99.0	double	accumulation
cu_sg	-99.0	double	accumulation
fe_sg	-99.0	double	accumulation
py_sg	-99.0	double	accumulation
s_sg	-99.0	double	accumulation
au_sg_nn	-99.0	double	accumulation
co_sg_nn	-99.0	double	accumulation
cu_sg_nn	-99.0	double	accumulation
fe_sg_nn	-99.0	double	accumulation
py_sg_nn	-99.0	double	accumulation
s_sg_nn	-99.0	double	accumulation
au_ppm_dw	-99.0	double	bm_es_calc_dw.bcf
co_ppm_dw	-99.0	double	bm_es_calc_dw.bcf

Variables	Default	Type	Description
cu_pct_dw	-99.0	double	bm_es_calc_dw.bcf
fe_pct_dw	-99.0	double	bm_es_calc_dw.bcf
py_pct_dw	-99.0	double	bm_es_calc_dw.bcf
s_pct_dw	-99.0	double	bm_es_calc_dw.bcf
au_ppm_nn_dw	-99.0	double	bm_es_calc_dw.bcf
co_ppm_nn_dw	-99.0	double	bm_es_calc_dw.bcf
cu_pct_nn_dw	-99.0	double	bm_es_calc_dw.bcf
fe_pct_nn_dw	-99.0	double	bm_es_calc_dw.bcf
py_pct_nn_dw	-99.0	double	bm_es_calc_dw.bcf
s_pct_nn_dw	-99.0	double	bm_es_calc_dw.bcf
mined_out_0_1	0.0	double	Artisanal mining 0-1 mined variable
sb_major_m	0.0	double	For mineralization envelope
sb_semi_m	0.0	double	For mineralization envelope
sb_minor_m	0.0	double	For mineralization envelope
au_xt	-99.0	double	es_calc_reg_wts.bcf
co_xt	-99.0	double	es_calc_reg_wts.bcf
cu_xt	-99.0	double	es_calc_reg_wts.bcf
fe_xt	-99.0	double	es_calc_reg_wts.bcf
py_xt	-99.0	double	es_calc_reg_wts.bcf
s_xt	-99.0	double	es_calc_reg_wts.bcf
cueq_2018_calc_xt	-99.0	double	es_calc_reg_wts.bcf
density_c_xt	-99.0	double	es_calc_reg_wts.bcf
ms_xt	-99.0	double	es_calc_reg_wts.bcf
cueq_2018_calc	0.000	double	Cu_eq_2018.ssc from Gemcom adapted to Vulcan
nsr_usdpt	-99.0	double	not including Co
nsr_usdpt_co	-99.0	double	including Co
cueq_pct	-99.0	double	not including Co
cueq_pct_co	-99.0	double	including Co
rock_0_1	0.00	double	mine with es_rock.00t then d_air=0 when 0
d_nn_lith	0	integer	for pit shell extension
d_nn_redox	0	integer	for pit shell extension
d_nn_fb	0	integer	for pit shell extension
da_nn_bearing	0	double	-
da_nn_plunge	0	double	-
da_nn_dip	0	double	-
d_categ_2h	-99.0	integer	-
cueq_pct_co_xt	-99.0	double	-
cueq_pct_xt	-99.0	double	-
nsr_usdpt_co_xt	-99.0	double	-
nsr_usdpt_xt	-99.0	double	-

Table 14-42: ES Regular Model Parameters

Block Model Details	
Model name:	bm_es_rsc_reg12x12x6m_eng
Format:	extended
Structure:	regular
Compressed:	no
Smooth:	no
Number of blocks:	1747200
Number of variables:	44
Number of schemas:	1
Origin:	395410.000000 7074470.000000 748.000000
Bearing/Dip/Plunge:	90.000000 0.000000 0.000000
Created on:	Mon Jul 4 10:16:42 2022
Last modified on:	Mon Jul 4 10:16:42 2022
Block Dimensions	
Schema <parent>	
Offset minimum:	0.000000 0.000000 0.000000
maximum:	2400.000000 1092.000000 576.000000
Blocks minimum:	12.000000 12.000000 6.000000
maximum:	12.000000 12.000000 6.000000
No of blocks:	200 91 96

Table 14-43: Block Model Key Variables

Variables	Default	Type	Description
rock_0_1	0	double	topography % (below topo)
d_lith	0	integer	Lithology
d_class	0	integer	Resource classification
d_potential	0	integer	Internal class by redox and area
d_area	0	integer	Area
d_redox	0	integer	majority d_redox
d_slope	0	integer	majority d_slope
density_c	0	double	Density gr/cm^3
tonnes_block	0	double	Tonnes
cu_pct_dw	0	double	Copper % (official)
fe_pct_dw	0	double	Iron % (official)
au_ppm_dw	0	double	Gold g/t (official)
co_ppm_dw	0	double	Cobalt ppm (official)
s_pct_dw	0	double	Sulphur % (official)
py_pct_dw	0	double	Pyrite % (official)
rmcu2	0	double	Estimated Cu recovery (%)
rmau2	0	double	Estimated Au recovery (%)
Ley_au_conc_m	0	double	Estimated Au grade in Cu con (in g/t)
aupay	0	double	Estimated proportion of payable Au in Cu con (as %)

Variables	Default	Type	Description
conc_cu3	0	Double	Estimated mass of Cu Conc (tonnes)
proc3	0	Double	Estimated Final processing cost in US$/t
nsr3	0	double	NSR (official incl Cu and Au) in US$/t

14.4.11 Classification

Definitions for resource categories used in this report are consistent with those defined by CIM (2014) and adopted by NI 43-101. In the CIM classification, a Mineral Resource is defined as “a concentration or occurrence of solid material of economic interest in or on the Earth’s crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction”. Mineral Resources are classified into Measured, Indicated, and Inferred categories. Economic value (using NSR calculations) was only assigned to blocks within the mixed and primary zones (14.4.13). Thus, mineral resources at ES only include material from within those two zones. Oxides are excluded from the resources at ES.

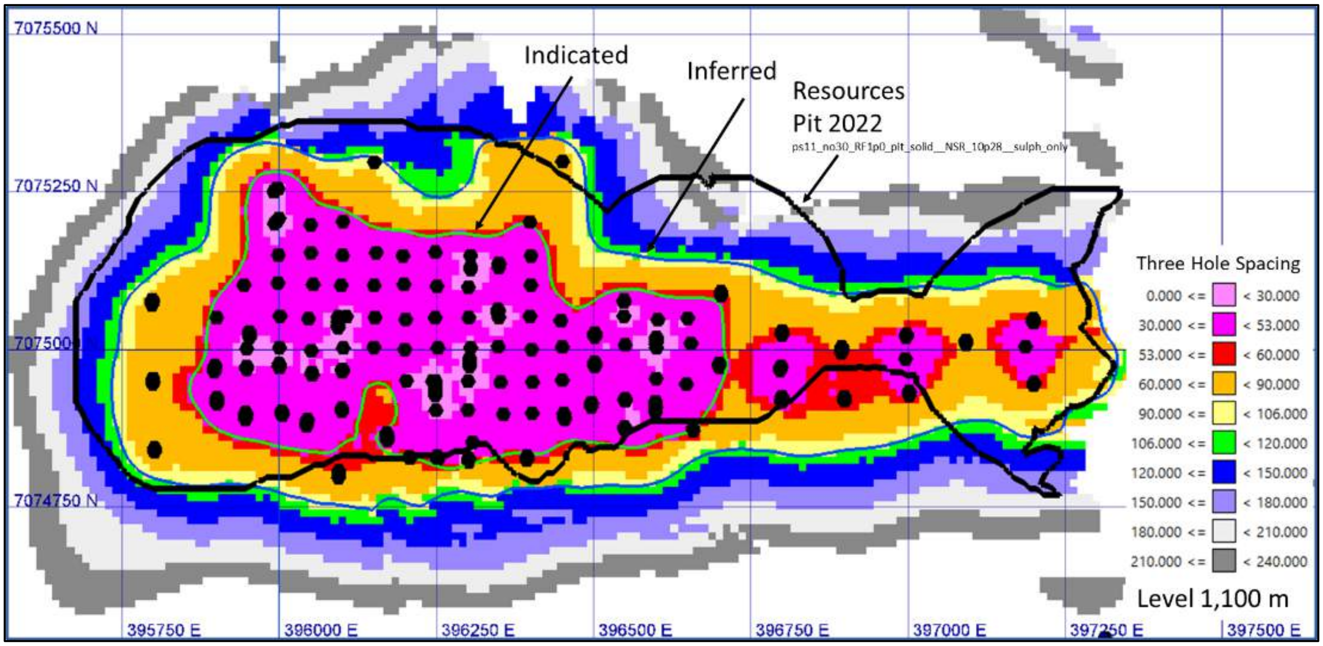
SLR used a scripting methodology to set block categories by interpolating a three-hole distance variable into the block model. An average two-hole distance for each block was then calculated in a script, and the block category was set based on the distance. SLR then reviewed the initial results and performed an iterative subsequent cleanup which excluded holes by indexed name.

In general accord with the variogram results, the classification results are nominally a 75 m two-hole spacing for Indicated material, and a 150 m two-hole spacing for Inferred material. There is no Measured Mineral Resource material at ES. At the outboard edges of the drilling, radii are applied which limit Mineral Resource class extrapolation to nominally half the two-hole distance: 35 m and 75 m (sometimes a little farther) extrapolation from the last hole for Indicated and Inferred, respectively.

Vulcan’s grade shell algorithm was then used to generate solids to flag the regularized model for Mineral Resource classification. A plan view of the classification schema at ES is shown in Figure 14-40 and a cross section is presented in Figure 14-41.

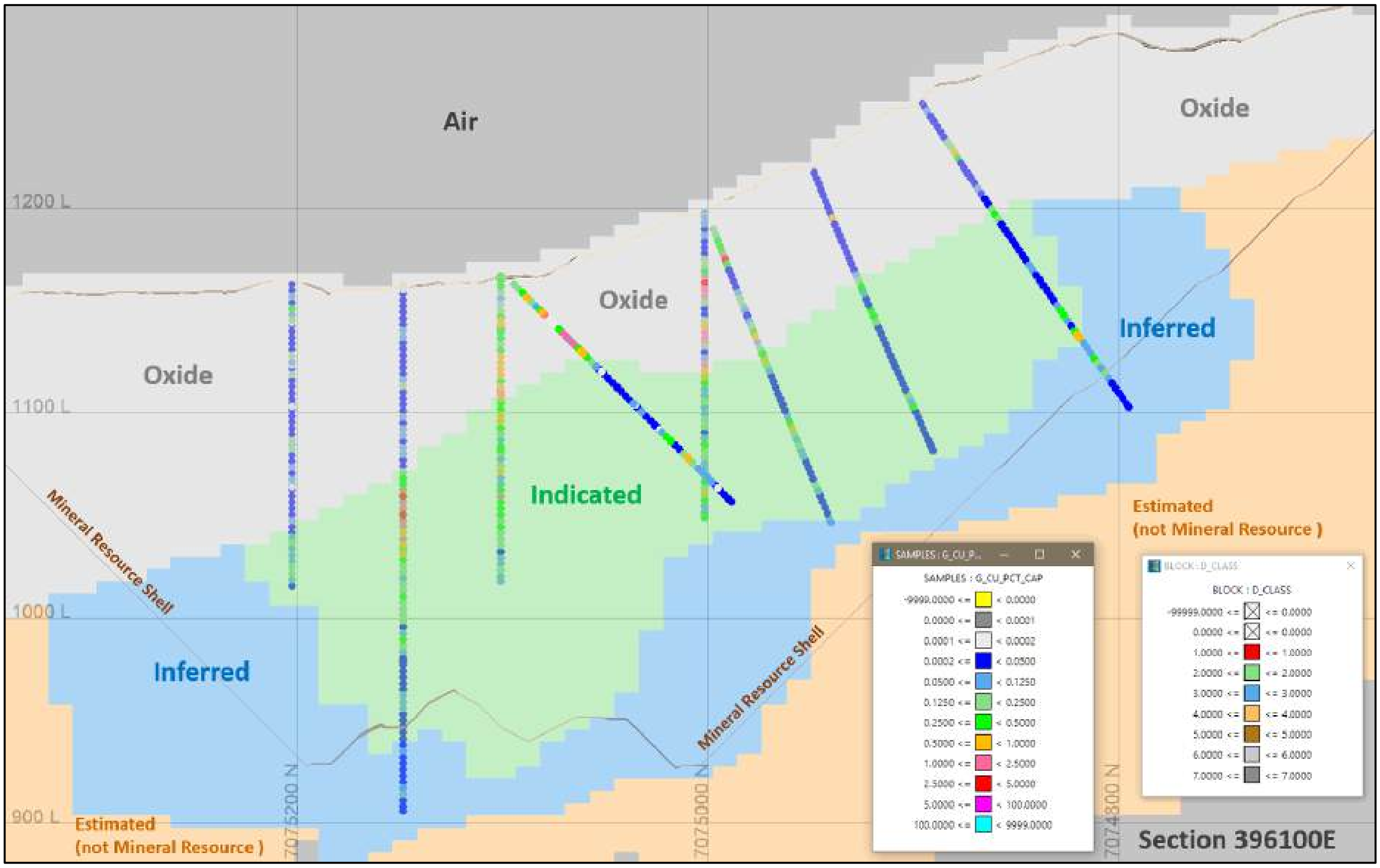
The QP reviewed the classification criteria used, and results obtained by SLR and consider them adequate for the deposit.

Figure 14-40: ES Mineral Resource Classification Plan View



Source: Figure prepared by SLR, 2023.

Figure 14-41: ES Mineral Resource Classification Section 396,100E



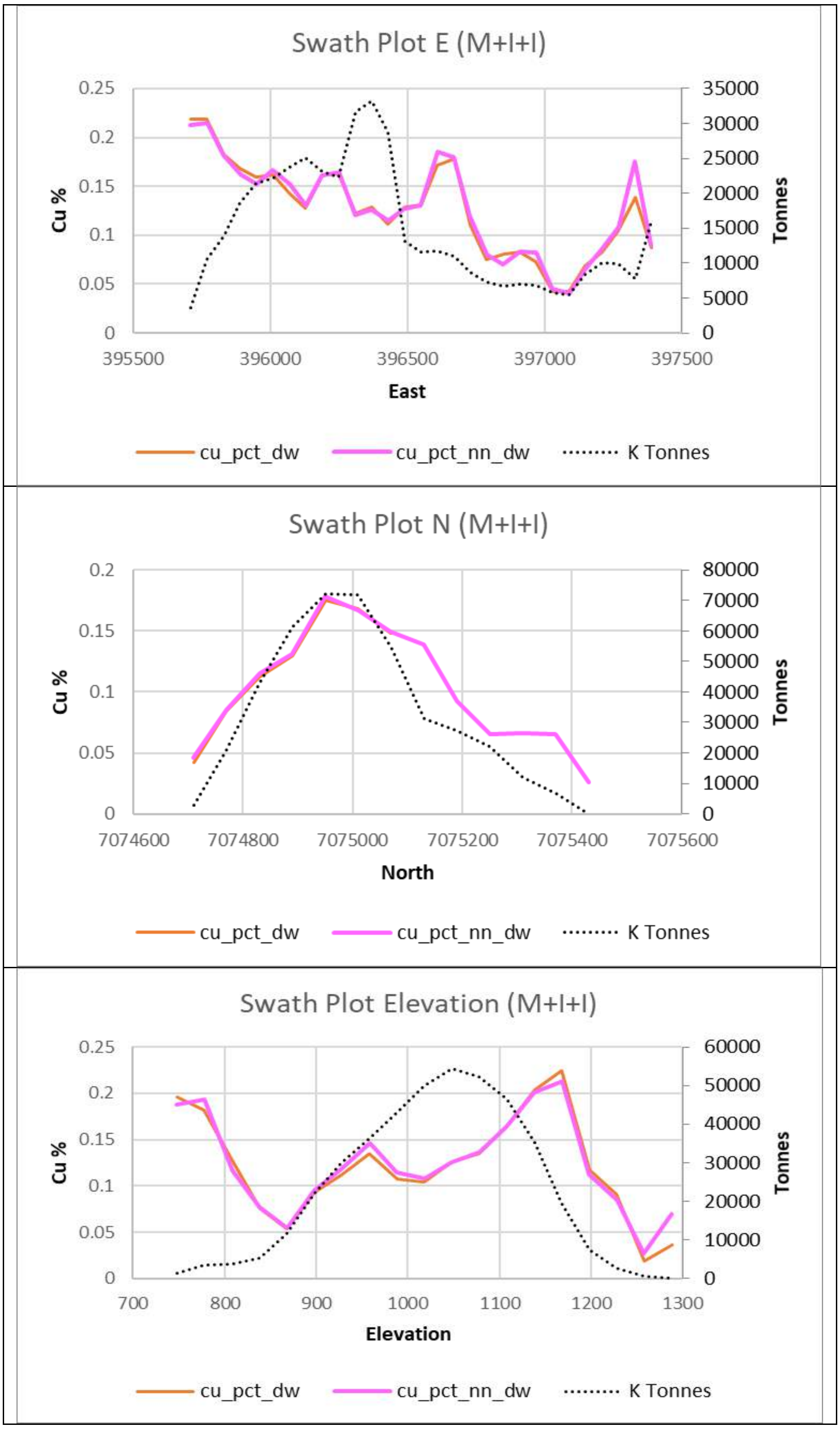
Source: Figure prepared by SLR, 2023.

14.4.12 Block Model Validation

SLR validated the block models using summary statistical comparisons to composites, and NN estimates to compare with the DW results. DA trends were visually validated in Vulcan against input surfaces and drilled grades. NSR and CuEq block model calculations were validated both in the original Excel COG work sheets and in Python-adapted versions of the Vulcan scripts used to calculate them. A summary of the composites versus block model comparison is provided in Table 14-44.

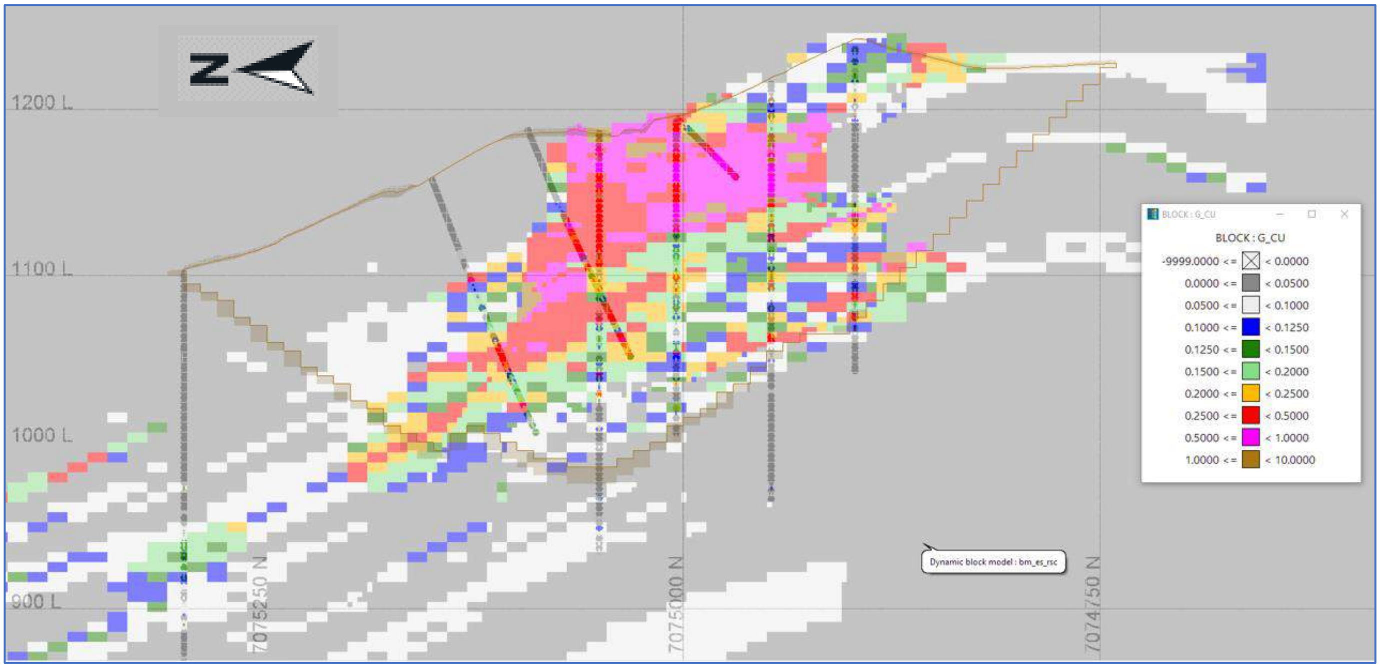
SLR found that visual checks of composite and drilled grades against block grades showed very good spatial control with little smearing, especially in areas of denser drilling. Local HYR was applied to control occasional overextrapolation. Swath plots versus NN estimates were very well correlated overall. Example swath plots are shown in Figure 14-42, and example visual validations are shown in Figure 14-43 through Figure 14-45.

Figure 14-42: ES Swath Plot Example: Cu



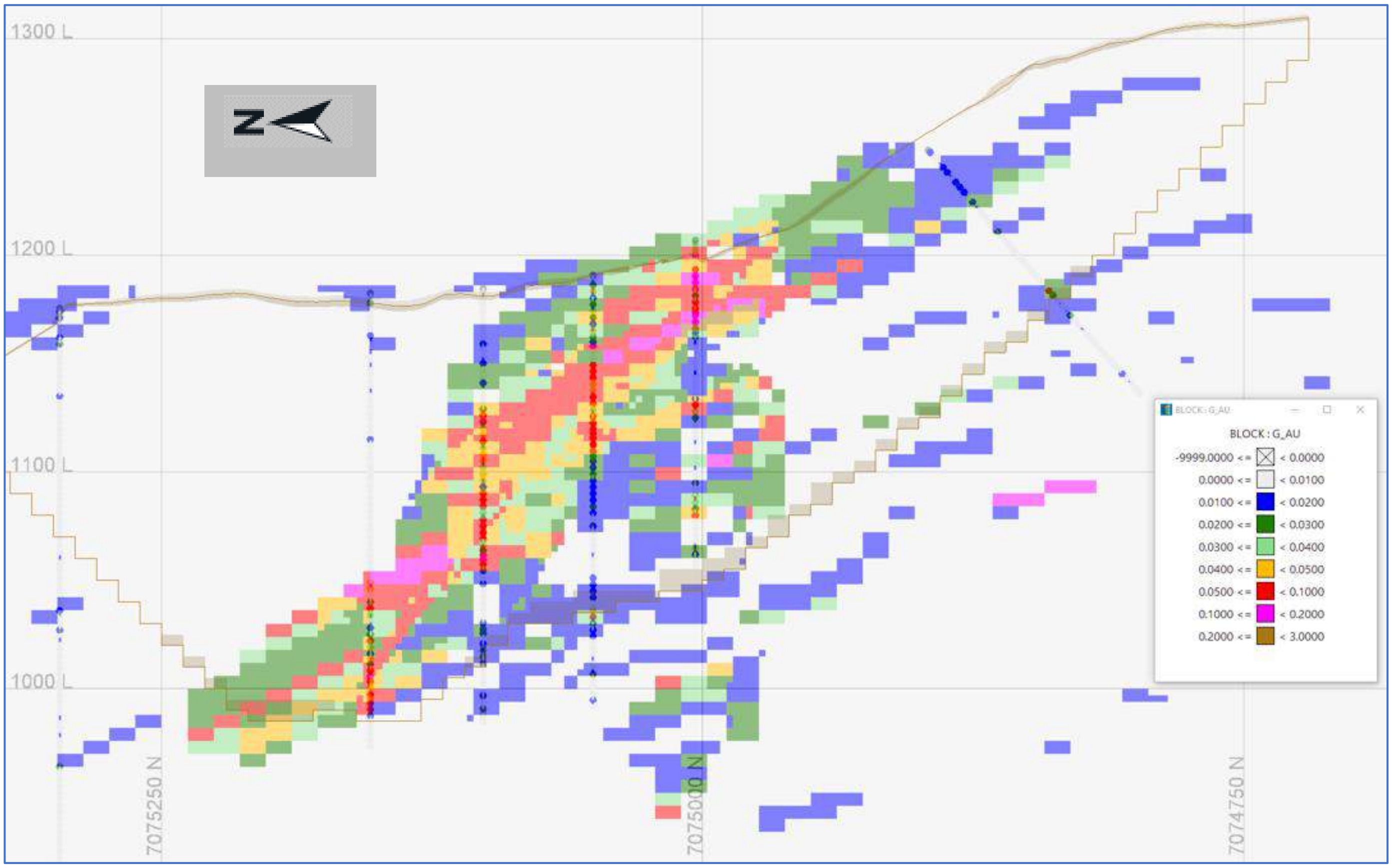
Source: Figure prepared by SLR, 2023.

Figure 14-43: ES Visual Validation Example: Cu Section 396,345E



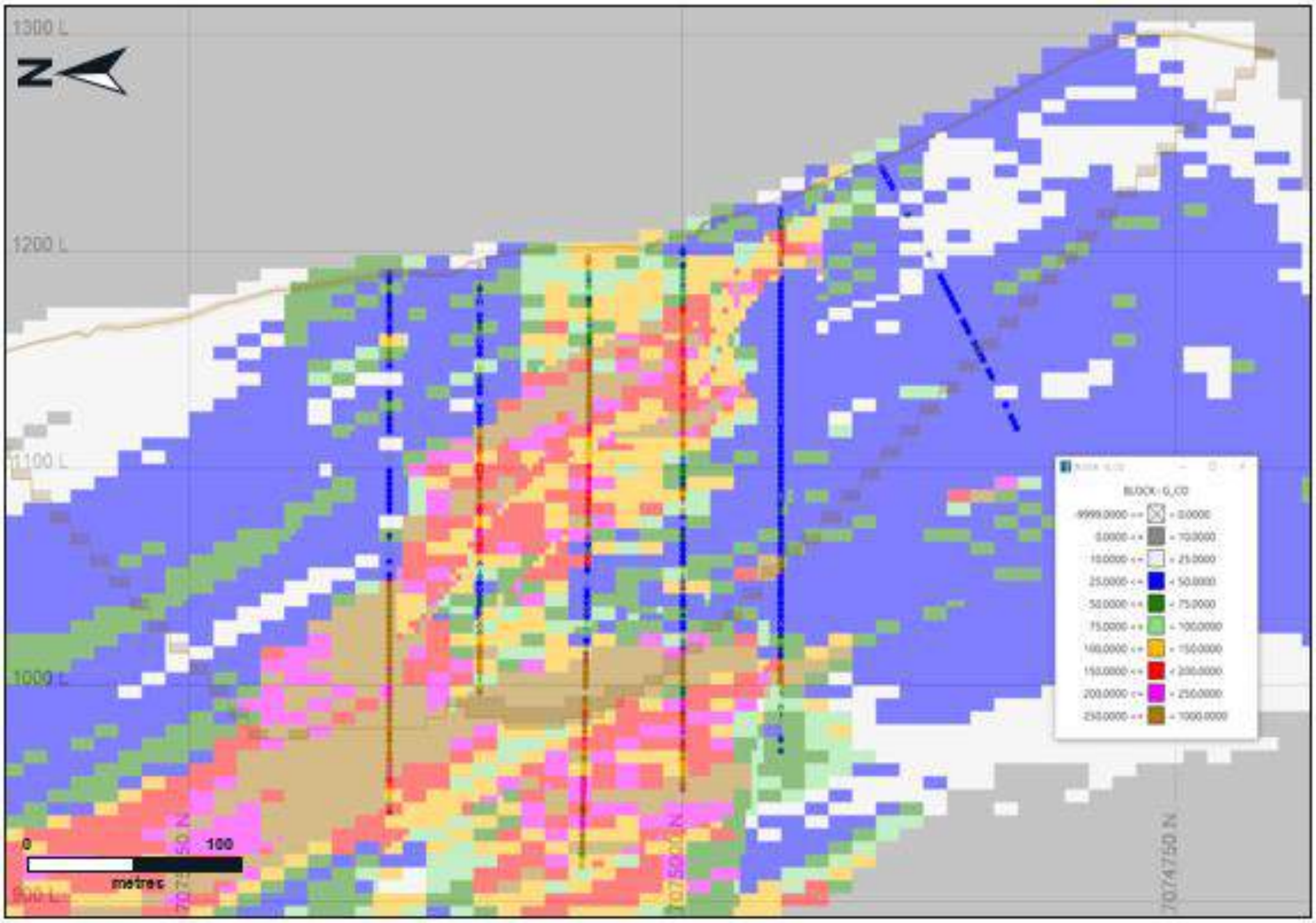
Source: Figure prepared by SLR, 2023.

Figure 14-44: ES Visual Validation Example: Au Section 396,145E



Source: Figure prepared by SLR, 2023.

Figure 14-45: ES Visual Validation Example: Co Section 396,195E



Source: Figure prepared by SLR, 2023.

Table 14-44: Estrellita Block and Composite Statistics Inside Mineral Resource Pit Shell

Column	Units	Domain	Comp Count	BM Count	Comp Min	BM Min	Comp Max	BM Max	Comp Mean	BM Mean	Diff Mean	Diff Mean (%)	Comp CV	BM CV	Diff CV (%)
au	g/t	fb31_minz0	715	57,859	0.0001	0.0001	0.143	0.263	0.004	0.004	0.000	0.0	3.53	2.15	-39.1
au	g/t	fb31_minz31	1,019	32,369	0.0001	0.0001	0.563	0.282	0.028	0.027	-0.001	-5.2	1.47	0.79	-46.3
au	g/t	fb33_minz0	956	67,939	0.0001	0.0001	0.132	0.106	0.003	0.003	0.000	-3.3	2.28	1.40	-38.6
au	g/t	fb3234_minz0	1,286	47,226	0.0001	0.0001	0.108	0.134	0.005	0.006	0.002	24.6	2.31	1.28	-44.6
au	g/t	fb33_minz31	575	21,089	0.0001	0.0001	0.686	0.328	0.027	0.027	0.000	1.1	1.55	0.95	-38.7
au	g/t	fb3234_minz31	2,750	67,612	0.0001	0.0001	0.687	0.356	0.035	0.034	-0.002	-4.4	1.29	0.84	-34.9
co	ppm	fb31_minz0	715	57,859	0.0001	0.0001	725.5	576.6	45.2	40.6	-4.6	-11.3	1.48	1.02	-31.1
co	ppm	fb31_minz31	1,019	32,369	0.0001	17.2471	1440.0	847.3	117.7	121.5	3.8	3.2	1.09	0.62	-43.1
co	ppm	fb33_minz0	956	67,939	0.0001	0.0001	1100.0	942.0	45.4	39.9	-5.5	-13.8	1.80	1.28	-28.9

Column	Units	Domain	Comp Count	BM Count	Comp Min	BM Min	Comp Max	BM Max	Comp Mean	BM Mean	Diff Mean	Diff Mean (%)	Comp CV	BM CV	Diff CV (%)
co	ppm	fb3234_minz0	1,286	47,226	0.0001	0.0001	1100.0	984.4	68.9	66.1	-2.9	-4.4	1.64	1.13	-31.1
co	ppm	fb33_minz31	575	21,089	0.0001	5.3022	1030.8	748.9	121.6	122.6	1.0	0.8	1.16	0.83	-28.4
co	ppm	fb3234_minz31	2,750	67,612	0.0001	0.0001	1618.9	1080.0	128.5	133.9	5.3	4.0	1.18	0.79	-33.1
cu	%	fb31_minz0	715	57,859	0.0001	0.0001	1.08	1.66	0.05	0.05	0.00	-1.9	2.20	1.38	-37.3
cu	%	fb31_minz31	1,019	32,369	0.0001	0.0053	3.42	2.15	0.29	0.27	-0.02	-7.5	1.31	0.67	-48.9
cu	%	fb33_minz0	956	67,939	0.0001	0.0001	0.43	0.87	0.04	0.04	0.00	2.9	1.13	0.77	-31.9
cu	%	fb3234_minz0	1,286	47,226	0.0001	0.0001	0.86	1.57	0.05	0.06	0.01	17.2	1.60	1.11	-30.6
cu	%	fb33_minz31	575	21,089	0.0001	0.0291	7.59	1.89	0.28	0.27	-0.01	-2.2	1.49	0.87	-41.6
cu	%	fb3234_minz31	2,750	67,612	0.0001	0.0001	5.41	3.06	0.35	0.34	-0.02	-5.0	1.23	0.79	-35.8
density	g/cm^3	fb31_minz0	715	57,859	2.72	2.74	3.27	3.45	2.79	2.83	0.04	1.3	0.02	0.02	0.0
density	g/cm^3	fb31_minz31	1,019	32,369	2.72	2.76	4.25	3.92	2.89	2.87	-0.02	-0.8	0.05	0.02	-60.0
density	g/cm^3	fb33_minz0	956	67,939	2.72	2.72	2.96	3.47	2.80	2.81	0.01	0.3	0.01	0.01	0.0
density	g/cm^3	fb3234_minz0	1,286	47,226	2.72	2.72	3.22	3.30	2.80	2.81	0.02	0.6	0.01	0.02	100.0
density	g/cm^3	fb33_minz31	575	21,089	2.74	2.75	4.36	3.71	2.89	2.87	-0.02	-0.9	0.06	0.03	-50.0
density	g/cm^3	fb3234_minz31	2,750	67,612	2.72	2.74	4.78	4.27	2.91	2.88	-0.02	-0.8	0.06	0.04	-33.3
fe	%	fb31_minz0	715	57,859	0.0001	0.0001	24.45	28.85	8.17	7.29	-0.88	-12.0	0.29	0.39	34.5
fe	%	fb31_minz31	1,019	32,369	0.0001	3.9924	44.48	41.12	12.53	12.42	-0.11	-0.9	0.39	0.21	-46.2
fe	%	fb33_minz0	956	67,939	0.0001	0.0001	16.29	16.97	8.07	7.92	-0.15	-1.9	0.36	0.28	-22.2
fe	%	fb3234_minz0	1,286	47,226	0.0001	0.0001	23.25	23.03	8.18	8.14	-0.04	-0.5	0.28	0.23	-17.9
fe	%	fb33_minz31	575	21,089	0.0001	4.568	44.20	36.64	12.37	12.20	-0.17	-1.4	0.43	0.31	-27.9
fe	%	fb3234_minz31	2,750	67,612	0.0001	0.0001	49.73	44.51	13.19	12.92	-0.26	-2.0	0.42	0.31	-26.2
s	%	fb31_minz0	715	57,859	0.0001	0.0001	3.20	3.89	0.10	0.09	-0.01	-7.5	3.34	2.39	-28.4
s	%	fb31_minz31	1,019	32,369	0.0001	0.0057	7.36	4.18	0.42	0.47	0.05	11.0	1.57	0.85	-45.9
s	%	fb33_minz0	956	67,939	0.0001	0.0001	6.00	4.53	0.09	0.08	-0.02	-22.0	3.90	2.95	-24.4
s	%	fb3234_minz0	1,286	47,226	0.0001	0.0001	5.00	3.88	0.24	0.21	-0.03	-16.3	2.29	1.80	-21.4
s	%	fb33_minz31	575	21,089	0.0001	0.0061	6.02	3.62	0.48	0.50	0.02	4.4	1.52	1.09	-28.3
s	%	fb3234_minz31	2,750	67,612	0.0001	0.0001	6.92	5.47	0.47	0.51	0.04	6.9	1.42	1.03	-27.5

14.4.13 Net Smelter Return Cut-off Value and Whittle Parameters

Revenue in the Estrellita project will be generated from the sale of copper concentrates. In addition, the copper concentrate contains payable quantities of gold. Only mixed and primary (sulphide) materials were considered in the estimation of the resources. Oxides are not considered part of the resource at this stage of the project.

Metal prices used for revenue estimates were advised by Capstone and were \$4.10/lb for copper and \$1,600/oz for gold.

14.4.13.1 NSR Cut-Off Value

To keep the resources at Estrellita consistent and comparable with the resources at SD-IN, the Mineral Resources at Estrellita are reported utilizing an NSR, based on metallurgical test results and current long term metal pricing supplied by Capstone. The NSR (\$/t) was assigned to each block by combining the contribution from each metal. Table 14-24 summarizes parameters used for the NSR estimate.

14.4.13.2 Whittle Parameter

The parameters for the NSR calculation at Estrellita are the same as those used for SD-IN (Table 14-25), except that Estrellita only accrues value (and their associated costs) from Cu and Au. Prior to the export of the resource model into the Whittle pit optimization software, the in-situ NSR value was first calculated and coded into each block in the model, to allow the pit optimization to be carried out on the in-situ NSR values.

The Whittle model with block size of 12 m x 12 m x 12 m contains material types (oxide, sulphide mineralization, and barren rock) and pre-calculated NSR values. The optimizer uses the LG algorithm to determine the economic pit limits based on the block NSR value and estimates for pit slope angles.

Pit overall slope angle of 43° was used, which is an average angle of the Santo Domingo and Iris Norte pit slopes.

The internal (or mill) cut-off value of $9.63/t milled is estimated for the processing option without iron. The NSR cut-off value incorporates all operating costs except mining. The operating costs were provided by Capstone.

As was stated earlier for Santo Domingo, no Copper Equivalent values are used, and results are tabulated based on the NSR cut-off value.

14.4.14 Estrellita Mineral Resources

14.4.14.1 2024 Estrellita Mineral Resources

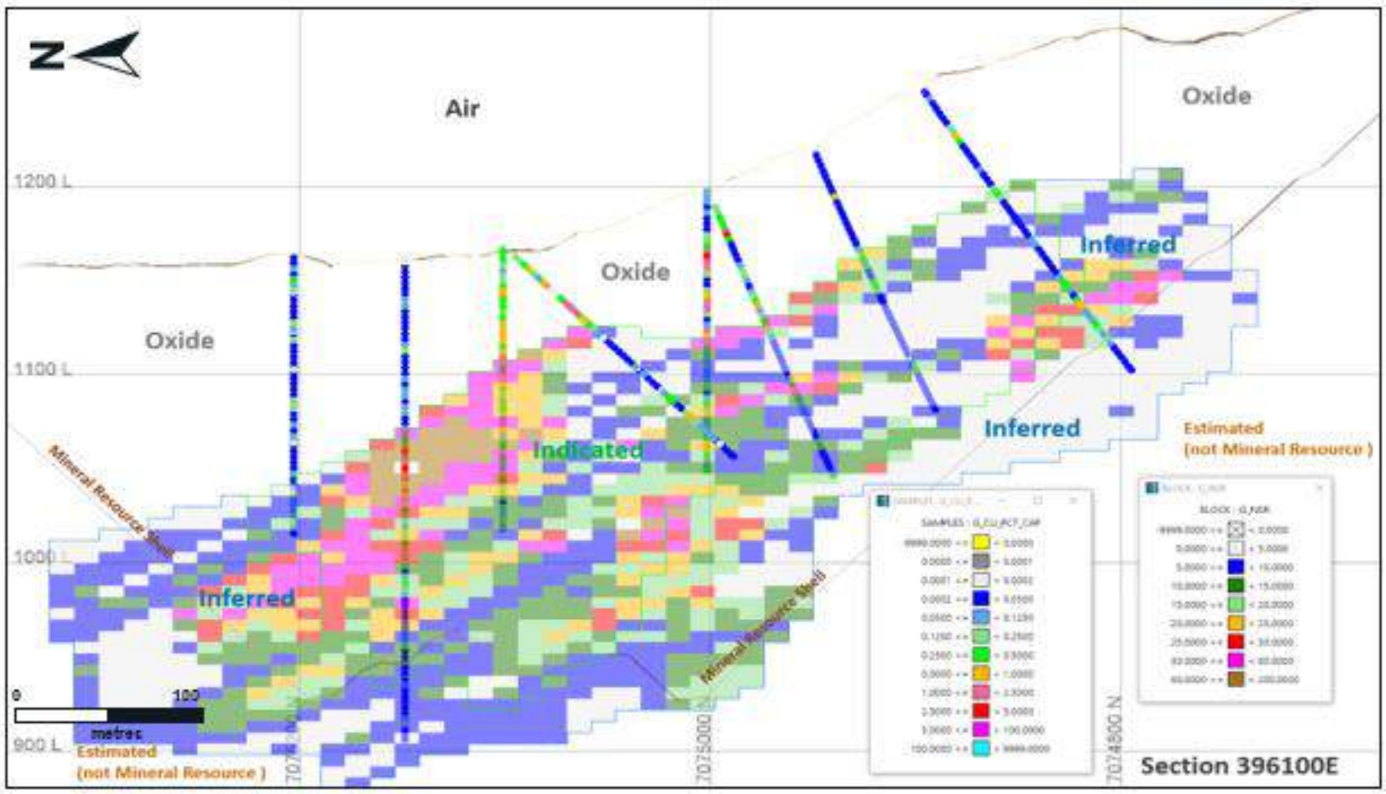
The Mineral Resource estimates for ES with an effective date of 31 March 2024, are summarized in Table 14-45. The Mineral Resource estimate is based on a Whittle shell optimization at an NSR cut-off value of US$9.63/t. The regular block model was used to report the final Mineral Resource estimate. A cross-section showing the relationships between the Whittle shell, NSR block grades and Mineral Resource classes is provided in Figure 14-46.

Table 14-45: Estrellita Mineral Resources as of 31 March 2024

Category	Tonnage (Mt)	Grade			Contained Metal	
		NSR ($/t)	Cu (%)	Au (g/t)	Cu (kt)	Au (koz)
Indicated	41	24	0.32	0.03	133	44
Total Indicated	**41**	**24**	**0.32**	**0.03**	133	**44**
Inferred	27	25	0.34	0.03	93	29
Total Inferred	**27**	**25**	**0.34**	**0.03**	93	**29**

Notes: **1.** Mineral Resources are classified according to Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves dated May 10, 2014 (CIM (2014) definitions). **2.** Mineral Resources are reported inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. **4**. Mineral Resources for the Estrellita deposit have an effective date of 31 March 2024. **5.** Mineral Resources for the Estrellita deposit are reported using an NSR cut-off value of US$9.63/t. NSR is calculated using average long-term prices of US$4.10/lb Cu, and US$1,600/oz Au. **6.** Only copper, and gold were considered in the NSR calculation; iron, cobalt and sulphur were excluded. **7.** Only mixed and primary material were recognized in the NSR calculation; oxides were excluded. **8**. Mineral Resources are constrained by preliminary pit shells generated using a Lerchs–Grossmann algorithm and the following assumptions: pit slopes 43º; mining cost of US$1.55/t, processing cost of US$9.46/t (including G&A cost); processing recovery are calculated based in the recovery curves for copper and gold, and metal prices of US$4.10/lb Cu, and US$1,600/oz Au. **9.** Rounding as required by reporting standards may result in apparent summation differences. **10.** Tonnage measurements are in metric units. Copper is reported as percentage, and gold as grams per tonne.

Figure 14-46: ES Mineral Resource Classification Section 396,100E



Source: Figure prepared by SLR, 2023.

14.4.14.2 Sensitivity of Resources to Cut-off

Table 14-46 shows the sensitivity of the total reported resources at SD-IN to changes to the NSR cut-off used, for a range of cut-offs both lower and higher than the one used in reporting (US$9.63/t, highlighted in gray in the tables). At higher cut-offs, the increase in grade does not compensate for the decrease in tonnage, yielding lower contained metal. Estrellita is much more sensitive to changes in the cut-off than SD-IN.

Table 14-46: ES Sensitivity of Resources to Cut-off

NSR Cut-off ($/t)	Tonnage (Mt)	Grade			Contained Metal	
		NSR ($/t)	Cu (%)	Au (g/t)	Cu (kt)	Au (koz)
7.70	78	23	0.30	0.03	237	77
8.67	73	24	0.32	0.03	232	75
9.63	68	25	0.33	0.03	226	73
10.59	64	26	0.34	0.03	220	71
11.56	59	27	0.36	0.04	213	69
12.52	55	28	0.37	0.04	206	67
14.45	48	30	0.40	0.04	193	63
Relative Differences						
NSR Cut-off ($/t)	Tonnage (Mt)	Grade			Contained Metal	
		NSR ($/t)	Cu (%)	Au (g/t)	Cu (kt)	Au (koz)
7.70	14%	-8%	-8%	-8%	5%	5%
8.67	7%	-4%	-4%	-4%	3%	2%
9.63	0%	0%	0%	0%	0%	0%
10.59	-6%	4%	4%	4%	-3%	-3%
11.56	-13%	9%	9%	9%	-6%	-6%
12.52	-19%	13%	13%	13%	-9%	-9%
14.45	-30%	22%	22%	22%	-15%	-15%

14.4.14.3 Comparison with the ES 2020 Mineral Resource

Comparisons to the ES 2020 Mineral Resources are provided in Table 14-47 and Table 14-48. Changes, updates, and improvements to the Mineral Resource estimate overall resulted in significant reductions in tonnage and grade for the Indicated resources: tonnage decreased by 25%, copper grade decreased by 15%, gold grade decreased by 20% , resulting in a reduction of contained metal by 36% for copper and 37% for gold in Indicated Mineral Resources; this is partly compensated by approximately five times more contained copper and gold in Inferred Mineral Resources.

The 2024 Mineral Resource estimate incorporates substantial improvements and changes to nearly every component of the modelling process. As such, comparison with the previous model is only valid in a global sense. Changes from the 2020 Mineral Resources include:

- Updated NSR calculations based on revised metal price assumptions to reflect 2024 pricing, and updated recovery factors and payable metal assumptions based on the most up to date metallurgical test work;
- Additional drilling at ES in 2007 and 2008 since the last Mineral Resource model was constructed in 2007;
- Dynamic Anisotropy to vary estimation direction with mineralized trends;
- Discarded old domaining system in favour of domaining by fault block, since DA covered lateral extension and lithology did not appear to control mineralization as well as structure;
- Completely recompiled and validated Mineral Resource database to increase confidence;
- Updated density database with additional samples, discarded lower quality subset;

- Updated, more sophisticated regression formulae for density estimate;
- New geological interpretation and model based on geochemical analyses of rock types;
- New oxide model;
- Density weighted estimation.

Table 14-47: ES Comparison with Previous Mineral Resource with Co

Category	2024 Mineral Resource Estimate						2020 Mineral Resource Estimate					
	Tonnage (Mt)	Grade			Contained Metal		Tonnage (Mt)	Grade			Contained Metal	
		NSR ($/t)	Cu (%)	Au (g/t)	Cu (kt)	Au (koz)		NSR ($/t)	Cu (%)	Au (g/t)	Cu (kt)	Au (koz)
Indicated	41	24	0.32	0.03	133	44	55	0.40	0.38	0.04	209	69
Total Indicated	**41**	**24**	**0.32**	**0.03**	133	**44**	**55**	**0.40**	**0.38**	**0.04**	209	**69**
Inferred	27	25	0.34	0.03	93	29	5	0.32	0.31	0.03	15	5
Total Inferred	**27**	**25**	**0.34**	**0.03**	93	**29**	**5**	**0.32**	**0.31**	**0.03**	15	**5**

Notes: **1.** Mineral Resources are classified according to Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves dated May 10, 2014 (CIM (2014) definitions. **2.** Mineral Resources are reported inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. **3.** The Qualified Person for the February 13, 2020, Mineral Resource estimate was Mr. David Rennie, P. Eng., an associate of SLR Consulting (Canada) Ltd. (formerly Roscoe Postle Associates Inc.). **b 4**. Metal Prices and COG/COV: **a.** February 13, 2020, Mineral Resources are reported using a cut-off grade of 0.125% CuEq. CuEq grades are calculated using average long-term prices of US$3.50/lb Cu, US$1,300/oz Au and US$99/dmt Fe conc. Only copper, gold and iron were recognized in the CuEq calculation, cobalt was excluded. **b.** 31 March 2024, Mineral Resources are using a NSR cut-off value of US$9.63/t. NSR is calculated using average long-term prices of US$4.10/lb Cu, and US$1,600/oz Au. Only copper and gold were used in the NSR calculation. **5.** Whittle Pit Shell Parameters: **a.** February 13, 2020 Mineral Resources are constrained by preliminary pit shells derived using a Lerchs–Grossmann algorithm and the following assumptions: pit slopes averaging 45º; mining cost of US$1.90/t, processing cost of US$7.27/t (including G&A cost); processing recovery of 89% copper, 78.4% Co, and 79% gold; and metal prices of US$3.50/lb Cu, US$1,300/oz Au and US$99/dmt iron concentrate. **b.** 31 March 2024, Mineral Resources are constrained by preliminary pit shells generated using a Lerchs–Grossmann algorithm and the following assumptions: pit slopes 43º; mining cost of US$1.55/t, processing cost of US$9.46/t (including G&A cost); processing recovery are calculated based in the recovery curves for copper and gold, and metal prices of US$4.10/lb Cu, and US$1,600/oz Au. **6.** Rounding as required by reporting standards may result in apparent summation differences. Tonnage measurements are in metric units. Copper is reported as percentage and gold as grams per tonne.

Table 14-48: ES Comparison with 2020 TR Mineral Resource: Relative Differences

Category	Tonnage (Mt)	Grade		Contained Metal	
		Cu (%)	Au (g/t)	Cu (kt)	Au (koz)
Indicated	-25%	-15%	-17%	-36%	-37%
Total Indicated	**-25%**	**-15%**	**-17%**	**-36%**	**-37%**
Inferred	443%	10%	12%	502%	488%
Total Inferred	**443%**	**10%**	**12%**	**502%**	**488%**

Note: Relative Difference calculated as: relative difference = 100%*(value2024 – value2020)/value2020

14.5 Summary Resource Estimate for the Santo Domingo Project

The combined Resource Estimates for the whole Santo Domingo property, including Santo Domingo, Iris Norte and Estrellita are shown on Table 14-49. The Mineral Resources for the Santo Domingo property have an effective date of 31 March 2024.

Table 14-49: Santo Domingo Project Mineral Resources as of 31 March 2024

Category	Tonnage (Mt)	Grade				Contained Metal		
		NSR	Cu	Fe	Au	Cu	Fe	Au
		($/t)	(%)	(%)	(g/t)	(kt)	(Mt)	(koz)
Measured								
Santo Domingo	134	46	0.51	26.9	0.07	679	36	293
Total Measured	**134**	**46**	**0.51**	**26.9**	**0.07**	679	**36**	**293**
Indicated								
Santo Domingo	298	34	0.26	25.7	0.04	786	77	362
Iris Norte	74	28	0.14	24.0	0.02	106	18	43
Santo Domingo + Iris Norte	372	33	0.24	25.4	0.03	892	95	405
Estrellita	41	24	0.32	-	0.03	133	-	44
Total Indicated	**413**	**32**	**0.25**	**N/A**	**0.03**	1,025	**95**	**449**
Total Measured + Indicated	**547**	**35**	**0.31**	**N/A**	**0.04**	1,704	**131**	**742**
Inferred								
Santo Domingo	173	28	0.20	22.2	0.03	349	38	157
Iris Norte	30	28	0.12	23.9	0.01	35	7	14
Santo Domingo + Iris Norte	203	28	0.19	22.5	0.03	384	46	171
Estrellita	27	25	0.34	-	0.03	93	-	29
Total Inferred	**230**	**28**	**0.21**	**N/A**	**0.03**	477	**46**	**200**

Notes: **1.** Mineral Resources are classified according to CIM (2014) definitions. **2.** Mineral Resources are reported inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. **4.** Mineral Resources for the Santo Domingo, Iris Norte and Estrellita deposits have an effective date of 31 March 2024. **5.** Only mixed and primary material were recognized in the NSR calculation; oxides were excluded. **6.** The average Iron grade for the Project Total Indicated, Total Measured plus Indicated, and Total Inferred Resources cannot be calculated because Estrellita does not contain iron resources. **7.** Notes specific to the Mineral Resources for the Santo Domingo and Iris Norte deposits: **a.** Mineral Resources for SD include Iris. **b.** Mineral Resources are reported using a net smelter return (NSR) cut-off value of US$9.85/t. NSR is calculated using average long-term prices of US$4.10/lb Cu, US$1,600/oz Au, and Fe prices that depend on the expected grade of the Fe concentrate (US$94.75/dmt or $129.77/dmt or $140.26/dmt Fe concentrate). **c.** Only copper, gold, and iron were recognized in the NSR calculation; cobalt and sulphur were excluded. **d.** Mineral Resources are constrained by preliminary pit shells derived using a Lerchs–Grossmann algorithm and the following assumptions: pit slopes 36.3°- 47.9°; mining cost is calculated using a function that depends on where the material comes from (Santo Domingo or Iris Norte) and its destination (dumps, plant or stock), processing cost based on Fe concentrate routing code (including general and administrative (G&A) costs); processing recovery are calculated based in the recovery curves for copper and gold. Iron recoveries are calculated based on magnetic susceptibility. Metal prices used are those listed on Note 5b. **8.** Notes specific to the Mineral Resources for the Estrellita deposit: **a.** Mineral Resources are reported using an NSR cut-off value of US$9.63/t. NSR is calculated using average long-term prices of US$4.10/lb Cu, and US$1,600/oz Au. **b.** Only copper, and gold were considered in the NSR calculation; iron, cobalt and sulphur were excluded. **c.** Mineral Resources are constrained by preliminary pit shells generated using a Lerchs–Grossmann algorithm and the following assumptions: pit slopes 43º; mining cost of US$1.55/t, processing cost of US$9.46/t (including G&A cost); processing recovery are calculated based in the recovery curves for copper and gold, and metal prices of US$4.10/lb Cu, and US$1,600/oz Au. **9.** Rounding as required by reporting standards may result in apparent summation differences. **10.** Tonnage measurements are in metric units. Copper and iron are reported as percentages (%) and gold as grams per tonne (g/t).

The risk factors that could potentially affect the Mineral Resources estimates include mostly the assumptions used to generate the conceptual data for consideration of reasonable prospects of eventual economic extraction. Those include:

- Commodity price assumptions
- Exchange rate assumptions
- Density assumptions

- Geotechnical and hydrogeological assumptions
- Operating and capital cost assumptions
- Metal recovery assumptions
- Concentrate grade and smelting/refining terms
- Changes in interpretations of mineralization geometry and continuity of mineralization zones.

There are no environmental, permitting, legal, title, taxation, socio-economic, marketing, political or other relevant factors known to the QP, other than those discussed in this Report, which could affect the Mineral Resource estimates.

14.6 Conclusions and Recommendations

14.6.1 SD-IN Conclusions

Regarding the Mineral Resource estimate for Santo Domingo–Iris Norte, the QP makes the following conclusions:

- The new geological model improves greatly upon the previous model.
- Updated structure and oxidation modelling also improves upon previous.
- The Mineral Resource database is well constructed and sufficient for Mineral Resource estimation.
- Capping levels for all metals are light since grade distribution shows long continuous tails at higher grades.
- Analysis of metal grades at SD-IN shows correlation between gold and copper, and iron and cobalt, but not iron and copper.
- Dynamic anisotropy is an effective tool to interpolate grade along its orientations in the deposit.
- Variograms and visual review show strongest continuity up to 75 m from holes and lesser continuity out to 130 m to 150 m from holes.
- Mass recovery estimates, including the chosen cut-off grades for magnetic susceptibility, are sensible given the intensity and continuity of iron mineralization through the different lithologies.
- Density weighting (DW) produces approximately 2-7% more grade than non-DW estimation.
- New NSR and class criteria produce more tonnage at slightly higher grade (for Cu and Au) and slightly lower grade for Fe.
- Density sampling is insufficient to split out regression populations by SD and IN, while for the combined SD-IN, three regression populations, for oxide, mixed, and sulphide, have been produced.
- Quadratic curves fit the density information better than linear.

- The oxidation boundaries contain some uncertainty, because the sequential copper dataset contains less complete, irregular coverage, approximately 8% of the samples at SD-IN have soluble copper information, compared to the total copper assays. There are few available soluble copper assays to define a mixed domain.

14.6.2 SD-IN Recommendations

Regarding the Mineral Resource estimate for Santo Domingo–Iris Norte, the QP makes the following recommendations:–Drill new holes between SD and IN to delineate the transitional volume, especially considering that the host rock of the mineralization appears to be shallower in this area.

- Explore the unestimated northwest area of the Santo Domingo model to determine the depth of mineralization and outline any continuity with the adjacent Estrellita deposit.
- Correct minor interpretation inconsistencies in the IN geological model where they do not match SD rock types.
- Review sequential copper and metallurgical data further and refine oxide and mixed boundaries.
- Incorporate lithogeochemical and sequential copper analyses into the workflow to refine oxidation interpretation and ore characteristics.
- Review and refine the oxide model contacts in context of additional cyanide and soluble copper assays.
- Ensure that density sampling in oxide and mixed material is carried out at regular intervals in all future drilling.
- Drill additional holes at SD to upgrade Inferred Mineral Resource material to Indicated Mineral Resources.
- Refine redox, geological, and structural model to unify SD and IN. Create one large model spanning the entire block model volume.
- Continuously monitor the database for negative values and reset them according to database conventions.
- Re-assay overlimit assays. If unable to re-assay, set overlimits at overlimit value + 0.0001 to avoid artificially reducing grade in the model.
- Continue detailed correlation work on metals by area and lithology.

14.6.3 ES Conclusions

Regarding Mineral Resource estimate for Estrellita, the QP makes the following conclusions. In a comparison of the updated domains, classifications, and resource pit volume versus those from the 2020 resource estimation:

- The new Mineral Resource excludes oxide material.
- The inclusion of data from additional drill holes (41 holes for a total of 6,424 m, including 10 twin holes, or 31 holes for a total of 4,908 m without the twins) tends to reduce the average grade.
- The DA interpolation in the block model extrapolates along lithological orientations.
- Grade often truncates against new interpreted fault boundaries, better representing drilled grades.

- The new classification schema encloses larger volumes, especially in Inferred Mineral Resource material.
- Sequential copper analysis suggests that oxide boundaries may be inaccurate in the range of five to fifteen metres in places.
- Mineralization appears to be less restricted by lithology and more restricted by structure than SD but is still banded to some extent.
- Variograms show similar behaviour to IN, but with shorter ranges.
- Updated density regression formulae for copper oxide and sulphide material resulted in small differences from the previous formula.
- There are 130 total drill holes (comprising 25 km of drilling) in the ES Mineral Resource estimate. This includes 41 drill holes drilled in 2007 and 2008 that were not available for the 2007 block model, which was used for the 2018 and 2020 estimates. This includes 27 drill holes that were drilled inside the 2020 modelled pit shell. Inclusion of these 27 additional drill holes results in a potential reduction in average grade of approximately 13%.
- There appears to be little displacement on central fault blocks 32 and 34 and visual validation of the grade estimates supports treating them as one structural domain given the drill density.
- The DA estimation methodology works well to control search orientation while domaining of mineralized material and fault blocks constrains interpolation appropriately.
- Block grade estimation is well controlled overall.

14.6.4 ES Recommendations

Regarding the Mineral Resource estimate for Estrellita, the QP offers the following recommendations:

- Refine the geological and structural model.
- Review sequential copper and metallurgical data and revise the oxidation model to better understand the character of the oxide material.
- Drill diamond holes in key areas to assess metallurgical recoveries, oxide contacts, and fault locations and displacement, and to obtain additional sequential copper data.
- Revisit mining equipment and bench height and adjust block size and estimation parameters accordingly.
- Review the amenability of the oxide material to mineral processing to determine if some of the oxide material could be classified as Mineral Resources.
- Perform lithogeochemical studies similar to those performed for SD-IN.
- Analyze metal grades at Estrellita to determine correlations between gold, copper, iron, and cobalt.

15 MINERAL RESERVE ESTIMATES

The Mineral Reserve was developed by Capstone and is the total of all Proven and Probable category material planned for processing in the LOMP. Development of the final open pit design and Mineral Reserve was based on the best economic limit.

The Mineral Reserve is based on industry standard mine planning practices, as are applied at similar open pit mines. The Lerchs-Grossmann algorithm (LG) combined with practical phase designs and trials of alternative mine production schedules, were used to set the LOMP and production schedule. The Mineral Reserve is the total of all material planned for processing within the mine production schedule.

Early in the development of this LOMP, a series of preliminary phases or push backs were designed that initially targeted a LG shell that was based on a copper price of $3.75/lb.

A revenue factor of 0.84 was used for the LG shell that was employed as the guide for the practical design for both the Santo Domingo and Iris Norte pits (refer to discussion in Section 16). This selected revenue factor is conservative and as such allows for changes in metals pricing before any significant effect on the Mineral Reserves estimate will occur.

15.1 Block Model

The Mineral Resource block model was updated, effective June 1, 2024. The block model includes Mineral Resources in the Measured, Indicated, and Inferred categories. Pit optimization, mine design and mine planning were carried out on Mineral Resources in the Measured and Indicated categories for conversion to Mineral Reserve. Inferred Mineral Resources were treated as zero-grade waste.

A block size of 12 m E x 12 m N x 15 m high was selected for the block model. The selected block size was based on the geometry of the domain interpretation and the data configuration.

15.2 Supporting Assumptions for Pit Optimization

The mining cost estimate for the pit optimization process is based on studies developed by NCL during 2023 and 2024. The estimated average mining cost was separated into various components such as fuel, explosives, tires, parts, salaries and wages, benchmarked against similar current operations in Chile. Each component was updated for third quarter 2023 prices and the exchange rate from Chilean Pesos to US dollars. This resulted in an estimated mining cost of approximately $1.55/t. The metal prices, processing costs, refining costs, and processing recoveries were provided by Capstone Copper. Oxides were considered as waste during mine planning.

A summary of the initial input parameters used in the constraining LG pit shell is included in Table 15-1.

A number of calculations were performed in the model in order to determine the NSR of each individual block. The internal (or mill) cut-off of $9.77/t milled incorporates all operating costs except mining. This internal cut-off is applied to material contained within an economic pit shell where the decision to mine a given block was determined by the pit optimization and was applied to all of the Mineral Reserve estimates.

Table 15-1: Pit Optimization Parameters

Item	Unit	Value
Metal Price		
Copper	$/lb	3.75
Gold	$/oz	1,400
Iron (62%-65%-67%)	$/dmt	69.0 - 104.02 - 114.51
Recovery to Concentrate		
Copper (Mixed)	%	-0.011*Co+11.967*%CuT-13.740*%CuT^{2} +5.502*%CuT^{3}-0.920*CuT^{4}-0.20*68.714 +94.068
Copper (Sulphides)	%	-0.011*Co+11.967*%CuT-13.740*%CuT^{2} +5.502*%CuT^{3}-0.920*CuT^{4}-0.05*68.714 +94.068
Gold (Mixed)	%	-142.598 * g/t Au + 44.752*%CuT-10.095*%CuT^{2} - 0.20*46.680 + 51
Gold (Sulphides)	%	-142.598 * g/t Au + 44.752*%CuT-10.095*%CuT^{2} - 0.05*46.680 +51
Mass recovery for magnetite concentrate	%	%Fe >= 65% : Mass recovery for magnetite was estimated at the block model.
		60.3 <= %Fe <65.0 : Mass recovery is penalized by a factor of 0.7872 (0.82*0.96)
		55.0 <= %Fe <60.3 : Mass recovery is penalized by a factor of 0.7296 (0.76*0.96)
Concentrate Grade		
Copper (Mixed)	%	-1.701*%CuT+0.2*7.767+(CuT/S)*23.362 + 22.330
Copper (Sulphides)	%	-1.701*%CuT+0.05*7.767+(CuT/S)*23.362 + 22.330
Gold	g/t	(Rec Au * g/t Au) * (Rec Cu * %CuT / Grade Conc Cu)
Moisture content	%	8%
Magnetite Concentrate Grade		
Iron	%Fe	Variable on a block-by-block basis (> 62%)
Moisture content	%	8
Smelter Payables		
Payable copper	%	96.55
Payable Au		
0 - 3 (g/t)	%	0
1 - 3 (g/t)	%	90
3 - 5 (g/t)	%	92
5 - 7 (g/t)	%	94
7 - 10 (g/t)	%	95
10 - 15 (g/t)	%	96
> 15 (g/t)	%	96.5
Gold deduction in all concentrate	g/t in concentrate	0
Off-Site Costs		
Cu conc. treatment	$/dmt conc.	80
Cu refining charge	$/lb pay Cu	0.08
Au refining charge	$/oz pay Au	5.0
Shipping copper concentrate	$/wmt concentrate Cu	40
Shipping magnetite concentrate	$/dmt concentrate Fe	25.75
Operating Cost		
Waste mining cost	$/waste tonne	1.55

Item	Unit	Value
Ore mining cost	$/ore tonne	1.55
Processing + G&A + Sust Cap	$/t processed	9.77
Average Overall Pit Slope Angle		
SDS/Iris & Iris Norte	Overburden	36.6°
	Sector 1	36.6°
	Sector 2	38.6°
	Sector 3	40.30°
	Sector 4	45.0°
	Sector 5	40.0°
	Sector 6	45.6°
	Sector 7	39.2°
	Sector 8	46.2°
	Sector 9	45.3°
	Sector 10	41.8°
	Sector 11	45.6°
	Sector 12	46.5°
	Sector 13	47.3°
	Sector 14	47.9°
	Sector 15	43.2°
Other		
Grade factor (1-Dilution)	%	100
Mining recovery	%	100
Royalties	%	2
Discount rate	%	8

Note: The indicated Waste and Ore mining cost corresponds to the average cost. However, the optimization considers mathematical expressions that allow the variable transportation component to be reasonably estimated based on the elevation of each block to be extracted. A slightly conservative Au recovery was used during mine planning, but this is not expected to significantly impact the outcome.

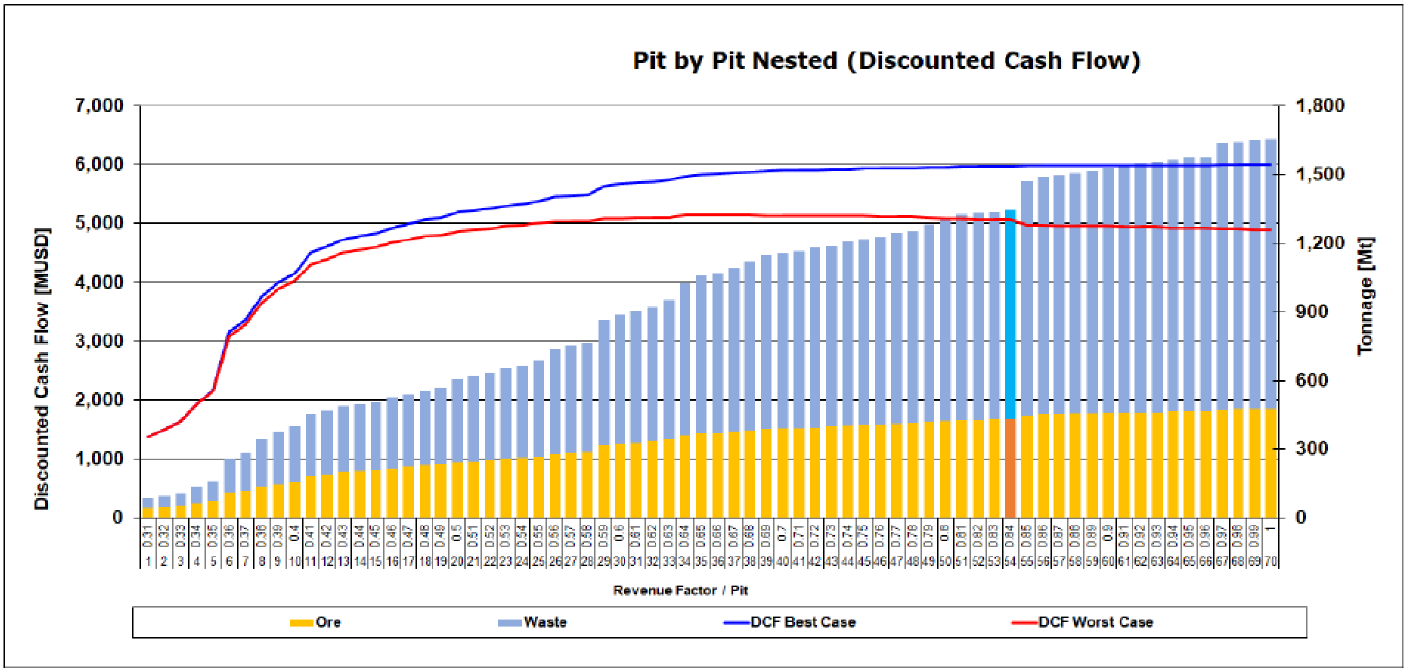
Nested pit shells were generated for several revenue factors. Whittle shell #70 is the revenue factor 1 shell for Santo Domingo and Iris Norte. However, after analyzing the results (and reviewing total, as well as incremental, values), the Santo Domingo and Iris Norte optimized pit shell #54 (revenue factor 0.84) was chosen as the basis for the detailed ultimate pit design. The difference between pit shell #54 and #70 is an expansion to the south of the Santo Domingo pit the case of the latter, adding 44 Mt of higher risk, low-grade material with a strip ratio of 6.1. The pit shells are summarized in Table 15-2. Figure 15-1 shows the discounted cash flow results for different values of the revenue factor. These Discounted Cash Flow results are used to define the optimal pit only, since they do not include Capital Investment (CapEx) values. The choice of pit 54 corresponds to a step change in waste from pit 55 onwards.

Table 15-2: Pits Shells Results

Pit	Revenue	Ore				Waste	Total	Strip Ratio
N°	Factor	Mt	%CuT	%Fe	Au g/t	Mt	Mt	
1	0.31	43.9	0.57	26.5	0.08	42.1	86.0	1.0
2	0.32	47.8	0.57	26.6	0.08	47.9	95.7	1.0
3	0.33	52.9	0.57	26.7	0.08	54.6	107.5	1.0
4	0.34	61.7	0.56	27.2	0.08	71.9	133.6	1.2
5	0.35	70.1	0.56	27.5	0.07	87.8	158.0	1.3
6	0.36	110.3	0.53	28.0	0.07	150.3	260.6	1.4
7	0.37	119.3	0.53	28.2	0.07	165.9	285.2	1.4
8	0.38	136.1	0.52	28.6	0.07	209.1	345.2	1.5
9	0.39	146.7	0.51	28.7	0.07	227.6	374.3	1.6
10	0.40	155.7	0.50	28.8	0.07	242.7	398.4	1.6
11	0.41	182.2	0.48	28.5	0.07	267.9	450.1	1.5
12	0.42	189.2	0.47	28.5	0.06	277.8	467.0	1.5
13	0.43	198.4	0.47	28.5	0.06	288.0	486.4	1.5
14	0.44	203.7	0.46	28.5	0.06	293.6	497.4	1.4
15	0.45	208.0	0.46	28.5	0.06	296.3	504.3	1.4
16	0.46	215.7	0.45	28.5	0.06	313.4	529.0	1.5
17	0.47	222.7	0.45	28.4	0.06	318.7	541.3	1.4
18	0.48	231.1	0.44	28.3	0.06	326.8	558.0	1.4
19	0.49	234.0	0.44	28.3	0.06	332.4	566.4	1.4
20	0.50	243.8	0.44	28.2	0.06	363.1	606.9	1.5
21	0.51	248.5	0.44	28.2	0.06	373.2	621.7	1.5
22	0.52	251.6	0.44	28.2	0.06	378.7	630.3	1.5
23	0.53	258.9	0.43	28.2	0.06	393.2	652.0	1.5
24	0.54	262.8	0.43	28.1	0.06	400.7	663.5	1.5
25	0.55	269.0	0.42	28.1	0.06	420.9	689.9	1.6
26	0.56	280.4	0.41	28.1	0.06	455.8	736.2	1.6
27	0.57	285.4	0.41	28.0	0.06	466.2	751.6	1.6
28	0.58	288.4	0.41	28.0	0.06	472.2	760.6	1.6
29	0.59	316.7	0.39	27.6	0.05	548.4	865.0	1.7
30	0.60	324.9	0.39	27.5	0.05	565.1	890.0	1.7
31	0.61	329.4	0.38	27.5	0.05	574.6	904.0	1.7
32	0.62	335.0	0.38	27.4	0.05	587.1	922.1	1.8
33	0.63	342.1	0.38	27.3	0.05	611.3	953.4	1.8
34	0.64	358.3	0.37	27.2	0.05	666.3	1,024.6	1.9
35	0.65	366.2	0.36	27.1	0.05	690.3	1,056.5	1.9
36	0.66	369.6	0.36	27.1	0.05	700.6	1,070.2	1.9
37	0.67	375.3	0.36	27.1	0.05	717.6	1,092.9	1.9
38	0.68	382.0	0.35	27.0	0.05	739.0	1,121.0	1.9
39	0.69	389.1	0.35	27.0	0.05	760.1	1,149.2	2.0
40	0.70	391.5	0.35	27.0	0.05	767.2	1,158.6	2.0

Pit	Revenue	Ore				Waste	Total	Strip Ratio
N°	Factor	Mt	%CuT	%Fe	Au g/t	Mt	Mt	
41	0.71	393.5	0.35	27.0	0.05	771.4	1,164.9	2.0
42	0.72	396.7	0.35	27.0	0.05	782.9	1,179.6	2.0
43	0.73	399.6	0.34	27.0	0.05	788.5	1,188.0	2.0
44	0.74	403.9	0.34	26.9	0.05	801.3	1,205.2	2.0
45	0.75	406.9	0.34	26.9	0.05	810.2	1,217.0	2.0
46	0.76	408.4	0.34	26.9	0.05	816.3	1,224.7	2.0
47	0.77	412.5	0.34	26.9	0.05	831.7	1,244.3	2.0
48	0.78	415.2	0.34	26.9	0.05	839.7	1,254.9	2.0
49	0.79	420.5	0.34	26.8	0.05	861.8	1,282.3	2.0
50	0.80	424.5	0.34	26.7	0.05	879.8	1,304.2	2.1
51	0.81	428.3	0.33	26.7	0.05	896.0	1,324.3	2.1
52	0.82	429.9	0.33	26.7	0.05	901.2	1,331.1	2.1
53	0.83	431.7	0.33	26.6	0.05	906.8	1,338.5	2.1
54	0.84	433.0	0.33	26.6	0.05	910.7	1,343.7	2.1
55	0.85	448.8	0.33	26.4	0.05	1,018.6	1,467.4	2.3
56	0.86	451.0	0.33	26.4	0.05	1,038.7	1,489.7	2.3
57	0.87	453.2	0.33	26.4	0.05	1,045.5	1,498.7	2.3
58	0.88	454.5	0.33	26.4	0.05	1,050.9	1,505.4	2.3
59	0.89	456.2	0.33	26.3	0.05	1,061.2	1,517.4	2.3
60	0.90	458.0	0.33	26.3	0.05	1,072.6	1,530.6	2.3
61	0.91	458.7	0.33	26.3	0.05	1,077.2	1,535.9	2.3
62	0.92	460.9	0.33	26.3	0.05	1,089.9	1,550.8	2.4
63	0.93	461.7	0.33	26.3	0.05	1,092.8	1,554.5	2.4
64	0.94	463.4	0.33	26.3	0.05	1,102.7	1,566.2	2.4
65	0.95	464.5	0.33	26.3	0.04	1,107.2	1,571.7	2.4
66	0.96	465.2	0.33	26.3	0.04	1,108.6	1,573.8	2.4
67	0.97	474.0	0.32	26.2	0.04	1,164.0	1,637.9	2.5
68	0.98	474.7	0.32	26.2	0.04	1,166.9	1,641.6	2.5
69	0.99	475.8	0.32	26.2	0.04	1,171.8	1,647.6	2.5
70	1.00	476.8	0.32	26.2	0.04	1,177.0	1,653.9	2.5

Figure 15-1: Pit by Pit Nested Results



Source: Figure prepared by Capstone Copper, 2024.

These LG shells were employed as guides for the practical designs for both the Santo Domingo and Iris Norte pits (refer to discussion in Section 16).

15.3 Geotechnical Considerations

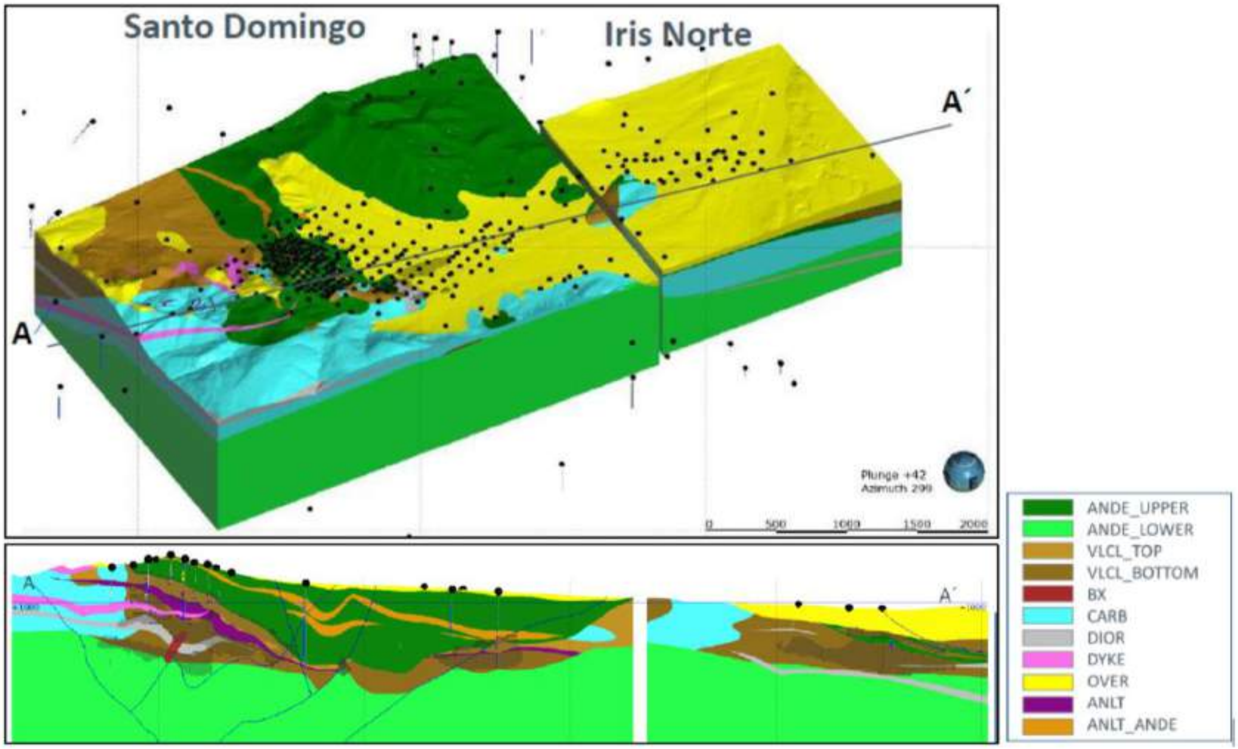
The rock mass of the MSD mine has been determined mainly based on lithology. Open pit mappings and various geotechnical drilling programs have been completed to inform the current study. These studies were completed by Capstone and its geotechnical consultants, including DERK (2012), AMEC (2015-2018) and INGEROC (2023). INGEROC was commissioned to update the geotechnical model for MSD, thus generating the update of the current geotechnical model of the Santo Domingo and Iris Norte pits, and the design variables of the pits.

Rock mass characterization has been completed by mapping and drill core using various geotechnical indices such as rock quality designation (RQD), rock mass ratio (RMR) and geological strength index (GSI). In addition to laboratory strength testing of core samples, these classifications have been used to develop the rock mass strengths used in the stability analysis.

The geological model is a lithological model supported by litho-geochemical classification, where the following geological units are recognized:

- Overburden (OVBR). Corresponds to the cover of unconsolidated material and is located at the upper end of the stratigraphic column. This unit groups all types of Quaternary deposits.
- Andesites (ANDE). They correspond mainly to amygdaloidal andesites, porphyritic andesites and breccias, and to a lesser extent crystalline tuffs. A distinction is made between two subunits; ANDE_UPPER and ANDE_LOWER, where the difference between them corresponds to their location within the stratigraphic column.
- Volcaniclastic Units (VLCL). This unit groups different fine- to medium-grained volcanic rocks with a variety of clastic textures and contains most of the replacement of iron oxides and sulphides. However, two sub-units VLCL_TOP and VLCL_BOTTOM, are differentiated depending on their stratigraphic position with respect to the ANLT unit.
- Andesitic Lithic Tuff (ANLT). Corresponds mainly to lapilli tuffs, lithic tuffs and crystalline tuffs. It is possible to differentiate them from the rocks of area due to the size of the lapilli clasts.
- Breccia (BX). Corresponds to a clastic-supported hydrothermal breccia where andesite clasts and volcaniclastic rocks are in a matrix of chlorite, actinolite and iron sulphides. The clasts within the breccia may show mineralization. This unit is located towards the central and southern sectors of the deposit.
- Carbonate rocks (CARB). Corresponds to the grouping of carbonate rocks such as limestones, carbonate sediments and calcareous sandstones.
- Dioritic and Andesitic porphyries (DIOR). They correspond to intrusives of andesitic and dioritic composition that cut the volcanic sequence of the deposit.
- Feldspathic porphyry (DYKE). Corresponds to feldspar-rich, dioritic/andesitic composition emplaced after the mineralization events of the of the deposit.

Figure 15-2: Lithological Model



Source: Figure prepared by INGEROC, 2023.

The RQD variable corresponds to the most widely sampled feature in the database, with a population of 14,848 data in 180 boreholes. The GSI variable has been described in less detail within the study area, with 1,795 records distributed over 34 boreholes.

The rock mass characteristics of the main lithologies and geotechnical units used to develop the recommended slope angles are summarized below. For each lithological unit, input properties were defined for the model based on historical information collected during the life of the project. Table 15-3 is a simplified summary of the GSI behavior per geotechnical unit.

Table 15-3: GSI by Lithological Unit

	GSI			
Unit	Average	Standard Deviation	Maximum Value	Minimum Value
ANDE_LOWER	65	9.3	77	38
ANDE_UPPER	69	10.3	86	28
ANLT	70	8.2	77	40
ANLT_ANDE	69	10.2	77	38
BX	64	9.1	72	29
CARB	72	6.1	77	40

GSI				
DIOR	69	9.7	77	43
DYKE	71	6	77	57
VLCL	72	7.6	77	38
VLCL_BOTTOM	67	11.5	77	20
VLCL_TOP	64	11.9	86	16

The geology of Iris Norte is characterized by seven lithological units and one major fault, while the Santo Domingo Sur sector is made up of 11 lithological units and four major faults.

MSD has been subject to an extensive campaign of laboratory tests including: 1) Densities, 2) Uniaxial Compressive Strength, 3) Triaxial Tests, 4) Indirect Tensile Tests, 5) Direct Shear Tests on natural structures intended to present the properties of intact rock as well as the properties of structures according to the Mohr-Coulomb failure criteria.

Table 15-4 shows, as an example, uniaxial compressive strength (UCS) for each major lithology.

Table 15-4: UCS by Lithological Unit

UCS					
Unit	Samples	Average (MPa)	Standard Deviation (MPa)	Minimum Value (MPa)	Maximum Value (MPa)
ANDE_LOWER	22	153.3	65.2	58.7	378.3
ANDE_UPPER	27	90.6	47.4	26.1	206.1
ANLT	5	132.1	35.3	86.9	178.1
ANLT_ANDE	5	99	63.2	34.1	204
BX	3	93.4	60.5	31.4	152.3
CARB	13	104.5	42.2	19.9	169.3
DIOR	12	131.1	56.6	51.9	257.9
DYKE	4	225.8	87.6	99.6	302
OVER	2	7.2	4	4.4	10.1
VLCL	13	42.5	45.5	4.2	138.3
VLCL_BOTTOM	46	126.1	70.5	25.8	363.2
VLCL_TOP	22	134.8	59.4	44.1	241.7

The structural domains were estimated based on structural information from the GD drill holes (optical-acoustic mapping) and the study of fault type structures generated by Asgeomin SpA in 2023. The methodology to define the structural domains is based on the comparison of pole density diagrams delimited by macroblocks and faults.

The design criteria for defining the geometry of the walls of the Santo Domingo and Iris Norte ditches were carried out considering double bench of 30 m in height, using the Ritchie criterion for determining berm widths.

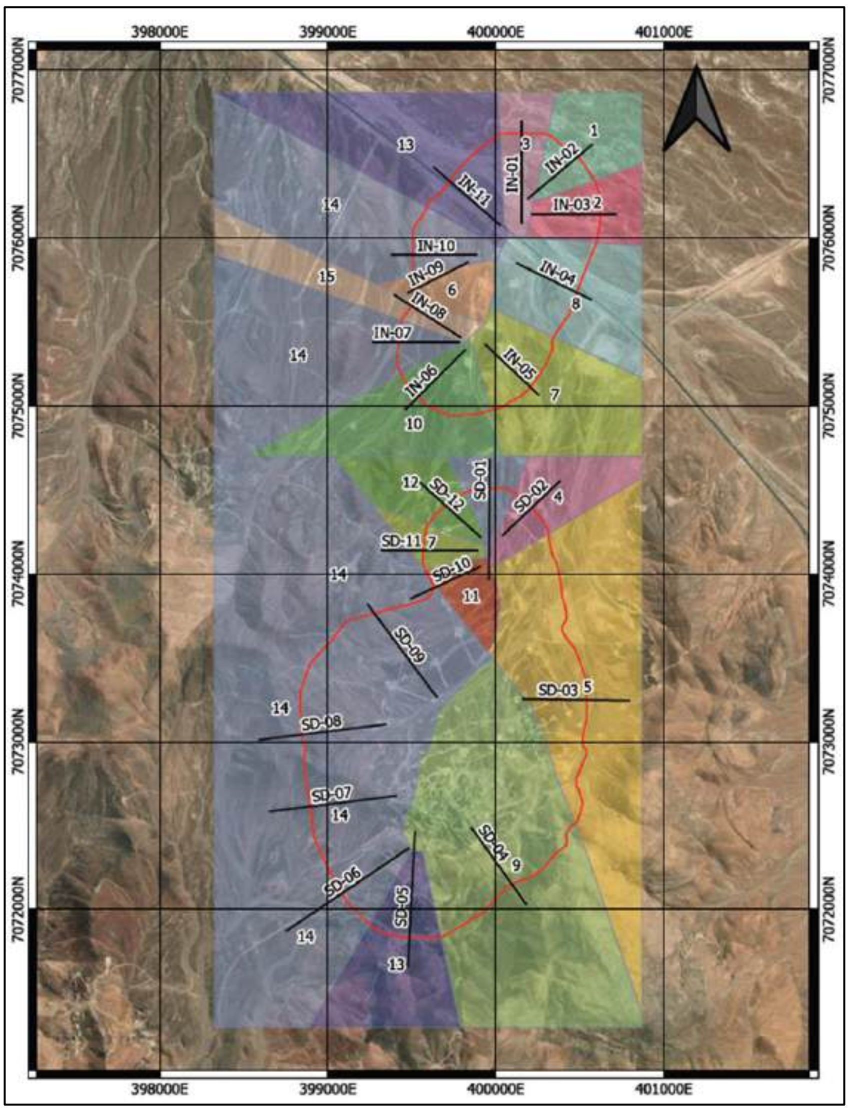
Criteria described by Read and Stacey (2009), which are widely used nationally and internationally, were satisfied.

15.4 Geotechnical Recommendations

Design parameters follow geotechnical zone recommendations. There are 15 geotechnical zone, each with distinct parameters. The Overall Angles vary from 36.3° to 49.7°, batter angle is between 69° and 75° and the inter-ramp angles vary from 49° to 58°.

The phase designs have 15m bench heights, using electric and hydraulic shovels and loaders. Figure 15-3 shows the geotechnical slope domains and Table 15-5 summarizes the pit slope angles and parameters.

Figure 15-3: Geotechnical Slope Domains



Source: Figure prepared by INGEROC, 2023.

Table 15-5: Pit Slope Angle and Parameters

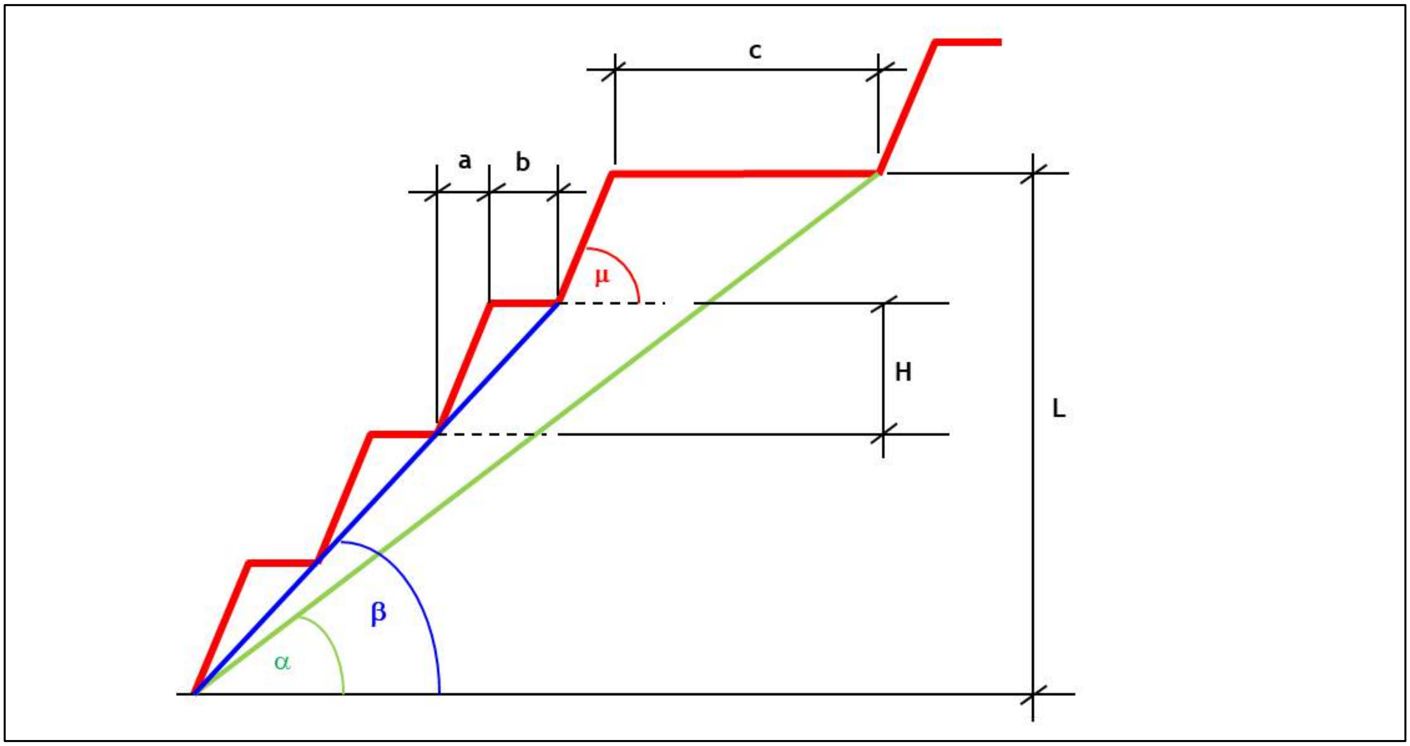
Sector	Inter-ramp Slope					Overall Slope		
	IRA	Face Angle	Height	Backbreak	Berms	Catch Berm	Slope Height	Slope Angle
	β (º)	µ (º)	H (m)	a (m)	b (m)	c (m)	L (m)	α (º)
1	49	63	30	15.3	10.5	40	80	36.3
2	53	68	30	12.1	10.5	40	80	38.6
3	54	69	30	11.5	10.5	40	90	40.3
4	54	70	30	10.9	10.5	40	140	45.0
5	55	71	30	10.3	10.5	40	80	40.0
6	55	71	30	10.3	10.5	40	140	45.6
7	52	75	30	8.0	15.4	40	90	39.2
8	56	72	30	9.7	10.5	40	140	46.2
9	57	73	30	9.2	10.5	40	120	45.3
10	58	75	30	8.0	10.5	40	80	41.8
11	58	75	30	8.0	10.5	40	110	45.6
12	58	75	30	8.0	10.5	40	120	46.5
13	58	75	30	8.0	10.5	40	130	47.3
14	58	75	30	8.0	10.5	40	140	47.9
15	52	75	30	8.0	15.4	40	140	43.2

The geometric parameters indicated in Table 15-6 are described below and Figure 15-4 illustrates these parameters in a typical section.

Table 15-6: Geotechnical Parameters

Geotechnical parameter	Description
IRA (β)	Angle measured between the toe of a bench and the toe of the bench immediately above
Face angle (µ)	Angle measured between the horizontal plane and the wall of the bench.
Height (H)	Vertical distance measured between the toe and the crest of the same bench
Backbreak (a)	Horizontal distance measured between the toe and the crest of the same bench
Berm (b)	Horizontal distance measured between the crest of the lower bench and the toe of the upper bank
Catch Berm (c)	Horizontal distance between a set of benches
Slope Height (L)	Vertical distance measured between catch berms
Slope Angle (α)	Angle measured between toe and toe of catch berms

Figure 15-4: Wall Slope Design Parameters



Source: Figure prepared by Capstone Copper, 2024.

Final slope angles used for the pit design process were a result of analysis carried out by the MSD mining team, NCL and geotechnical specialists Ingeroc SpA as follows:

- A pit optimization was carried out with an initial set of overall slope angles for the geotechnical domains.
- A pit shell was selected for detailed mine design, adding haul roads, safety and geotechnical berms and applying detail bench configuration (batter height, batter angle, berm widths).
- The obtained overall angles per slope domain were measured and compared with the initial assumptions.
- The detailed pit design was re-analyzed, updated configuration was generated, and the final mine design was developed.

15.4.1 Hydrogeological Considerations

The Santo Domingo pits are expected to be relatively dry as a result of their location within an arid region, supported by a hydrogeology model. A system of pumping wells would need to be implemented to manage the drainage of the pit and these will be replaced as the Pit progresses to ensure that the dewatering capacity is adequate and meets the development requirements. The quantity and quality of water encountered as pit depth progresses will be monitored and appropriate management systems put in place, such as evaluation of water quality for pumping to the processing plant.

If required, horizontal drilling campaigns and installation of piezometers will be carried out to detect specific pores in the pit wall as it deepens and target new areas to depressurize.

15.5 Dilution and Mine Losses

An analysis was carried out to review the resource model for individual blocks that were surrounded by waste (potential loss) and waste blocks that were surrounded by ore (potential dilution). The potential for losses and dilution are minimal, based on the limits of the mineralized areas, typically sharp transitions between ore and waste units, as well as the planned use of shovels equipped with high-precision GPS instruments. Any potential impacts to ore feed that might arise due to ore loss or dilution were not considered material, and no provisions for ore loss or dilution were included in the mine plan.

Careful grade control will need to be practiced during mining operations to avoid sending unfavorable material to the plant, particularly for the magnetite ore due to the importance of preserving high concentrate grade. These efforts should include the following standard procedures:

- Implement a systematic program of sampling, mapping, laboratory analyses and reporting
- Use of shovels with high precision GPS to selectively mine ore zones
- Maintain high quality laboratory staff, equipment, and procedures to provide accurate and timely assay reporting
- Use trained geologists and technicians to work with shovel operators in identifying, marking and selectively mining and dispatching ore and waste
- Implement and validate a procedure that incorporates the concept of NSR within the mineral control processes for the management of Cu-Fe-Au commodities.

15.6 Cut-off Grades

For mine production schedule purposes an NSR in $/t was calculated to take into account assumptions listed in Table 15-1, which result in the recoverable, payable value of copper, gold and iron, net of off-site costs (transport, smelting and refining).

The internal (or mill) cut-off of $9.77/t milled incorporates all operating costs except mining. Mining is treated as a sunk cost for the purposes of the cut-off determination. This internal cut-off is applied to material contained within the mining phases, defining the difference between ore and waste.

15.7 Mineral Reserves Statement

Mineral Reserves are summarized in Table 15-7 and have an effective date 31st March 2024. The Mineral Reserve was developed by tabulating the contained measured and indicated (proven and probable) material above the mill cut-off grades inside of the designed pit (described in Section 16). Mine optimization and phase design were performed using slightly different parameters for mining cost and gold recovery than what was assumed in Section 22 Economic Analysis. The impact from these changing assumptions will have a negligible impact on the mine design.

Table 15-7: Mineral Reserve Statement

Reserve Category	Stage	Tonnage (Mt)	Ore Grade			Contained Metal		
			Cu (%)	Au (g/t)	Fe (%)	Au (koz)	Cu (kt)	Magnetite Conc. (Mt)
Proven Mineral Reserves	Santo Domingo	130.9	0.52	0.07	27.2	291	675	12.6
	Iris Norte							
Total Proven Mineral Reserves	**Total**	**130.9**	**0.52**	**0.07**	**27.2**	**291**	**675**	**12.6**
Probable Mineral Reserves	Santo Domingo	251.2	0.27	0.04	26.6	309	673	45.1
	Iris Norte	53.9	0.16	0.02	24.3	36	88	10.8
Total Probable Mineral Reserves	**Total**	**305.1**	**0.25**	**0.04**	**26.2**	**346**	**761**	**55.8**
Total Mineral Reserves	Santo Domingo	382.1	0.35	0.05	26.8	600	1,347	57.6
(Proven and Probable)	Iris Norte	53.9	0.16	0.02	24.3	36	88	10.8
Total Mineral Reserves	**Total**	**436.1**	**0.33**	**0.05**	**26.5**	**637**	**1,435**	**68.4**

Notes to Accompany Mineral Reserves Estimate: **1**. Mineral Reserves have an effective date 31st March 2024. **2**. Mineral Reserves include the sulfide and mixed zone, excluding oxides are reported as constrained within Measured and Indicated Resources and pit designs supported by a mine plan featuring variable throughput rates and cut-off optimization. The pit designs and mine plan were optimized using the following economic and technical parameters: metal prices of US$3.75/lb Cu, US$1,400/oz Au and Fe Concentrate prices ranging from US$69/dmt to US$114.51/dmt based on the Fe grade in concentrate (net of Fe concentrate transport costs); average recovery to concentrate is 90.1% for Cu and 56.3% for Au, with magnetite concentrate recovery varying on a block-by-block basis; copper concentrate treatment charges of US$80/dmt, U$0.08/lb of copper refining charges, US$5.0/oz of gold refining charges, US$40/wmt and US$25.75/dmt for shipping copper and iron concentrates respectively; average waste mining cost of $1.55/t, average ore mining cost of US$1.55/t and process + G&A + Susex costs of US$9.77/t processed; average pit slope angles that range from 36.3º to 47.9º; a 2% royalty rate assumption and an assumption of 100% mining recovery. **3**. Rounding as required by reporting standards may result in apparent summation differences between tonnes, grade and contained metal content. **4**. Tonnage measurements are in metric units. Copper and iron grades are reported as percentages, gold as grams per tonne. Contained gold ounces are reported as troy ounces, contained copper as thousand tonnes and contained iron as metric million tonnes.

15.8 Factors That may Affect the Mineral Reserves Estimate

In the opinion of the QP, the main factors that may affect the Mineral Reserves estimate are metal prices, metallurgical recoveries, operating costs (fuel, energy, and labour) and block model accuracy. However, the Mineral Reserves are not considered sensitive to moderate changes in these factors, due in part to the use of a 0.84 Revenue Factor LG shell used to guide pit design. Other factors that may impact the Reserves include land tenure or permitting, water supply and geotechnical stability of pit walls and tailings facilities.

15.9 Risks

As reserve models are an estimate based on certain assumptions and interpretations, they have certain inherent risks. Risks to the MSD Mineral Reserve as outlined in this report include, but may not be limited to:

- Changes to the Mineral Resource estimate, potentially resulting from revised interpretation and/or the results of additional drilling and sampling
- Changes to assumptions in metals prices and operating costs
- Adjustment to the Pit Design due to future reinterpretation of Geomechanical parameters in some sectors.

15.10 Opportunities

There is upside potential for the Project. The current mine plan so far only considers the processing of Cu and Fe concentrate, identifying the following opportunities:

- Blocks interpreted as Oxides are not considered for processing in the Mining Plan presented in this report. However, future technical studies that indicate the economic viability of processing these minerals could represent a potential increase in reserves.
- The mineralization within the pit shell that is currently classified as Inferred, and therefore set to waste for this study, may, with infill and blast hole drilling during the mining process, be upgraded to higher confidence categories. These resource blocks represent a future opportunity for potential incorporation into the mine plan.
- Future studies may be able to demonstrate positive economics from extraction of cobalt, which could have a positive impact on reserves.
- The material falling outside the 0.84 RF pit but within the 1.00RF pit may represent a future opportunity to add reserves, particularly if commodity prices move favorably.

15.11 Conclusions

Mineral Reserves amenable to open pit mining were constrained by practical pit designs that had been guided by an optimized pit shell (LG). An internal cut-off of $9.77/t milled was applied to all the Mineral Reserve estimates, as well as during the process of generating the optimized pit shell.

A Mine plan was developed to process generally 72/65 kt/d (26.3 to 23.7 Mt/y) with a peak total mining rate of 90 Mt/y. Inferred Mineral Resources were treated as waste in the mine plan.

Pit optimization and mine planning were performed without introducing any additional factors to account for dilution. Any potential impacts to ore feed that might arise due to ore loss or dilution were not considered material, and no provisions for ore loss or dilution were included in the mine plan.

Proven and Probable Mineral Reserves total 436.1 Mt grading 0.33% Cu, 0.05 g/t Au and 26.5% Fe.

The main factors that may affect the Mineral Reserve estimates are metal prices, metallurgical recoveries, operating costs (fuel, energy, and labour) and block model accuracy. However, the Mineral Reserves are not considered sensitive to moderate changes in these factors, due in part to the use of a 0.84 Revenue Factor LG shell used to guide pit design. Other factors that may impact the Reserves include land tenure or permitting, water supply and geotechnical stability of pit walls and tailings facilities.

15.12 Recommendations

Implement a robust ore control system to ensure minimal dilution of the ore grades, particularly for preserving higher grade of Fe concentrate.

15.13 QP Comments

The QP considers that the methodologies described in Section 15 represent suitable application of industry standard approaches for use in Mineral Reserve disclosure, as well as to support detailed mine planning and scheduling.

16 MINING METHODS

Santo Domingo is planned as an open pit mine, producing ore bearing copper sulphide and magnetite. Conventional open-pit mining utilizes the cycle of drilling, blasting, loading, and hauling of material to the respective destinations. Ore is hauled to the primary crusher for processing and waste rock material is hauled to waste storage facilities.

Mining is accomplished on benches of 15 m height and the LOMP is reported in metric tonnes.

This Technical Report incorporates, according to the hardness properties of the mineral, a variable mill throughput until the reserves are completely extracted. The mine production schedule was developed with the goal of maintaining mill feed and maximizing the mine's net present value (NPV).

The LOMP schedules movement of an average of 250 ktpd (90 Mtpa) of total material from Year 01 to 15. Beginning in Year 16 the plant feed and waste mined begins to fall, and the total material movement reduces.

16.1 Pit Design

Initial pit design considerations are included in Section 15.

The final pit design was based on the economic shells obtained at revenue factor 0.84 for Santo Domingo and Iris Norte, with variable overall slope angles according to geotechnical domains ranging from 36.3° to 47.9°. The mine design parameters are summarized in Table 16-1.

A road width of 40 m was selected to accommodate 290 t trucks. MSD used a 10% road gradient which is common in the industry for this type of truck. The current mine plan is designed with 15 m benches stacked to 30 m (i.e. double benching) for the fresh rock material. Mining costs are based on blasting 15 m benches for the waste zones and for the ore.

Additional 40 m wide safety berms were included in the design when the slope height exceeds between 80 and 140 m in accordance with geotechnical recommendations.

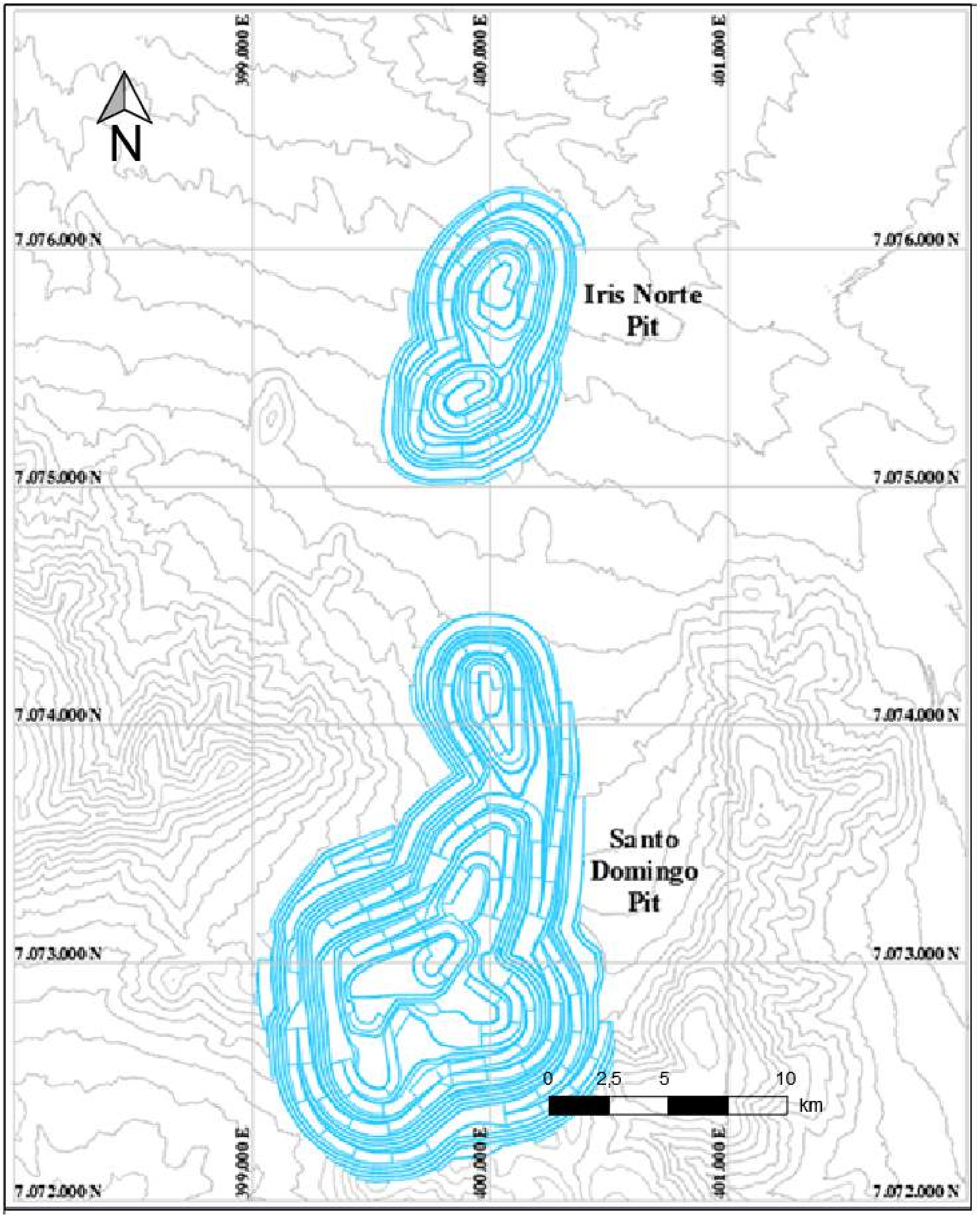
The Santo Domingo pit will have one exit on the west side to provide access to the stock pad area and the primary crusher. On the east side there will be another exit to access the main waste rock storage area. The final pit will be 2,350 m long in the north–south direction and 1,500 m wide in the east–west direction. The pit bottom will be at the 694 masl elevation. The highest wall will be 435 m and is situated on the central west side of the pit. The total area disturbed by the pit will be approximately 232 ha. Figure 16-1 shows the final Santo Domingo and Iris Norte pit layout.

Table 16-1: Mine Design Parameters

Parameter	Unit	Value
Haul road width	m	40
Haul road grade	%	10
Bench height	m	15
Stacked bench height with 2 benches stacked (fresh rock)	m	30
Nominal minimum mining phase width	m	100
Batter angle	°	As per geotechnical domains
Berm width	m	
Catch berm width every 80 m/140 m of pit wall	m	40

Geotechnical Domains	Batter Height (m)	Batter Angle (°)	Berm Width (m)
Sector 1	30	63	10
Sector 2	30	68	10.5
Sector 3	30	69	10.5
Sector 4	30	70	10.5
Sector 5	30	71	10.5
Sector 6	30	71	10.5
Sector 7	30	75	15.4
Sector 8	30	72	10.5
Sector 9	30	73	10.5
Sector 10	30	75	10.5
Sector 11	30	75	10.5
Sector 12	30	75	10.5
Sector 13	30	75	10.5
Sector 14	30	75	10.5
Sector 15	30	75	15.4

Figure 16-1: Santo Domingo and Iris Norte Pit Layout



Source: Figure prepared by Capstone Copper, 2024.Grid indicates scale. Grid squares are 1 km x 1 km.

The Iris Norte pit will have one exit access to the stock pad area, primary crusher and to access the waste rock storage area. The final pit will be 1,350 m long in the north–south direction and 700 m wide in the east–west direction. The pit bottom will be at 709 m elevation. The highest wall will be 260 m and is located on the north side of the pit. The total area disturbed by the pit is 76 ha.

16.2 Pit Phases

Ten pit phases are planned; eight for Santo Domingo and two for Iris Norte (refer to Figure 16-2 and Table 16-2).

In Santo Domingo, Phase 1 (SD01) targets the ore with the highest grade and lowest strip ratio in the central area, down to 889 masl elevation. Phases 2 and 3 (SD02 and SD03) are successive expansions to the west and east, down to 739 masl and 784 masl elevation, respectively. Phase 4,5,6,7,8 in Santo Domingo (SD04, SD05, SD06, SD07 and SD08) is the final expansion to in all orientations, deepening the central portion down to 694 masl. This expansion includes the Iris sector, which is mined together with Santo Domingo in the northern portion of the pit. This sector has separate access on the east side and goes down to the 799 masl elevation.

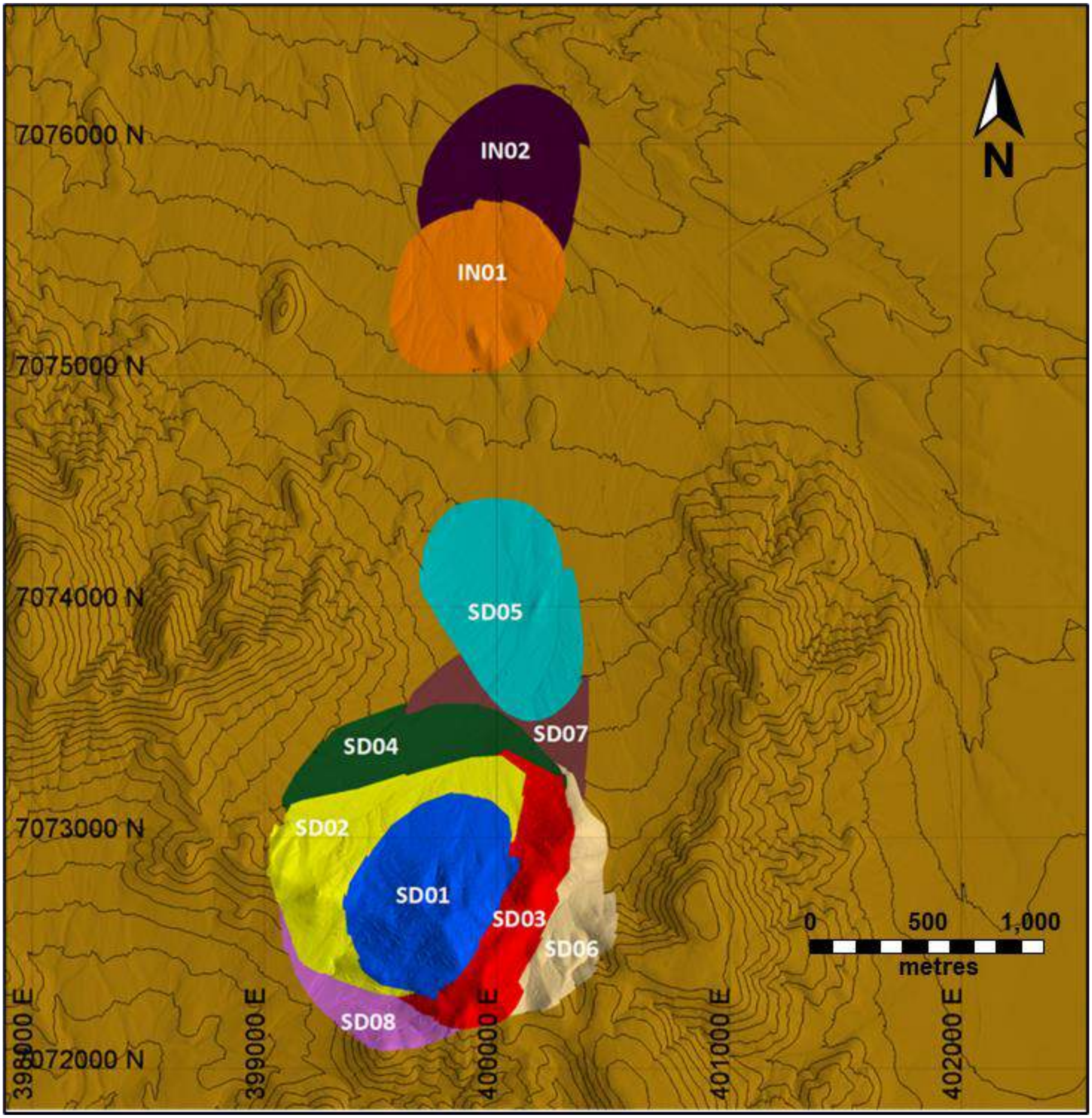
Two mining phases were designed in Iris Norte (IN01, IN02), which are successive expansions from south to north, going down to 739 masl and 709 masl elevation, respectively. Each phase has access from the east side.

16.3 Production Schedule

A mine production schedule was developed to show the ore tonnes, metal grades, waste material and total material by year, throughout the LOM. The distribution of ore and waste contained in each of the mining phases was used to develop the schedule, ensuring that criteria such as continuous ore exposure, mining accessibility and consistent material movements were met.

MSD used a traditional Whittle process to guide final pit limits, and mine sequence was optimized using Simshed from Mining Math Software Ltda. to get the macro sequence guidance for phase design. Phase design was done within Maptek Vulcan software. Sensitivity analysis on total annual mine movement was performed using MineMax Scheduler to assess stockpiling and rehandle strategies, considering the restrictions of maximum mill throughput and maximum capacity of the pipeline to transport Fe concentrate. Final detailed scheduling was performed in MinePlan by David Carkeet Software EIRL.

Figure 16-2: Mining Phases



Source: Figure prepared by Capstone copper, 2024. Grid squares are 1 km x 1 km.

Table 16-2: Phase Summary

Phase	Ore (Mt)	CuT (%)	Fe (%)	Au (g/t)	Co (ppm)	MassRec (%)	Oxides (Mt)	Waste (Mt)	Strip ratio	Total Rock (Mt)
SD01	79	0.50	27.3	0.07	207.9	14.9	5	111	1.40	195
SD02	66	0.38	29.8	0.05	319.9	18.8	1	162	2.46	230
SD03	79	0.39	27.2	0.05	240.2	11.4	17	84	1.06	179
SD04	39	0.13	28.1	0.02	253.6	25.1	2	144	3.69	185
SD05	30	0.33	22.7	0.04	244.7	9.5	10	114	3.84	154
SD06	38	0.38	24.2	0.05	271.7	5.1	2	104	2.70	145
SD07	31	0.09	28.0	0.01	99.0	26.8	2	86	2.80	119
SD08	20	0.37	20.4	0.06	200.9	8.0	0	83	4.09	104
IN01	28	0.21	21.6	0.03	166.1	15.4	6	89	3.13	124
IN02	25	0.11	27.3	0.01	212.1	25.1	2	97	3.82	124
Total	**436**	**0.33**	**26.5**	**0.05**	**232.4**	**15.7**	**49**	**1,075**	**2.46**	**1,560**

Table 16-3: Mine Production Schedule Summary

Period	Ore Feed	Ore Stockpile		Oxide	Waste	Total Moved
	Tonnage	In Tonnage (kt)	Out Tonnage (kt)	Tonnage (kt)	Tonnage (kt)	Tonnage (kt)
Y-1	-	1,149	-	992	8,859	11,000
Y0	-	5,126	-	2,553	26,321	34,000
Y1	19,939	1,106	-	2,239	66,716	90,000
Y2	26,279	989	-	472	62,260	90,000
Y3	21,319	74	4,962	527	62,080	88,962
Y4	25,086	9,251	1,267	2,578	50,408	88,590
Y5	26,278	6,623	-	13,370	43,729	90,000
Y6	23,726	13,522	-	1,615	51,137	90,000
Y7	23,722	6,634	-	1,555	58,265	90,177
Y8	16,311	1,928	7,480	1,609	62,652	89,980
Y9	18,838	2,104	4,887	461	62,960	89,250
Y10	23,726	8,559	-	236	57,480	90,000
Y11	18,234	692	5,491	47	66,275	90,739
Y12	17,929	995	5,862	4,677	60,415	89,878
Y13	21,884	-	1,841	3,138	61,860	88,723
Y14	11,486	-	12,240	7,832	56,808	88,365
Y15	15,140	-	8,586	2,620	62,241	88,586
Y16	18,352	-	-	509	66,139	85,000
Y17	16,641	-	2,320	1,676	54,252	74,889
Y18	17,588	-	1,400	200	24,250	43,437
Y19	14,826	-	2,418	-	9,652	26,896
Total	**377,303**	**58,753**	**58,753**	**48,905**	**1,074,757**	**1,618,471**

Note: Ore Feed from mine corresponds to the mineral that will be sent to the process directly from the pit. A total of 58.75 Mt of mineral will be mined and stockpiled (Stockpile Ore) for later re-handle (Rehandle Ore) for a total LOM mill throughput of 436 Mt.

Several iterations at different total material movement rates were done to determine an adequate production schedule strategy to identify the total mine movement that maximizes the NPV for the Project. This employed a strategy of stockpiling lower-grade ore during years of surplus production. Subsequent operational adjustments were made in different software packages to generate a feasible mine sequence in conjunction with the plant requirements.

The schedule is based on process plant throughput of 72,000 t/d for the first 4 years (except for year 01 which is considered Ramp up, that is to say, for years 2 to 5) and 65,000 t/d for most of the remaining life, which considers a plant utilization of 93% (Grinding). The mined material movement peaks at 90 Mt/y. The production schedule was limited by a maximum sinking rate of 8 benches per year in each phase. The total mined waste considers two main destinations for the material: the main waste rock storage areas and TSF for the embankment construction:

- Waste requirements for the TSF construction were provided to MSD by Knight Piésold and are based on the schedule for the dam embankment raises
- The material to be sent to the mine waste storage areas corresponds to the difference between the total mined waste from the mine production schedule and the requirement for the TSF
- One waste rock storage facility (WRSF) principal area at the west of the pits have been designed (refer to Section 18).

The mined ore will be hauled to the primary crusher for direct tipping. Blending material will be mined and hauled to a stockpile located between the Santo Domingo and Iris Norte pits. This material will be re-handled and will become part of ore feed to regulate the Fe grade.

The oxide material is treated as waste in the mine plan. No economic process has been defined to treat this material. However, a stockpile area for the oxide material with copper content greater than 0.2% was set aside so that this material can be stockpiled for possible future processing.

The plan assessed the pre-stripping on a quarterly basis; and first year of commercial production on a yearly production basis. Additionally, the pre-stripping period was analyzed with a monthly resolution, ensuring mine equipment allocations as a short range mine schedule.

The pre-production period requires the mining of 45 Mt of total material to expose sufficient ore to start commercial production in Year 1. The pre-production period will be approximately 18 months. The ore mined during pre-production will be stockpiled in the Stock pad area and will make up part of the LOMP ore production. The production plan showing material sent to mill and to stockpile is provided in Table 16-4.

Table 16-4: Plant Feed Schedule

Period	Plant Feed										
	Tonnage (kt)	Cu (%)	Rec. Cu (%)	ConCu (kt)	Cu Fines (kt)	Fe (%)	MassRec (%)	ConFe (Mt)	Au (g/t)	Rec. Au (%)	Au (koz)
Y-1	-	-	-	-	-	-	-	-	-	-	-
Y0	-	-	-	-	-	-	-	-	-	-	-
Y1	19,939	0.50	90.6	320	91	25.0	6.7	1.3	0.07	70.2	31
Y2	26,279	0.59	91.6	500	142	29.0	13.9	3.7	0.08	71.0	47
Y3	26,281	0.48	91.2	417	115	26.4	15.8	4.1	0.07	68.9	38
Y4	26,353	0.49	91.0	436	118	29.4	16.6	4.4	0.07	68.1	39
Y5	26,278	0.43	88.5	374	99	30.0	17.1	4.5	0.06	66.0	32
Y6	23,726	0.45	89.0	366	95	32.0	18.0	4.3	0.07	66.0	33
Y7	23,722	0.39	90.4	324	84	30.7	15.9	3.8	0.06	64.8	27
Y8	23,791	0.26	90.0	220	56	26.6	18.1	4.3	0.04	60.6	18
Y9	23,725	0.22	89.5	188	47	26.7	18.9	4.5	0.03	58.9	13
Y10	23,726	0.30	89.7	257	65	28.4	17.6	4.2	0.04	61.3	21
Y11	23,725	0.33	90.1	280	71	25.8	11.3	2.7	0.05	62.1	23
Y12	23,791	0.31	90.0	262	67	23.2	10.7	2.5	0.05	62.5	22
Y13	23,725	0.22	89.4	185	47	21.4	14.4	3.4	0.03	59.5	14
Y14	23,726	0.20	88.7	168	42	21.9	11.7	2.8	0.03	58.2	12
Y15	23,725	0.29	90.1	236	62	21.2	8.2	1.9	0.04	62.6	18
Y16	18,352	0.20	90.4	129	33	24.1	15.5	2.8	0.03	58.8	10
Y17	18,961	0.12	89.7	78	21	25.8	22.9	4.3	0.02	57.1	6
Y18	18,988	0.11	88.7	74	19	26.4	23.1	4.4	0.01	57.1	5
Y19	17,243	0.13	89.2	82	19	27.5	25.7	4.4	0.01	53.0	4
Total	**436,056**	**0.33**	**90.1**	**4,896**	**1,293**	**26.5**	**15.7**	**68.4**	**0.05**	**64.7**	**412**

16.4 Blasting and Explosives

The drilling equipment will consist of diesel units capable of drilling 9⅞" diameter holes for ore and 12¼" diameter holes for waste. Additionally, support units capable of drilling 6½" diameter holes for pre-splitting are included. Two units will be required for the pre-production period. During commercial production from Year 1 through Year 08 five units will be required. Support unit requirements are one during pre-production and LOM.

A general design for the drilling and blasting patterns has been carried out, using the assistance of experts to design the patterns. According to the drill pattern specified, a blasting powder factor between 131 g/t and 311 g/t were estimated, as a function of the rock type, see Table 16-5. Both estimated values are common for fresh rock material.

Table 16-5: Rock Type

Rock Type	Ore	Volcaniclastic Bottom	Breccia	Carbonate	ANLT Ande	Andesite Lower	Volcaniclastic	Volcaniclastic Top	Andesite Top	Diorit e
Powder factor [g/t]	311	172	182	193	189	179	134	131	143	143

16.5 Mining Equipment

Mine equipment requirements were calculated based on the annual mine production schedule, the mine work schedule and equipment annual production capacity estimates. This represents the equipment necessary to perform the following duties:

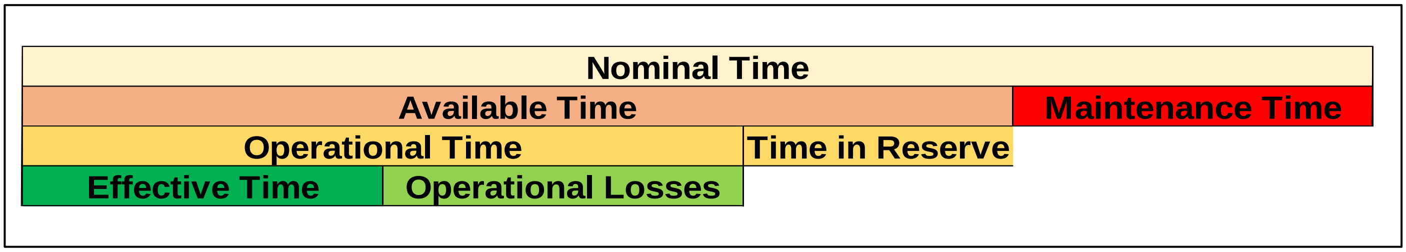
- Build pioneer roads to the initial mining areas as well as to the crusher, waste rock storage areas and stockpiles. Construct additional roads as needed to support mining activity
- The pre-production development required to expose ore for initial production
- Mine and transport ore to the primary crusher
- Mine and transport waste from the pit to the waste rock storage areas
- Maintain all the mine work areas, in-pit haul roads and external haul roads; and maintain the waste rock storage areas
- Re-handle the ore and marginal material (load, transport and auxiliary equipment) from the stockpiles to feed the primary crusher.

The mine major equipment was selected based on the mine production schedule, 18 months of pre-production and approximately 19 years of commercial mining operations. The pre-production period will include preparing roads, preparing bench openings and pre-production stripping. The total material mined during pre-production is 45 Mt. Re-handling will be required for Ore mined during pre-production to complete the plant feed requirement.

An average dry bank density of 3.1 t/m^3 was used for ore and 2.8 t/m^3 for waste. The density values are based on the resource block model values for the various materials tabulated from the mine production schedule. The material handling swell was estimated at 30% (3.1 x 0.70 = 2.2 t/m^3). A moisture content of 2% was assumed, which represents the weight percent of the wet weight of the material. The density of wet, loose material was used to calculate truck allowable payload limits.

An operational factor of 83%, to allow for operational losses, was used to estimate all major units of equipment and productivities; this corresponds to 50 minutes per operating hour. An operational factor of 85% was used for the haul trucks.

Figure 16-3: Operational Time



Source: Figure prepared by Capstone, 2024

The mining plan considers operating 24 hours a day, 365 days a year (Nominal Time). The Operational Time (Figure 16-3) is the product of the Available Time and the Utilization Factor, where the Utilization Factor considers the time in which the equipment is not being used (Time in Reserve) due to scheduled delays such as blasting, shift changes, and fuel supply. The Effective Time is the Operational Time minus any the Operational Losses such as losses due to loading front preparation, road repair, and control of blasting oversize, among others. Certain operational delays such as 'queue to dump' and 'queue at crusher' are considered as part of a truck's cycle time and thus are excluded from the Utilization Factor and Operational Losses.

The mine plan considers a fleet of electric shovels, hydraulic excavators, front-end loaders, and trucks with a capacity of 290 t. and equipment selected are able to achieve the annual total material movement of 90 Mt while providing sufficient selectivity for good grade control. The fleet will be complemented with blasthole drill rigs that will also be used in ore and waste delineation. Auxiliary equipment will include track dozers, wheel dozers, motor graders and water trucks. The mine fleet will also include the necessary equipment to re-handle the ore from the stockpiles to the primary crusher. This operation will be carried out using a front-end loader and the same 290 t trucks used in the open pit.

The peak equipment requirements for the pre-production and mine life are included as Table 16-6. Fleet requirements by year are included in Table 16-7. During pre-production the hydraulic excavator and front-end loader will be required. These will be complemented with two electric shovels during the commercial production period, from Year 1 through Year 15, after which the number will drop as less material is mined towards the end of the mine life.

The number of front-end loaders required is less than one for of the entire the mine life. The front-end loader will also be used as back-up for production loading activities.

Table 16-6: Peak Fleet Requirements for Pre-Production and Commercial Production

Type of Equipment	Peak Pre-Production	Peak Requirement
Front-end Loader 40-50 yd3	1	1
Hydraulic shovel 50-55 yd3	1	1
Electric shovel 68-73 yd3	-	2
Haul truck 290t	8	26
Production drill 10-13”	2	5
Support drill 6-8”	2	2
Bulldozer (610 horsepower)	1	2
Bulldozer (934 horsepower)	-	1
Wheel dozer (530 horsepower)	1	2
Wheel dozer (853 horsepower)	-	1
Motor grader	2	2
Water truck 70-75m3	2	2
Backhoe	1	1
Fuel truck	1	1
Mobile crane 200t	1	1
Lowboy truck	1	1
Tire handler	1	1
Lighting plant	5	11

Truck requirements for each period were calculated using cycle times for the various routes, as well as the productivity corresponding to each combination of truck and loading unit type.

The total haulage distance varies from a minimum of 3.2 km to a maximum of 6.6 km. Truck speeds were determined using typical values obtained from supplier information and similar operations. The truck cycle assignments include fixed times for loading, dumping and queuing. Two and a half minutes have been added to every cycle for dumping and queuing.

Operational indices considered for the trucks were:

- Availability (MA): Variable profile according to vendor and fleet life
- Utilization: 86.1%
- Operational losses: 85% (accounting for operational factors (including queue time and truck acceleration /deceleration), inspection and training).

Table 16-7: Fleet Requirements by Year

Name	Y-1	Y0 H1	Y0 H2	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17	Y18	Y18
FEL 40-50 yd3	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Hydraulic shovel 50-55 yd3		1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1		
Electrical Shovel 68-73 yd3				2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1	1
Haul truck 290t	6	7	8	16	21	22	24	24	22	24	24	25	26	26	26	26	24	26	26	26	18	11
Production drill 10-13”	1	1	2	4	4	4	5	5	5	5	4	4	5	4	4	4	3	4	4	4	3	2
Support drill 6-8”	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Bulldozer (610 HP)	1	1	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1
Bulldozer (934 HP)				1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Wheel dozer (530 HP)	1	1	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1
Wheel dozer (853 HP)				1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Motor grader	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1
Water truck 70-75 m3	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Backhoe	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Fuel truck	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mobile crane 200t	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Lowboy truck	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Tire handler	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Lighting plant	4	4	5	10	10	10	11	11	11	11	10	10	11	10	10	10	9	10	10	10	5	4

Note: FEL= front-end loader.

The number of trucks required during pre-production is 8. The requirement gradually increases from 16 units in Year 1 to a maximum of 26 units in Years 10, then decreases to the end of mine life as less material is mined.

The primary duties that will be assigned to the auxiliary equipment are as follows:

- Mine development including access roads, drop cuts, temporary service ramps and safety berms
- Waste rock storage area control, including maintaining access to the dumping areas and maintaining the travel surfaces
- Ore stockpile storage area control, including maintaining access to the stockpile areas and maintaining the travel surfaces
- Maintenance and clean-up in the mine and WRF areas
- Drilling for pre-splitting.

Equipment types included in the auxiliary mine fleet are:

- Track Dozer (610 HP)
- Track Dozer (934 HP)
- Wheel Dozer (530 HP)
- Wheel Dozer (853 HP)
- Grader (417 HP)
- Water Truck (75 m^3)
- Production Drill 10-13”
- Support Drill 6-8”

16.6 Mine Rotation Schedule

The mine is scheduled to work seven days per week, 365 days per year. Each day will consist of two 12-hour shifts. Four mining crews will rotate to cover the operation (two working and two on time off).

16.7 Risks

While the mine plan considers high-capacity equipment and qualified personnel to achieve the planned productions, future adjustments to the pit design during operations may require a greater amount of material to be removed. Below are the risks identified for the Mining Method and Plan:

- Availability of skilled labor to achieve mine plan

- Potential increases to ancillary equipment due to slope cleaning, oversize on blasting and mitigation of environmental commitments
- Potential increases in waste movement due to adjustments to pit design.
- The mine plan assumes that the electric shovels will employ double-sided loading whenever reasonably possible. This will require operational discipline to achieve; if it is not achieved then an additional loading unit will be required to reach planned production rates.

16.8 Opportunities

The mining method must continue to be analyzed and optimised. There may be optimization opportunities related to the equipment selected and the production rates, among others. The following potential opportunities have been identified:

- Continue to review the mining fleet to optimize type and size of equipment used
- Continue analyzing the economic optimum of the rate of total mine movement and its impact on the Plant production plan
- Share equipment for synergies with Mantoverde and Mantos Blancos, such as critical spares, ancillary equipment easily transported, etc.

16.9 Conclusions

The QP notes the following interpretations and conclusions, based on the review of data available for this Report:

- Pit designs are based on optimized LG shells at a revenue factor of 0.84 with variable overall slope angles according to geotechnical domains ranging from 36.3° to 47.9°.
- Ten pit phases are planned: eight for Santo Domingo and two for Iris Norte.
- The mine plan will generally process 72,000 t/d to 65,000 t/d of feed (26.3 to 23.7 Mt/y) with a peak total mining rate of 90 Mt/y.
- The 18-month pre-production period requires mining 45 Mt of total material to expose sufficient ore to start commercial production in Year 1. To allow for this, pre-production mining activities (pre-stripping) accesses and platforms are needed ahead of time. This is accounted for as an allowance and will be performed by a contractor.
- The total mined waste considers two main destinations for the material, the main waste storage areas and the tailings storage facility for the embankment construction.
- The mined ore will be hauled to the primary crusher for direct tipping. The mine plan employed an elevated cut off grade and stockpiling strategy during years of surplus production. Blending material will be mined and hauled to a stockpile located between the Santo Domingo and Iris Norte pits. This material will be re-handled and will become part of ore feed to regulate the iron grade. Oxide material is considered as waste in the mine plan.

- The mine is scheduled to operate seven days per week for 365 days per year. Each day will consist of two 12-hour shifts. Four mining crews will rotate, two working and two on time off, to cover the operation.
- The mining operation was developed for use of electric shovels, hydraulic excavators, front loaders and trucks with a capacity of 290 t. The fleet will be complemented with blasthole drill rigs that will also be used for ore and waste delineation. Auxiliary equipment will include track dozers, wheel dozers, motor graders and water trucks.

16.10 Recommendations

Based on the review of data available for this Report, the QP has the following recommendations related to the mine plan or mining methods, to be completed in-house by Capstone:

- Analyze the economic optimum of the rate of total mine movement and cutoff grade strategy, and their impact on the plant production plan.
- Analyze effect of increasing the pre-stripping to optimize the feed to the process plant, improving the revenue generation.

16.11 QP Comments

The QP considers that the methodologies described in Section 16 represent suitable application of industry standard approaches for use in mill scheduling and economic analysis.

17 RECOVERY METHODS

17.1 Overview

The Santo Domingo flowsheet produces copper concentrate and iron concentrate. The mineral processing plant has been designed for a processing rate of 65 kt/d at the nominal design P_{80} of 150 µm but can sustain higher annual averages during the initial years when softer ore feeds the plant. Milling and beneficiation have been designed to allow throughputs of up to 75 kt/d when processing the softer ore or operating at a coarser grind size and the downstream unit processes are expected to also be able to utilise design contingencies during the high throughput in the early years.

Ore is subject to crushing and grinding to produce an ore slurry amenable to beneficiation, through a gyratory crusher and a grinding mill circuit. The copper concentrate is produced through flotation and subsequently dewatered, filtered and stockpiled in the process plant location prior to transportation by truck to the port.

There is allowance for future production of pyrite concentrate (containing cobalt and copper) by flotation and recovery from the copper circuit cleaner-scavenger tails. The concentrate will be dewatered and stored in holding tanks prior to pumping of pyrite concentrate to a pyrite processing facility at the Mantoverde site. This pyrite facility is accommodated by the process design and plant layout but is not included in the costs or financial analysis in this report, as the data supporting this process is still in development.

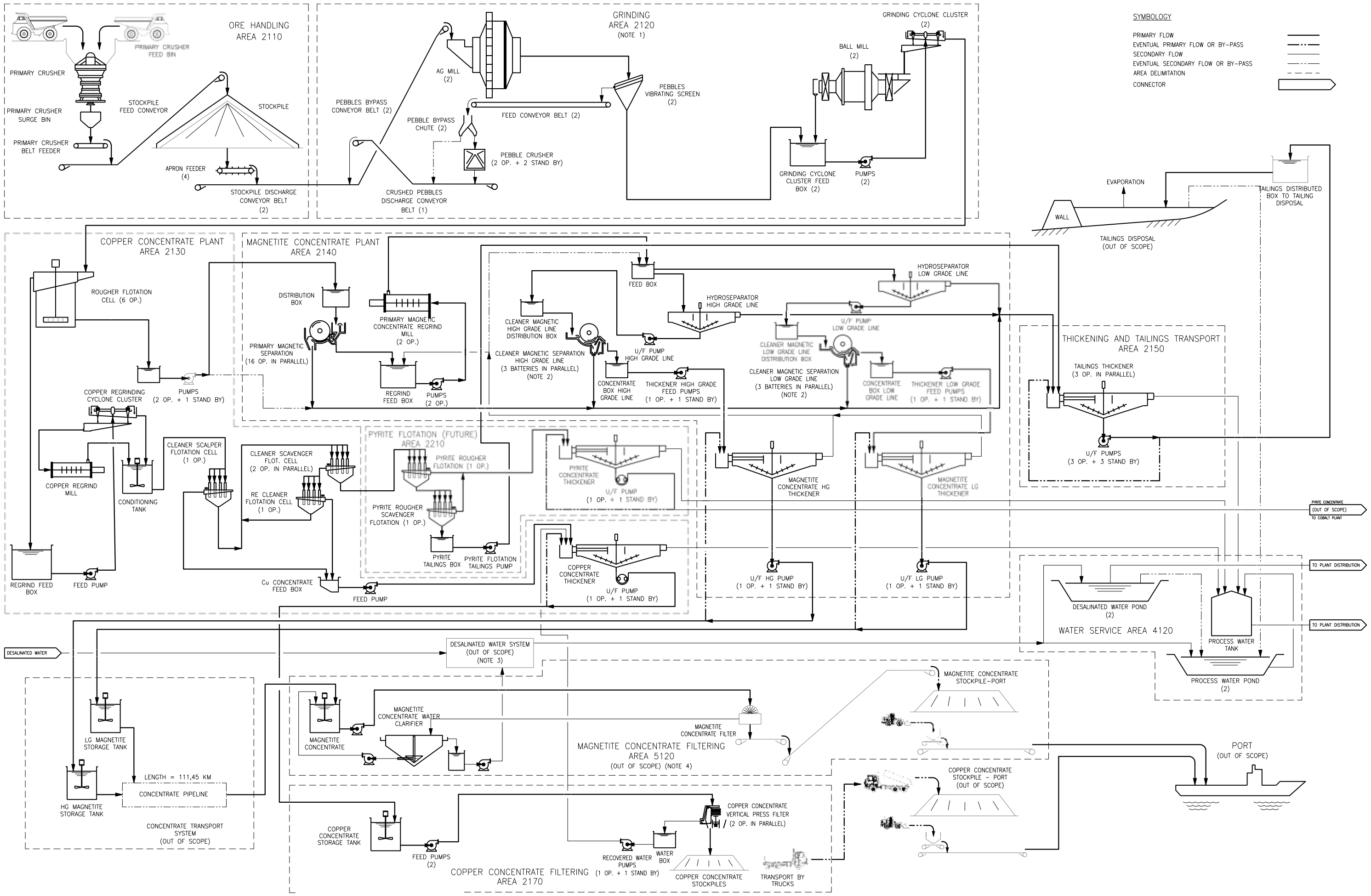
Magnetite concentrates are produced by magnetic separation in a dedicated magnetite plant located adjacent to the copper (and future pyrite) recovery plant. The magnetic separation plant processes copper circuit rougher tailings and produces two different grades of magnetite concentrate. The magnetite concentrates are dewatered in thickeners and stored in separate holding tanks according to iron grade. The magnetite concentrates are pumped via pipeline to the dewatering facility at the port in batches.

Water for processing is provided by a desalination plant located near the port and delivers desalinated water to the desalinated water ponds through use of a dedicated water pumping and pipeline system. Water from dewatering of magnetite concentrate at the port facility is also returned to the plant via the desalinated water pipeline.

17.2 Process Flowsheet

The general process flowsheet is shown in Figure 17-1. It illustrates the process flow in the following areas: crushing, ore handling; grinding; copper and (future) pyrite flotation, magnetite concentration; magnetite and (future) pyrite concentrate transportation pipeline systems, copper concentrate dewatering and filtering; copper concentrate stockpile and handling, and copper concentrate hauling to port for stockpiling and shipping.

Figure 17-1: Process Flowsheet



Source: Figure prepared by Ausenco, 2024.

17.2.1 Coarse Ore Handling and Crushing

The primary crushing plant processes ROM feed in an open crushing circuit. Trucks (290 t) unload material into a 450 t live capacity dump pocket with two truck dump stations; a rock breaker is provided for management of oversized rocks.

Primary crushing is carried out using a 60” x 89” gyratory crusher with 750 kW of installed power. The crusher product discharges in the 450 t surge bin, from where crushed ore is reclaimed using a belt feeder. The belt feeder transfers ore to the stockpile feed conveyor, and ultimately discharges fresh ore over a covered conical coarse ore stockpile.

The coarse ore stockpile has a live capacity equivalent to twelve hours of mill feed for 65 kt/d average daily rate. Ore is reclaimed from the stockpile using four apron feeders (two per line) to two mill feed conveyors, which feed two parallel grinding circuits.

17.2.2 Grinding and Classification

The grinding circuit consists of two parallel trains, each train with an autogenous grinding mill (AG mill), one AG mill discharge screen, pebble crushers (duty and standby), one cyclone cluster and one ball mill with trommel discharge.

The AG mills are 40’ x 25’ EGL mills with 18 MW of installed drive power, with variable speed dual pinion drives. The AG mills are fed by conveyor belts which deliver fresh ore reclaimed from the stockpile, including recirculating pebbles transferred from the pebble crusher stations. The AG mill product slurry discharges over a horizontal single deck vibrating screen (aperture 40 x 10 mm) for pebble washing. Pebbles report to the pebble conveyors.

The pebble conveyors report to the pebble crushing station bins (one per line). The bins are reclaimed with two feeders to feed the two pebble crushers (duty and standby, each of 750 kW installed power and base dimensions of 3,900 mm x 3,900 mm). Periods of high pebble generation are managed using both pebble crushers. Crushed pebbles are returned to the AG mill by a conveyor belt which discharges to the mill feed conveyor.

The design provides the option of bypassing the pebble crushers and allows recycling uncrushed material to the AG mill using the same belt conveyor system, when required. Pebbles can also be discharged to grade for occasions when there is need to run pebbles out of the circuit, such as for mill grind-out.

The AG mill screen undersize constitutes the fresh feed to the ball milling circuit and is discharged into the grinding cyclone cluster feed hopper, where it is mixed with the ball mill product. Slurry from the feed hopper is pumped to the cyclone cluster, from where cyclone underflow discharges into the ball mill and the cyclone overflow (P_{80} 150 µm) reports to the flotation circuit. One pump is installed, and a spare pump is in the vicinity for quick change-out.

The design also allows discharging grinding cyclone underflow to the AG mills, to enable operation of the AG mills as single stage mills. This allows operation at reduced tonnage for each grinding line when the corresponding ball mill is offline. Single stage operation of the AG mills allows greater flexibility for maintenance of the ball mills, without requiring a full grinding circuit shutdown.

Each ball mill is an overflow type ball mill (23’ x 32’) with 9 MW of installed power (single pinion drive with VSD for variable speed control). Ball milling operates in reverse closed circuit with a cluster of 33” cyclones.

17.2.3 Copper Flotation and Regrinding

Copper flotation consists of the following stages:

- Conventional rougher flotation;
- Regrinding and classification;
- Jameson cleaner circuit with cleaner scalper, cleaner scavenger, and re-cleaner cells.

The slurry product from both grinding lines flows through two MSA-type metallurgical samplers and is combined in the rougher flotation feedbox. Copper rougher flotation is carried out in a bank of six 630 m³ mechanical, forced air tank cells arranged in a 1-1-1-1-1-1 configuration. Flotation rougher concentrate produced from the rougher cells flows by gravity to the regrind mill feed box where it is pumped to a cyclone cluster operating in open circuit with a stirred mill for the copper concentrate regrind duty.

The products from the stirred mill (10,000 L) and regrind cyclone (20" diameter) overflow (P_{80} 34 µm) report to an agitated conditioning tank. Reagents are added to the conditioning tank and the tank overflow feeds the cleaner circuit.

The cleaner scalper stage consists of one Jameson cell (5.4 m diameter). The concentrate from this cell reports to final concentrate, while tailings gravitate to the cleaner scavenger stage (two 5.4 m diameter Jameson cells operating in parallel). The cleaner scavenger cells are also fed by recirculating recleaner stage tailings. Tailings from both cleaner scavenger cells constitute the feed to the future pyrite flotation cells.

Concentrate from both cleaner scavenger cells feed into the cleaner scavenger concentrate reception box and is pumped to the re-cleaner stage.

The re-cleaner flotation stage consists of one Jameson cell (5.4 m diameter). The tailings from this stage are recirculated to the cleaner scavenger stage. The concentrate from this stage is combined with the concentrate from the cleaner scalper stage, constituting the final copper concentrate which is pumped to the copper thickening stage.

Tailings from the rougher train discharge from the final cell through an MSA type metallurgical sampler to the primary flotation tails pump box and then are pumped to the magnetic separation plant. For occasions that require bypassing of the magnetite plant, tails can be pumped to the final tailings thickeners.

17.2.4 Future Pyrite Flotation

Copper cleaner scavenger cell tailings will gravitate by launder to the future pyrite flotation circuit, which includes two Jameson Cell pyrite flotation cells. Pyrite flotation reagents will be added to a feedbox to condition the pyrite flotation stages. These include sulfuric acid for pH adjustment, collector, and copper sulphate (if required) for pyrite flotation enhancement.

Concentrates from both pyrite flotation cells will be recovered in the pyrite concentrate hopper and from there pumped to the pyrite thickening stage prior to storage in agitated tanks. Pyrite will be transported by a dedicated transport pipeline to pyrite processing facilities located at Mantoverde.

17.2.5 Copper Thickening

The final copper concentrate is thickened in a 20 m diameter high-rate thickener with the aid of flocculant. The copper concentrate thickener underflow discharges at approximately 60% solids w/w and is pumped to the copper concentrate holding tank located in the filtration plant area.

Thickener overflow water gravitates to the process water tank adjacent to the tailings thickeners.

17.2.6 Future Pyrite Thickening

The final pyrite concentrate will be thickened in a 28 m diameter high-rate thickener with the aid of flocculant. The pyrite concentrate thickener underflow will discharge at approximately 63% solids w/w and be pumped to the pyrite concentrate holding tank located near the copper filtration area, from where pyrite concentrate will be transported using a dedicated pipeline.

Thickener overflow water will gravitate to the process water tank.

17.2.7 Copper Filtration and Load Out

Thickened copper concentrate from the thickener underflow discharges into the copper concentrate storage tank (6 h residence time)through a static trash screen which rejects possible trash and other oversize tramp materials that could compromise filter press performance. Trash reports to a bunker located at grade, for subsequent reclaim and disposal. The storage tank allows surge management between concentrate delivery, and batch feeding of filter presses.

Copper concentrate is filtered in two 108 m^2 pressure filters. Recovered water from these filters is sent to the copper concentrate filtrate tank, from where it is pumped to the concentrate thickener for clarification.

The filtered concentrate cake discharges to grade at approximately 9 % moisture content. Mobile material handling equipment reclaim and transport concentrate to the concentrate stockpile area, located in an extension of the filter area building.

The concentrate will be loaded into trucks from the stockpile using front end loaders and transported to the port.

17.2.8 Magnetic Separation

The magnetic separation plant is designed to produce up to two products, which may consist of two of the following three concentrate grade categories:

- 67% iron grade (or greater) as feed for direct reduction plants;
- 65% to 67% iron grade as feed for blast furnace plants, and
- 62% to 65% iron grade concentrate as feed for iron agglomeration processes, such as a pelletization.

The iron concentration area includes the following stages:

- Rougher magnetic separation;
- Regrinding;
- Hydro-separation (higher and lower grade);
- Cleaning magnetic separation (higher and lower grade);
- Magnetite concentrate thickening (higher and lower grade).

As noted above, the plant is designed as a dual product processing circuit that allows simultaneous production of two products of different grade through use of similar unit operations. The two process circuits use different magnetic intensities in the magnetic cleaner separation stage.

One of the circuits targets higher grade concentrate and includes cleaner stages of lower magnetic intensity than the lower grade producing circuit, to favour higher overall selectivity and production of a higher-grade concentrate. The higher grade circuit also has flexibility to report higher grade cleaner tails to the lower grade cleaner stages, to allow scavenging of iron product to lower grade concentrate.

17.2.8.1 Rougher Magnetic Separation and Regrinding

Tailings from the copper rougher flotation stage are pumped to two central distribution boxes which feed two parallel lines of eight back-to-back type wet counter-rotation magnetic drums (1,150 Gauss). Primary magnetic concentrate is collected in the mill feed hoppers.

Primary magnetic concentrate is pumped to the regrinding stage which consists of two parallel lines, each with one stirred mill. The circuit has been designed to produce a P_{80} of 40 µm. The solids density of the primary magnetic concentrate slurry is suitable for direct feed to the mill (20,000 L) and no classification is required prior to the regrinding circuit. Both mills discharge into the primary magnetic regrind product hopper, from where the reground concentrate is pumped to the hydro-separation stage. A dedicated pump feeds the higher grade hydro-separator, while another pump feeds the lower grade hydro-separator.

Rougher magnetic concentration tailings report to the final plant tailings stream.

17.2.8.2 Hydro-separation

This area consists of two hydro-separators (21 m diameter each) operating in parallel, one dedicated to the higher grade circuit and the other to the lower grade circuit.

Feed to the hydro-separators pass through magnetic flocculators to enhance settling of magnetic solids, and nonmagnetic and colloidal material (associated to silica) is removed by entrainment in the overflow stream. Deslimed magnetite concentrate reports to the underflow and is pumped to the magnetic cleaner stages. Hydro-separator underflows pass through demagnetizing coils prior to feeding downstream processes.

The combined overflow stream is fed to the tailings thickening stage while underflow is fed to the cleaner magnetic separation stage. The higher grade hydro-separator feeds the higher grade magnetic cleaner circuit and lower grade hydro-separator feeds the lower grade magnetic cleaner circuit.

17.2.8.3 Cleaner Magnetic Separation

The magnetite cleaning circuit consists of two parallel lines, each one with three magnetic separation units. One line is dedicated to higher grade, and the other to lower grade. Each line consists of a total of eight drums divided in two groups: two units with three counter current drums in series and a third unit in series with two counter current drums in series.

The lower grade line operates with a magnetic field intensity of 750 Gauss for the first two triplet units, and final double drum unit uses 500 Gauss for final finishing. The higher grade line operates with a magnetic field intensity of 500 Gauss for all units.

The final magnetite concentrate produced from each line reports to concentrate hoppers (one for higher grade and one for lower grade) and is pumped to the magnetite concentrate thickener area, where two thickeners recover water, each dedicated to one type of concentrate. Tailings from the cleaner magnetic stage (higher grade and lower grade) are combined with rougher tailings and sent to the final tailings stream.

An operating feature of the higher grade cleaner magnetic circuit is that tails from the cleaner units can be diverted to a dedicated recycle launder that does not report to tails and thus allows recycling higher grade cleaner tailings to the lower grade hydro-separator. This allows magnetite in the higher grade tailings to report to the lower grade cleaner stages and be recovered to the lower grade concentrate.

17.2.9 Magnetite Thickening

The final magnetite concentrate from each grade type is collected and discharged to hoppers and pumped to dedicated high-rate thickeners (20 m diameter each) after previously passing through a magnetic flocculator to enhance settling properties of the concentrate.

Overflow water from the each of the two concentrate thickeners is recovered and returns to the process.

Thickener underflow is pumped to magnetite storage tanks with separate storage for higher grade and lower grade concentrates. A dedicated concentrate pipeline system transfers the magnetite slurry to receiving tanks located at the port, in alignment with a batch strategy that allows segregation of the two types of products, higher grade and lower grade.

17.2.10 Lime and Reagent Preparation Plants

The lime and reagent preparation plants (including storage and distribution systems) is located near the flotation area; the flocculant preparation plants is located near the tailings thickeners.

The flotation reagent plant includes the primary collector, secondary collector and frother systems, each of which have reception, storage, and distribution facilities. The metering pump systems for lime distribution, primary collector, frother and flocculant supply the reagents to each of the required points in the process. The storage tanks are designed for 3.6 days capacity for lime and 7 days capacity for the collector, frother and flocculant. Reagents are programmed to be received on a regular basis.

Pyrite specific reagent preparation and distribution systems (future facilities) are in the vicinity of the two pyrite flotation cells.

17.2.11 Grinding Media

Separate grinding media handling systems serve the ball mills, the copper concentrate regrind mill and the magnetite concentrate regrind mills.

17.2.12 Tailings Thickening

Tailing thickening has been configured to make use of single stage of high-density thickening , based on thickening test work that supports target underflow densities from high-density thickeners. The thickeners use drive units currently supplied by two thickener vendors and are of a type currently in use in existing operations.

The test work was based on representative 2023 tailings sample and supported a sizing exercise which identified a settling rate of 0.5 $t/h/m^2$ for the design, resulting in the selection of 3 x 48 m diameter high density thickeners. A Maximum Operating Torque (MOT) of 14.1 MNm was specified for each thickener to achieve the target design underflow density of 67% solids w/w.

The tailings from the iron concentration plant (or directly from the copper rougher flotation tails if the magnetite plant is bypassed) are combined with copper scavenger flotation tailings (in the absence of pyrite flotation). Final thickening for disposal takes place near the process plant in three 48 m diameter high compression thickeners. Flocculant is added at a design dosing rate of approximately 30 g/t of tailings feed, to aid thickening and maximize water recovery.

The underflow from the thickener (approximately 67% solids by weight) is pumped to the tailings thickener underflow reception and pumping system, from where two trains of centrifugal pumps (five operating in series and five stand-by for first three years, then seven operating in series and seven stand-by) pump the tailings to the TSF.

Thickener overflow water is recovered in a process water tank located in the vicinity of the thickeners, as recovered water is preferentially recycled back to the process from this tank. The tank is also fed from the process water pond to make-up water requirement.

17.2.13 Plant Water Distribution System

The water distribution systems are designed to meet the water requirements for the distinct needs of the plant, including:

- Desalinated water for:
  - Magnetite plant
  - Copper concentrator services
  - Reagent preparation
- Process water for:
  - Copper concentrator
  - Magnetic separation plant
- Potable water for:
  - Diego de Almagro community
  - Plant Facilities
  - Non-Process infrastructure (offices, change house, etc.).

Two desalinated water ponds are provided at the plant site, each with a capacity of approximately 20,000 m^3, to receive and store desalinated water from the desalinated water plant located at the port area. Pumps are installed to pump desalinated water to the following consumption points from the desalinated water ponds: gland seal water tank, water treatment plant, crushing area dust suppression system, desalinated water and fire water tank, from where water is further distributed to the magnetite plant and other destinations. Desalinated water is preferentially used in the magnetite process to improve performance. This water then reports to the main process through the process water tank.

The desalinated water and fire water tank allows a reserve volume for firefighting duty with dedicated pumps.

The desalinated water pipeline that delivers desalinated water from the port to desalinated water pond is also used to transport recovered water from the magnetite concentrate filter plant located at the port. The intent is to operate the water pipeline as a batch operation, pumping desalinated water (to desalinated water ponds) and recovered water (to process water ponds) as separate periodic batches.

For process water inventory, two process water ponds are provided with a capacity of approximately 40,000 m^3 each, which will serve as main reservoir and inventory for process water. TSF water and magnetite filtrate reports to this pond as overflow from the process water tank.

17.2.14 Plant Auxiliary Facilities

The air distribution system in the plant will provide compressed air for consumption as plant air and instrument air.

A system based on compressors (operating and stand-by), accumulators and dryers will provide instrument air. This system will supply plant and instrumentation air to the stockpile, grinding, conventional flotation, magnetic separation, tailings thickening, lime plant and reagents areas.

Copper circuit rougher conventional flotation cells will use dedicated blowers with a standby unit. Copper circuit cleaner and future pyrite circuit cells do not require air services for flotation, given that they are self-aspirated Jameson cells.

Compressed air for primary crushing and copper filtration will be provided by dedicated systems. The plant and instrument air for primary crushing will be provided by an air service system, which includes dryers and/or accumulators for the instrument air.

Compressed air for copper filtration comes from two dedicated compressed air systems located next to the filtering building.

17.2.15 Port

The copper concentrate will be delivered to the port on trucks in sealed containers, which can be unloaded directly at the storage building or deposited in the handling and storage area, from which they will be sent to the storage building.

The copper concentrate is reclaimed using loaders which transfer the copper concentrate from the stockpiles to the belt feeder which feeds the copper concentrate onto the ship loader conveyor belt. The conveyor is fully enclosed to minimize dust emission. The conveyor has auxiliary equipment such as metal detector, magnet, sampler and belt scale.

Based on the current mining plan for the first years of operation, the following are the expected approximate peak production rate and stockpile capacity requirements:

- Copper concentrate peak (MSD & MV): 720 kt/y;
- Stockpile capacity for copper: 50,000 t;
- Platform Container Handling Area: 45,000 t.

17.2.15.1 Magnetite Concentrate

Magnetite concentrate is received from the magnetite slurry pipeline at the port in agitated storage tanks, one for each concentrate quality. The magnetite concentrate is then be pumped directly to the filter plant holding tank. The filter plant initially contains two ceramic disc filters, expanding to five filters after Year 4, each with a filter area of 144 m². The requirement is for two filters for the first year of operation, two additional filters for the third year and one additional filter from Year 4 onwards.

There is a common belt feeder which receives discharged filter cake from each pair of filters. Both belt feeders then discharge onto a conveyor which transfers the filtered concentrate to the mobile stacker at the magnetite stockpile. The mobile stacker runs along the north side of the magnetite concentrate stockpile area. The filtered concentrate has a moisture content of approximately 8%.

The filtrate from the magnetite concentrate filters reports to a clarifier (21 m diameter). The underflow from the clarifier is recirculated to the magnetite concentrate storage tank. The overflow from the clarifier reports to the desalinated water area in Bahía Flamenco, from where it is pumped to the plant. The magnetite concentrate stockpile area will be equipped with a fixed windbreak to reduce dust generation and comply with applicable environmental regulation.

Based on the current mining plan for the initial years of operation, the magnetite concentrate production peak is 5.4 Mt/y (dry). A magnetite stockpile capacity of 500,000 t is provided.

17.2.16 Port Auxiliary Facilities

17.2.16.1 Plant Air

The port requires both plant and instrument air. The compressor plant consists of two 110 kW compressors, one accumulator and one dryer with an accumulator for instrument air. The distribution networks consist of carbon steel piping and valves, oil filters, moisture traps, cut-off valves, quick connections, and controls. Air is not required for the filter plant operation other than instrument air which is provided from the instrument air accumulator. The plant and instrument air are provided via distribution ring main systems.

17.2.16.2 Dust Control

Dust suppression systems installed at concentrate transfer points use specialized nozzles to produce extremely small water droplets in a dispersed mist. These nozzles operate by atomizing water with compressed air. This type of dust control system consumes water at a rate of about 0.1 L/t to 0.5 L/t of copper concentrate.

The dust suppression system in stockpiles uses large volume water nozzles. For copper concentrate loading and conveyor transfers points, the dust is collected by dry bag filter systems.

17.2.17 Water Supply Facilities

Desalinated water for all of the facilities (mine, plant, and port) is provided by a build, own, operate, transfer (BOOT) contractor sea water desalination facility and delivered via pipelines to Capstone at the port and the mine site.

The water desalination facilities consist of a sea water intake, filtration, treatment, and a reverse osmosis desalination plant located at the port area.

For construction stage, the potable water for the port is supplied by a sanitary company from a main pipe in the port area. Potable water for the plant and camp facilities is also supplied by the sanitary company through a main pipe near the bypass of the C-17 route.

17.3 Plant Design

17.3.1 Design Criteria

The Santo Domingo flowsheet produces copper concentrate and iron concentrate. A pyrite concentrate may be produced in the future. The concentrator has been designed to sustain an average feed rate of 65 kt/d at a design P_{80} of 150 µm but will be able to achieve 75 kt/d when processing softer ore early in the mine life.

The main process design criteria include:

- Design capacity: 65,000 t/d;
- Milling and beneficiation has been designed to allow a throughput of 75,000 t/d when processing softer ore or operating a coarser grind size;
- Operating period: 365 days per year, with effective yearly hours of operation according to area utilization.

Table 17-1 provides the projected utilization for the various plant area and components and Table 17-2 summarized main ore characteristics. For this chapter, utilization is defined as the percent of total time the circuit is operating and nominal design capacity, based on 24 hours per day and 365 days per year.

Table 17-3 provides a summary of the planned crushing and grinding design criteria. The copper concentrate circuit design summary is included in Table 17-4, the magnetite circuit in Table 17-5, and the tailing thickener circuit in Table 17-6.

Table 17-1: Area Utilization

Area	Utilization (%)
Primary crusher	75
Grinding	93
Copper and pyrite Flotation	93
Tailings thickening	93
Copper and pyrite thickening	93
Reagent (lime – flocculant)	93
Magnetic separation	93
Magnetite concentrate thickening	93
Magnetite concentrate pipeline	98.5
Filters and copper conc. handling	80
Filters and magnetite conc. handling	90

Table 17-2: Ore Characteristics

Parameter	Unit	Value
Ore Specific Gravity		
Range	t/m^3	2.6 – 4.2
Design	t/m^3	3.2
Bulk Density		
Range	t/m^3	1.56 - 2.52
Design	t/m^3	1.6
Moisture, % H_2O		
Range	%w/w	2 - 6
Design	%w/w	3.0

Parameter	Unit	Value
JK DWT, A x b		
Average	-	45.1
Design	-	36.7
Bond Crushing Work Index, CWi		
Average, design	kWh/t	13.6
Bond Rod Mill Work Index, RWi		
Average	kWh/t	14.5
Design	kWh/t	16.4
Bond Ball Mill Work Index, BWi		
Average	kWh/t	12.3
Design	kWh/t	14.1
Abrasion Index, Ai		
Range	g	0.01-0.37
Design, percentile 75	g	0.18

Table 17-3: Design Criteria for Grinding Circuit

Parameter	Unit	Value
Grinding Plant Capacity		
Circuit configuration	-	ABC
Grinding circuit capacity, design (dry)	t/h	2,912
Grinding circuit feed size, F_{80}	mm	118
Grinding circuit product size, P_{80}	µm	150
Autogenous Mill (AG)		
AG mill size	ft	40 x 25
Discharge pulp density	% solids (w/w)	72
AG mill speed, % of critical	-	-
Maximum	%	80
Design	%	72
Minimum	%	60
AG mill filling	-	-
Design	% v/v	29
Maximum operating	% v/v	33
Pebble Crusher		
Surge bin capacity per line, live	t	55
Pebble rate, per grinding line	-	-
Design	% of new feed	45
Maximum	% of new feed	68
Design	t/h	990
Number of crushing stations	-	2
Number of pebble crushers per station	-	2
Pebble crusher specific energy	kW/h/t	1.33

Parameter	Unit	Value
Crusher feed size, F_{100}	mm	80
Crusher feed size, F_{80}	mm	40
Crusher product size, P_{80}	mm	12
Ball Mill		
Type	-	overflow
Ball mill size	ft	23 x 32
Number of ball mills	-	2
Circuit configuration	-	closed
Ball mill circuit P_{80}	µm	150
Specific energy	kWh/t	5.63
Ball mill discharge pulp density	% solids (w/w)	75
Circulating load, design	%	250
Circulating load, maximum	%	350
Ball mill speed, % of critical	%	75
Ball mill ball charge	-	-
Design	% v/v	28.8
Maximum operating	% v/v	31.5
Structural design	% v/v	40.0
Ball mill discharge screen	-	-
screen type	-	trommel
aperture type	-	slotted
aperture size	mm	12 x 50
Classification Cyclones		
Hydrocyclone cluster, per line	-	1
Hydrocyclone, per cluster	-	12
Operating	-	9
Stand by	-	3
Feed pulp density, nominal	% solids (w/w)	56
Feed pulp density, design	% solids (w/w)	60
Overflow density	% solids (w/w)	33.0
Underflow density	% solids (w/w)	78.3

Table 17-4: Copper and Future Pyrite Circuit

Parameter	Unit	Value
Rougher		
Feed	-	-
Pulp density	% solids (w/w)	33
Pulp pH	-	9.5
Specific gravity	t/m^3	3.2
Cells	-	-
Type	-	Conventional

Parameter	Unit	Value
Number of cells	-	6
Flotation cell size	m^3	630
Cell arrangement	-	1-1-1-1-1-1
Stage recovery	-	-
Copper, design	%	95.9
Sulphur, design	%	97.1
Mass, design	%	9.23
Concentrate grade		
Copper, design	%	7.28
Specific gravity	-	3.6
Pulp density	% solids (w/w)	23.7
Cleaner Scalper		
Feed	-	-
Pulp density	% solids (w/w)	23.2
Pulp pH	-	12.0
Specific gravity	-	3.6
Cells	-	-
Type	-	Jameson Cell
Number of cells	-	1
Flotation cell size	-	B5400/18
Number of downcomers	-	18
Stage recovery	-	-
Copper, design	%	76.0
Mass, design	%	21.3
Concentrate grade	-	-
Copper, design	%	26.0
Specific gravity	-	4.10
Cleaner Scavenger		
Feed	-	-
Pulp density	% solids (w/w)	19.3
Pulp pH	-	12.0
Specific gravity	-	3.49
Cells	-	-
Type	-	Jameson Cell
Number of cells	-	2
Flotation cell size	-	B5400/18
Number of downcomers	-	18
Stage recovery	-	-
Copper, design	%	94.5
Mass, design	%	20.5
Concentrate grade	-	-
Copper, design	%	12.30

Parameter	Unit	Value
Specific gravity	-	3.6
Re-Cleaner		
Feed	-	-
Pulp density	% solids (w/w)	23.0
Pulp pH	-	12.0
Specific gravity	-	3.60
Cells	-	-
Type	-	Jameson Cell
Number of cells	-	1
Flotation cell size	-	B5400/18
Number of downcomers	-	18
Stage recovery	-	-
Copper, design	%	70.0
Mass, design	%	33,1
Concentrate grade	-	-
Copper, design	%	26.0
Specific gravity	-	4.10
Pyrite Rougher (Future)	-	-
Feed	-	-
Pulp density	% solids (w/w)	15.0
Pulp pH	-	9.5
Specific gravity	-	3.44
Cells	-	-
Type	-	Jameson Cell
Number of cells	-	1
Flotation cell size	-	B5400/18
Number of downcomers	-	18
Stage recovery	-	-
Sulphur, design	%	60.0
Mass, design	%	45.7
Concentrate grade	-	-
Sulphur, design	%	46.8
Specific gravity	-	5.0
Pyrite Rougher Scavenger (Future)	-	-
Feed	-	-
Pulp density	% solids (w/w)	15.0
Pulp pH	-	9.5
Specific gravity	-	3.44
Cells		-
Type	-	Jameson Cell
Number of cells	-	1
Stage recovery	-	-

Parameter	Unit	Value
Sulphur, design	%	62.5
Mass, design	%	35.1
Concentrate grade		-
Sulphur, design	%	46.8
Specific gravity	-	5.0
Regrinding hydro-cyclones	-	-
Cyclone overflow pulp density	% solids (w/w)	15.1
Mass fraction report to overflow	%	50.0
Cyclone underflow pulp density	% solids (w/w)	45.0
Feed size (F_{80})	µm	84
Product particle size (P_{80})	µm	34
Specific gravity	-	3.6
Hydrocyclones	-	-
Type	-	Conical
N° of cluster per line	-	1
N° of hydrocyclones per cluster	-	8
Operating	-	6
Size	mm	508
Operating pressure for hydrocyclones, design	kPa	70
Regrind Mill	-	-
Type	-	Stirred Mill
N° of mill per line	-	1
Installed motor power	kW	3,000
Product size (P_{80})	µm	34
Specific energy	kWh/t	9.1
Media filling (Jb)	%	45
Copper Thickener	-	
Type	-	High rate
Diameter	m	20
N° of thickeners	-	1
Unit area	t/h/m^2	0.25
Feed pulp density	% solids (w/w)	25
Underflow pulp density	% solids (w/w)	60
Flocculant	-	-
Dosage	g/$t_{concentrate}$	3.0
Pyrite Thickener (Future)	-	-
Type	-	High rate
Diameter	m	28
N° of thickeners	-	1
Unit area	t/h/m^2	0.25
Feed pulp density	% solids (w/w)	25
Underflow pulp density	% solids (w/w)	63

Parameter	Unit	Value
Flocculant		
Dosage	$g/t_{concentrate}$	3.0
Copper Filtration	-	-
Type	-	Vertical press with horizontal plates
N° of filters	-	2
Chamber thickness	mm	60
Filtration area required (per filter)	m^2	108
Product moisture, design	%	9
Specific gravity	t/m^3	4.1
Storage tank	-	-
N° of tanks	-	1
Total volume	m^3	700
Residence time @ design rate	h	6.5

Table 17-5: Magnetite Circuit

Parameter	Unit	Value
Primary Magnetic Separation		
Feed	-	-
Pulp density, nominal	% solids (w/w)	28
Grade total Fe, range	%	26 – 33
Grade total Fe	%	32.6
Grade magnetic Fe	%	16.1
Specific gravity	-	3.14
F_{80}	µm	160
Equipment	-	-
Type	-	Twin-back wet counter rotation drum tank magnetic separator
Magnetic Intensity	Gauss	1,150
Working gap width	mm	50
Maximum magnetic loading	t/h/m	50
N° of lines	-	2
N° of units, per line	-	8
N° of total units	-	16
Concentrate		
Total mass recovery, design	%	30.8
Fe magnetic recovery	%	90.0
Fe total recovery	%	51.5
Pulp density	% solids (w/w)	65.4
Pulp density, diluted	% solids (w/w)	54
Specific gravity	t/m^3	4.53
Product size (P_{80})	µm	125
Cleaner Magnetic Separation		

Parameter	Unit	Value
Equipment		
Type of unit	-	Triplet or doublet wet counter current drum tank magnetic separator
Magnetic intensity, higher grade	Gauss	500 all units
Magnetic intensity, lower grade	Gauss	750 for triplet, 500 for doublet
Working gap width	mm	44.5
Maximum magnetic loading	t/h/m	35.0
N° of units	-	8
1st Cleaner Magnetic Separation		
Feed		
Pulp density	% solids (w/w)	20.0
Specific gravity	t/m^3	4.53
Concentrate		
Pulp density	% solids (w/w)	55.0
Specific gravity	t/m^3	4.62
Tailings		
Pulp density	% solids (w/w)	5.13
Specific gravity	t/m^3	3.50
Recovery		
Total mass recovery	%	82.0 for higher grade 83.0 for lower grade
Fe total recovery	%	92.2 for higher grade 92.2 for lower grade
Fe magnetic recovery	%	96.2 for higher grade 96.2 for lower grade
2nd Cleaner Magnetic Separation		
Feed		
Pulp density	% solids (w/w)	20.0
Specific gravity	t/m^3	4.62
Concentrate		
Pulp density	% solids (w/w)	55.0
Specific gravity	t/m^3	4.67
Tailings		
Pulp density	% solids (w/w)	2.26
Specific gravity	t/m^3	3.29
Recovery		
Total mass recovery	%	92.5 for higher grade 93.5 for lower grade
Fe total recovery	%	94.0 for higher grade 94.5 for lower grade
Fe magnetic recovery	%	98.3 for higher grade 98.3 for lower grade
3rd Cleaner Magnetic Separation		
Feed		

Parameter	Unit	Value
Pulp density	% solids (w/w)	20.0
Specific gravity	t/m^3	4.67
Concentrate		
Pulp density	% solids (w/w)	55.0
Specific gravity	t/m^3	4.74
Tailings		
Pulp density	% solids (w/w)	1.92
Specific gravity	t/m^3	3.57
Recovery		
Total mass recovery	%	93.7 for higher grade 94.7 for lower grade
Fe total recovery	%	95.9 for higher grade 95.9 for lower grade
Fe magnetic recovery	%	98.5 for higher grade 98.5 for lower grade
4th Cleaner Magnetic Separation		
Feed		
Pulp density	% solids (w/w)	20.0
Specific gravity	t/m^3	4.74
Concentrate		
Pulp density, nominal	% solids (w/w)	55.0
Specific gravity	t/m^3	4.81
Tailings		
Pulp density, nominal	% solids (w/w)	0.77
Specific gravity	t/m^3	3.57
Recovery		
Total mass recovery	%	97.5 for higher grade 98.0 for lower grade
Fe total recovery	%	98.4 for higher grade 98.8 for lower grade
Fe magnetic recovery	%	98.5 for higher grade 98.5 for lower grade
5th Cleaner Magnetic Separation		
Feed		
Pulp density	% solids (w/w)	20.0
Specific gravity	t/m^3	4.81
Concentrate		
Pulp density	% solids (w/w)	55.0
Specific gravity	t/m^3	4.88
Tailings		
Pulp density	% solids (w/w)	0.77
Specific gravity	t/m^3	3.57
Recovery		

Parameter	Unit	Value
Total mass recovery	%	97.5 for higher grade 98.0 for lower grade
Fe total recovery	%	98.4 for higher grade 98.8 for lower grade
Fe magnetic recovery	%	99.0 for higher grade 99.0 for lower grade
6th Cleaner Magnetic Separation		
Feed		
Pulp density	% solids (w/w)	20.0
Specific gravity	t/m³	4.81
Concentrate		
Pulp density	% solids (w/w)	55.0
Specific gravity	t/m³	4.88
Tailings		
Pulp density	% solids (w/w)	0.77
Specific gravity	t/m³	3.57
Recovery		
Total mass recovery	%	97.5 for higher grade 98.0 for lower grade
Fe total recovery	%	98.4 for higher grade 98.8 for lower grade
Fe magnetic recovery	%	99.0 for higher grade 99.0 for lower grade
7th Cleaner Magnetic Separation		
Feed		
Pulp density	% solids (w/w)	15.0
Specific gravity	t/m³	4.81
Concentrate		
Pulp density	% solids (w/w)	53.0
Specific gravity	t/m³	4.88
Tailings		
Pulp density	% solids (w/w)	0.42
Specific gravity	t/m³	3.57
Recovery		
Total mass recovery	%	98.0 for higher grade 98.0 for lower grade
Fe total recovery	%	99.0 for higher grade 99.0 for lower grade
Fe magnetic recovery	%	99.1 for higher grade 99.1 for lower grade
8th Cleaner Magnetic Separation		
Feed		
Pulp density	% solids (w/w)	15.0
Specific gravity	t/m³	4.81

Parameter	Unit	Value
Concentrate		
Pulp density	% solids (w/w)	53.0
Specific gravity	t/m^3	4.88
Tailings		
Pulp density	% solids (w/w)	0.42
Specific gravity	t/m^3	3.57
Recovery		
Total mass recovery	%	98.0 for higher grade 98.0 for lower grade
Fe total recovery	%	99.0 for higher grade 99.0 for lower grade
Fe magnetic recovery	%	99.1 for higher grade 99.1 for lower grade
Regrind Circuit		
Feed particle size (F_{80})	µm	125
Regrind Mill		
Type of circuit	-	Direct feed to mill, without classification
Regrind mill	-	Stirred Mill
Specific energy	kWh/t	11.4
Power per mill	kW	5000
N° of lines	-	2
Product particle size (P_{80})	µm	40
Hydro-separator		
N° of units	-	2
Diameter	m	21
Rise rate	$m^3/h/m^2$	8.65
Grade total Fe, underflow	%	54.7
Mass recovery	%	95.0
Feed pulp density	% solids (w/w)	12
Underflow pulp density	% solids (w/w)	35
Thickener (Higher grade and Lower grade)		
Type	-	High rate
Diameter	m	20
N° of thickeners	-	2
Solids Loading	$t/h/m^2$	1.08
Feed pulp density, nominal	% solids (w/w)	53
Underflow pulp density, nominal	% solids (w/w)	65
Flocculant		
Type	-	Magnetic Flocculator
Magnetite Filtration		
Type	-	Ceramic disc filter
N° of filters	-	5

Parameter	Unit	Value
Chamber thickness	mm	60
Filtration area required (per filter)	m^2	144
Product moisture, design	%	8
Specific gravity	t/m^3	4.9

Table 17-6: Tailings Thickener

Parameter	Unit	Value
Thickener		
Type	-	High density
N° of thickeners	-	3
Thickener diameter	m	48
Feed pulp density (35% only when magnetite plant is bypass)	% solids (w/w)	7.2%
Unit area thickening rate	$(t/h)/m^2$	0.50
Underflow pulp density, 1st year	% solids (w/w)	58
Underflow pulp density, 2nd year	% solids (w/w)	65
Underflow pulp density, design	% solids (w/w)	67
Flocculant		
Type	-	Magnafloc 1011 + Rheomax 1050 (50:50)
Dosage	$g/t_{tailings}$	30

17.4 Comminution Circuit Selection and Throughput Modelling

A techno-economic assessment of the different comminution circuits was performed by Ausenco, first as a dedicated trade-off study (Ausenco Comminution Circuit TOS, November 2022) and later revised as part of the feasibility study. These studies concluded that an autogenous grinding circuit with ball mill and pebble crusher circuit returned the maximum value due to the reduced operating costs (lower steel media consumption).

A data set of test results was used that included RQD, JK Drop Weight Test (Axb), Drop Weigh Index (DWi), Bond rod mill work index (RWi), Bond ball mill work index (BWi), and specific gravity (SG) as measured in the SMC test. The design SG is important as the power draw of the AG mill is influenced by the rock SG and the mill filling.

The average rock density (3.2 t/m^3) at 28% volumetric filling was used for the design pinion power of the AG mill to ensure the mill can draw the power required at lower ore SG.

The 75th percentile rock density (3.4 t/m^3) at 32% volumetric filling was used to determine the installed motor power to ensure that the mill can still operate during periods of high-density feed.

RQD data reviewed were not typical Andean ore that have RQD values of 30-40%. From the database, 73% of the samples were reported to have RQD values greater than 80%. As a result, the typical Andean AG/SAG specific energy correction was not applied to the Santo Domingo design.

A geometallurgical assessment and data evaluation was done using Cancha. Transmin's Cancha is an integrated geometallurgy software package that streamlines sample selection, results interpretation, prediction modelling and reporting. In Cancha, mining pit shells for the MSD project for years 5, 7, 8 and 9 were assessed and cross-referenced with the drill core to evaluate spatial distribution of the samples and how the design parameters change over time.

Variability was observed on different scales within the Santo Domingo Sur pit. On a >100 m scale, there is a clear distinction between more central andesite units and a larger proportion of volcaniclastic and manto-style lithologies at the southern end of the pit, alongside a higher proportion of carbonates and some diorite. On a smaller scale, the more prominent variability is between Manto-style lithology and andesitic tuff (ANDT).

Looking at distribution of copper within the deposit, most higher-grade copper zones coincide with the 'Manto' style lithology. Other higher grade lithologies are volumetrically insignificant or represent a fault breccia (FAULT) that could also include fragmented Manto-style lithology (i.e., this is a structural overprint of the actual lithology). Other lithologies, such as andesite (ANDE) and tuff (ANDT) tend to coincide with lower copper grades. This suggests that the Manto lithology and its variability will be important from a comminution perspective, as the plant will likely process a larger amount of this material relative to (typically) lower copper grade lithologies such as andesite.

Taking this general copper distribution forward and looking at the predicted BWi it was clear that Manto-style lithology is at the softer end of material found in Santo Domingo Sur, whilst andesitic tuff is considerably harder. This shows that there will be metre-scale variability of the ore in terms of comminution characteristics. Given the relatively small scale of this variability relative to the scale of the benches, it is expected that this will be homogenized within the crushing circuit. Nonetheless, this variability will require consideration in the blending strategy to ensure that it does not detrimentally impact AG mill operation.

For design of the comminution circuit the following process design criteria were selected:

- The mill motors were sized appropriately for the 75th percentile SG at maximum filling at 80% of critical speed;
- The AG mills were sized for a design speed of 72% of critical, to improve the ratio of attrition breakage to impact breakage;
- Pebble production rates are a function of the ore hardness, ball load and the grate design. Considering the design conditions and the data reviewed, 45% pebble production with a maximum of 68% was selected for design of the pebble handling and crushing equipment.

The geometallurgical analysis calculated the mill throughput and grind size using the 75th percentile ore characteristics for each year. The estimated throughput and grind size were based on the AG and ball mill motor power not exceeding 90% of the motor nameplate power.

With the lower ore competence (DWi value) in years 1-4, the design throughput of 65 kt/d can be exceeded with throughputs of up to 75 kt/d for the design P_{80} of 150 µm. However, the ball mill constrains the throughput in years 4-7 at the design grinding circuit product size and the 75th percentile BWi values. At lower BWi values, this constraint is relaxed accordingly.

Overall analysis of the data provided the basis for design for comminution in Table 17-7.

Table 17-7: Ore Characteristics for Process Design from Geometallurgical Analysis

Parameter	Units	Value
Axb	-	36.7
SG	-	3.2
DWi	kWh/m³	8.7
RWi	kWh/t	16.4
BWi	kWh/t	14.1
Total Specific Energy	**kWh/t**	**15.4**

17.5 Production Plan

Table 17-8 presents the production plan derived from the mine plan and the metallurgical models for copper and iron recovery at yearly average treatment rates. The annual peak production is 500 kt of copper concentrate and 4.5 Mt of magnetite concentrate, which both occur during the first 5 years of production.

Figure 17-2 shows the grades of copper and iron in the plant feed. The head grade will vary between 0.39% Cu and 0.59% Cu during the first 7 years of production. After the Seventh year, the head grade is projected to decrease to between 0.33% Cu and 0.20% Cu for nine years, before reaching about 0.12% Cu for the final three years of mining.

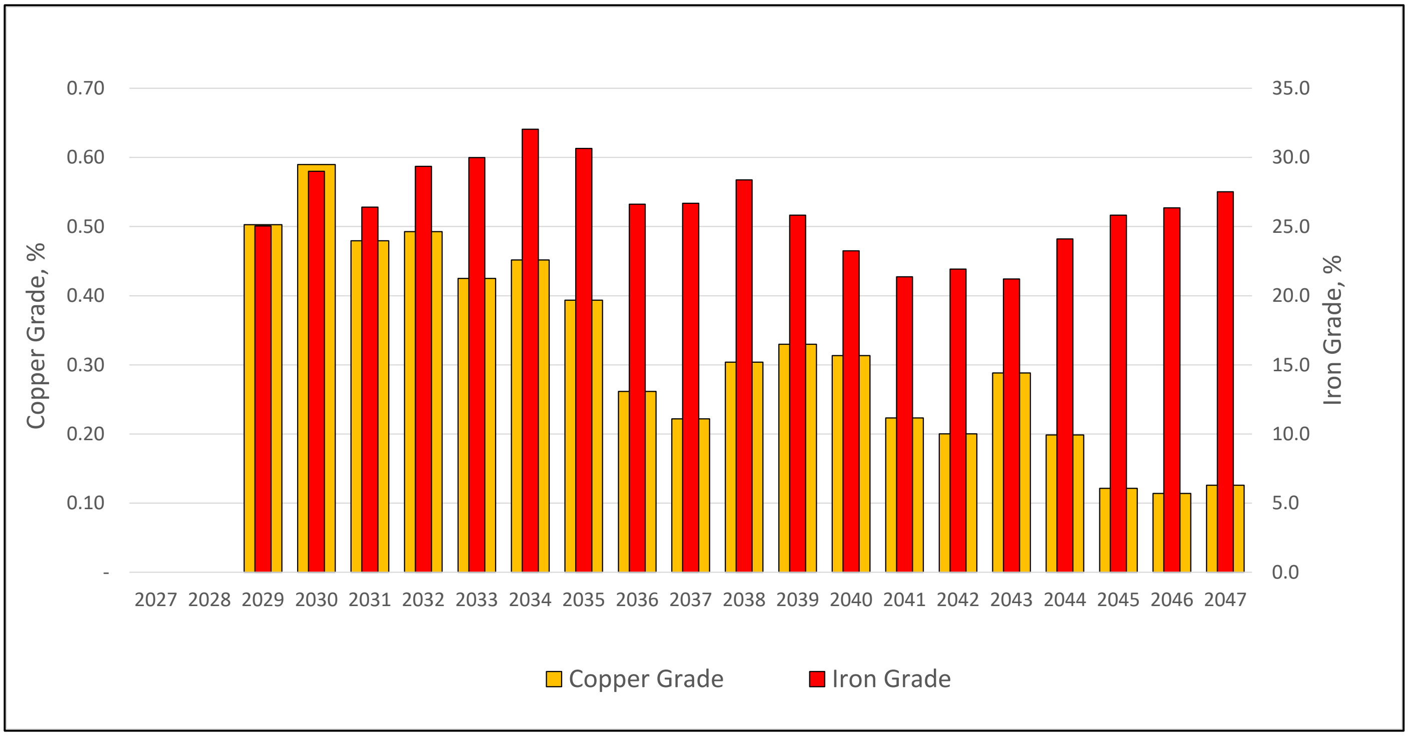
For the first 7 years, the head grade will average 29.0% Fe with variation over the LOM. After the seventh year, the head grade for iron is projected to vary between 21.2% Fe and 28.4% Fe, with average of 24.8 Fe%.

Table 17-8: Production Plan

Year	Mineral (kt)	%CuT	Rec Cu (%)	Rec Au (%)	Conc Cu (kt)	Cu Metal in Conc (kt)	Au Metal (koz)	Fe (%)	Conc. Iron Total (kt])	Mass Recovery (%)
2027	-	-	-	-	-	-	-	-	-	-
2028	-	-	-	-	-	-	-	-	-	-
2029	19,939	0.50	90.6	70.2	320	91	31	25.0	1,345	6.74
2030	26,279	0.59	91.6	71.0	500	142	47	29.0	3,663	13.9
2031	26,281	0.48	91.2	68.9	417	115	38	26.4	4,139	15.8
2032	26,353	0.49	91.0	68.1	436	118	39	29.4	4,380	16.6
2033	26,278	0.43	88.5	66.0	374	99	32	30.0	4,494	17.1
2034	23,726	0.45	89.0	66.0	366	95	33	32.0	4,274	18.0
2035	23,722	0.39	90.4	64.8	324	84	27	30.7	3,772	15.9
2036	23,791	0.26	90.0	60.6	220	56	18	26.6	4,305	18.1
2037	23,725	0.22	89.5	58.9	188	47	13	26.7	4,480	18.9
2038	23,726	0.30	89.7	61.3	257	65	21	28.4	4,182	17.6
2039	23,725	0.33	90.1	62.1	280	71	23	25.8	2,682	11.3
2040	23,791	0.31	90.0	62.5	262	67	22	23.2	2,545	10.7
2041	23,725	0.22	89.4	59.5	185	47	14	21.4	3,421	14.4

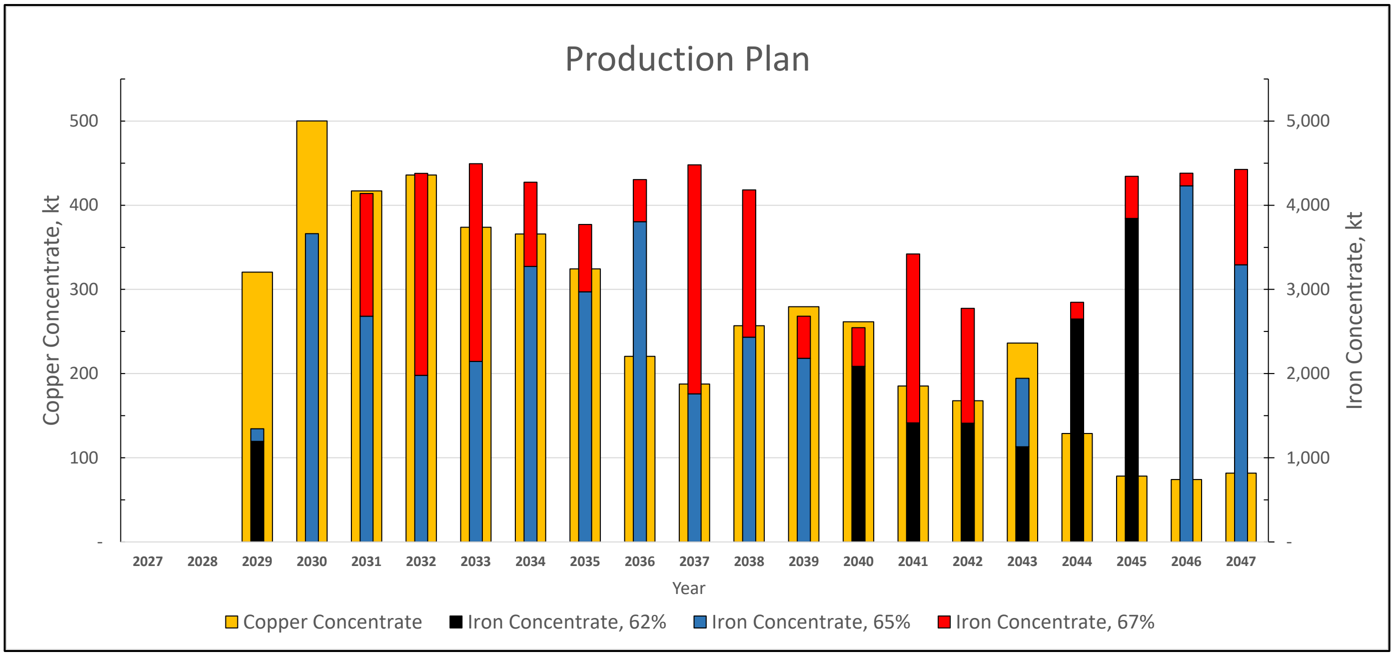
Year	Mineral (kt)	%CuT	Rec Cu (%)	Rec Au (%)	Conc Cu (kt)	Cu Metal in Conc (kt)	Au Metal (koz)	Fe (%)	Conc. Iron Total (kt])	Mass Recovery (%)
2042	23,726	0.20	88.7	58.2	168	42	12	21.9	2,775	11.7
2043	23,725	0.29	90.1	62.6	236	62	18	21.2	1,944	8.19
2044	18,352	0.20	90.4	58.8	129	33	10	24.1	2,848	15.5
2045	18,961	0.12	89.7	57.1	78	21	6	25.8	4,345	22.9
2046	18,988	0.11	88.7	57.1	74	19	5	26.4	4,381	23.1
2047	17,243	0.13	89.2	53.0	82	19	4	27.5	4,425	25.7
Total	**436,056**	**0.33**	**90.1**	**64.7**	**4,896**	**1,293**	**412**	**26.5**	**68,398**	**15.7**

Figure 17-2: **Iron and Copper Grade of Plant Feed**



Source: Figure prepared by Ausenco based on data provided by Capstone, 2024.

Figure 17-3: Production Plan for Copper and Iron Concentrate



Source: Figure prepared by Ausenco based on data provided by Capstone, 2024.

17.6 Energy, Water and Process Materials Requirements

The peak power demand to sustain operations is estimated to be approximately 130 MVA, excluding the desalinated water system in the port.

As noted in Section 17.2.13, the mineral processing plant will use fresh desalinated and recycled water.

Desalinated water is used for both process requirements and for production of potable water for process plant use, including safety showers / eye wash when required, and camp consumption. Permitting commitments also define a reserved stream to be provided to nearby community (10 L/s).

Over the life of mine, the average annual consumption, including non-process consumption and commitments, amounts to the equivalent of approximately 1050 m^3/h (290 L/s).

The forecast water requirement is satisfied by the 366 L/s capacity of the desalinated water delivery system as authorized in RCA No. 122-2020 "Change of Desalination Plant"

Reagents required for the process plant operation include lime, primary collector (PEX), flocculant and frother (MIBC). Desalinated water is used for reagent preparation and, where needed, dilution is performed with process water at the delivery points, such as flocculant delivery to thickeners.

Steel balls are required for the ball mills and ceramic media is required for copper and magnetite concentrate regrinding mills.

Reagents required at the port and desalination plant include:

- acid for washing the ceramic filter plates;
- hypochlorite for desalination plant intake;
- corrosion inhibitors;
- antiscalants;
- various other reagents as required.

18 PROJECT INFRASTRUCTURE

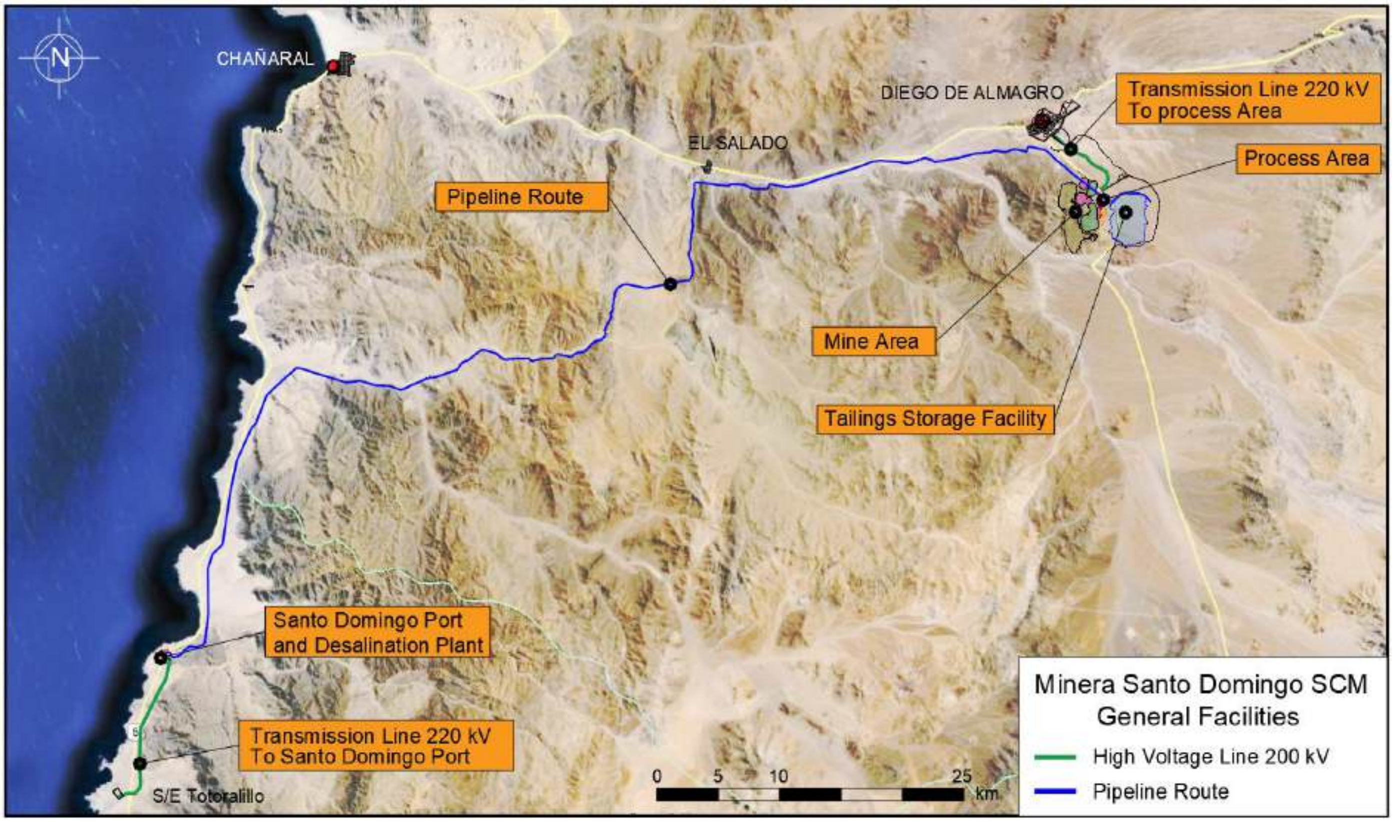
18.1 Introduction

The centre of the Santo Domingo property is located approximately at geographic coordinates 26° 27' 46.69” S latitude and 69° 59' 6.74" W longitude and Universal Transverse Mercator (UTM) 397,815 m E and 7,073,620 m N (datum: WGS 84, Zone 19S). The principal facilities are planned to be located at the following sites:

- Construction and Operations accommodation camp: located on site about 2.5 km north from the mine and plant.
- Tailings storage facility (TSF) designed for storing up to 361 M dry tonnes of thickened tailings: located on site about 2 km southeast from the mine and plant.
- High voltage transmission line, length approximately 9 km, from San Lorenzo substation to the main substation at the concentrator plant.
- Magnetite concentrate and water pipelines: approximately 111 km long between the Santo Domingo plant site location and the Santo Domingo port site at Punta Roca Blanca.
- Port facilities: The proposed port, Puerto Santo Domingo at Punta Roca Blanca, will be located approximately 43 km north of the town of Caldera, in the Atacama Region.
- A desalination plant located in the port area operated by a third party.
- High voltage transmission line: from the Totoralillo substation to the MSD port and desalination plant infrastructure.

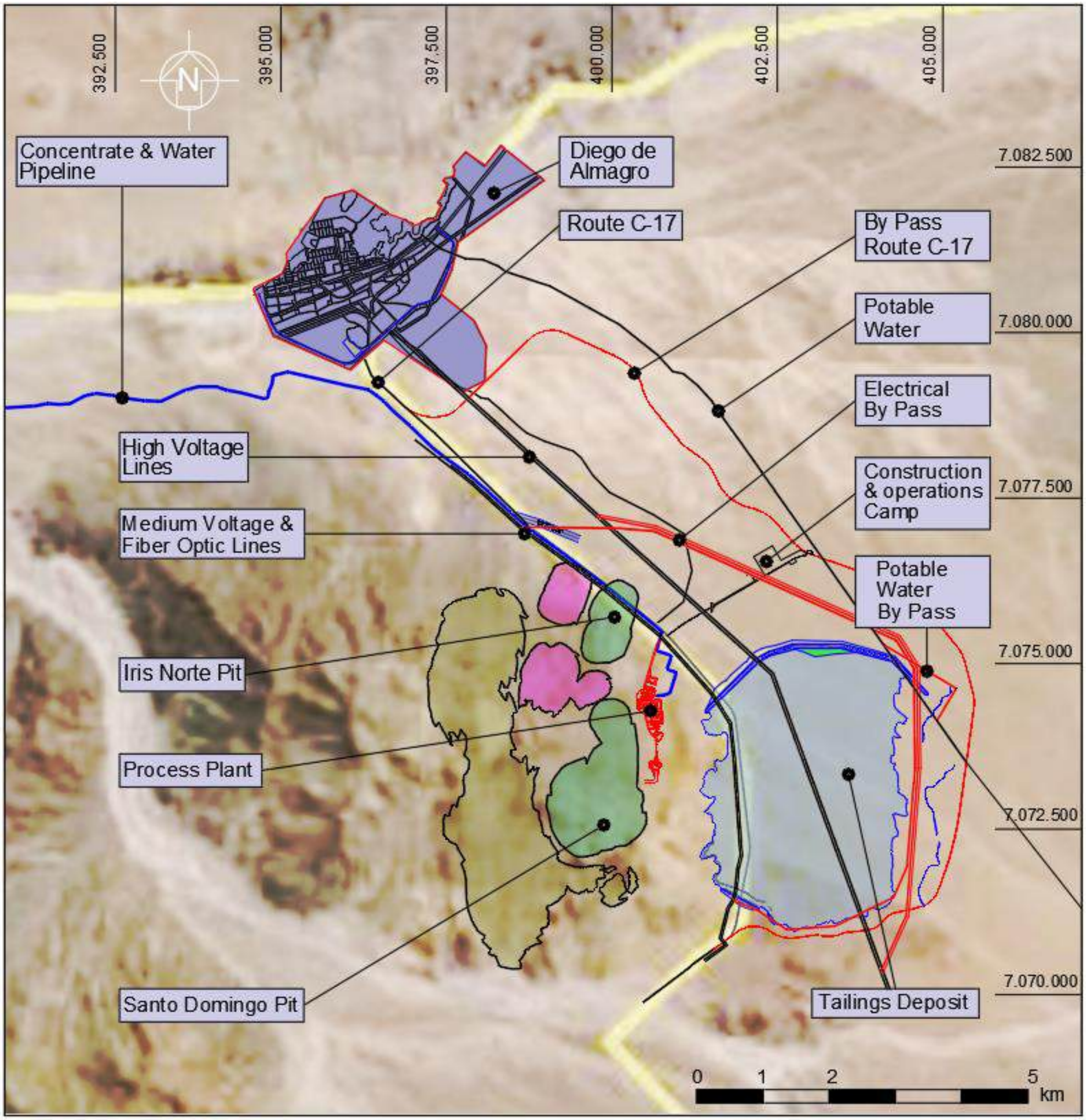
Figure 18-1 shows the proposed locations of principal facilities, including the route for the water and magnetite concentrate pipelines from the mine site to the proposed port location. Figure 18-2 shows the details of the proposed mine site and plant layout.

Figure 18-1: Location Plan Showing Principal Facilities, including Proposed Pipeline Route



Source: Figure prepared by Capstone, 2024. Backdrop is based on Google Earth image. As an indicator of map scale, it is approximately 11 km from the proposed process plant location to the proposed port site.

Figure 18-2: Proposed Mine Site and Plant Layout



Source: Figure prepared by Capstone, 2024. Backdrop is based on Google Earth image.

The following list of relocations are planned where the future locations of project infrastructure will clash with existing infrastructure.

- Relocation of National High Voltage Power Lines (HVL's), a Medium Voltage Power Line (MVL) and Fiber Optic lines, referred to as Electrical By Pass in Figure 18-2, that currently pass through the future locations of mine infrastructure, plant infrastructure and the TSF. The aforementioned existing infrastructure clashes with future project infrastructure for a length of approximately 8.5 km, requiring by-pass infrastructure lengths ranging from 10.5 km to 12.5 km.
- Relocation of approximately 11 km of existing C-17 public road, referred to as By Pass Route C-17 in Figure 18-2, that passes through the future locations of mine infrastructure, plant infrastructure and the TSF. The C-17 by-pass road will be approximately 18.5 km long. The construction of C-17 by-pass road has begun, with approximately 50% progress at the effective date of this report.
- Relocation of approximately 1.6 km of municipal potable water pipeline feeding Diego de Almagro town, referred to as Potable Water By Pass in Figure 18-2, to bypass the east side of the TSF's main dam and diversion channel. The potable water by pass will be approximately 1.6 km long.

18.2 Roads and Logistics

18.2.1 Access

The planned route for transporting cargo, staff and equipment to the Santo Domingo site is from the south of the mine site by Route C-17 (Figure 18-3) and from the north by Route C-13 (Figure 18-4).

The closest commercial airport is the Desierto Atacama Airport, 113 km south from Chañaral, which has regular scheduled flights to Santiago. The closest airport to the Santo Domingo site is the El Salvador Airport, a private airport, 44 km from the site.

The planned port for transport and shipment of heavy machinery, equipment and materials for construction is Punta Angamos in Mejillones, Antofagasta Region, 520 km from the plant site. This port is a year-round operation and is accessed directly from Route 5 North (Figure 18-5).

18.2.2 Onsite Access

The access to Santo Domingo site is located at about 8 km south of Diego de Almagro. The access is via the Main Access Road that will connect the by-pass of C-17 route with the camp area, guardhouse and the rest of facilities like the mine pit, waste dumps, concentrator plant, administrative buildings, TSF among others. At the Santo Domingo site, approximately 13 km of roads will be built in order to connect the plant areas. The roads will be of two types: 7 m wide road for Standard Vehicles or 40 m wide road for Mining Vehicles. These roads will be used for service, operations and/or access of mining trucks to the infrastructure of the Mine area. All roads will have a gravel surface constructed from placed controlled fill.

18.2.3 Copper Concentrate Haulage Route

A concentrate haulage study report was completed during the 2018 DFS stage of the project for the approximately 117 km distance between the planned process plant site and the proposed Santo Domingo port. Based on the findings of the 2018 report, the public roads used for concentrate haulage will be C-17, C-13, C-209, C-225, C-261 & Route 5 North.

The 2018 concentrate haulage study report recommended a by-pass option around the town of El Salado that will allow for future town growth.

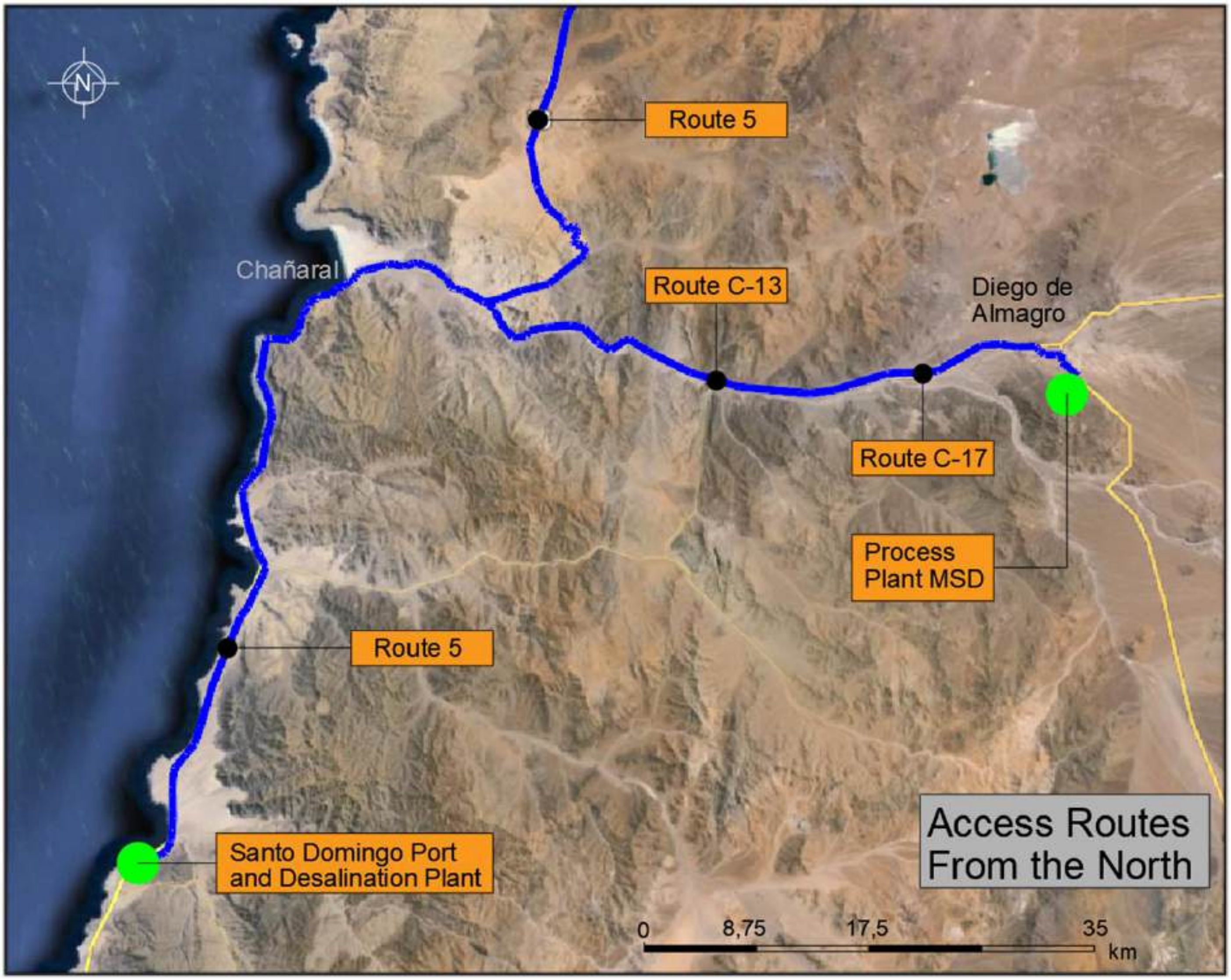
The 2018 concentrate haulage study report also made recommendations for the preparation of contingency plans for spill management and accidents and for training of an incident management team.

On February 15th, 2022, Chilean Law 21.425 was officially published to regulate mineral transportation to avoid environmental impact. Compliance with this new regulation requires using sealed transport containers for copper concentrate transportation from the mine to the port. The project has considered infrastructure for addressing this requirement at the plant site and port, based on the experience of Capstone’s adjacent Mantoverde project. Allowances have been added to the project to account for these infrastructure items including but not limited to storage areas for sealed transport containers, earthworks platforms and mobile equipment.

Figure 18-3: Access Routes from the South

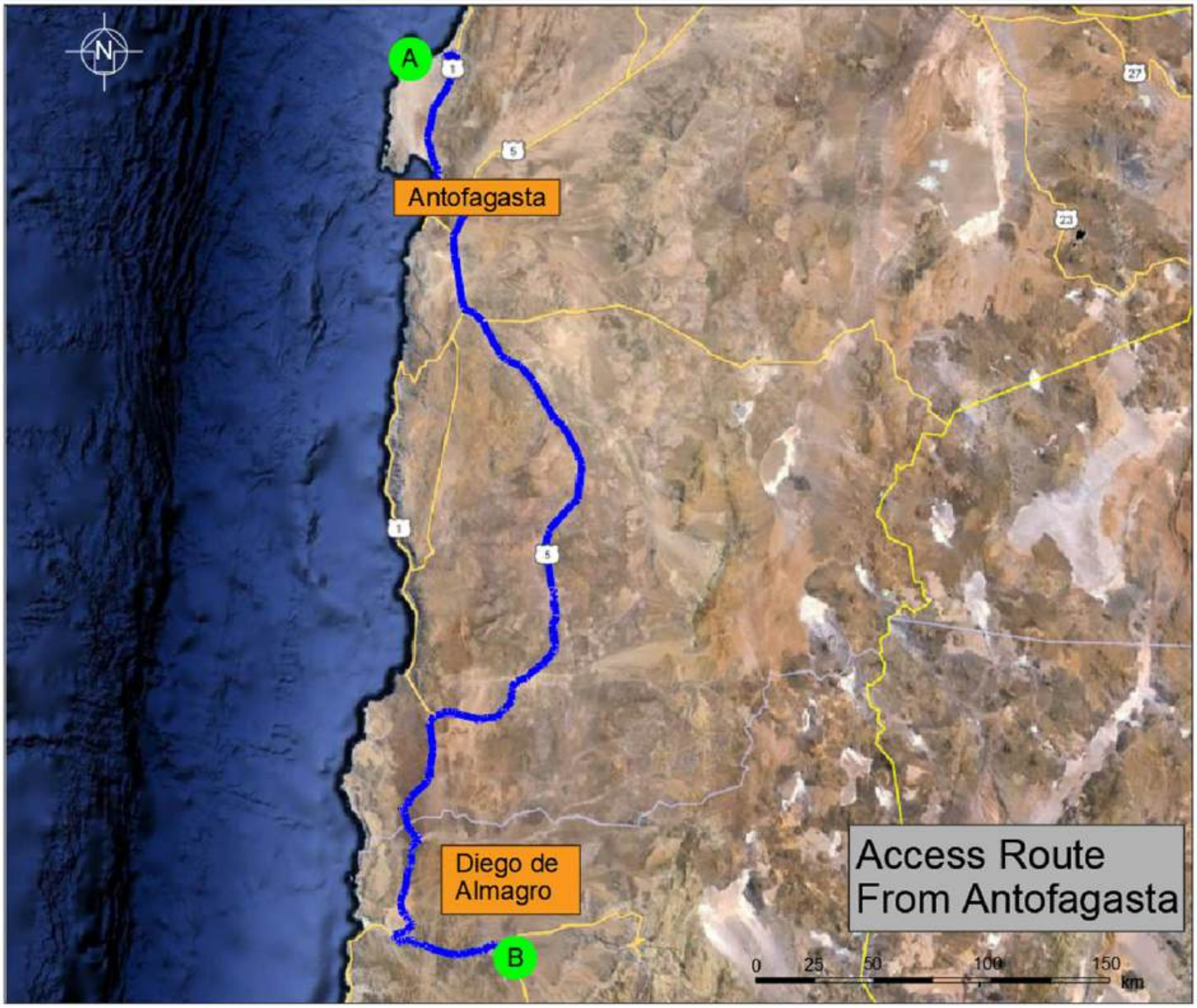
Source: Figure prepared by Wood, 2013. Figure uses Google Earth backdrop, modified by Wood, 2013. As an indicator of map scale, it is approximately 117 km from the proposed process plant location to the proposed port site.

Figure 18-4: Access Routes from the North



Source: Figure prepared by Wood, 2013. Figure uses Google Earth backdrop, modified by Wood, 2013. As an indicator of map scale, it is approximately 117 km from the proposed process plant location to the proposed port site.

Figure 18-5: Access Route between Santo Domingo Mine/Plant Site and Punta Angamos Port, Mejillones



Source: Figure prepared by Capstone 2024. Figure uses Google Earth backdrop. Point A shown on the plan = Mejillones; point B = Santo Domingo site. Map north is to top of plan. The distance from Chañaral to Taltal is approximately 143 km as an indicative scale for the plan.

18.2.4 Pipeline Route Studies

The proposed route for the magnetite concentrate pipeline was optimized using a single 15 m wide Right-of-Way (RoW) and a common trench for the concentrate pipeline and the desalinated water pipeline. The route is designed to run parallel to the existing roads and uses existing RoW access to avoid the construction of new roads. Ongoing access to the pipeline route during operation will be along the pipeline earthworks platform and construction road.

18.3 Waste Rock Storage Facilities

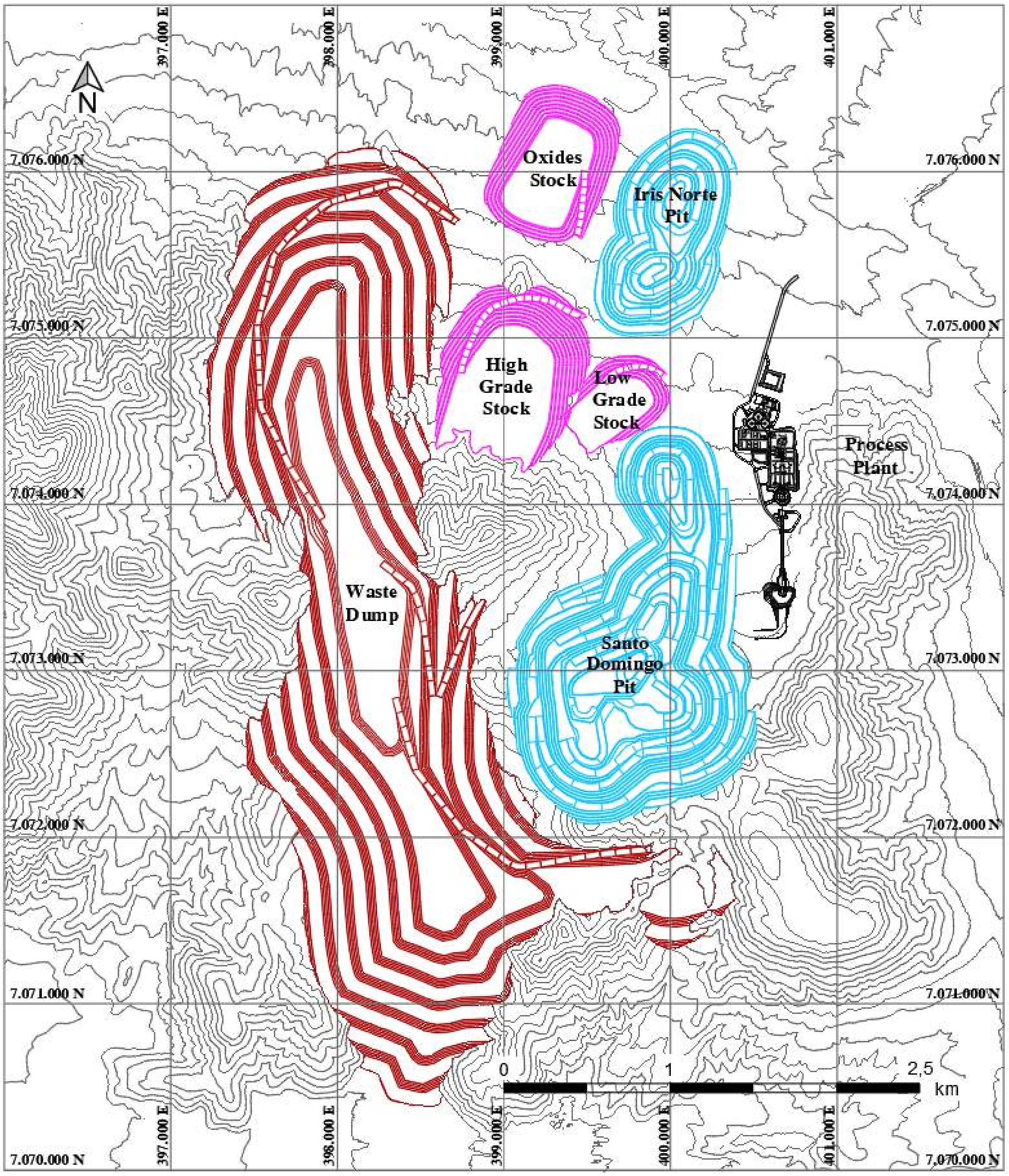
The WRF area to be located to the west and south of the Santo Domingo Pit. The final configuration is shown in Figure 18-6.

Pre-stripping activities will generate approximately 45 Mt of rock. From this amount, approximately 22 Mt will be transported by trucks to the WRFs and approximately 13 Mt will be used in the TSF starter dam construction, with consideration for Non Acid Generating (NAG) or Potential Acid Generation (PAG) material requirements The rest will be deposited in the Sulphide and Oxide stocks.

The facilities were designed in 45 m lifts. Each lift will be constructed at an approximate angle of repose of 37° and bench of 75 m. This represents an overall slope of 23.3 degrees which will facilitate re-sloping during mine closure. The maximum design capacity is 1,265 Mt for a constant 2.0 t/m^3 loose density assumed in the design. In the northern sector the maximum height of the dump is 284 m, while in the south side the maximum height of the dump is 325 m. The closest facility to the foot of the dump is the Santo Domingo Pit, about 150 m away, (Phase 8). Static and pseudo-static stability analyses have been performed to confirm that the WRF designs conform to industry standards for Factor of Safety and Probability of Failure.

No significant acid generation is expected by Capstone from the mined waste, as the dry climate conditions are not expected by Capstone to produce sufficient water to generate drainage through the WRFs that would mobilize any acid solutions.

Figure 18-6: Final Pit, Stockpile and Waste Rock Facility Configuration



Source: Figure prepared by Capstone Copper, 2024.

18.4 Stockpiles

The ore stockpiles (High and Low grade) and oxide stockpile will be located between the Santo Domingo and Iris Norte pits. The stockpiles are designed with 10 m lifts and 10 m setbacks to facilitate later re-handling (Table 18-1).

During the pre-production period, the ore stockpile will be constructed for later re-handling to the primary crusher. The total ore to be stockpiled during this period amounts approximately to 6.3 Mt.

The design capacities and maximum requirements scheduled in the mine plan are shown in Table 18-1. A constant 2.2 t/m^3 loose density was assumed in the design.

Table 18-1: Design Capacities and Maximum Planned Utilization

Stockpile Facility	Design Capacity (Mt)	Maximum Planned Utilization (Mt)
High and Low grade Stock	77	38
Oxides Stock	50	48

18.5 Tailings Storage Facility

18.5.1 Introduction

The TSF will be located approximately 2 km southeast of the process plant and approximately 8.5 km southeast and upgradient from the city of Diego de Almagro (Figure 18-2).

A total of approximately 361 M dry tonnes of tailings are planned to be produced over the life of mine. The tailings will be thickened in three parallel high-density thickeners to a target solids content of 67% but allowance is made in the first two years of operations for reduced levels of efficiency of the thickeners with flexibility to have lower solids contents limits of 58% and 65% in those years, respectively.

The scope of work considers the tailings pumping and delivery pipe system from the thickeners to the TSF, the tailings deposition pipework in the TSF and the surface water management and drainage management facilities in the TSF. The thickeners are being designed by Ausenco. This section presents the current design of the TSF, and the systems recently mentioned, including updated tailings characterization testing and site investigation studies. The design was updated from previous studies in 2013 by Knight Piésold (KP) to account for changes in the mine plan, the updated tailings and site characterization data, and revisions to applicable TSF international standards. The facility design is aligned with Capstone's corporate commitment to adopt the Global Industry Standard on Tailings Management (GISTM, 2020) for all its TSFs.

18.5.2 General Arrangement of the TSF

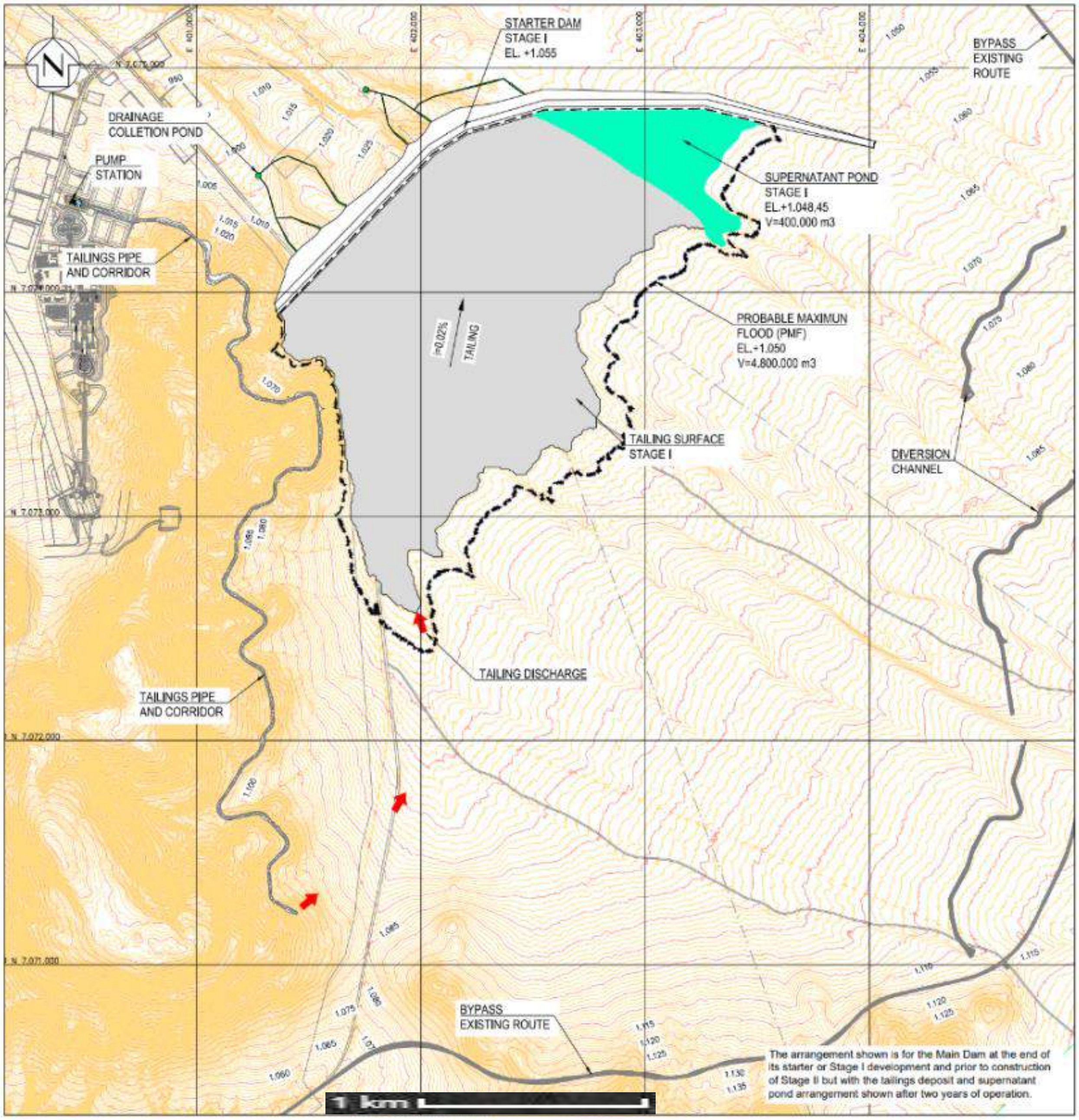
The TSF will be developed on a broad alluvial plain that slopes upward to the south and east from a low point in the northwest. Tailings containment will be provided behind and upslope of a west to east earthfill and rockfill downstream dam that will be constructed across the northwest low point. This is termed the Main Dam. Over time the tailings deposit will rise and expand to the south and east and the dam will be progressively raised and extended to the east to continue providing the containment. Six stages of Main Dam development are planned through the life of mine with Stage I serving as the starter dam. Stage I is planned to provide tailings and water storage containment for the first two years of operation. On the west side of the site, tailings containment will be provided by a high, prominent, bedrock ridge that runs from north to south. The Main Dam will have its west abutment against this ridge.

Two local low points or saddles around the upper perimeter of the site will require construction of saddle dams. Containment in the first area in the southwest side of the site will be by a dam termed the Saddle Dam while containment in the second area immediately to the east will be by a dam termed the Auxiliary Dam. These dams will be developed as earthfill and rockfill dams like the Main Dam but since the saddles are at higher elevations they will be smaller than the Main Dam and will not be developed initially.

Figure 18-7 shows the Stage 1 arrangement of the TSF in plan view. The tailings deposit and supernatant water pond are shown at the end of year two of operations. At this time the Stage II raise to the Main Dam will be completed but it is not shown here to illustrate the starter dam.

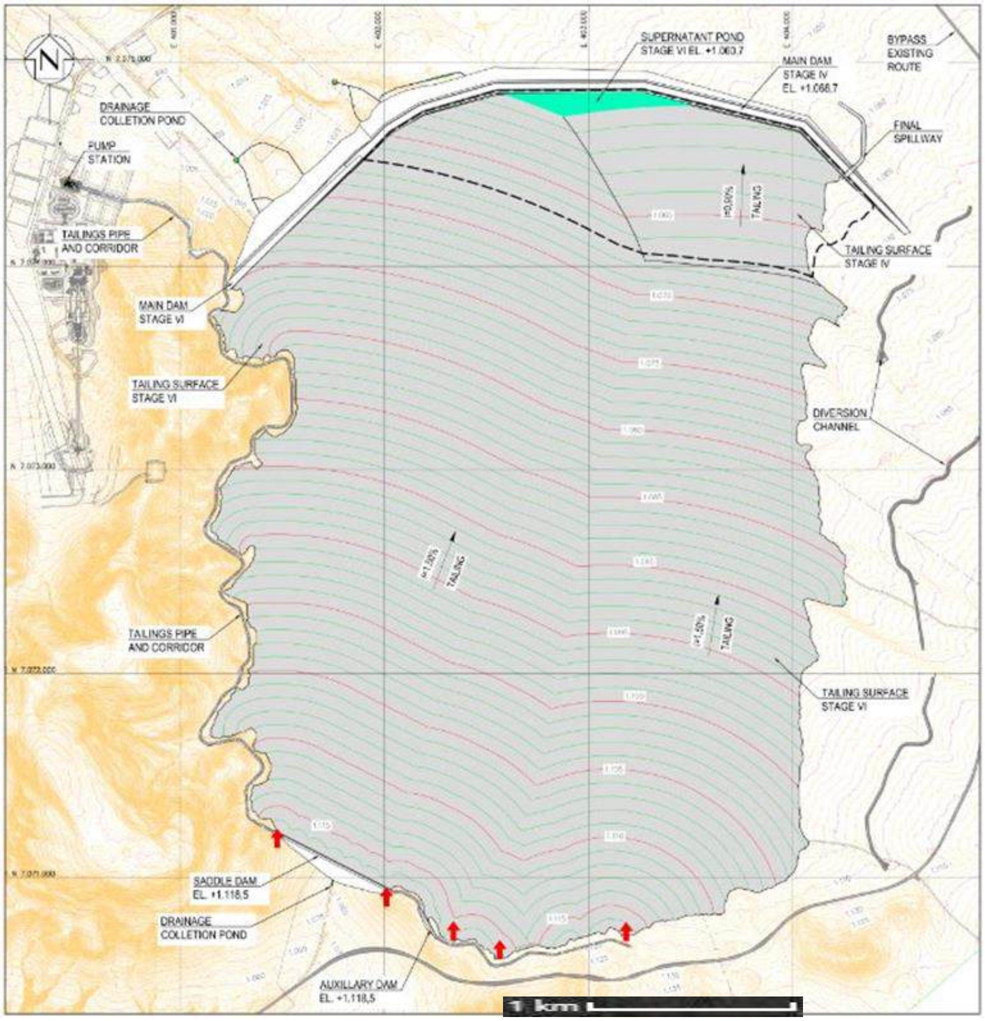
Figure 18-8 shows the fully developed TSF in plan view containing 361 M dry tonnes of tailings. The storage volume is estimated at 245 Mm^3 with an overall average dry density of 1.47 t/m^3. These tailings are planned to be deposited over 20 years.

Figure 18-7: Plan View of Stage I Arrangement of the TSF Showing the Starter Main Dam Together with the Tailings Deposit and Supernatant Water Pond at the End of Year Two of Operations



Source: Figure prepared by Capstone Copper, 2024.

Figure 18-8: Plan View of TSF at Full Development



Source: Figure prepared by Capstone Copper, 2024.

18.5.3 Foundation Conditions and Site Preparation

The foundation under the TSF site consists largely of deep, ancient, alluvial, angular to sub-angular sand and gravel deposits that extend beyond the northern, eastern, and southern limits of the site. The natural slopes on the alluvium together with the Saddle and Auxiliary Dams will provide the needed tailings containment to the south and east while the Main Dam will provide containment to the north.

The alluvium is relatively stiff and with generally low soluble salt (calcium carbonate “caliche” and gypsum) content, but some lenses or zones of loose soils and/or higher salt content exist. Investigations to date indicate that these are typically at shallow depths of up to a few metres, and they will be removed in the foundation preparation work for the dams. For the current design a consistent depth of 2 m of removal under the dams has been assumed except for the rock abutments where 1 m has been assumed. Further investigations are planned for the next stage of design to confirm or modify this understanding and establish a more precise sub-excavation plan.

The natural groundwater level in the alluvium has been measured at between 58 and 72 mbgs, and thus at a significant depth, in two monitoring wells installed to date at the site. Measured hydraulic conductivities in the alluvium gave low to very low values (between $7x10^{-4}$ cm/s and $6x10^{-7}$ cm/s) with a trend of decreasing conductivity with depth. Additional perimeter wells will be installed in a next stage of project planning for further monitoring.

The bedrock ridge on the west side of the site is comprised of limestone and sandstone. Investigations to date have indicated that karst development in the limestone is very minor with only small, non-continuous dissolution features noted largely on the surface of the rock. However, further investigations are planned for the next stage of design to confirm this understanding. Measured hydraulic conductivities in the rock both in the ridge and underlying the alluvium support a low karst assessment with low values in the order of 10^{-5} cm/s in the upper weathered and fractured zone that rapidly decrease to very low values of approximately $1x10^{-8}$ cm/s in the harder rock underneath.

18.5.4 Main Dam

The Main Dam will be constructed out of clean, non-acid generating mine waste rockfill that will be placed, levelled, and compacted in designated and controlled horizontal lifts under a rigorous QA/QC program to form a dense, stiff and stable structure. It will be built in six stages (I to VI) up to an ultimate elevation of 1,078 masl. Its maximum height will be approximately 63 m, which will be above the northwest low point of the TSF. Its ultimate length will be approximately 3,970 m and it will require a total of 14.16 Mm^3 of rockfill and 0.34 Mm^3 of earthfill to construct. The starter dam will have a crest elevation of 1,055 masl that will be constant from west to east as there will be no cross slope on the tailings then, and it will have a maximum height of 45 m and a length of approximately 3,000 m. It will require a total of 5.65 Mm^3 of rockfill and 0.18 Mm^3 of earthfill. The five raises will be built using the downstream method of construction such that no reliance will be made of the tailings for support or stability.

To improve the efficiency of rockfill placement, maintain a reasonably constant rate of placement and match the fill demand with the supply from the mine, the stages have been designed with variable crest and bench widths and bench elevations. In the first five stages, the rockfill will be placed with mine haul trucks via direct haul from the mine to the dam. For the final stage, smaller and lower capacity trucks will be required to suit the final crest width of the dam (20 m).

The rockfill will be placed in designated zones from upstream to downstream with the allowable lift thicknesses and maximum particle sizes progressively increased in each zone towards the downstream. This will create a dense stiff zone with a smoother face on the upstream side, which will be lined, and a coarse graded downstream zone for maximum erosion protection. The intermediate zone(s) will transition from one to the other to allow for gradual changes in gradation and stiffness.

The upstream sloping face of the dam will be inclined at 2.5H:1.0V and will be lined with a 1.5 mm HDPE geomembrane placed directly over a geotextile layer that will be installed on a trimmed and compacted surface of a lower permeability soil layer that will be on the upstream side of the rockfill. The soil and geotextile layers will provide a firm and kind bedding surface for the geomembrane. This layered liner system will be continued laterally and vertically upward over the entire upstream face of the dam to its full build out arrangement.

The downstream sloping face of the dam will be inclined at 2.0H:1.0V, which will apply to the ultimate final slope as well as interim slopes for Stages I to V.

The starter dam (Stage I) will be constructed up to 1,055 masl to provide sufficient capacity for the first two years of TSF operation. This elevation is based on the early tailings deposit forming a negligible beach slope upward to the south and east into the basin. As time passes and the tailings deposit grows, and its surface gradually steepens upward to the south and east, the rate of rise of the tailings deposit will reduce. This will reduce the required rate of rise of the dam. The currently anticipated raise schedule for the crest of the dam based on the design basis tailings slopes, and after including for the operating and extreme event ponds as well as a minimum 2 m of freeboard allowance, is as follows:

- Stage I 1,055.0 masl
- Stage II 1,062.5 masl
- Stage III 1,067.0 masl
- Stage IV 1,071.0 masl west side and 1,068.7 masl east side
- Stage V 1,074.0 masl west side with the east side maintained at 1,068.7 masl
- Stage VI 1,078.0 masl west side with the east side maintained at 1,068.7 masl

In the latter stages, the west side of the dam will become higher than the east side due to a west to east slope on the tailings deposit that will be developed then. The purpose of this will be to position the low point on the final tailings beach surface in the northeast part of the TSF where post-closure surface water drainage will be removed after placement of the closure cover. During operations, this will result in the supernatant pond being progressively displaced from west to east after Stage III against the lined face of the dam.

A drainage collection system will be constructed under the dam during Stage 1 to intercept any small flows that may pass through the liner. These flows will be directed into a lined set of ponds immediately downstream of the dam for re-cycle to the process plant.

18.5.5 Adjacent Lined Basin Area

The basin surface in the area just upstream of the Main Dam will be lined with a 1.5 mm HDPE geomembrane over a graded and compacted lower permeability soil layer, similar to the liner system layers on the dam. Both layers will be continuations of the layers on the dam to form a contiguous high quality seepage barrier that extends into the basin.

The upstream limits of the liner have been set by water balance analyses that were conducted to size the supernatant water pond during the initial period of operation when the pond will be directly on the basin floor. The liner may be extended further in a future design stage to provide coverage for a potentially larger pond if an updated water balance confirms the need for this. Care will be taken during the initial operating period to keep the pond over the liner. However, over a short period of time thereafter, in the order of 6 months, the lower portion of the tailings deposit will begin to form above the liner and expand progressively upslope to the south and east over the unlined basin floor, and these tailings will then separate the basin floor from the pond to limit any seepage beyond the liner. After that, the thickness as well as the southward and eastward limits of the tailings will grow, and any periodic lateral expansions of the pond will have little effect on seepage.

Two excavations will be designed in a follow-up stage of engineering in the lined basin area to serve as alluvium borrow sources for the soil layer in the liner system on the dam and for any soil required to be imported for the basin liner (the majority of this is expected to be from in-situ soil). Once complete, these excavations will be lined and will add some initial capacity for storing water during the start-up of operations and for overall tailings management.

18.5.6 Saddle and Auxiliary Dams

The Saddle Dam will be developed in two stages in Years 5 and 8 using the downstream method while the Auxiliary Dam will be developed in one stage in Year 8. Both will be constructed in a manner similar to the Main Dam out of clean, mine waste rockfill that will be placed and compacted in controlled horizontal lifts to form dense, stiff and stable structures. Neither will rely on the tailings for support or stability. They will be significantly smaller than the Main Dam with final heights of 34.5 and 4.5 m respectively up to a final crest elevation of 1,118.5 masl. The Saddle Dam will have a maximum height of 34.5 m and a length of 647 m, and it will require 0.65 Mm^3 of rockfill and 0.02 Mm^3 of earthfill. The Auxiliary Dam will be built to a height of 4.5 m and a length of approximately 170 m, and it will have a volume of less than 10,000 m^3 of fill. These dams will not be lined because they will be located well away from the supernatant pond on the tailings surface and will have long, drained and air-dried beaches in front of them. The upstream and downstream faces of both dams will be inclined at 2.0H:1.0V.

The Saddle Dam will have a drain system constructed under its footprint to intercept any small flows that may drain from the beach periodically. Similar to the Main Dam, these flows will be directed into a lined pond immediately downstream of the dam.

18.5.7 Hazard Consequence Classifications of the Main and Saddle Dams and the TSF

The hazard consequence classifications of the Main and Saddle Dams are based on the potential consequences of a failure of those dams. Based on the results of two previous dam breach analyses (KP, 2013; HATCH; 2019), an "Extreme" classification has been assigned to the Main Dam and Saddle Dam using the Global Industry Standard on Tailings Management (GISTM, 2020) guidelines. No consequence classification has been assigned to the Auxiliary Dam due to its very low height and negligible downstream impact potential.

Under Chilean regulations, given the capacity and proposed height of the TSF, the facility has been classified as a "Category C" reservoir under the Water Authority (DGA) Decree 50 (Ministerio de Obras Públicas, 2015).

18.5.8 Stability Analyses

The design of the dams has been conducted to meet the following limit equilibrium factors of safety:

- Long-term static conditions - minimum 1.5 (CDA, 2021; ICOLD, 2022)
- Seismic loading conditions using the pseudo-static method - minimum 1.2 (Chile Ministry of Mining, 2007)

- Post-earthquake conditions - considering reduced strengths in materials that may liquefy or undergo strain-softening behavior - minimum 1.2 (CDA, 2007 and 2019).

These required minimum factors of safety are the highest of the applicable regulations or guidelines for the stated loading conditions. The specific regulations or guidelines that provided these factors of safety are in brackets.

The analyses completed for this design gave factors of safety at least equal to these minimum requirements and thus dam stability can be considered acceptable in accordance with the applicable standards.

18.5.9 Tailings Characteristics

The tailings physical and geotechnical characteristics based on recent testing, can be summarized as:

- Particle size distribution - approximately 30% sand and 70% fines including silt and clay size particles
- Liquid Limit (LL) 21%, Plastic Limit (PL) between 13% and 14%, Plasticity Index (PI) 7% to 8% (the PI is moderately high for tailings and suggests that some clay like minerals appear to be included)
- Unified Soil Classification System (USCS) classification - CL-ML
- Specific gravity - ranging from 3.02 to 3.18 – (these values are relatively high due to iron content).

Bench scale settling, draining, air drying and consolidation testing to estimate the range of possible dry densities in the deposit gave the following results:

- Settled undrained dry densities - range of values between 1.39±0.16 t/m^3 for an initial slurry solids content of 58% and 1.47±0.15 t/m^3 for an initial slurry solids content of 67%
- Settled drained dry densities - range of values between 1.56±0.12 t/m^3 and 1.65±0.10 t/m^3 for the same two initial solids contents
- Air drying maximum dry density - 1.83±0.12 t/m^3 (representative of very large stresses induced by air drying - there was little difference in the results between the slurries at the two initial solids contents)
- Oedometer consolidation dry density – average of 1.53 t/m^3 (at the stress level associated with the maximum height of the tailings - there was little difference in the results between the slurries at the two initial solids contents).

Based on the above, the following in-situ dry densities have been used for the design of the TSF at this time:

- Years 0 – 3: 1.30 t/m^3
- Years 4 – 20: 1.53 t/m^3 .

The beach slope profiles used for the design have been interpreted based on 2023 tailings slurry rheology testing and beach slope prediction modelling conducted by Tailpro with the technical assistance of Water, Waste, and Land. The slope predictions are very important because after the first few years of operation a substantial portion of the tailings in storage will be in the tailings deposit that extends upslope to the south and east in the basin. For design purposes, the following slopes have been used at this time:

- Year 0 – 2: 0.0% - 0.1%
- Year 3 – 5: 0.3% - 0.5%
- Year 5 – 10: 0.5% - 1.0%
- Year 10 – 20: 1.0% - 1.5%.

In selecting the above values, care has been taken to avoid overestimating the slopes and therefore risk overestimating the storage capacity. However, the use of slopes that are too low would result in the need for a substantially larger Main Dam. A best estimate balance between these two factors has been attempted for this design, but during the operation regular beach slope surveys will be conducted to evaluate opportunities for optimizing the tailings deposit and the staged development of the Main Dam and the other dams.

18.5.10 Tailings Delivery to the TSF

The tailings pipe will be delivered in an HDPE-lined steel pipe from the mill process plant that will reach the TSF from the north, from a route downstream of the west abutment of the Main Dam. The pipe will be extended up over the west side of the dam on a ramp and then continue further upslope from there it will be extended as an HDPE pipe further upstream to the south along the west side of the TSF on an access road and pipe service bench. At select points, a tee and lateral connection will be made to extend the pipe to the east on pre-determined alignments above the progressively expanding growing south side of the deposit. This segment will contain a series of valve-controlled spigot offtake points that will deliver the tailings slurry into the basin. As time passes and the deposit rises the pipe and bench on the west side will be raised and as the deposit expands progressively further to the south the spigot distribution line will be repositioned further upslope to the south. The line will also be extended further east with additional spigot points added. Ultimately, the spigot distribution line will be raised to a point just above and beyond the final southern limit of the TSF and extended just beyond to the eastern side to place the last tailings.

During the initial period of operations, the tailings will be discharged into two shallow northwest trending sub-valleys in the base topography using only a few dedicated spigot offtakes. These spigot offtake lines will extend far enough downslope to the north in the sub-valleys to deliver the first tailings onto the lined portion of the basin without tailings flowing down the slope and eroding the basin and possibly undermining the liner anchor trench. The ends of these lines will be equipped with energy dissipating measures to prevent the slurry from damaging the liner. The deposit will rapidly rise out of these sub-valleys and expand progressively upslope, and to allow for the on-going slurry discharge the lines will be provided with outlet points at progressively higher levels. As the deposit rises it will bury the liner anchor trench and remove the potential for any future erosion or liner undermining concerns.

With additional time and further growth of the deposit upward to the south and east, there will be a reduction in the rate of rise and an increase in the extent of air drying of the tailings surface. Both factors will serve to stiffen, strengthen, and increase the density of the tailings deposit, which will steepen the beach slope and increase the storage efficiency. Key measures to better realize these opportunities will involve positioning and regularly alternating the active spigot offtake points to maximize the overall deposit area, reduce the comingling of individual streams on the beach, and form thin, uniform layers. This is the sub-aerial method of deposition.

18.5.11 Water Management

Water management in the TSF will involve the temporary containment of all surface water in the facility including supernatant water draining from the tailings and net runoff from precipitation falling on the facility. This will be re-cycled to the process plant. A runoff diversion channel will be constructed just beyond the eastern side of the TSF and extended upslope beyond the southern limits of the TSF to divert any upgradient runoff to a point downstream of the Main Dam. The site has a very arid climate, and the month-to-month water balance will be in a significant deficit condition such that there will be no routine discharges. The water contained in the facility will be temporarily stored on top of the tailings against the Main Dam, and the dam will be sized to safely store it. The design for this includes the volumes and depths for:

- The predicted maximum monthly normal supernatant pond draining from the tailings, plus,
- The design basis extreme event water pond, which consists of the 72-hour duration PMF on top of the tailings and upstream catchment (assuming the diversion channel is inoperative), which is calculated to be 3.8 Mm^3, plus,
- A minimum additional freeboard of 2 m for Stages I to VI of the Main Dam.

The provision of storage capacity for the PMF in the TSF per bullet 2 above will apply to Stages I through III but in Stages IV through VI an emergency spillway will be added to the TSF near the east end of the Main Dam. This will reduce the extreme event storage requirement.

18.5.12 Spillway

The spillway to be installed for Stages IV through VI will be an emergency high level overflow chute and channel system. It will be designed to pass the peak flow from a 24-hour PMF falling on the TSF and upstream catchment area assuming the diversion channel is inoperative. This amounts to a flow rate of 20.4 m^3/sec. After Stage VI it will be converted to a closure and post-closure spillway.

The base width of the spillway will be 12 m wide while the side slopes will be 2.0H:1.0V and for Stage IV the inlet invert elevation will be 1,065.7 masl. The calculated depth for passing the peak flow for the 24-hour PMF for this arrangement is 1 m, and with the Stage IV dam crest at 1,068.7 masl the 2 m freeboard requirement will be maintained. Similar configurations will be adopted for the Stages V and VI spillways with the inlet invert levels set 3 m below the dam crest elevations.

The base and slopes of the spillway channels are planned to be lined with grouted riprap at least over their upstream lengths near or adjacent to the dam for protection against erosion.

18.5.13 Site Climate Characteristics

The climate at the TSF site can be characterized as arid with high evaporation and low seasonal precipitation that occurs usually in short duration intense storms. Mean annual precipitation is 29 mm and evaporation are 2,175 mm.

Temperatures and wind speed vary significantly throughout the day from warm and windy in the afternoons to cold and still at night.

The design PMF precipitation depth values used for the TSF were calculated using a Log-Pearson type statistical distribution based on precipitation data from regional stations at Las Vegas, Pastos Grandes, Potrerillos, El Salvador, Inca de Oro and Llanta. The results are:

- 24-hour PMF depth - 94 mm (used for calculating the peak flow to pass through the spillway for Stages IV to VI of the Main Dam
- 72-hour PMF depth - 136 mm (used for calculating the extreme event storage volume requirement for Stages I to III of the Main Dam)

18.5.14 Site Seismicity

As noted in Section 5.0, the seismic hazard of the site can be characterized as high and within Zone 3 and Zone 2 according to the Chilean standard NCh 2369. This classification is due largely to its close proximity to the tectonic subduction zone off the west coast of Chile. The seismogenic sources for the area consider three types of earthquakes:

- thrust-type subductive interplate earthquakes,
- intraplate earthquakes of intermediate depth, and
- superficial or crustal intraplate earthquakes.

Large historic earthquakes of types 1 and 2 have occurred relatively close to the site and they have had a significant influence on the seismic hazard assessment and ground motions used for the design.

In accordance with the “Extreme” hazard consequence classification of the Main and Saddle Dams (GISTM, 2020) and the “Category C” classification of the TSF (Chile Water Authority (DGA)), the design of the TSF and dams has been based on the Maximum Credible Earthquake (MCE) that has been determined from the seismic hazard assessment for earthquakes types 1 and 2, and the key parameters of those events are given in the Table 18-2.

Table 18-2: Seismic Hazard Assessment – PGA (SyS, 2015)

Seismic Source	Magnitude	Soil Type	Peak Ground Acceleration, PGA (g)
Interplate	8.8	Hard Rock	0.36
Intraplate	8.0	Hard Soil	1.02
Intraplate	8.0	Hard Rock	0.51

An update of the seismic hazard assessment is recommended for the next stage of design as a good practice step as the current one is over 8 years old. While this will likely change some of the ground motions from the design earthquakes, the TSF dams are not expected to be significantly affected because they will be constructed out of compacted dense rockfill using the downstream method of raising to produce inherently stable structures. The foundations will also be dense and stable and seepage from the tailings will be minimized by the liner and the long-drained tailings beaches elsewhere.

18.5.15 Instrumentation and Monitoring

A rigorous and extensive instrumentation and monitoring program will be implemented for the TSF. The key objectives will be to monitor the following critical control elements:

- Deformation of the dams under static and earthquake conditions
- Pore pressures in and under the dams (negligible pore pressures are expected but monitoring will be used to confirm this or to detect any that do develop for subsequent analysis)
- Tailings beach slopes, densities, deposit strengths, and the rate of rise and expansion of the deposit.

Geotechnical instrumentation will be installed in the TSF to monitor for the first two critical controls. For deformation, the following instrumentation items will be installed:

- Survey monuments – for surface deformation – will be installed on the crests of the dams after each stage.
- Clino-extensometres – for internal deformation – will be installed on the dam crests and anchored 5 m below the foundation in select locations and stages.
- Accelerometres – for seismic ground motions - will be installed with the first on the rock on the west abutment of the Main Dam, the second on the alluvial soil just downstream of the Main Dam and the third on the crest of each stage of the Main Dam (this one will be relocated with each raise).
- Horizontal inclinometres – for measuring potential differential settlements along or across the Main Dam – will be installed near the west abutment of the Main Dam where a significant change in the stiffness of the foundation occurs between the west side rock ridge and the deep alluvium that could lead to some localized differential settlements in the dam.

For pore pressure measurements, five vertical cross sections will be established through the Main Dam, and one will be established through the Saddle Dam that will contain multiple vibrating wire and Casagrande piezometers in the dam fill zones, drains, foundation materials and the tailings immediately upstream of the dams to create a robust network. No piezometers are currently planned for the Auxiliary Dam because it will be small will have a very low potential for having pore pressures develop in or under it due to the long-drained beaches that will be maintained in front of it.

Regular tailings surface surveys will be conducted to monitor the beach slopes, rate of rise and expansion and assist in calculating the density of storage. Periodic geotechnical investigations will also be conducted using sampling with lab testing and in-situ methods (SCPTu, etc.) to verify the density values and assess the deposit strength.

In the event that any drainage water is detected in the monitoring wells or the investigation points, additional pumping wells may be installed to intercept the source. The water recovered will be either treated for release or returned to the TSF for re-cycling in the process. After Year 10 any nominal flows from the TSF are expected to be intercepted by the Iris Norte well system that will be installed to dewater the pit north of the TSF.

18.5.16 Closure

Closure of the TSF will include removal of any residual post-operational supernatant water, covering the surface of the tailings with non-acid generating (NAG) granular material and modifying the Stage VI emergency spillway in the northeast side of the TSF to serve as the long-term post-closure drainage outlet. The spillway will have sufficient capacity to convey the probable maximum precipitation (PMP) event throughout the post-closure phase of the facility.

18.5.17 International Standards

The design of the TSF is being conducted to conform with the new Global Industry Standard on Tailings Management (GISTM, 2020). In August 2023, a presentation was made to the Independent Tailings Review Board (ITRB) on the design developed up to that point and the ITRB made some recommendations to further support GISTM conformance. Some of these have been incorporated and the remainder, which are associated with on-going investigations and studies as well as future operational plans, will be incorporated when appropriate. The ITRB's recommendations are clearly identified and registered in Capstone Copper's internal information management systems, where frequent monitoring and tracking of progress is carried out, and conformance level dashboards are generated.

18.6 Water Management

18.6.1 Hydrology

18.6.1.1 Pre-2018 Hydrology update

A feasibility level analysis of meteorological and hydrological data was carried out by KP based on regional stations and historical data and site studies for water resources carried out by third parties on the proposed Santo Domingo mine site area since 2009. The purpose of this analysis was to characterize the water resources in the area and to support the development of the environmental impact assessment of the Santo Domingo property. Specific outputs from this work included a general description of the site climate, development of regional and site-specific meteorological and precipitation statistics, and hydrologic run-off statistics.

18.6.1.2 2018 Hydrology update

In 2018 Hatch Engineers and Consultants conducted an updated hydrological assessment focused on addressing queries from the General Water Directorate (DGA) regarding the environmental sectoral permit for the Tailings Storage Facility. The report provides a refined analysis of significant hydrological events that occurred in 2015 and 2017 and employs modelling techniques to estimate Probable Maximum Precipitation (PMP) and Probable Maximum Flood (PMF) for various return periods. Key enhancements include the use of updated precipitation data, a comprehensive evaluation of the basin's geomorphological features, and the application of modern statistical methods for flood flow and volume estimation.

18.6.2 Surface Water Management

In addition to the specific TSF surface water management described in Section 18.5, the project considers natural water diversion works that will allow the diversion and return of rainwater downstream from the project facilities. These works would include:

- Surface water diversion channel to the North of the Iris Norte pit
- Surface water diversion channels in the waste landfill.

Any contact water from the process plant area will be incorporated into the process water circuit.

18.6.3 Construction Water

The industrial water requirement during the construction phase will be provided by an authorized water supplier. For industrial water at the mine and plant site, a 152 mm diameter water delivery pipeline has already been constructed to the process plant site from the Water Treatment Plant near Diego de Almagro, to supply industrial water for road dust control, bulk earthworks for moisture content adjustment at the plant site and TSF, and for dust control at the mine during the pre-stripping activities. For construction at the port, potable water will be used, provided by an authorised water supplier through a tie-in to an existing potable water pipeline running beside the port area.

The potable water requirements during the construction phase at the mine and plant sites will be provided by an authorized water supplier. A tie-in to the existing municipal potable water pipeline that feeds Diego de Almagro will provide a potable water supply point near the planned construction camp. This water will be used at the camp and at the concrete batch plant.

18.6.4 Process Water

The mineral processing facility will use desalinated sea water and recycled water. The current plan is to produce the desalinated water under a build, own, operate, transfer (BOOT) or build, own, operate (BOO) contract with a third party. The desalinated water supply capacity is 366 [L/s] as authorized in RCA No. 122-2020 "Change of Desalination Plant".

Two high-density HDPE-lined open ponds for process water will be located near the process plant and will have a total capacity of 80,000 m^3. These ponds will store water reclaimed from the copper and magnetite concentrate thickeners and tailings thickener overflows and make-up from the plant desalinated water storage ponds. Recycled water from the magnetite concentrate filtration at the port will also be pumped to site after treatment to improve its quality.

18.6.5 Potable Water

During the operational phase of the project Capstone will operate potable water treatment plants that treat the desalinated water so it can be used for human consumption in the mine and port areas, as well as supplementing water resources in the town of Diego de Almagro.

18.7 Magnetite Concentrate Pipeline

The magnetite concentrate pipeline, and the desalinated water pipeline will run parallel and will be buried in a common trench for the majority of the pipeline route. At the port and plant locations, each line will be routed separately to their respective facilities.

18.7.1 Magnetite Concentrate Pipeline Operating Philosophy

The magnetite concentrator plant produces two qualities of magnetite concentrate, which are stored in tanks located in the plant. The concentrate transport system is designed to alternately transport these two types of concentrate to the concentrate filtering plant in the port, with an intermediate water drive to effectively isolate the differentiated concentrations.

18.7.2 Magnetite Concentrate Pipeline Route

The main pumping station, located in the process plant area, consists of a system of three positive displacement pumps (2 operating, 1 standby).

The transportation of the concentrate will be carried out through a steel pipeline 111.6 km long. The system considers:

- A station for drainage of the line, located at the lowest point of the route profile (at km 36.10, 418 masl)
- A PMS monitoring station, located at the highest point of the pipeline layout (km 51.35), to monitor the internal pressure of the pipeline and ensure optimal flow conditions in the concentrate pipeline (e.g. avoid slack flow)
- An intermediate choke station located at km 65, 448 masl
- A pressure relief station to allow smooth discharge of the slurry to the port slurry storage tank. Drainage choke and terminal stations will have emergency ponds.

The electrical power supply for the magnetite concentrate pump station will be provided from the electrical system at the plant site. The electrical power supply for the terminal station at the port will be provided by the port electrical system. Process control for the concentrate transportation system will be by a dedicated, independent control system, connected to the main process control system (PCS).

18.8 Pyrite Concentrate Pipeline

If the future pyrite recovery opportunity proceeds as described in Section 24.4.1 of this technical report, the preliminary pipeline design route is from Santo Domingo to El Salado, then on to Mantoverde operations for further concentrate processing.

18.9 Building Infrastructure

The roofed buildings at the mine - plant site and port are summarized in Table 18-3.

Table 18-3: Site Buildings at Mine–Plant and Port

Area	Building	Description
Mine-Plant site	Truck shop	Workshop for mine trucks, including warehouse, light vehicle workshop, offices, dining room & change room.
	Mine truck operator's lunchroom	Lunchroom
	Primary crusher dump pocket dust enclosure	Roof & walls to contain dust from dump trucks discharging into primary crusher
	Primary Crusher control room	Control room for operating Primary Crusher rock breaker
	Coarse ore stockpile	Domed structure covering coarse ore stockpile
	Copper concentrate filter building	Roofed & walled building containing copper concentrate filters
	Electrical rooms	Area electrical rooms for electrical power supply
	Mine-plant administration area	Offices, process plant control room and dispatch office, change house and training building, dining room for process plant and mine, gatehouse
	Assay laboratory	Offices, sample preparation area, wet laboratory and service facilities and an open sided, roofed area for sample storage
	Maintenance workshop	Mechanical maintenance, welding, warehouse and offices, lubricants shop, change house
	First aid and emergency	Offices, training room, examination rooms, a bathroom and a roofed area for the ambulance
	Accommodation camp	Accommodationn buildings, dining room, recreation building, camp gatehouse. Initially for construction workforce, then downsized to permanent operations camp.
Port	Port office	Police, customs, biosecurity & marine services
	Port administration area	Port dining room, offices, control room, laboratory, change rooms
	Maintenance workshop	Mechanical, electrical and instrumentation maintenance workshop & offices
	Access control	Gatehouse
	Copper concentrate storage	Negative pressure building internally divided into 2 storage areas
	Magnetite filter building	Magnetite filtering/dewatering
	Electrical rooms	Area electrical rooms for electrical power supply
	Desalination plant building	Desalination plant

Capstone will provide building and maintenance areas to be used by service contractors for plant and mobile equipment maintenance. The areas will include provision for mine trucks and equipment, light vehicles, a mine truck wash bay and tyre shop, a welding shop, spare parts storage area, offices, maintenance dining room and a change house. The heavy vehicle workshop building will have service bays for mobile mine equipment, light trucks, mine trucks and tracked vehicles for maintenance and routine servicing.

Mine equipment maintenance will be done under a MARC contract for the first 5 years; after that Capstone will carry out its own maintenance.

The warehouse area will consist of buildings and open areas for general, lubricant, reagent, and gas bottle storage.

Accommodation for construction and operations personnel will be in one camp at the mine site using temporary units to increase the capacity during construction. These units will be removed when construction is complete. The planned location of the camp is 2.5 km from the mine and process area.

During construction the camp will have capacity for up to 3,100 beds (including 307 beds for operations staff). The camp will have a dedicated and modular sewage treatment plant allowing for staged construction and later decommissioning. The proposed permanent camp will accommodate approximately 500 people. There is no plan to retain the construction camp buildings once operations start. For ongoing construction and maintenance activities, it is planned to accommodate personnel in off-site accommodations in Diego de Almagro or other nearby locations.

Ancillary infrastructure will include fire protection, compressed air systems, dust control and solid waste management facilities.

18.10 Port

The proposed port, Puerto Santo Domingo, will be located in the Punta Roca Blanca area, in the Atacama Region.

Based upon current Capstone concentrate production in the district, including production from Capstone's adjacent Mantoverde project, the maximum required annual port capacity is 5.4 Mt/y of magnetite concentrate and 0.72Mt/ y of copper concentrate. It is planned to ship magnetite concentrate using a mixture of Panamax- and Cape-size vessels. Copper concentrate will be shipped using Panamax- and Handymax-size vessels.

A total occupancy (berth commitment time) of 68% has been estimated for handling both Santo Domingo and Mantoverde productions. This appears slightly conservative when compared to the 75% and 85% figures given as berth commitment thresholds for similar port conditions in report 184 published in 2019 by the Permanent International Association of Navigation Congresses (PIANC). In the next design stage of the project, an operational simulation study will be carried out to refine this total occupancy estimate (berth commitment time).

Capstone has developed an updated berth alignment analysis to estimate port availability for ship-loading operations and its impact on the upstream operation. Based on this assessment, dock design, concentrate loading and storage arrangements were updated according to new requirements.

The port's storage capacity for copper concentrate is 50 kt, with an additional 45 to 50 kt of storage capacity available in the sealed transport container handling area. The storage building will operate with an internal pressure below atmospheric pressure, to control copper concentrate dust leakage. The use of a negatively pressurised storage building, the change to sealed transport containers and the development of infrastructure associated with them is to ensure compliance with Chilean regulations for the transportation and loading of concentrated minerals (Law n° 21,425 August 2022).

The port's storage capacity for iron concentrate will be 500 kt, in a stockpile that will be constructed in a single stage. Constructing the iron concentrate handling area in a single stage (500 kt) will allow the parallel management of two stockpiles for concentrate qualities, one for "bulk concentrate" and another for "premium concentrate".

The soil mechanics of the land area of the port shows 3 strata; the first (surface stratum) is made up of gravel and sand, the second stratum is made up mainly of sand and contains shells, and the third is composed of strongly fragmented igneous type rock. For land works, structural foundations will be placed with their underside in contact with the rock stratum, as the soil mechanics study recommends founding footings and piles in this stratum. In the case of superficial foundations, it is recommended to carry out structural backfilling until reaching the rock stratum; in the case of deep foundations, driving the piles using an Oslo point is considered.

The geophysics of the maritime area shows a base of meta-sedimentary rock that gradually subsides into an 8 to 22m thick sandy-gravel mantle on the seabed. Given this condition, it is estimated that a portion of the piles will be driven into rock. To ensure these piles driven into rock maintain their correct location during installation, the rock surface will be prepared with a local excavation to guide the pile's entry into the rock stratum.

In 2022, a new maritime campaign was carried out that carried out the drilling of core samples. The information obtained allowed the identification of the configuration of the seabed and its stratigraphy. With the background provided by this drilling campaign, it was possible to specify the typology of the anchors for the piles. This was very useful when updating the pile design in accordance with the Seismic and Tsunami Regulations, since it was determined that some piles must be anchored in the rock to resist tensile loads, and each pile loading case could be evaluated.

Capstone has applied new regulatory requirements to the port design in relation to tsunami and seismic design based on recently updated current codes. The seismic standard (Nch 2369/2023), in addition to a review of the load combinations, proposes software modelling applied in three dimensions simultaneously and the incorporation of the main equipment to the model, to analyze its interaction with the structure and mainly with anchors to the seabed.

The review of port infrastructure behavior under Tsunami conditions was carried out considering personnel safety and operational continuity, in a dynamic model that simulates the three waves of flooding and the point of greatest stress on the structures.

These design verifications were carried out comply with recent seismic regulatory requirements, obtain design certification and thereby ensure current port design is appropriate from a seismic and tsunami perspective.

18.11 Electrical Power Supply

High voltage electrical power will be supplied to the mine-plant site and port site via connection to the Chilean national grid (Sistema Eléctrico Nacional or SEN). The Chilean national grid can provide a reliable and continuous supply for the project's electricity needs.

Capstone has entered into a long-term power purchase agreement (PPA) with one of the major power companies that operate on the national grid and supply several of Chile's major mining companies. This is consistent with the strategy used by other mining companies to contract for long-term electricity supply in Chile. The terms and conditions of the PPA are considered customary and competitive in the Chilean electricity market.

18.11.1 Mine-Plant Site Electrical Power

The connection point to feed the Mine and Plant is the San Lorenzo substation, about 9 km from the Plant, located in the municipality of Diego de Almagro. The connection point has already been constructed and commissioned as a BOO contract with the local power transmission company ENLASA. Power transmission will be done through the construction of a new 220 kV power line from the connection point at San Lorenzo substation to the main substation in the mine- plant area.

Estimated approximate electrical power demands at the mine-plant site during operations:

- 123,394 MW total maximum (peak) demand with a power factor of 0.995
- 108,032 MW, average demand with a power factor of 0.998
- 946,361 MWh annually.

The electric power inside the mine-plant will be distributed in 23 kV medium voltage lines from the main substation to the different unit substations in the areas. These unit substations will lower the voltage to 4.16 kV, or 0.4 kV as required for connection to the electrical loads.

A mine loop has been designed to supply electrical power to the mine shovels and general services within the Santo Domingo and Iris Norte pits. The mine loop will be a 23 KV medium voltage overhead line located around the perimeter of the pits in a ring distribution system, feeding mobile substations located near the mine shovels.

18.11.2 Electrical Power at The Port and Desalination Plant:

The closest connection point to MSD's Port and Desalination Plant is the Totoralillo substation, about 14 km from the port area. The feasibility for connection has been confirmed and feasibility level engineering was developed in 2014 by Amec for this solution. Power transmission will be done through the construction of a new 220 kV power line from the connection point at Totoralillo substation to the port area, feeding the desalination plant through a line tap-off.

19 MARKET STUDIES AND CONTRACTS

19.1 Market Capabilities

Capstone currently markets copper concentrate from its four mining operations and has established a reputation as a reliable supplier of high-quality concentrates and copper cathodes. Capstone has existing commercial sales contracts covering all production of varying quantities and tenors. Current production is delivered to counter parties in Mexico and Chile, as well as exported to Asian and European destinations. During 2023, Capstone generated gross revenue of $1,422.4 million primarily from the sale of 110,179 dmt concentrate and 50,015 t of cathode, with approximately 230,000 dmt per year to be added from its 70%-owned Mantoverde Mine starting in 2025 as a result of a full year of concentrator operation. Santo Domingo's production is expected to be around an average of 410,000 dmt during the first 5 full years of operation.

Capstone's studies of iron concentrate (magnetite) marketing show most of Santo Domingo's production will align with market conditions to support pellet feed demand projections, with a minor portion of it to be marketed as bulk iron concentrate for sinter plants when average grade is below 63%, with the corresponding premium reduction.

Capstone's commitment to excellence in mining, marketing, and overall business operations has consistently propelled us to the forefront of the industry, ensuring sustained growth and success for all involved.

19.2 Copper Concentrate Market

Since 2000, China has played a leading role in the global copper market, both as a significant importer of raw material in the form of copper concentrate and as the world's leading consumer of refined copper. The volume of copper concentrate imported by China has surged from virtually nothing in 2000 to 27.5 Mt in 2023, and it now accounts for more than half of the global refined copper consumption.

In 2023, global refined copper production reached approximately 25.8 Mt, with a steady average growth rate of 1.9% per annum projected until 2033. The peak of this growth is expected between 2024 and 2025, coinciding with substantial capacity expansions in Asia.

China's smelting and refining capacity is projected to expand by 3.3 Mt to 13.8 Mt by 2025, and further increase to 14.8 Mt by 2033. This growth trend is mirrored in other parts of Asia, where new refined production facilities in India and Indonesia will contribute to an average regional growth rate of 3.1% per annum, leading to a total capacity of 4.6 Mt by 2033.

Global refined and total copper consumption is forecast to grow at a Compound Annual Growth Rate (CAGR) of 2.2% (copper cathodes) and 2.7% (total of all copper consumption) for an overall average growth of 1.9%, between 2023 and 2033. These projections reflect a strong demand outlook for copper, particularly in emerging markets, and highlight the need for sustained investments in production and refining capacity to meet future demand.

The demand for refined copper is a pivotal indicator of growth for developed and developing nations. While copper smelters are dispersed across various regions, the bulk of these facilities are concentrated in Asia. However, the ultimate choice of smelter location hinges on logistical considerations, as this can significantly influence long-term offtake arrangements and financial returns. Secondary factors include the potential to link offtake agreements with financing arrangements and strategic diversification goals.

A range of potential smelter counterparts are available for direct sales contracts:

Domestic:

- Smelters within Chile, such as Codelco's Potrerillos.

Export:

- Brazil (Caraiba)
- Germany, Sweden, Bulgaria, Spain (Aurubis, Hamburg; Aurubis, Pirdop; Boliden, Ronnskar; Freeport, Huelva)
- China (Jiangxi, Tongling, Jinlong, Daye, XGC, Jinchuan, Yunnan)
- Japan (Saganoseki, Naoshima, Onahama, Niihama, Hibi)
- Korea (LS MnM)
- Philippines (PASAR)
- India (Hindalco, Birla, Vedanta, Sterlite (if it resumes operations); Adani, Kutch Copper, Gujarat).

Geographical diversification serves as a risk mitigation strategy in marketing, albeit with attention also on factors like credit risk and performance risk. The preference for direct sales agreements will likely focus on Asian smelters unless finance-linked contracts offer a compelling alternative.

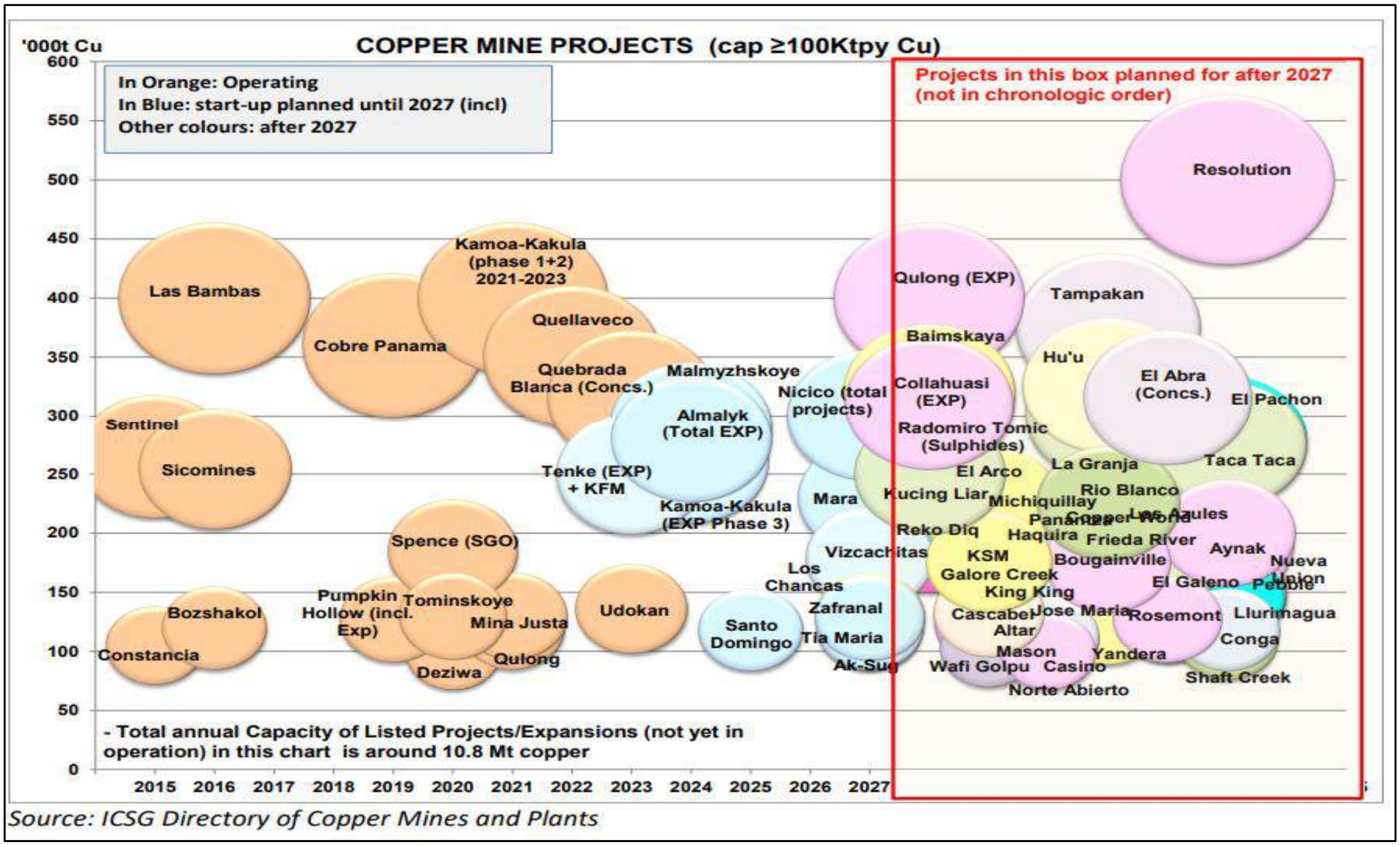
19.2.1 Supply

The current level of mine supply is expected to be insufficient to meet demand beyond 2026, especially after the peak in production at 26 Mt. In the subsequent forecast period to 2033, mine output is expected to dwindle by 4.6 Mt to 21.4 Mt. This decline in mine production is primarily attributed to the declining ore grades at existing mines, coupled with insufficient capital investment to offset these production reductions.

While a few new mines are projected to begin production before 2026, most capital expenditures are allocated towards maintaining the production levels of existing mines. This underinvestment relative to the projected demand has been primarily driven by depressed copper prices that have fallen below project hurdle rates. These economic challenges are further compounded by escalating risks such as water scarcity, more stringent environmental regulations, increased energy costs, political instability, and resource nationalism.

Despite the existence of numerous copper resource projects worldwide, shown in Figure 19 1, it is estimated that only about 9 Mt of potential annual supply is considered economically viable. This quantity barely offsets the decline in baseline copper supply over the period up to 2033, underlining the precarious balance between supply and demand in the global copper market.

Figure 19-1: Copper Mine Projects



Source: The planned start-up date for Santo Domingo Project shown in the figure is an interpretation by International Copper Study Group presented in "The World Copper Factbook 2023". Capstone has not sanctioned a construction decision as of the date of this Technical Report.

19.2.2 Demand

China remains the primary catalyst for the expected rise in global copper consumption, with industrial production (IP) forecast to grow at a steady 2.7% annually until 2030, shown in Table 19 1. This projection, based on a 2022 baseline, implies a corresponding 2.7% annual increase in copper demand during the same period, with China contributing around 40% of this overall growth. By this estimation, copper demand is poised to grow by a substantial 6.3 Mt by 2030, compared to the 2022 levels.

A key driver of this growth is the burgeoning renewable energy infrastructure sector, which is expected to emerge as the primary engine of global copper demand expansion. The imperative to integrate numerous small-scale electricity generation units into the power grid is set to significantly boost copper demand. This trend is exemplified by China's installation of 216 gigawatts (GW) of new solar generation capacity in 2023, surpassing any other nation's historical total, and a remarkable 20.7% increase in wind generation capacity (75.9 GW) in the same year.

Furthermore, the transition to offshore generation, alongside the proliferation of electric vehicles (EVs) and their charging infrastructure, will also drive significant demand for copper. While EV growth, particularly in China, is currently modest, manufacturers in the Western world are gearing up for substantial production escalations. The combined global increase in passenger EV production is projected to boost copper demand by an additional 1.5 Mt by 2030 (see Table 19-1).

This forecast underscores the critical role of China and the renewable energy and EV sectors in the global copper market's future growth trajectory. The sector's evolution is poised to fundamentally reshape copper demand dynamics, underscoring the pivotal role that innovation and sustainable energy solutions will play in shaping the future of the industry.

Table 19-1: Copper Consumption by Components Through 2030 (in kt unless stated otherwise)

End Use	2019	2020	2021	2022	2023f	2024f	2025f	2026f	2027f	2028f	2029f	2030f	CAGR%	
													'22-'25	'22-'30
Global total consumption	24,944	24,436	26,010	26,324	27,114	27,727	29,041	29,904	30,507	31,208	31,913	32,651	3.3	2.7
(semis consumption excluding fabrication losses) (%)	1.1	-2.0	6.4	1.2	3.0	2.3	4.7	3.0	2.0	2.3	2.3	2.3	-	-
Decarbonization (Power gen, EV, batteries, decarb grid)	1,410	1,739	2,262	2,813	3,850	4,461	5,370	5,902	6,366	6,911	7,456	8,043	24.1	14.0
Consumption change due to Decarbonization activities y/y (%)	2.9	23.4	30.1	24.4	36.9	15.9	20.4	9.9	7.9	8.69	7.9	7.9	-	-
Global share (%)	5.7	7.1	8.7	10.7	14.2	16.1	18.5	19.7	20.9	22.1	23.4	24.6	-	-
Power generation	1,037	1,217	1,463	1,544	2,241	2,542	2,980	3,166	3,307	3,503	3,668	3,823	24.5	12.
Wind	268	392	462	320	450	470	595	617	595	641	709	742	22.9	11.1
Solar	495	543	704	919	1,471	1,764	1,978	2,147	2,310	2,469	2,569	2,700	29.1	14.4
Battery grid storage (BESS)	17	34	58	183	237	250	297	295	323	358	395	432	29.7	11.4
Cu battery content	12	23	37	151	153	136	154	139	146	158	163	163	17.7	1.0
Cu in battery installation infrastructure	6	11	21	32	85	114	143	155	177	200	232	269	0.7	30.6
Renewables related grid expansion	177	226	267	312	500	594	680	742	796	860	916	973	65.4	15.3
Core grid expansion	2,677	2,425	2,323	2,604	2,625	2,642	2,671	2,690	2,702	2,713	2,724	2,735	29.7	0.6
Ex-China grid ex-power gen	537	491	304	267	270	268	275	282	285	288	292	295	0.9	1.2
China grid ex-power gen	2,140	1,934	2,018	2,336	2,355	2,374	2,396	2,408	2,417	2,425	2,432	2,440	1.0	0.5
China core grid development	1,070	967	1,009	1,168	1,156	1,145	1,133	1,122	1,111	1,100	1,089	1,078	0.8	-1.0
China grid on cyclical trend	1,070	967	1,009	1,168	1,198	1,229	1,263	1,286	1,306	1,325	1,344	1,362	-1.0	1.9
Electric vehicles and charging	179	262	474	774	871	1,076	1,413	1,699	1,939	2,190	2,478	2,816	2.6	1.5
Passenger vehicles	130	178	362	573	722	839	1,069	1,287	1,471	1,641	1,822	2,054	22.2	17.5

End Use	2019	2020	2021	2022	2023f	2024f	2025f	2026f	2027f	2028f	2029f	2030f	CAGR%	
													’22-‘25	’22-‘30
Commercial vehicles and buses	31	28	38	64	69	86	133	197	235	285	333	392	23.1	17.3
Supply-chain stock adjustments	9	45	49	97	27	82	116	80	64	60	76	76	27.9	25.5
E-bikes and E-trikes	2	3	4	4	4	5	7	9	11	12	14	17	27.1	21.5
EV charging infrastructure	6	8	23	37	50	64	86	127	158	192	232	277	33.2	28.8
Other automative	2,300	1,996	2,000	2,060	2,190	2,257	2,174	2,016	1,921	1,849	1,778	1,696	1.8	-2.4
ICE (PV+CV)	2,384	1,990	1,954	1,891	1,909	1,907	1,822	1,743	1,693	1,631	1,551	1,477	-1.2	-3.0
Hybrid	96	105	132	188	193	243	239	247	239	231	235	247	8.4	3.5
Supply-chain stock adjustments	-189	-126	-142	-24	38	111	105	343,	-3	-17	-20	-20	-	-
All other/cyclical	18,557	18,276	19,425	18,848	18,449	18,366	18,825	19,296	19,518	19,735	19,955	20,177	0.0	0.9
Consumption change due to All other/cyclical activities y/y (%)	2.2	-1.5	6.3	-3.0	-2.1	-0.4	2.5	2.5	1.1	1.1	1.1	1.1		
Building and construction	6,964	7,007	7,236	6,726	6,442	6,403	-	-	-	-	-	-	-	-
Consumer goods	4,163	4,118	4,416	4,121	4,4033	3,926	-	-	-	-	-	-	-	-
Machinery and capital goods	2,935	2,759	3,083	3,272	3,337	3,337	-	-	-	-	-	-	-	-
Other	4,496	4,392	4,690	4,728	4,686	4,700	-	-	-	-	-	-	-	-

Source: Citi Research, ICA, Bloomberg, BNEF, 2023-2024.

19.2.3 Supply/Demand Gap

A shortage of copper is likely over the period 2026 to 2030. Potential mine expansions will only cover the loss in production from maturing mines.

Growth in demand could subside for a while, as occurred during the global COVID pandemic. Substitution could also occur to some degree, e.g., aluminum for medium voltage wiring if price differentials become large.

The supply/demand gap will naturally rise over time. Over the period 2026 to 2030, an average annual supply gap of just over 5 Mt/y is predicted. Given that deficits of this level do not (and cannot) occur in commodity markets, prices will rise to incentivize projects to fill the gap.

19.2.4 Price Projections

Analyst consensus long-term copper price outlook was used in section 22 of this Report. As at the date of this Report, the long-term consensus copper price was $4.10 per pound which represents the median and average from 20 independent and global investment banks. Over the last 12 months, the long-term consensus copper price has increased approximately $0.35 per pound as analysts factor in higher capital and operating costs as well as other sector-specific and macro factors that have impacted incentive pricing.

According to various industry sources, to incentivize sufficient copper supply, it is expected copper prices, at a minimum, will need to remain over $8,280/t or $3.75/lb in the long-term. The median consensus estimates from over 20 analysts around the world expect copper prices to average approximately $4.10 per pound in the long-term at the time of this report. This moved up $0.26 per pound over the past 12 months as analysts factor in higher capital and operating costs amongst other factors including geopolitical, environmental, and skilled labour tightness concerns.

Predicting prices as far out as 2033 poses challenges. Projections must factor in several key considerations:

Return on Investment for Miners: For the long-term stability of the industry, miners must see a reasonable return on their investments, ensuring their ability to replace depleting resources with new production. Thus, price growth needs to outpace marginal cost growth.

Inventory Adjustments: A drawdown in inventories, whether due to changes in market demand or industry practices, can impact price dynamics.

Long-Term Price Levels: Prices must remain high enough over the next decade to fill the gap from 2026 to 2033.

In terms of supply, analysis of recent projects such as Quebrada Blanca 2, Mantoverde, and Reko Diq, alongside projections for existing mines, helps estimate total copper concentrate supply. The addition of solvent extraction/electro-winning (SX/EW) copper production provides a complete picture of the copper units available for the market. Historical production forecasts often have a 5% margin of error due to unforeseen factors like labour strikes, natural disasters, or equipment failures.

On the demand side, smelter production forecasts rely on historical operating rates. The difference between supply and demand indicates the market's state – whether it's in surplus or deficit. This approach offers insight into the market's trajectory and potential influences on future prices, despite the inherent uncertainty of long-term forecasting.

19.2.5 Treatment and Refining Charges

The pricing structure for the treatment and refining of copper concentrates (TC/RCs, in units of $/t of concentrate and cents/lb of payable Cu, respectively) is a key indicator of the broader supply-demand balance. Traditionally, the yearly benchmarks have been pivotal in guiding producers and smelters in negotiations. More often now, producers and smelters are forming unique agreements, tailored to individual value propositions. This shift allows for greater flexibility and can lead to arrangements that capitalize more effectively on added value.

For smelters, the stability of a fixed-term agreement spanning a year is often preferable. Many miners are willing to operate under such terms as well. However, this approach does not always align with the pursuit of more lucrative terms based on value propositions, thereby complicating the prediction of TC/RCs over a longer term.

Currently, the average Chinese copper smelter requires a TC/RC of $65/6.5 cents to reach a breakeven point. Beyond this, by-product credits and currency exchange rates can influence profitability.

Considering the expected shortage of concentrate in the future, a reasonable expectation is a decrease in TC/RCs. This projection is significant given that, despite the relatively balanced market of 2017 and 2018, where benchmark TC/RCs were agreed at $82.25/8.225 cents, the 2024 benchmark terms settled at $80.80/8.08 cents, consistent with 2019. This decision, made considering the expected deficit in concentrate supply, suggests that even a relatively small deficit could considerably reduce TC/RCs, potentially pushing them below smelter breakeven points. The possibility of this deficit persisting could lead to the closure of some high-cost smelters, subsequently driving up copper prices and TC/RCs as the market seeks equilibrium.

Longer-term TC/RCs of $75/7.5 cents are perceived as reasonable, offering smelters an operating margin while still acknowledging a market likely to be in deficit for an extended period.

19.3 Copper Concentrate

19.3.1 Santo Domingo Likely Product Specifications

Capstone analysed various industry sources to assess the marketability, including the impact of deleterious elements and penalties, of Santo Domingo’s copper concentrate.

China has imposed strict regulations on the importation of copper concentrates. Specifically, importing materials that exceed limits for certain deleterious elements is prohibited. Copper concentrate must contain less than 0.5% arsenic, 6% lead, 1,000 ppm fluorine, 500 ppm cadmium, or 100 ppm mercury for any one element, by weight, per dmt of concentrate.

As the world's leading consumer of seaborne copper concentrates in terms of tonnage, China's regulations significantly impact the marketability of copper concentrates. Any concentrate surpassing the specified limits is likely to face substantial discounts compared to 'clean' copper concentrates in the market.

In this context, the Santo Domingo concentrate is considered average quality. It is expected to have an average copper content of 26.4% Cu (23.4% Cu to 28.6% Cu) while containing negligible levels of deleterious elements (for further detail, refer to Section 13).

Regarding chlorine and fluorine, these elements are typically below the penalty thresholds, and even if they occasionally exceed these limits, only nominal penalties are expected.

The global copper concentrate market also features a sizable secondary market, with commodity traders facilitating the trade of over 17 Mt annually. These traders often offer more favorable net terms compared to smelters, along with increased flexibility in terms of delivery locations.

In summary, the copper concentrate market is governed by stringent regulations set by leading consumers like China. For suppliers, adhering to these standards is crucial for maintaining market access and optimizing pricing. The Santo Domingo concentrate is particularly well-positioned due to its quality, but attention must be paid to ensure compliance with evolving regulatory requirements.

The marketing strategy for Santo Domingo is expected to target both the premium and conventional markets. Capstone’s marketing group has been proactively engaging with various parties from both segments, and there have been encouraging expressions of interest, contingent upon the project advancing to construction.

Given the expected ‘clean’ composition of the Santo Domingo concentrate, Capstone foresees substantial demand from trading companies that specialize in blending complex materials with cleaner concentrates. These companies typically prefer concentrates like Santo Domingo’s due to their compatibility with blending processes and enhanced value proposition.

Moreover, the inherent appeal of high-quality concentrates also makes it highly coveted by both smelters and traders alike. This further supports the expected strong demand for Santo Domingo’s copper concentrate in the market.

19.3.2 Marketing Strategy

Copper concentrates can be sold under several different agreements, including long-term offtake agreements or frame contracts, mid-term agreements or mid-terms, evergreen and spot contracts and trader offtake agreements.

Copper concentrates, when delivered to end users, are sold based on a payment which is the sum of the addition of all the component ‘payable’ metals (copper, gold, silver and sometimes platinum and palladium) less the sum of the TCs, less the sum of the RCs for copper, silver, and gold, less the sum of any penalties and discounts. The amount of payable metal and TC/RCs vary from contract to contract.

Copper content is paid for at 96.5% of the full and final assayed quantity (after final assays are agreed), but this would typically be subject to a minimum deduction of one unit of copper grade. For all practical purposes, if the copper content drops below 28.6% in the concentrate, the payable copper will be the copper content less one unit (e.g., if the copper content is 28% the amount payable would be 27% (i.e., 28 - 1)). The price paid for the copper content is usually an average price based on a quotation period of the London Metal Exchange (LME) quoted cash copper settlement price (i.e., the seller's price of copper at the midday close on the LME) on each day during the average period.

For precious metal payments there are two different methods commonly used to determine the payable quantities:

Asian-style pricing: silver is paid for at 90% of the full and final assayed quantity of the silver, provided that the silver content is above a minimum of 30 g per dmt. Below this threshold, silver would not normally be payable. Gold is payable on a percentage based on a sliding scale of the full and final assayed quantity, provided that there is a minimum of 1 g per dmt of gold contained. Below this threshold, gold is not payable.

European-style pricing: silver is payable on the full and final assayed quantity of silver less a deduction of 30 g. Any content below 30 g per dmt would not be payable. In higher silver content concentrates, there is often a deduction of 50 g per dmt instead of 30 g per dmt. Gold is payable on the full and final assayed quantity of gold less a deduction of 1 g per dmt. In concentrate containing less than 1 g per dmt there would be no payment.

Another variation on the actual price paid for each metal (copper, silver, and gold) occurs when copper concentrates are exported to the United States. Prices paid for the payable metals are based on the Comex (the New York Mercantile Exchange's Commodity Exchange division) traded first position (essentially the spot month). This is only used for concentrate delivered from overseas to, or internally within, the United States.

Copper concentrate long-term frame contracts are typically highly sought-after by smelters. Smelters, especially in China, have been operating at well below capacity. Over the last decade, spot TC/RCs for concentrate supply have been running at a discount of $15 - $20 on the long-term contract rates. There is a worldwide trend for concentrates to have a higher average arsenic content. The trend is partly a result of general trends in large orebodies currently being mined but is also due to higher commodity prices for contained copper, gold and silver in concentrates. This results in many high arsenic mines (e.g., in Peru, Mexico, the Philippines and Bulgaria) continuing production, despite very high penalties for the arsenic content of the concentrates produced compared to the clean concentrate market.

19.3.3 Concentrate Marketing Assessment

It is expected that the copper concentrate produced will have a low gold content (around 3 g per dmt) and a low silver content (around 27 g per dmt). As a result, there will be considerable value to pricing the material on an Asian-style basis as opposed to a European-style pricing. This will be accentuated when, occasionally, the silver content rises above 30 g per dmt. If this percentage is payable using European terms the payment will be very low, but with Asian terms over 89% would be payable.

Several factors must be taken into consideration when assessing the best contract partners for Capstone on a long-term basis. Factors such as freight, assay bias, geographic location and contractual party reliability must be considered. The normal contract split for mines of the proposed size of Santo Domingo are:

- 60% to 70% on long-term frame contracts with four or five major smelters

- 10% to 20% to traders on 3–5-year fixed TC/RCs or TC/RCs to be negotiated annually
- 20% to 30% spot contracts for up to 1 year with traders at fixed terms.

Long-term contracts should be adjusted to incorporate the high copper concentrate production in the first years of operation and the gradual decline in copper concentrate production thereafter. Consideration must also be given to the terms and timing of the contract renewals so that renewals do not all occur at the same time.

Capstone may obtain signed letters of intent (LOIs) or memorandums of understanding (MOUs) from smelters, followed by full long-term contracts, to support arrangements for project financing

19.3.4 Logistics

The copper concentrate will arrive at the port in containers sealed on trucks. The port area has a pressurized storage building with a capacity of 50,000 tonnes in two sections. Additionally, in the port area, the construction of a platform for the handling of sealed containers has been considered.

The copper concentrate will be shipped in Handymax class ships (60,000 deadweight tonnes).

Ocean freight rates are primarily driven by two factors: energy (fuel) costs and the supply/demand of vessels. The International Maritime Organization (IMO) requirement that vessels reduce emissions is expected to contribute to higher freight rates as higher cost low sulphur fuel are being used. Other alternative fuels are being considered but are also likely to come with a higher cost.

For the period January 2019 to January 2024, indicative time charter rates for Panamax/Supramax/Handymax vessels, shown in Figure 19 2 (the primary type of vessels used in transporting copper concentrate) have been volatile but are currently about $11,000 per day.

Figure 19-2: Indicative Time Charter Rates

Source: World Wide Shipping & Chartering, 2024.

Currently there is an adequate supply of vessels and new builds are effectively replacing scrapped vessels. The dry bulk market is expanding at about 3% per year. However, current order books are thin as current freight rates do not provide an incentive for owners to expand their fleets beyond replacement. If this scenario continues, an annual increase of 5% in freight rates is expected. A price of $40/dmt was used in the economic analysis in this Report.

19.3.5 Opportunities for Synergies with Mantoverde Mine Logistics

The proximity of the Santo Domingo and Mantoverde mines provides economies of scale and shared use of logistic resources that will provide reduced logistics costs for both operations. Land transport (trucks with containers) can be deployed to either mine under a single service agreement that would reduce the unit costs. Port storage facilities would be shared with the product from each mine segregated but could be blended or substituted depending on grades. The increased volume of exports also provides for lower marine freight rates by combining cargos on the same vessel which is attractive to vessel owners and provides some leverage in negotiating lower freight rates in Contracts of Affreightment.

19.4 Iron Concentrate

Magnetite concentrate fines with high iron content (Fe >62%) will be produced for shipment overseas in Cape-sized vessels to iron and steel makers.

To supplement information on the international market for the Santo Domingo iron ore concentrate, Capstone contracted the following:

- Mr. David Trotter, DAZMIN CONSULTING, (global iron ore and commodity marketing consultant), who prepared a study entitled "Pellet Feed Market Characterization and Forward Pricing Outlook" for Capstone, dated September 2018. This initial report was updated in November 2023.
- A report from Braemar (March 2024) provided updated freight projections.

The Trotter (2023) forward pricing report prepared for Capstone estimated that prices for 62% Fe content sinter fines (Platts Iron Ore Index or IODEX) cost-and-freight (CFR) Qingdao delivery (considered the standard product for CFR China delivery) can be expected to be $113.94/dmt (based on the average price from 2018 to 2023). Premiums for 65% Fe concentrate ($15.04/dmt), value-in-use (VIU) for 66% Fe ($17.83/dmt) and for 67% Fe ($26.32/dmt), in all cases compared to Platt 62% Fe.

19.4.1 Iron Ore Market

Iron ore is globally traded with hematite (Fe_2O_3) and magnetite (Fe_3O_4) ores making up most of the world seaborne trade with most of the supply coming from South America and Australia.

Iron ore is used either in a blast furnace (BF) or in a direct reduction (DR) furnace to make metallic iron for use in steelmaking. Globally and in the key import markets, over 75% of steelmaking plants use hot iron from a blast furnace. The input of iron into the blast furnaces can be in the form of pellets (agglomerated fine pellet feed from a pellet plant), lump (naturally occurring agglomeration) and sinter fines (agglomerated coarser iron ore fines in a sinter plant).

Steel production dictates the demand for iron ore. Global steel production in 2023 was 1.89 billion tonnes (World Steel Association, 2024). Global world steel demand is expected to increase to 2.2 billion Tonnes by 2040, shown in Table 19-2.

Table 19-2: Global Steel Demand Forecast 2020 – 2040

Global Crude Steel Demand Forecast 2020-2040					
	2020	2025E	2030E	2035E	2040E
Billion Tons	1.886	2.038	2.103	2.151	2.199
Kg per capita	242	249	246	242	239

Source: World Steel Association, 2024.

China dominates the global production of steel and thus the consumption of iron ore. China produced 1,019 Mt of steel in 2023. Other top producers in 2023 were India at 140 Mt, Japan at 87 Mt, and South Korea at 67 Mt.

The pellet feed market demand outlook is considered the strongest segment of iron ore demand, being linked to strong finished steel demand and the strong pellet market. By looking at the demand and then supply of pellets, the future requirement of pellet feed can be estimated.

Despite the limited growth in steel demand in China the demand of blast furnace pellet feed is expected to be strong and the dominant growth segment in the iron ore market. It is being driven by the change in requirements of ore in China. The key factors in China include:

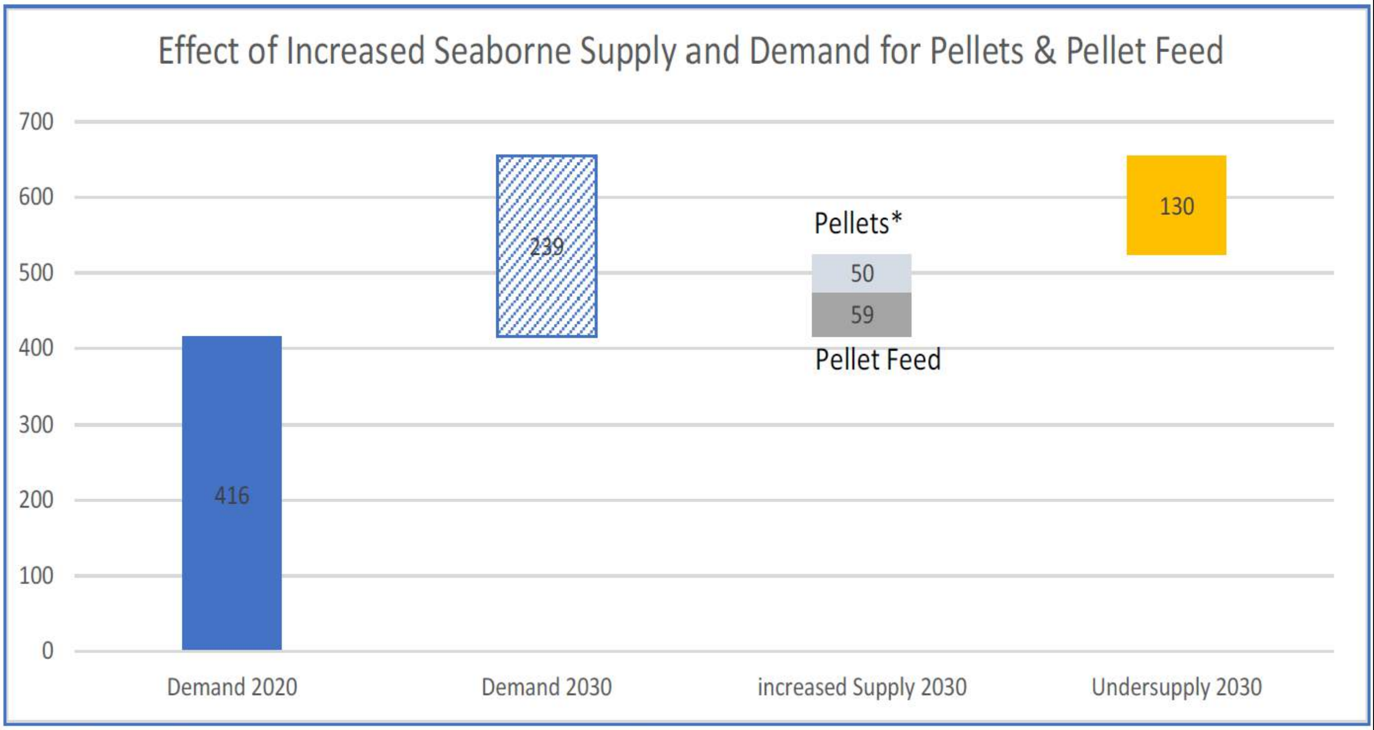
- Environmental Restrictions on new blast furnace construction
- Environmental restrictions on new sinter plant capacity
- Improving efficiencies in blast furnaces to 90%
- Lack of high-grade domestic pellet feed which average grade declines
- New large-scale pellet plants within China
- Higher margins for high quality steel
- Increasing use of pellet feeds and higher-grade ores in sintering
- Individual customers demand and technical limits changing.

19.4.2 Pellet and Pellet Feed Supply and Demand Summary

The pellet feed supply situation is expected to tighten with the increase in direct reduced iron (DRI) production. The supply situation in 2032 is also highly uncertain due to disruptions in the Ukraine and Russia and recovery in Samarco production.

Various supply demand forecasts in the shortage of pellet feed supply by 2032 is from 45 Mt/y in some industry sources to 120 Mt/y (CRU) shortage (see Figure 19-3).

Figure 19-3: Pellet Feed and Pellet Supply/Demand (2020–2030)



'*includes BF and DR Pellets data from supply and demand project announcements and CRU data.

Source: Trotter, 2023.

The combined pellet feed and pellet market is likely to be undersupplied by 126 Mt by 2030, shown in Figure 19-3with current supply and assuming Samarco restart. An inferred supply gap of seaborne pellet feed of at least 50% with the other 50% being made from captive pellet plant producers. The short fall of pellet feed is likely therefore to be between 63 to 130 Mt per annum by 2030 in current supply forecasts.

19.4.3 Pricing of High Grade Iron Ore Fines (>65% Fe)

The price of iron ore is widely accepted as the 62% Iron Ore Index as reported by price reporting companies in dollars per dmt (Table 19-3).

Index basis for 62 and 65 index are based on hematite for the BF market. The DRI market is only estimated to be currently around 120Mt of seaborne demand and is not driving the pricing indices.

Magnetite and Hematite ores have different Silica, Alumina and FeO content ratios making the use of these indexes more difficult for magnetite. The DRI market is only estimated to be currently around 100 Mt of demand in 2032.

Platts Iron Ore Index, or IODEX (IODBZ00), is a benchmark assessment by S&P Global Commodity Insights of the spot price of physical iron ore. The assessment is based on a standard specification of iron ore fines with 62% iron, 2.25% alumina, 4% silica and 0.09% phosphorus, among other gangue elements.

Metal Bulletin 65 Index, or MBIOI-65-BZ is an assessment by Fast Markets used for fine iron ore with compositions above 63.5% iron. The iron content base is 65%, silica 1.7%; alumina 1.5%; phosphorus base 0.08%; sulphur base 0.01%; moisture base 9%.

The prices are quoted in US Dollars per dry metric tonne and the lump premium is in US Dollars per dry metric tonne iron units.

Table 19-3: Pricing Summary for Magnetite Iron Ore Fines Key Indexes From 2018 to 2023 (US$)

Index	2018	2019	2020	2021	2022	2023*	Average
58 Nominal	57.5	86.6	96.6	136.4	101.9	111.4	100.6
58 Real	60.7	86.7	99.0	137.5	102.5	111.4	102.4
62 Nominal	69.4	93.3	108.2	160.0	112.1	122.8	113.9
62 Real	73.3	96.8	110.9	161.1	112.7	122.8	115.9
65 Nominal	90.1	104.2	121.8	185.0	123.2	139.2	129.8
65 Real	95.2	108.1	124.9	185.0	123.2	139.2	131.7
Lump Premium	16.0	17.2	9.0	22.9	13.7	7.0	14.3

Source: published prices Platts and Fast track markets, 2023.

The premium should be considered a major part of any price forecast for iron ore products including pellet feeds. This has been mirrored to an extent by the low-grade ore index; the low-grade ores are under reduced demand and are increasingly being blended with high grade ores.

To forecast this premium's value over the next 10 years, shown in Table 19-4, assuming the major factors remain unchanged, the other drivers will be coking coal cost (the other major steelmaking cost) and steel margins.

Table 19-4: Nominal Price Forecast (US$) on VIU Pricing 2023-2033, Applicable to Magnetite Iron Ores

	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033
(Actual)											
IODEX 62% Fe, CFR Qingdao China	113.94	113.94	113.94	113.94	113.94	113.94	113.94	113.94	113.94	113.94	113.94
+ Differential 65% Fe Index	15.83	15.83	15.83	15.83	15.83	15.83	15.83	15.83	15.83	15.83	15.83
+ Fe VIU for 64.95 (Product 3)	-0.79	-0.79	-0.79	-0.79	-0.79	-0.79	-0.79	-0.79	-0.79	-0.79	-0.79
+ Fe VIU for 66.06 (Product 2)	2.79	2.79	2.79	2.79	2.79	2.79	2.79	2.79	2.79	2.79	2.79
+ Fe VIU for 67.08 (Product 1)	8.49	8.49	8.49	8.49	8.49	8.49	8.49	8.49	8.49	8.49	8.49
Median Fright in 180 kt average vessel (Santo Domingo)	-25.75	-25.75	-25.75	-25.75	-25.75	-25.75	-25.75	-25.75	-25.75	-25.75	-25.75
Pricing Capstone Pellet Feed FOB Chile	114.51	114.51	114.51	114.51	114.51	114.51	114.51	114.51	114.51	114.51	114.51

Source: Trotter 2023. Table considers estimated freight cost by Mr. Trotter.

Using iron-making models it is possible to predict the scenarios which would affect the premium. Assuming a steel margin in the range \$30/t to \$100/t and a derived coke price between \$170/t and \$260/t the scenarios show a long-term premium between \$19/t and \$33/t. This is a sizable percentage of the 65% Iron Ore Index price component which is expected to remain disconnected from 62% Iron Ore Index.

The limits on alumina also help to drive up the premium for ores above 65% Fe because by mass balance the total oxides should be less than 6.0%. The market is expected to be alumina-constrained for several years and, given the small impact that low alumina pellet feed will have on a year-on-year basis, this may continue.

For the financial modelling and economic analysis in Section 22, Capstone selected a conservative approach to long-term iron concentrate pricing, particularly with respect to the Trotter 62% and 65% (CFR China) forecasts. Analyst consensus estimates were used as the basis for the long-term iron ore price forecast in this Report. As at the effective date of this Report, the long-term consensus 62% and 65% (CFR China) prices were \$85/t and \$110/t, respectively. An additional premium of \$10/t (CFR-China) is assumed for the Project's 67% Fe product.

For reference, as of May 31, 2024, 62% and 65% Fe prices (CFR China) averaged approximately \$118/t and \$149/t, respectively over the past 5 years. Over that same period, the premium between the 62% and 65% iron ore index price (CFR-China) averaged approximately \$25/t.

With respect to the forecast FOB Chile prices for iron ore, a \$25/t freight charge is assumed based on a Chile-to-China route and reflects historical and forecast data from Trotter as well as Braemar, a global leader in the shipping industry.

Table 19-5 provides a summary of the key commodity prices and related assumptions used in the economic evaluation of the Project highlighted in section 22. Please refer to section 22.5 for sensitivity analysis with respect to the following commodity price assumptions.

Table 19-5: Iron Prices Selected for use in Economic Analysis in Section 22

Metric	Unit	Price (US$)
Iron Concentrate - 62% CFR China	US$/t	85
Iron Concentrate - 65% CFR China	US$/t	110
Iron Concentrate - 67% CFR China	US$/t	120
Freight Charge assumed in CFR China price	US$/t	25

19.4.4 Logistics

Because iron ore is a relatively low-value commodity, it is imperative that logistics costs are minimized to protect margins. Typically, this means using very large ore carriers (VLOC) vessels. However, given the probable capacity to berth vessels in a new Chilean port will be limited to 225,000 deadweight tonnes, capsize vessels with a capacity of 170,000 – 220,000 deadweight tonnes are most likely to be used.

The appropriate index from the port of Santo Domingo is C3 index for larger bulk carrier's basis 170,000 tonnes. The index captures the daily hire rates by wet metric tonnes (wmt) for these vessels from Tubarao port Brazil to Qingdao China for coal or iron ore.

In determining a freight strategy, the following factors need to be considered:

- Scheduling of shipments and certainty of commodity production
- Shipment size and discharge port flexibility
- Freight pricing mechanisms, including fixed and index linked
- Charterer's risk appetite
- Outlook for freight rates
- Owners hedging positions
- Counter party risk.

19.4.5 Iron Ore Freight Rates

Over the last 5 years the market has seen extremes for freight rates for Capsize vessels. The current freight C3 is \$28.87/wmt. The C3 index has a yearly average rate ranging between \$14.10 and \$37.40 for the period 2018 to 2023.

Given current bunker rates, port charges, vessel values and 60 kt/d load rate with 50 kt/d discharge rate, a projected freight rate of \$25.75/wmt was determined for a Chile–China routing. Braemar estimated the freight rate in 2027 for Cape-size vessels from Chile to north China would be \$23.53/wmt.

19.4.6 Santo Domingo Iron Ore Concentrate - Specifications

For assessing the marketability of the iron ore concentrate, Capstone expects to produce an iron ore concentrate with the specifications shown in Table 19-6. The specifications given in the table are from 2022 - 2023 pre-feasibility study metallurgical test work and are not smelter specifications.

The iron ore concentrate forecast in the production schedule (see Section 16) is a typical pellet feed currently used in pellet plants. Magnetite is the predominant mineral. The iron grade is high (variable at Fe from >62% through >65%) and the low alumina (Al_2O_3) and low phosphorus (P) make the concentrate suitable for most pellet plants. The suitability and demand for this pellet feed will be considered in the context of increasing use of pellets in iron making, the increased use of higher-grade ores generally and as a premium additive to sinter plants by blending.

Table 19-6: Iron Ore Concentrate Specification

SGA No. 7692 E17886 Chemical Analysis	Unit	MSD L 8-1 Concentrate Cleaner 6	MSD L 8-3 Concentrate Cleaner 7	MSD Concentrate MSD 2020 8
Fe_{tot}	%	66.94	67.22	64.94
FeO	%	25.74	25.53	25.08
SiO_2	%	3.13	2.91	4.60
Al_2O_3	%	0.79	0.75	1.08

SGA No. 7692 E17886 Chemical Analysis	Unit	MSD L 8-1 Concentrate Cleaner 6	MSD L 8-3 Concentrate Cleaner 7	MSD Concentrate MSD 2020 8
CaO	%	0.63	0.59	0.82
MgO	%	0.34	0.32	0.48
P	%	0.008	0.006	0.013
S	%	0.003	<0.005	0.008
Na_2O	%	0.100	0.100	0.180
K_2O	%	0.086	0.083	0.130
Mn	%	0.061	0.60	0.069
TiO_2	%	0.18	0.17	0.23
V	%	0.0067	0.007	0.007
Cu	%	0.0150	0.0050	0.0070
Loss of Ignition	%	-1.13	-1.21	-0.55
Specific Weight	g/cm^3	4.89	4.90	4.78
Blaine	-	2,960	2,699	2,320

19.4.7 Deleterious Elements and Penalties in Iron Concentrates

Each steel mill has different impurity allowances and tolerances. Each is unique in its requirement for feed and will try very carefully to blend the constituent elemental requirements in the iron ore concentrate. However, as a generality, it can be said that most mills prefer SiO_2 <3.5%, though in pellet plants this may be as high as 5.5%; Al_2O_3 <1%; Mn <0.5%; P <0.1%; S <0.1%; Cu <0.01%; and a combined Na_2O and K_2O <0.5%. The Al_2O_3 is a cost factor due to its endothermic reaction (and consequent heat absorption cost). Other rarely found elements in iron ore such as copper are also problematic beyond a certain concentration, but the reality is that copper can often be blended out in the charge feed mix since it is usually only found in trace amounts in most iron ore types. If impurities are higher than the levels discussed, then it becomes more difficult, but not impossible, to place material with mills.

The iron ore concentrate specifications, and the list of impurities are used by steelmakers to calculate a value in use for each blast furnace and in turn the value of the pellet feed. The main levels of impurities expected in the iron ore concentrate are silica and copper. Copper is expected to be below the threshold but may, in some circumstances, represent a non-preferred feed. Silica is only likely to be a cost factor or penalty element rather than a rejectable quality issue. Silica penalties are variable but would be of the order of $1.50–$2.00/t per each 1% above 3.5%.

Based on the test results and the earlier specifications for study, the five products in Table 19-7 were selected for assessment.

Table 19-7: Products for Assessment Including Product 5 for Comparison

Specification	Product 1	Product 2	Product 3	Product 4	Product 5
Wt%	Average Clean	Previous Study	Uncleaned	Potential product	Future Upgrade*
Fe	67.08	66.06	64.94	63.00	68.00
SiO_2	3.02	4.10	4.6	5.0	2.00
Al_2O_3	0.77	1.00	1.08	1.5	0.60
P	0.007	0.011	0.013	0.15	0.007
LoI	-1.17	1.34	-0.55	2	-1.5

*For comparison purposes only as no current data. Source: Trotter, 2023.

Product 1 specifications above 67.08% Fe and 3.77% (SiO_2 + alumina) is applicable to the Global traded Iron ore market for Direct Reduction Grade Iron Ore Concentrates (often referred to as DRI Pellet feed). This high iron content with low alumina and other impurities, is considered salable in global markets to pellet plants producing DRI pellets. Some blending may be needed for the sizing and slightly higher silica as historic definitions indicate a maximum of 3% silica + alumina. However, due to current and forecast undersupply of high-grade concentrates, the current requirement in Asian and Middle East DRI plants in less selective.

The Capstone Pellet Feed at 67% Fe (Product1) is a typical feed used for production of pellets for iron making including direct reduced iron. The value of other possible feeds, sintering, blast furnace and DRI, are illustrated for comparison in Table 19-8.

Table 19-8: Pellet Feed Value Premiums

Specifications		Contract Premiums Against Platts 62, US$/dmt		
Scenario	Average Fe Grade (%)	Sinter	BF Pellets	DRI Pellets
5	68.00	23.77	26.20	29.65
1	67.08	18.92	22.16	26.32
2	66.06	14.93	16.99	17.83
3	64.94	13.13	14.52	15.04
4	63.00	3.11	4.00	4.17
Grand Total	**65.82**	**14.77**	**16.77**	**18.60**

Source: Trotter, 2023.

Pricing considerations for Product 1, according to Mr. Trotter as shown in Table 19-9, which is the current cleaned product, include:

- A price premium of a least US$26.32/dmt above Platts 62 CFR China for Capstone pellet feed based on value in use. The price may be above the value in use due to demand of high grade ores and an Alumina constrained market, especially in China.
- A price expected of over $US 150/dmt CFR China (based on the previous 5 years average pricing for Platts 62).
- A price expectation of $114.51/dmt FOB Chile based on average pricing for the last 5 years and the median freight rate to Huasco.

The specification of the Capstone pellet feed is on the high side of current and developing pellets feeds. While an iron content of 67% or higher would be the targeted quality to maximize premiums, iron ore concentrate with an iron content of 62% or more would still be marketable but would have a lower premium.

19.5 Contracts

On March 25, 2021, Capstone entered a Precious Metals Purchase Agreement between Wheaton, Capstone Resources MSD Ltd. and Capstone with respect to the purchase and sale of gold. Wheaton paid upfront cash consideration of $30 million and additional deposits totaling $260 million for total consideration of $290 million. Wheaton will receive 100% of the gold production until 285,000 ounces have been delivered, thereafter dropping to 67% of the gold production. Wheaton will make ongoing payments equal to 18% of the spot gold price, until the deposit of $290 million has been reduced to zero, thereafter increasing to 22% of the spot gold price upon delivery. No additional contracts are in place for Santo Domingo’s production for copper or iron ore concentrate.

On March 24, 2021, Capstone completed a share purchase agreement with Korea Resources Corporation (“KORES”) to purchase KORES’ 30% ownership interest in Acquisition Co., Capstone's Chilean subsidiary that owns Minera Santo Domingo SCM, for cash consideration of $120 million and non-cash consideration of $32.4 million, enabling Capstone's consolidation of 100% ownership in Santo Domingo. As of 31 March 2024, an unsecured liability of $42.9 million (December 31, 2023 - $42.4 million) has been recognized in the consolidated statement of financial position equal to the discounted amount of the remaining $45 million of cash consideration to be paid to KORES on March 24, 2025. The discounted amount of the remaining $45 million will be accreted up to its face value at 5% per annum. During the three months ended 31 March 2024, $0.5 million (March 31, 2023 - $0.5 million) of accretion was recorded in accretion expense in the consolidated statements of loss.

19.6 Comments on Section 19

The QP has reviewed the marketing studies, metal price forecasts and precious metals purchase agreement. The information supports the assumptions in the Technical Report and is acceptable for use in the economic analysis in Section 22.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

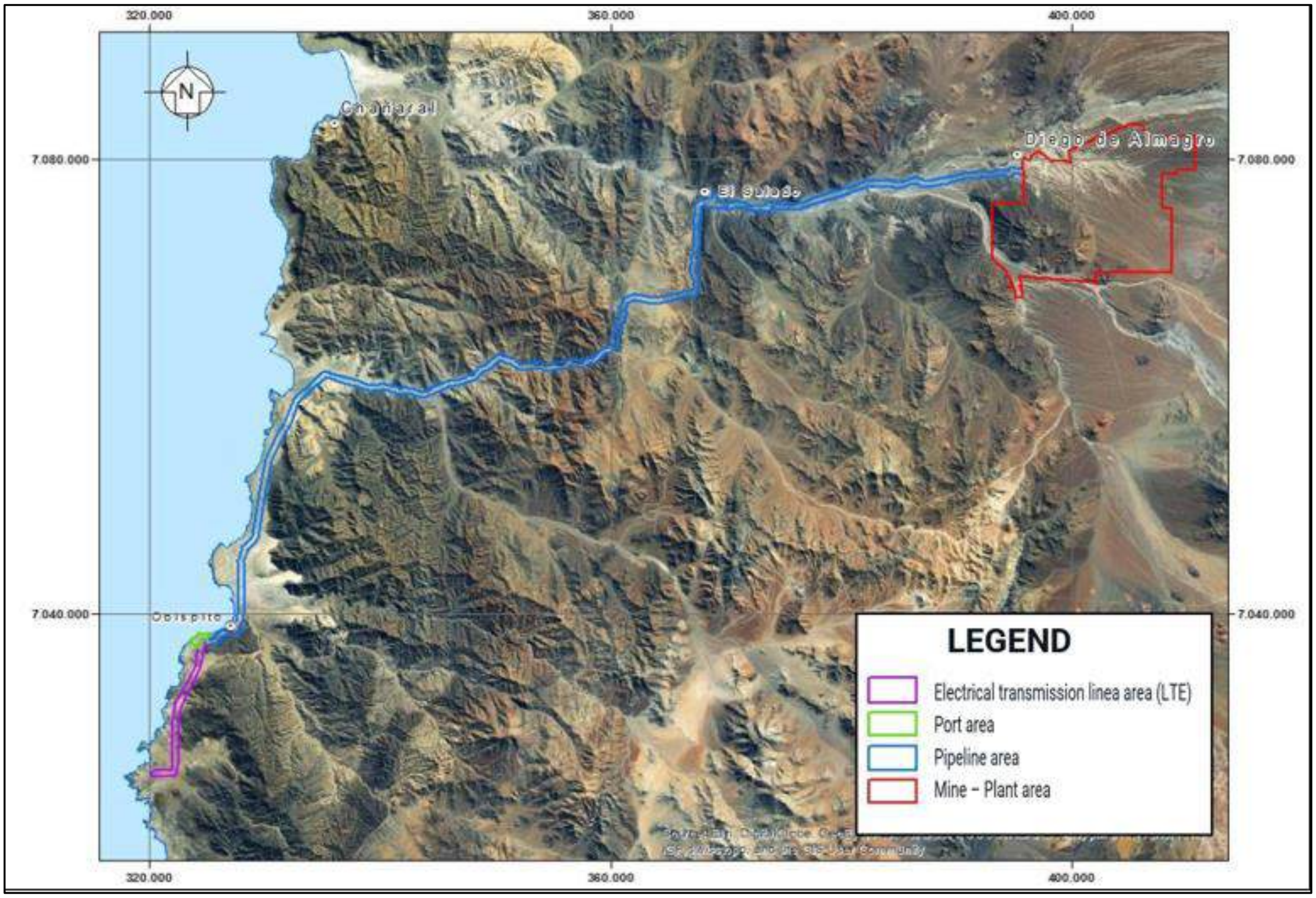
20.1 Introduction

This section provides an overview of the setting of the Santo Domingo Project. It outlines existing biological and physical baseline conditions, proposed new and ongoing baseline studies to support future permitting applications, existing permits, and future regulatory and permitting requirements, including required management plans for water, site environmental monitoring, and waste disposal. In addition, this section also discusses socio-economic baseline conditions, the status of community engagement, and mine closure and reclamation planning for the Project. This information was summarized in a previously published NI 43-101 report completed in 2020 (Santo Domingo Project, Region III, Chile, NI 43-101 Technical Report). Approved environmental licences were received for the Project in 2015 and 2020 based on the scope of the Project as presented in the 2020 NI 43-101 technical report (2020). Since that time, optimizations have been made to the 2020 Project scope, as presented in the Report, that will require Capstone to, at a later date, enter the Environmental Impact Assessment System (SEIA) through an Environmental Impact Declaration (DIA), indicating the updates compared to what is currently approved. The relevant modifications to be included in the DIA with respect to the previous approvals as follows:

- A new connection to an alternative existing electrical substation, located adjacent to the original substation. This required a modification to the substation which is operated by a third party.
- An updated TSF design within the area previously approved for the facility.
- Simplified tailings thickening system (one stage thickening instead of two as considered in the 2020 Project).
- Minor optimizations for the other areas (mine, process plant, port, pipelines, and tailings deposit); these changes would be made within the areas already assessed in the 2015 EIA. The port area requires additional information to be collected associated with a new access road and a small effluent treatment plant for iron concentrate filtrate.

The project is located in the Atacama Region, in the provinces of Copiapó and Chañaral, in the municipalities of Caldera, Chañaral and Diego de Almagro. Figure 20-1 shows the location of the Project sites.

Figure 20-1: Santo Domingo Project Location



Source: Figure from Project Santo Domingo Environmental Impact Study, 2013.

Given that the works and facilities of the Project are distributed in an extensive area that extends from Diego de Almagro to the Punta Roca Blanca sector in Caldera, the following areas have been considered as the study area:

Mine-Plant Area corresponds to the works and facilities to be used for the exploitation and processing of ore, and surrounding area including:

- the Santo Domingo and Iris Norte pits
- the tailings dams (north, southeast and southwest)
- the Concentrator Plant and the Thickened Tailings Deposit, among others.

This area is located 7 km southeast of the city and district of Diego de Almagro. The Project works are located outside the urban area of Diego de Almagro, with de exception of the power line route that supplies the Mine-Plant area.

Pipeline Area corresponds to the area where the magnetite concentrates pipeline, and the desalinated water transportation pipeline will run parallel and be buried in a common trench for most of the pipeline route. At the port and plant locations, each pipeline will be routed separately to their respective facilities.

Port Area: area for the location of works and facilities for washing, filtering, storage, and shipment of magnetite concentrate, as well as for the storage and shipment of copper concentrate. This area also includes a Desalination Plant, and the works attached to these facilities. This area is in the Punta Roca Blanca sector in the Caldera district.

Electrical Transmission Line Area (LTE): the alignment for the Electrical Transmission Line that will supply the Port Area, located within the district of Caldera.

20.2 Baseline and Supporting Studies

The 2015 EIA and the 2020 DIA presented bibliographic information and the results of field studies that characterized the Project area and surroundings. The sections below reproduce the results of the baseline environmental characterization studies, conducted between 2010 and 2013 to support the 2015 EIA and between 2019 and 2020 to support the environmental impact declaration (DIA) for the four areas defined, and include the relevant results of the most recent post-EIA/DIA baseline studies conducted.

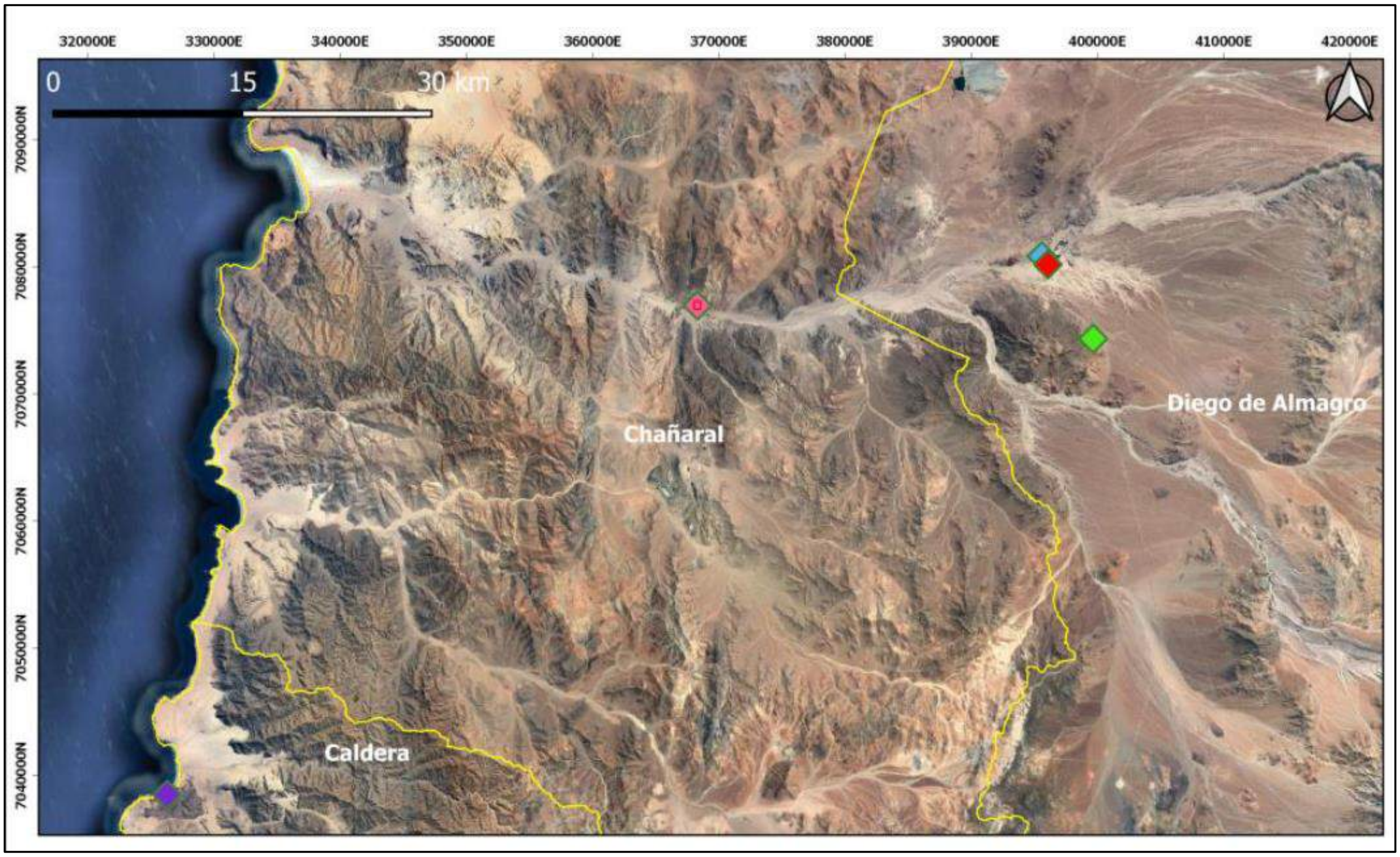
20.2.1 Weather and Meteorology

The study area is located in zones with a climatic development defined by its location in the Atacama Desert. There are two types of climatic zones present: Desert with Abundant Clouds and Normal Desert. These climates are typified by arid conditions and a significant lack of rainfall, in addition to marked daily temperature fluctuations.

The meteorological characterization of the study area is based on the analyses of meteorological records obtained from the monitoring network installed in the project study area, which consists of a total of six monitoring stations. The monitoring network commenced operations in 2010 at the Diego de Almagro and Mina stations, in November 2011 at the El Salado station, in June 2012 at the storage station and in December 2012 at the Obispito station.

The measurements have been collected continuously since the installation of each station up to the present. The locations of the monitoring stations network are shown in Figure 20-2.

Figure 20-2: **Air Quality**



Source: Figure prepared by SERPRAM, Air Quality and Meteorological Monitoring Santo Domingo Mine Report, December 2023.

For the Mine-Plant Area Sector, the meteorological records of the Diego de Almagro, Storage station, and Mine stations were analyzed. According to these records, the meteorology in the sector of the city of Diego de Almagro is characterized by a predominance of winds from WSW and SW throughout the year, with an average speed of 1.9 m/s during the monitoring period. On the other hand, the Mine-Plant Area (where the Mine station is located), is characterized by a predominance of WNW, NW, and NNW winds throughout the year, with an average speed of 3.1 m/s. It is of note that even though the stations are relatively proximal, the wind patterns vary due to the influence of local topography.

In addition, the Mine-Plant Area has a scarcity of precipitation. The average temperature corresponds to 18.6 °C, and the maximum temperatures are recorded in the summer months, where average monthly temperatures reach over 20.7 °C. The average monthly relative humidity corresponds to 42.6%.

For the Pipelines Area, meteorological records from the El Salado station were analyzed, which indicate a predominance of winds coming mainly from the WNW directions with an average wind speed for the monitoring period of 3.1 m/s.

Finally, for the Port Area, meteorological records from the Obispito station were analyzed. The sector where this station is located has coastal influence and is characterized by a predominance of wind direction from the W, followed by winds with WSW direction, and an average wind speed of 2.9 m/s. The average temperature of the area is 17 °C.

20.2.2 Geology, Geomorphology, and Natural Risks

For the geological component in the study area, two main types of geological units were identified: the rock units and the semi and unconsolidated deposits units. The rock units contain subunits of the intrusive rocks and pre-Tertiary stratified rock types. On the other hand, the semi and unconsolidated deposits units contain subunits including Atacama gravels, Ancient alluvial deposits, Caldera Strata, alluvial deposits and aeolian deposits. Regarding the geological structures, the Atacama Fault System predominates and is the main structural feature of the study area. For a detailed description of the geology for the study area please refer to Section 7 of this report.

For the geomorphology component, seven local geomorphologic units were identified in the study area, corresponding to the following: fluvio-marine plains, erosion platform, sedimentation plain, alluvial, alluvial fan, slope system, dunes and remnant forms.

Finally, three natural risks were identified, corresponding to: seismic and tsunami risk, rockfall or landslide risk and alluvial flow risk. These risks are related to the occurrence of extreme hydrometeorological events (i.e., floods) or earthquakes. Refer to Section 7 of this report for more detailed information on geology and geomorphology.

20.2.3 Geochemistry

Preliminary screening level geochemical studies were carried out in the study area (Knight Piesold, 2013), to assess the potential of acid generation and metal leaching from the waste rock, overburden and tailings material, the following tests were conducted:

- Acid-base accounting (ABA), which determines net neutralisation potential (NNP), representing the potential of the tailings material to release acid rock drainage (ARD) or to consume free acid
- Net acid generation tests, which measure the release of solute from the sulphide bearing tailings material.
- Synthetic Precipitation Leaching Process (SPLP), which measures the release of solute from non-sulphide barren material.
- Kinetic cell tests.

A quantity of samples was selected proportional to the mass of each waste material expected to be produced, including overburden, waste rock, and tailings (as of the mining plan in 2013). A total of 50 samples were analysed (36 from the Santo Domingo Pit and 14 from the Iris Norte Pit).

From the total waste material anticipated to be generated (estimated to be 1,248 Mt in 2013), it was expected that approximately 73% of the material from the Santo Domingo pit (689 Mt) and 65% of the material from the Iris Norte pit (198Mt) were expected to be non-potentially acid generating (NAG). Based on the limited number of samples that were tested, the following tonnages of potentially acid-generating (PAG) material were determined for the combined Santo Domingo and Iris Norte pits as of 2013.

- Overburden – no PAG material expected
- Andesite incl. tuff and volcanic sediments – 192 Mt PAG material expected
- Limestone – no PAG material expected
- Hornblende – no PAG material expected
- Iron oxide – 4.5 Mt PAG material expected
- Basalt (porphyritic or breccia) – no PAG material expected
- Fault or fault breccia – no PAG material expected

In consideration of the quantity of the material to be mined, the above results for waste rock were based on limited number of samples and testing that may not adequately characterize the potential for acid rock drainage and metal leaching. Waste rock high in sulphide exhibited a potential for early release of metals including elevated levels of certain metals and metalloids such as copper, zinc, and arsenic. However, over time, the concentrations of these elements decreased markedly, which suggested that their release is primarily associated with the initial oxidation of reactive sulfide minerals.

The ARD/ML (metal leaching) potential for tailings was assessed by means of a limited number of tests including analysis of tailings supernatant, ABA tests, kinetic tests, and leaching tests. The results indicated low concentrations of metalloids released from tailings, however, these results were based on a limited number of samples and test work that may not be representative of the variability of the ore body.

A detailed review of the geochemistry static and kinetic test work needs to be undertaken to confirm these results and preliminary conclusions of the KP (2013) study, especially in consideration of the current mine plan design and geological model of the deposit.

20.2.4 Soils

The soils of the study area classified as Aridisols for inland areas and as Entisols in the coastal zone. In general, the soils are moderately thick with dominant colours mainly in 7.5 YR (yellow, red) shade in dry and wet conditions. In terms of texture, the soils generally display sandy loam characteristics.

The organic matter content in the samples analyzed was low, with values between 0.05% and 1.6% in all the surface horizons analyzed.

In terms of chemical properties, it should be noted that the soils are moderately to strongly alkaline, in a pH range that varies in pH between 8.0 and 9.6. The soils are generally high in nitrogen (N), deficient in phosphorus (P) and average in potassium (K).

Regarding soil use capacity, the soils in the entire project study area are class VIII soils, which have little to no value for agricultural, livestock or forestry use.

20.2.5 Noise and Vibrations

The Project study area for the noise and vibration component was based on the identification of potential noise and vibration sensitive sites that will be potentially affected by the construction and operation of the Project, including transportation activities along the existing and new routes. The study area includes the populated localities that are close to the Project Areas, which correspond to Diego de Almagro, El Salado, and the coastal sector of Obispito.

Baseline sound levels and vibration measurements were conducted in sensitive areas near the Project areas. The measurements were taken at nineteen sampling points, which are effectively distributed throughout the Project areas.

The baseline measurements revealed that the main sources of ambient noise in the vicinity of the Project Area include wind interacting with foliage, vehicular traffic on nearby roads, wild birds, distant dogs, and breaking waves near the coast. Most of the monitoring points showed decreased noise levels at night due to reduced human activities like traffic and community noise. However, noise levels near the coast remained consistent throughout day and nighttime periods.

According to the limits established in Supreme Decree No. 38/2011 for each evaluation point, the maximum permitted levels vary between 60 and 65 dB(A) for the daytime period and between 45 and 50 dB(A) for the nighttime period. The analyses of the data obtained in the field campaigns with the maximum permitted levels shows that most of the values obtained would not exceed the maximum permitted levels for each sector, only being exceeded at limited monitoring points during the nighttime period. These points naturally present high values, due to the proximity to Route 5.

In addition, baseline vibration measurements were taken during the same measurement periods, which represent the characteristic conditions of each zone, with no external events influencing the recordings.

20.2.6 Air Quality

For the baseline characterization data, the records of respirable particulate matter (PM10), respirable fine particulate matter (PM2.5), sedimentable particulate matter (SPM) and gases from the monitoring stations (same stations used for weather and meteorology) that make up the Project's monitoring network were analyzed, with measurements taken between January 2010 and June 2013.

The main emission sources identified in the Mine-Plant Area, correspond to anthropogenic activity in the city of Diego de Almagro, the erosive action of the wind in waste sites and/or areas with material removal, nearby mining activity and vehicular traffic. The main emission sources detected in the Pipeline Area correspond to anthropogenic activity in the town of El Salado, vehicle traffic, nearby mining activity and, to a lesser extent, anthropogenic activity in the Obispito sector. Finally, in the Port Area, the main emission sources correspond to vehicle traffic on nearby unpaved roads, wind erosion in uninhabited sites and/or areas with material removal, and presumably the presence of marine aerosols.

Regarding the baseline concentrations of PM10, in the Mine-Plant Area, the values of the annual and 24-hour PM10 standard (Supreme Decree No. 12/2021) were not exceeded. The concentrations of PM10 were lower at the Mine station, which is explained by the fact that this is a rural sector with main source of emissions associated with vehicle traffic on unpaved roads. The highest PM10 values in the area were observed at the Diego de Almagro Station. This can be explained based on its location near the sectors of greatest urban activity in the city. On the other hand, the Storage Station records were lower than the Diego de Almagro station, which could be explained by its location (southern limit of the city), and the wind pattern in that area associated with the Salado River valley (see Figure 20-2 for these locations).

20.2.7 Water Resources

Surface and groundwater resources were characterized quantitatively and qualitatively. Both surface runoff (flow rates) around the Project Area and precipitation were determined as important groundwater recharge variables.

Hydrochemical parameters representing baseline quality of surface and groundwater in the study area were identified. The surface water and groundwater quality were characterized according to the quality criteria defined by the irrigation standard, NCh 1333/Of.78 (NCh. 1333/Of.78, Water Quality Requirements for Different Uses). Additionally, the interpretation of both surface and groundwater quality was developed considering parameters of environmental relevance for mining projects.

Based on the results by zone, most of the monitored points show a pH within the normal parameters indicated in NCh 1333/Of.78.

Regarding electrical conductivity (EC) and total dissolved solids (TDS) in surface waters, the areas with the highest measured values are the Salado River, followed by the Mina sector, Diego de Almagro Oriente, Depósito and finally the sector with the lowest EC and TDS is the Quebrada Angostura.

The El Salado River, approximately 8 km north of the open pit and the TSF, is the only surface waterbody with permanent pluvial runoff in the study area. The main headwaters of the El Salado River originate in the Vegas de Vicuña, at the western foot of Doña Inés Hill. It continues downstream to the town of Diego de Almagro, where the river waters infiltrate considerably, and then surface near the town of El Salado. Historically, El Salado River was used to transport tailings from the El Salvador mining process to the Chañaral coast.

Among the occasional flows is the Angostura stream, located between the Jerónimo hill and the Santo Domingo Mountain range, which feeds into the Salado River. Upstream, the Chañaral Alto stream flows into it, adjacent to Chañarcito hill.

The study area has a normal desert climate, characterized by a very dry atmosphere, a significant lack of rainfall and marked daily temperature fluctuations. The average annual temperature is around 17°C. Average annual precipitation in the study area ranges between 10 and 25 mm, while annual evaporation is around 2,500 - 2,700 mm.

The basins and sub-basins of interest, which are contributors to the project study area, considering the works projected for the Mine-Plant Area, are:

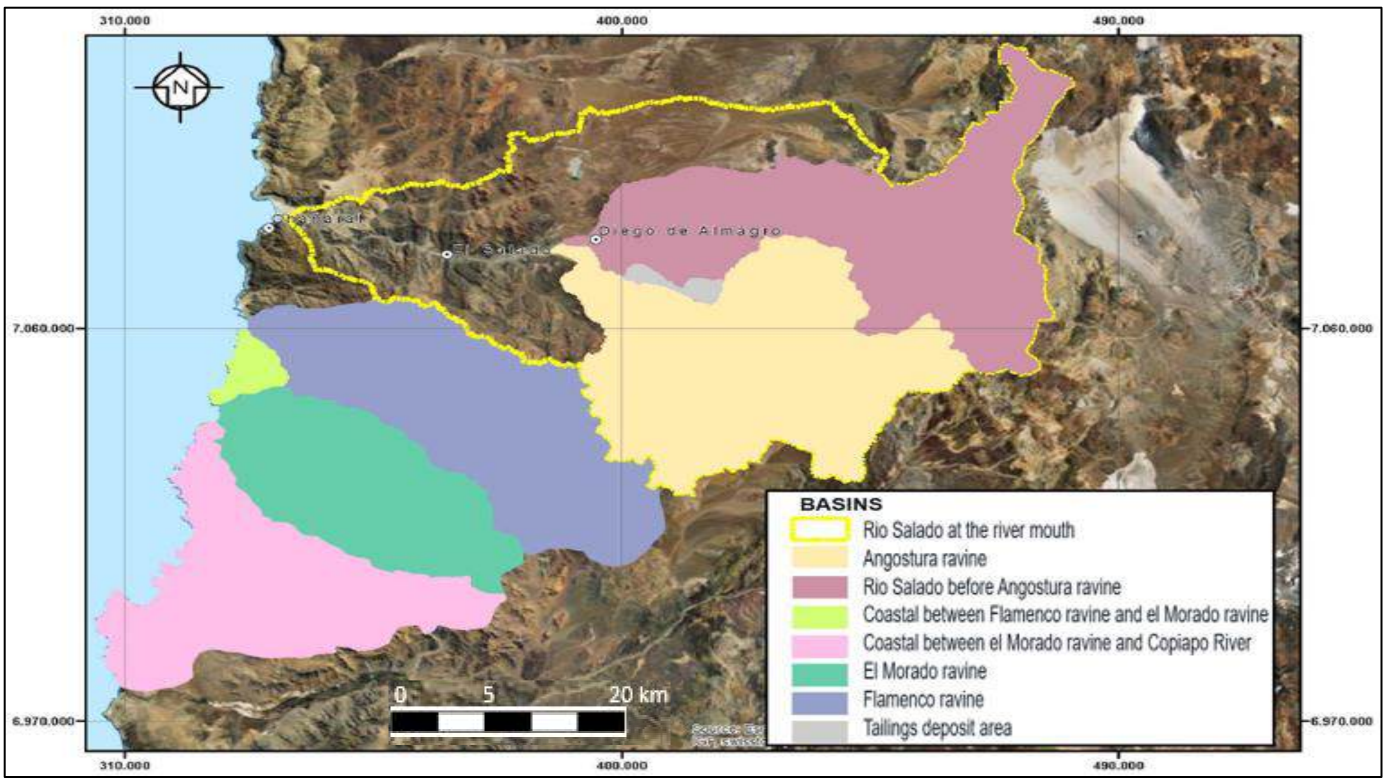
- Salado River before it joins with Angostura Creek (SAQA).
- Angostura Creek (QA)
- Tailings deposit (which is part of the Salado River basin before it meets Angostura Creek).

The following watersheds were considered for the generation of information for the baseline:

- Salado River at its mouth
- Obispito Creek
- El Morado Creek
- Coastal nearby Obispito
- Coastal between El Morado and Copiapó River.

Figure 20-3 shows the water resources study area considered and the various watersheds.

Figure 20-3: Water Resources Study Area



Source: Figure prepared by Project Santo Domingo Environmental Impact Study, 2013.

20.2.7.1 Hydrology and Precipitation

The basins studied are geographically located in one of the driest areas of the country. The Project is located in one of the most arid areas of the country, with high solar radiation, evaporation rate, and saline concentration in the soils.

Rainfall is occasional and irregular, and only occurs in some years during the winter period. As a result, the sector has no surface runoff, with the exception of the Salado River, which is the only river with seasonal runoff in the study area. According to the isohyet plan, the average annual precipitation in the basins considered varies between 12 and 40 mm. The highest precipitation occurs in the winter months (between June and August). The annual evaporation is around 2,500 - 2,700 mm.

Rainfall statistics and DGA publications were used to characterize precipitation in the study area, which allowed an adequate analysis of the hydrology for the sector.

Annual precipitation was estimated using isohyet curves, which were constructed using data from DGA stations (Chañaral, Caldera, Copiapó, Llanta Retén, Pueblo Hundido, Inca de Oro, El Salvador, Potrerillos, Las Vegas and Pastos Grandes). Using these same curves, different transposition factors were estimated for the ravines of interest, which allowed estimating the monthly precipitation in each study basin.

The maximum daily precipitation for the 10-year return period was estimated using the maximum daily precipitation isohyet curves from the publication "Precipitaciones Máximas en 1, 2 y 3 días" (DGA, 1991). The values obtained for each basin analyzed are presented below (Table 20-1).

Table 20-1: Summary of Maximum Daily Rainfall Per Basin (mm)

Basin	Maximum precipitation (mm) for ten-year return period
La Angostura ravine	29.7
Tailings dam (complete basin)	19.0
Salado river before it joins with La Angostura ravine	27.0
Flamenco ravine	21.1
El Morado ravine	20.0
Coastal between Flamingo and El Morado	20.0
Coastal between El Morado and Copiapó river	30.0

Source: Table information from Project Santo Domingo Environmental Impact Study, 2013.

The study area does not have fluviometric stations that allow direct characterization of the local fluviometry. For this reason, typical methods were used for flow estimation in basins without fluviometric control.

Based on annual precipitation and PMP values, the maximum instantaneous flows and respective volumes were obtained using the Synthetic Unit Hydrograph (SUH) method for different return periods, and for the probable maximum flood (PMF), considering the direct tailings deposit basin and the entire basin, considering a duration of 24 hrs. These values are presented in the following table (Table 20-2).

Table 20-2: Summary Values HUS - Complete Basin

Return Period (years)	Maximum Flow (m^3/s)	Overflow volume (m^3)
T=1000	9.89	624.103
T=500	6.71	423.274
T=100	3.50	221.049
T=50	2.21	139.342
T=25	1.55	98.147
T=20	108	68.306
T=2	0.31	19.825

Source: Table information from Project Santo Domingo Environmental Impact Study, 2013.

20.2.7.2 Hydrogeology

The Project is located within the Chilean Iron Belt (CIB), a metallogenic zone that extends between La Serena and Taltal. The predominant geological formation in the area is the north-south oriented Atacama fault. This fault controlled the mineralization of several CIB deposits in the area. The geological units (UG) that occur in the study area have been grouped into two main groups: Rock Units; and Semi and Unconsolidated Deposit Units. These groups are subdivided according to their origin and age. Refer to Chapter 7 for a detailed geologic description of the Project.

In general, low permeability values are estimated for the rock units, of the order of $7 \cdot 10^{-5}$ m/day in some cases, a value that increases considerably in the fracture zones of the same, reaching values of the order of 1 m/day. Regarding the semi-consolidated and unconsolidated deposit units, a relevant sub-unit in the study area is defined by the Atacama Gravels.

Six hydrogeological units were identified, grouped into three macro units. The aquifer systems are mainly found in the hydrogeological units linked to new and old alluvial deposits (UH-1). The geological unit corresponding to the Atacama Gravels geological unit (UH-2), is formed by sedimentary rocks and semiconsolidated sedimentary deposits that are partially saturated but with low productive potential. There are reports that indicate the existence of water in some areas of the lower basement unit (UH-3), associated with fault systems, whose productive potential is estimated to be low.

The water levels recorded in the observation wells indicate a stable level over time with no seasonal behavior, so it can be assumed that almost all the precipitation in the area evaporates. Recharge to the aquifers is very low, and therefore groundwater has a long residence time.

Groundwater flows in the plain NE of the Santo Domingo Mountain range have a general direction from southeast to northwest. In the rest of the sectors, groundwater flows follow approximately the same direction as the streams that contain them (Angostura stream and Salado River).

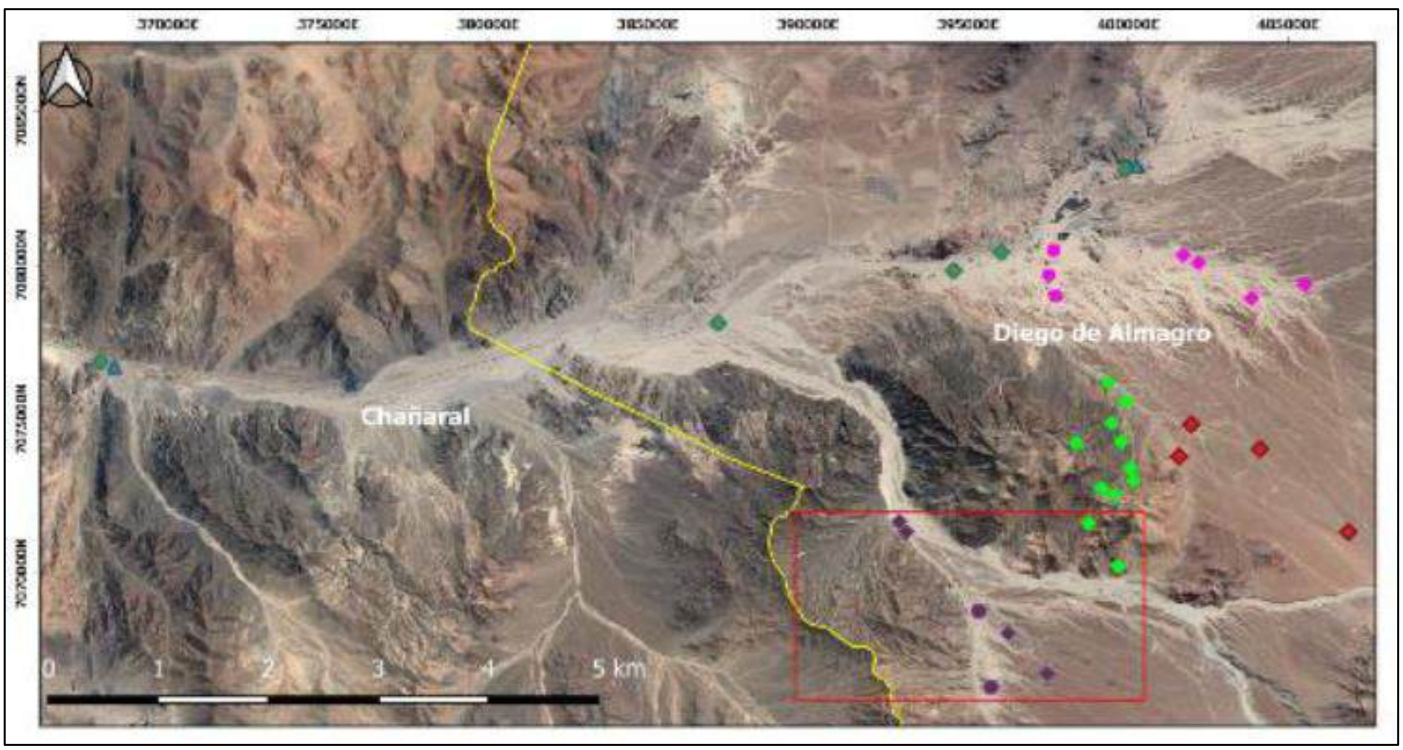
20.2.7.3 Hydrochemistry

Surface water and groundwater in and around the Project area has been monitored since 2013. A chemical characterization survey of surface and groundwater has been developed and, for both cases, temporal and spatial trends have been identified. The sample stations were ordered according to their geographic location within the Project, identifying five sectors designated as follows:

- Angostura Creek Sector
- Deposit Sector
- Mine Sector
- Diego de Almagro East Sector
- Salado River Sector.

Figure 20-4 shows the monitoring locations, each sector in a different color: Angostura Creek (purple), Deposit (red), Mine (light green), Diego de Almagro East (pink) and Salado River (dark green). The Mine and Deposit Sectors only have groundwater monitoring locations.

Figure 20-4: Surface Water and Groundwater Monitoring Locations



Source: Figure prepared by SERPRAM, Water Quality Monitoring Program Santo Domingo Mine Report, December 2023.

The current water quality monitoring plan is carried out on a monthly basis for pH, temperature, groundwater level, TDS and conductivity. A full set of chemical parameters, including oils, hydrocarbons, metals, sulphate, nitrate, fluoride, QOD, BOD, cyanide, among other parameters, are tested and reported biannually (May and November).

Also, cations and anions, such as sodium, calcium, magnesium, potassium, chlorides, sulphates, nitrates, carbonates and bicarbonates, as well as specific conductivity have been considered for the analyses.

The temporal analysis of each of the parameters and for each of the sectors defined above, showed that for the Angostura, Diego de Almagro Oriente, Deposit and Mine streams, most of the values present a temporal stability that can be seen in each of the fieldwork campaigns carried out.

On the other hand, the waters of the Salado River show high temporal variations in the quality parameters analyzed, which is mainly because this river is highly impacted by human activities located both upstream and downstream of Diego de Almagro.

In general terms, both surface water and groundwater in the Salado River, Diego de Almagro Oriente, Angostura Creek, Mine and Deposit sectors are grouped into two large groups. The first is related to the waters associated with the Project area (Angostura Creek Sector, Deposit Sector, Mine Sector, Diego de Almagro East Sector), which are of largely sulphate type composition. The second group is made up of the surface and ground waters of the Salado River sector, which are characterized by being of the Chloride-Sodium type, since they have a high saline composition (Na and Cl).

In general terms, the groundwater in the study area presents groups of waters with a temporally and spatially stable composition, presenting slight variations in the lower part of the Angostura Creek and in the upper part of the Diego de Almagro Oriente sector.

On the other hand, according to what was observed in the isoconcentration maps, particularly with the isoconductivity map, it is possible to establish the existence of two predominant subsurface flows in the sector. The first one corresponds to the Angostura Creek and the second one originating from the southeast between the Diego de Almagro Oriente and Mine-Deposit sectors. In both cases, the high specific conductivity provided by the Mina sector decreases due to dilution, when it meets the two predominant subsurface flows. The conductivity in both flows begins to increase as they approach the Salado River.

In addition, the information provided by the sulfate and chloride isoconcentration maps reveals the impact of the quality of the Salado River on the aquitard in the Project area, influencing the quality of the latter. An increase in both sulfate and chloride in the vicinity of the Salado River is observed. Lower concentrations of these two parameters are observed towards the southeast of Diego de Almagro.

It should be noted that groundwater concentrations of the parameters measured in the project area (cations, anions and specific conductivity) are much lower than those of the surface water and groundwater associated with the river. Values are much lower than those of the surface water and groundwater associated with the Salado River.

From the point of view of water quality, two major water systems are evident; one associated with the project area with no evidence of anthropogenic influence or alteration and another of very poor quality indicating an evident anthropogenic contamination corresponding to the Salado River system (almost all parameters above the Chilean irrigation standard), which can be associated with the mining activity that has historically developed in the sector.

20.2.8 Marine Environment

For the marine physical environment, wind regime, tides and currents were characterized. For the marine chemical environment, the quality of water and subtidal sediment was characterized, while for the biological environment, the intertidal benthic and subtidal benthic communities of hard and soft bottoms, planktonic communities, avifauna, and marine mammals associated with the study area of the Santo Domingo Project were studied.

20.2.8.1 Marine Physical Environment

In general terms, wind intensity was relatively low with a monthly average between 3.2 m/s (summer) and 2.4 m/s (winter). Regarding the prevailing winds, in both periods these blew from the SW or WSW associated with good weather conditions, with maximum magnitudes between 11.6 m/s (summer) and 9.8 m/s (winter).

A pattern with diurnal differentiation is distinguished, controlled by solar radiation and the topography of the sector. In the early morning, weak land-sea circulation (land breeze) or calm conditions predominate, and in the afternoon, sustained land-sea circulation (sea breeze) is common, with average magnitudes between 6.0 m/s and 8 m/s, but which dies out at sunset.

In summary, the daily variation is as follows: periods of calm or weak wind in the early morning and maximum intensity in the afternoon, declining at sunset.

A harmonic and non-harmonic analysis was carried out based on month-long records in order to define the tidal regime in the project area.

For the tidal regime, both in winter and summer, a mixed semi-diurnal regime was distinguished, with two high and two low tides in a lunar day, but with diurnal inequality. This inequality affected the high tide more than the low tide, with a monthly mean between 0.29 m (summer) and 0.35 m (winter).

No significant local anomalies were observed. In this sense, the tide has a typically regular and predictable behavior, since it responds mainly to the astronomical wave that originates as a compensation of the mean sea level, which has been altered by the action of the stars and altered by typical local factors, such as the depth and shape of the coast.

For the current regime, the results show a circulation pattern modulated by the wind regime, highly rotating, with flows and counterflows, entering and leaving the bay, in E and W direction, respectively.

Three circulation layers can be distinguished:

- The surface layer, with a marked response to the daily wind cycle. Sustained winds from the third quadrant induce the entry of water into the bay in an easterly direction, while weak winds favour a more rotational circulation, but generally out of the bay, in a westerly direction. Residual transport is to the NW in summer and to the SE in winter, which is due to the difference in wind regimes between the two periods.

- In summer, the intermediate water layer behaved in a dissimilar way to the surface water layer, since when the surface current enters the bay, the current of the intermediate layer leaves the bay in the opposite direction to that observed at the surface. It is postulated that the dynamics of this layer responds to a mass balance, to compensate for the outflow and/or inflow of water that occurs in the surface layer. The residual transport is verified towards the SW, so it is postulated the existence of a hurly circulation turn, as an average condition.
- The circulation of the bottom layer registers greater variability in the E-W direction than that reported in the upper strata, where there is a greater effect of the wind. Due to the above, the bottom layer shows a slight residual transport to the NW in both summer and winter, but close to the null value, 0.2 cm/s (summer) or 0.4 cm/s (winter).

Regarding the magnitudes of the currents reported, the relatively protected configuration of the study area, would condition a low intensity dynamic pattern in the intermediate and bottom water layer. Meanwhile, the surface water layer registers magnitudes in the range of moderate intensity.

For the swell regime, the modal condition of the waves is around a spectral period of the waves between 14 and 15 seconds, significant average heights between 0.63 m (summer) and 1.03 m (winter) and WNW direction. This last direction corresponds to the direction adopted by the ocean swell as it propagates towards the study area, since as the wave train advances towards the coast, it aligns with the isobaths of the sector.

Significant wave heights were comparatively higher in winter, which is characteristic of the seasonal wave regime.

20.2.8.2 Marine Chemistry Environment

Regarding baseline marine water quality, two monitoring events were carried out during 2013, one during summer and one during winter. The study area shows no evidence of contamination based on pH, transparency and fecal colimetry. The results from the monitoring events were compared with the limits established by the Chilean irrigation standard and by the Primary Quality Standard DS 144/2009 for marine waters. For marine waters, it was observed that these parameters showed values that met the regulatory standard and was indicative of the absence of contamination of the marine water column.

With respect to subtidal sediment in the study area, relatively low concentrations of trace metals were reported below the internationally accepted environmental quality criteria for the observation of toxic effects on biota. This indicates baseline subtidal sediments of good quality.

Al and Fe contents in subtidal sediments were low and below the reference values reported in other localities in the region. The levels of organic matter in sediments were reported in the moderate (summer) to low (winter) concentration range.

20.2.8.3 Marine Biological Environment

Regarding intertidal benthic communities, of the total number of species recorded, 19 were observed in summer and 20 were observed in winter, and two showed pelagic behaviour.

The species with the highest representation and abundance (greater than or equal to 4%) were as follows:

- *Chromis crusma* (burrito or castañeta, 46% summer and 67% winter)
- *Scartichthys viridis* (drunkard, 7% in summer and 4% in winter)
- *Girella laevifrons* (baunco, 6% in winter)
- *Aplodactylus punctatus* (jerguilla, 4% in winter)
- *Pinguipes chilensis* (Pinguipes chilensis, 6% in summer and 5% in winter)
- *Cheilodactylus variegatus* (bilagay, 6% in summer and 4% in winter).

The zooplankton community consisted mainly of chitinous zooplankton, herbivorous copepods of wide distribution and abundance in the Humboldt Current System, representing between 75% and 97% of the abundance.

The crustacean community showed a higher abundance of larval stages of cirripedes and decapod crustaceans in summer than in winter, coinciding with the reproductive cycles of these organisms.

The phytoplankton community was dominated by the diatom group, with low specific richness in summer and high in winter.

20.2.9 Flora and Vegetation

In general, most of the Mine-Plant Area is denuded and devoid of flora and vegetation. When present, it occurs at the bottoms of ravines, depressions, or alluvial cones, where it can form homogeneous units of vegetation with a coverage of less than 25% consisting typically of less than 11 species.

Between El Salado and Obispito (approximately 60% of the pipeline route) frequent vegetated areas can be observed. Between the Mine-Plant Area and El Salado (40% of the total route) it is mainly denuded or unvegetated. Where present, it is observed in creek bottoms, depressions or alluvial and/or colluvial cones, where they rarely form more than 1% coverage, typically consisting of less than 11 species.

The total flora detected in the study area was reported to be 107 species, of which 92.6% were native. Regarding the vegetative form, 48 were shrubs, 45 were herbs, 12 were succulents and two were arboreal.

On the other hand, the vegetation of the Port Area corresponds to a scrubland of low height (less than 1 m) and coverage that can reach 75%. Grasslands were scarce, mainly associated with land where the surface material was made up of low cohesion sands. In the Port Area, 35 species were observed, of which 97.2% were native. Regarding the vegetative form, 22 were shrubs, 10 were herbs and three were succulents.

The vegetation of the Power Line Area corresponds to a scrubland of low height (less than 1 m) and coverage that can reach 50%. The presence of grasslands was very scarce and associated with sandy soils. In the Power Line Area, the total flora detected amounted to 46 species, of which 95.6% were native. Regarding the vegetative form, 25 were shrubs, 16 were herbs and 5 were succulents.

In the surveyed area, the most frequent species are *Heliotropium floridum, Tetragonia maritima, Skytanthus acutus, Spergularia arbuscula, Nolana divaricata, Atriplex clivicola, Frankenis chilensis, Heliotropium pycnophyllum*, *Cristaria glaucophylla*, *Oenothera coquimbensis, Helenium atacamense, Copiapoa spp, Maihueniopsis glomerata* and *Puya boliviensis*, among others.

The uniqueness of the species present in the study area is defined according to their conservation status, their identification as xerophytic species and their regional endemism. Table 20-3 shows the species identified in the Project study area that have a conservation status category according to local regulations (which follow the IUCN conservation categories), and/or are xerophytic, and/or are endemic to the Atacama Region, which corresponds to a total of 29 species of flora. This table also shows the Project Area in which it was identified and the associated cartographic units.

Monitoring and management measures for flora and vegetation can be found in Section 20.3 of this report.

Table 20-3: Species Identification in The Study Area

Species	Category (1)	Xerophytic	Endemic	Area
Aphyllocladus denticulatus	Almost threatened	No	No	Pipeline
Copiapoa cinerascens	Rare	Yes	No	Pipeline and LTE
Copiapoa echinoides	Almost threatened	Yes	Yes	Pipeline
Copiapoa calderana	Vulnerable	Yes	Yes	Pipeline and Port
Copiapoa marginata	Vulnerable	Yes	Yes	Pipeline
Eriosyce aurata	Vulnerable	Yes	No	Pipeline
Eriosyce rodentiophila	Vulnerable	Yes	No	Pipeline
Eriosyce sociabilis	In critic danger	Yes	Yes	Pipeline
Eriosyce villosa	Vulnerable	Yes	No	Pipeline
Eulychnia acida	Minor concern	Yes	No	Pipeline, LTE and Port
Gypothamnium pinifolium	Almost threatened	No	No	Pipeline
Maihueniopsis glomerata	Almost threatened	Yes	No	Pipeline, LTE and Port
Oxalis caesia	Vulnerable	No	No	Mine-Plant
Prosopis chilensis	Vulnerable	Yes	No	Mine-Plant
Prosopis flexuosa	Vulnerable	No	No	Mine-Plant
Geoffraea decorticans	Uncategorized	Yes	No	Mine-Plant
Salix humboltiana	Uncategorized	Yes	No	Mine-Plant
Schinus areira	Uncategorized	Yes	No	Mine-Plant and Pipeline
Pyrrhocactus eriosyzoides	Vulnerable	Yes	No	Pipeline
Puya boliviensis	Vulnerable	Yes	Yes	Pipeline, LTE
Senecio microtis	Vulnerable	No	Yes	Pipeline
Cristaria integerrima	Uncategorized	Yes	No	Pipeline
Euphorbia lactiflua	Uncategorized	Yes	No	Pipeline, Port and LTE
Oxalis gigantea	Uncategorized	Yes	No	Pipeline and LTE
Quillaja saponaria	Uncategorized	Yes	Yes	Pipeline

Species	Category (1)	Xerophytic	Endemic	Area
Skytanthus acutus	Uncategorized	Yes	No	Pipeline, Port and LTE
Heliotropium floridum	Uncategorized	No	Yes	Pipeline, Port and LTE
Heliotropium glutinosum	Uncategorized	No	Yes	Pipeline
Nolana patula	Uncategorized	No	Yes	Pipeline

(1) Local sources that list species under conservation categories: Supreme Decrees N°151/2007, N°51/2008, N°50/2008, N°23/2009 N°33/2012, N°41/2012, N°42/2012, N°19/2013 and N°13/2013; The Red Book of Terrestrial Flora of Chile; Bulletin 47 of the National Museum of Natural History.
Source: Table information from Project Santo Domingo Environmental Impact Study, 2013.

20.2.10 Wildlife

The presence of 68 wildlife species were identified throughout the study area, of which 29 were found in the Mine-Plant Area, 29 in the Pipeline Area, 43 in the Port Area and 26 species in the LTE Area. The environments for the fauna present in the study area are seven and correspond to: Chañarcito, Interior Scrubland, Interior Transitional Scrubland, Coastal Scrubland, Coastal Edge, Denuded Zone and Sea.

Same as with flora and vegetation, the uniqueness of the species present in the study area is defined according to their conservation status and their regional endemism, according to local regulations (Hunting Law and the Supreme Decrees that follow the IUCN conservation categories). Of the total number of species found, 15 are in some category of conservation, of which three species are Endangered (*Spalacopus cyanus* in the central zone of Chile and *Tursiops truncatus*); one is Inadequately Known (*Pseudalopex griseus*); five Rare species (Li*olaemus atacamensis, Liolaemus velosoi, Liolaemus bisignatus, Liolaemus platei and Homonota gaudichaudii in the northern zone of Chile*); five Vulnerable species (*Callopistes palluma, Microlophus atacamensis, Pelecanoides garnotii, Phalacrocorax bougainvili, Lama guanicoe and Lontra felina*) and one species of Least Concern (*Otaria flavesceens*).

Monitoring and management measures for flora and vegetation can be found in Section 20.3 of this report.

20.2.11 Land Use and Territorial Planning

Regarding to territorial planning, the Mine-Plant and Pipeline Areas incorporate localities zoned under Regulatory Plans; these correspond to Diego de Almagro and El Salado. The Pipeline Area and the Power Transmission Line Area incorporate areas zoned under the Coastal Intercommunal Regulatory Plan.

Regarding land use in the study area, the analysis of this component indicates that the land is classified as barren and destined for other non-usable uses. A smaller portion, close to 0.2%, corresponds to land used for cultivation, fallow and rest; this classification was only identified in the Diego de Almagro Commune.

Water rights in the study area are held mostly by mining companies.

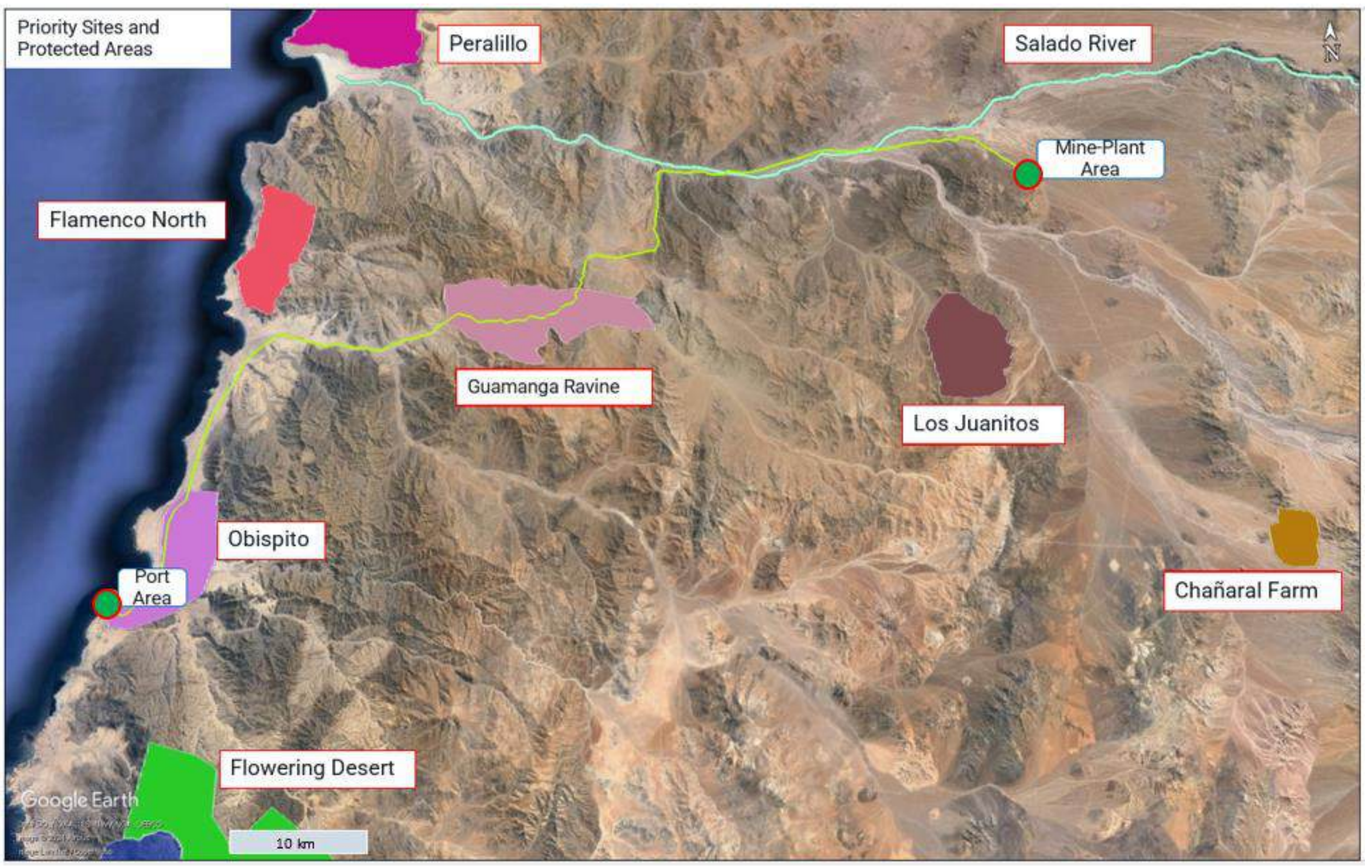
In the coastal zone of the study area, there are the benthic resource management areas of Punta Roca Baja, Torres del Inca, Caleta Obispo and Punta Obispito, the latter being the closest to the port area, located 4.3 km from it.

20.2.12 Priority Sites and Protected Areas

The priority sites for biodiversity conservation in the proximity of the project are the following: Salado River (*Río Salado*), Peralillo, Flamenco North (*Flamenco Norte*), Guamanga Ravine (*Quebrada Guamanga*), Los Juanitos, Chañaral Farm (*Finca Chañaral*) and Obispito. As can be seen in Figure 20-5, the Guamanga stream is directly crossed by the pipeline route and the Obispito site is adjacent to the port area. A Although these sites generate environmental restrictions, they do not limit activities within them.

The Flowering Desert (*Desierto Florido*) has a higher degree of environmental restriction as it is a national park, however, the closest distance to it is 10 km from the port area.

Figure 20-5: Priority Sites and Protected Areas



Source: Ausenco 2024.

20.2.13 Human Environment

The closest city to Santo Domingo is Diego de Almagro located approximately 5 km northwest of the mine and plant site areas. Further west, halfway along the pipeline route, is the town of El Salado. For the port area, the closest city is Obispito, a very small coastal locality located approximately 2 km to the west.

In the past, the Province of Chañaral and its municipalities have undergone an accentuated process of depopulation as a consequence of the local economic-productive dynamics, highly dependent on the extractive mining and marine industries. In fact, the successive cycles of mining expansion and contraction have been key in the generation of moments of economic development and depression in the area, which have determined that this province has high levels of poverty and social vulnerability during periods of mining decline and closure of mining operations.

At the present time, there is evidence of a change in the depopulation trend experienced by the province in previous decades, at least in most of its urban centres, except for the city of El Salvador, which, as the camp site for El Salvador mine, is directly affected by the decline and depletion of the mine. In Diego de Almagro, El Salado and Chañaral, on the other hand, a process of recomposition and even growth of the local population can be observed, with current evidence of immigration processes and pressure on services, housing and infrastructure that local governments are striving to address.

The same is true for the commune of Caldera, where a sustained growth of its population has been observed for two decades now, a trend that has been reinforced by the generation of new productive developments linked to aquaculture, port infrastructure and tourism.

Nevertheless, the communes of Chañaral and Diego de Almagro and, to a lesser extent, Caldera, are still highly dependent on mining activity and therefore vulnerable to the cycles of this industry. This industry is affected by phenomena such as the depletion of veins and deposits, the decline in the price of certain metals, lower ore grades produced by small-scale artisanal mining, global economic mining crises, global and local economic crises, as well as the scarcity and high price of energy.

Parallel to this, in the study area there is a shortage of other productive enterprises and activities that can provide alternatives for economic development. Factors such as the natural aridity of the area and the lack of water resources impede agriculture-related activities. In addition, there is weak industrial and tertiary industrial development and the scarcity of technical and professional skills that could support the development of non-extractive productive activities.

There is particular concern about water resources, currently limited to a minimum for human consumption, which does not allow for urban expansion and threatens productive activities. There is also a concern for the sustainable exploitation of marine resources, which constitute an important source of subsistence for the population in coastal areas and a complementary activity for the most vulnerable families. In this area, there is a trend towards more rational organisation and exploitation of marine resources by a growing number of fishermen's and fish gatherers' unions and groups, which seek to protect their sources of work and demand greater control to prevent the illegal exploitation of marine resources.

20.2.14 Archaeology

During 2011, 2012, and 2013, archaeological surveys were carried out in the area of the future site of the Santo Domingo Project. This work was aimed at identifying archaeological sites protected by Law No. 17,288 (National Monuments Law) and to assess those areas of greater and lesser heritage relevance with a view to evaluating the best alternatives for the location of the future Project.

A total of 262 archaeological sites were identified within the study area, classified in 12 different types of findings, such as stone accumulations, structures, lithic and ceramic concentrations, isolated fragments, shell heaps (conchal in Spanish), and trails, among others, covering Pre-Hispanic, Historical and Recent periods.

The inland sites, mostly identified as trails, wagon tracks, old railway lines and other evidence of transport routes, correspond to evidence of the historical human activities between the late nineteenth and mid-twentieth centuries, mostly related to small scale independent mining activities (*pirquineros* in Spanish) and to the droving of cattle and other goods (*arriería* in Spanish), an activity carried out as a small scale economic activity by non-indigenous and indigenous people. Archaeological sites related to this last activity were determined by the landscape, which is why it is possible to find a significant number of structures, both rectangular and circular, as well as windbreaks in flat areas, which made it possible to take refuge while passing through the Atacama Desert. On the coast, most of the sites found correspond to shell heaps of pre-Hispanic nature, associated with earlier coastal human settlements subsisting mainly on marine resources such as fish, shellfish and shore resources.

Conservation state of the archaeological sites varies with the better-preserved ones mostly located far from main or secondary roads. The conservation state diminishes as they get closer to Route 5, generally due to the anthropic intervention of the sites, as evidenced by the tracks of motorised vehicles, as well as the effect of occasional watercourses. In addition, some sites presented a poor degree of conservation, mainly due to the anthropic action observed in vehicle tracks near the sites, the illegal excavation of shell heaps and the deterioration of the sites due to environmental conditions. Two of these sites were very poorly preserved, as they were illegally excavated and impacted by the widening of Route 5.

Of the 262 sites identified, 27 will be directly under the project footprint and therefore impacted or lost because of project construction:

- Sixteen are located in the Mine-Plant Area, corresponding to seven structures, three windbreaks, four trails and old railway lines, one lithic concentration and one mine shaft.
- Eight are located in the Pipeline Area corresponding to five shell heaps, one structure, one trail or old railway line and one lithic concentration.
- Three are located in the Port Area corresponding to shell heaps.
- Capstone's current permits and Chilean regulation currently have requirements to manage the above sites by means of avoidance or mitigation. Cultural resource management plans will be developed for use by Capstone employees and contractors prior to land disturbance near these sites. An archaeological chance find procedure will be in place.

20.2.15 Visual Landscape

The purpose of the landscape baseline study was to identify, characterise and assess the landscape in which the Santo Domingo Project is located. The specific work consisted of defining the landscape units, characterising, and evaluating the quality and visual fragility of the landscape of each unit.

In general terms, at a South American level and from a biogeographical point of view, the study area is part of the so-called Xeromorphic Ecosystems, whose main characteristic is extreme aridity (high diurnal temperatures, wide thermal oscillations, minimal and cyclical precipitation), specifically the study area is part of the Tropical Perarid Desert Ecoregion, which extends from the rocky coastline to the Andean foothills located above 2,500 m of altitude and includes an intermediate zone called the desert plain.

On a national scale, the area corresponds to the Desert Region, which stretches from the northern limit of the country to the Elqui River in the Coquimbo region, constituting the southernmost section of the desert on the Pacific coast of South America. Although the western limit is the oceanic coast, the study area is mainly located in an inland desert, with an average altitude of approximately 1,500 masl, encompassing the steep coastal cliffs, the mountain ranges of the Cordillera de la Costa, the large inland depressions, the western slopes of the Andes Mountains and the coastal plains.

Ten Landscape Units, differentiated by different types of attributes, are recognised in the study area. To assess the visual quality or aesthetic value of each landscape unit, the elements defined in the “environmental impact assessment manual: concepts and basic background” were considered. Morphology of the sector, presence of vegetation, whether there is anthropic action, among others, were considered.

To characterise the visual fragility of each landscape unit, presence of slopes, size and shape of the visual basin, accessibility, among others, were considered as assessment elements. A summary of the characteristics of these landscape units is presented in Table 20-4.

Table 20-4: Summary of Visual Landscape Results by study area

Study area	Landscape unit	Visual quality (value)	Fragility (value)
Mine Plant area	Diego de Almagro Gravels of Atacama Sierra Santo Domingo	Medium – low (1.6) Low (1.1) Medium (1.9)	Medium (2.1) Low – Medium (1.5) Medium – Low (1.8)
Pipeline area	El Salado valley El Salado town Ravine Las Animas Ravine Flamenco – Guamanga Coastal border	Medium – Low (1.7) Medium – Low (1.7) Medium – Low (1.6) Medium – Low (1.7) Medium – High (2.3)	Medium (1.9) Medium (2.0) Medium (1.9) Medium (1.9) Medium – High (2.3)
Port area	Obispito	Medium (2)	Medium (2)
Electrical Transmission Line Area	Totoralillo	Medium – Low (1.7)	Medium (2)

Source: Table information from Project Santo Domingo Environmental Impact Study, 2013.

In the Mine Plant Area, three Landscape Units were defined. Of these, the Sierra Santo Domingo Landscape Unit is the one with the highest value of Visual Quality (Medium Quality), a unit that has as one of its main characteristics the Chañarcito Oasis, which is an atypical visual landmark within the mostly arid unit and without other types of characteristic visual landmarks. It should be noted that the Chañarcito Oasis is perceptible only within the visual basin that contains it. The unit with the lowest Visual Quality within the Mine-Plant Area is the Gravels de Atacama Unit (Low Quality), which is a unit made up of visual basins of arid plains and wide vision. In terms of Visual Quality values, for this area, the most fragile unit would be the Diego de Almagro unit, with a medium rating, which acquires a traditional value for being part of an area with historical relevance in Chile's mining activity. The lowest Fragility value is obtained by the Gravels de Atacama unit with a Low - Medium rating, this unit consisting of arid plains typical of the desert of Northern Chile.

In the Pipeline Area, five Landscape Units were defined, where the one with the highest Visual Quality rating is the Coastal Border Unit, mainly due to the presence of the sea. The Unit with the lowest Visual Quality corresponds to Pueblo El Salado and Las Animas Ravine, which could be justified by being areas with little chromatic contrast, arid, and common in the region. Regarding the Visual Fragility of the Landscape Units of the Pipeline Area, these are maintained at average values, the most fragile being, due to its physical and visual The Coastal Border Unit is the most fragile due to its physical and visual accessibility characteristics.

In the Port Area, only one Landscape Unit was defined, the Obispito Unit, which has a medium value for Visual Quality and Fragility. Its main characteristics correspond to irregular visual basins, most of them with limited visual amplitude. It is a unit where rocky areas predominate and where there is no anthropic action. The predominant visual landmark in this unit is the Pacific Ocean, which contributes positively to visual quality.

The Totoralillo Unit was defined in the Power Transmission Line Area, with typical characteristics of the northern coast of Chile, with aridity and low chromatic variation. Its Visual Quality was rated as medium-low, which is consistent with the anthropic effect exerted by the port of Totoralillo, as well as the existing power transmission line. The Visual Fragility of this area is of medium value.

20.3 Environmental Monitoring and Management

In accordance with letter f) of Article 12 of Law N° 19,300, letter k) of Article 18 and Article 105 of D.S. N° 40/2012 (SEIA Regulation), a DIA needs to present a Monitoring Plan, which aims to ensure that the relevant environmental components that were the subject of environmental assessment evolve as projected. For the Santo Domingo Project this plan includes not only the environmental components associated with significant impacts, but also the monitoring of air quality and water quality parameters to confirm the absence of additional impacts not considered previously.

This plan specifies the environmental component that will be subject to measurement and control, the associated environmental impact and measures, the location of the control points, the parameters that will be used to characterize the state and evolution of the component, any limits allowed or committed, the duration and frequency of the monitoring plan for each parameter, the method or procedure for measuring each parameter, and the deadline and frequency for the delivery of monitoring reports to the environmental authority.

Table 20-5 shows the monitoring activities established by the project, stating the associated element and its current status. These activities may be modified or supplemented as a result of an updated environmental assessment.

Table 20-5: Monitoring for Significant and Non-Significant Impacts

Item	Monitoring Description	Status
Significant impacts		
Air quality	Monthly monitoring is carried out at the 5 stations of the project (Mina, Diego de Almagro (2), El Salado and Obispito areas). In addition, when there has been work on site, the roads are wetted on a daily basis and the effectiveness of the measure is monitored to ensure that it is at least 75% effective.	Active
Flora and vegetation	Before entering a work area, environmental mitigation is carried out. Seeds are collected from native and conservation species that are to be nursed and transplanted. Cactaceae and bromilaceae are rescued and relocated with periodic monitoring until 3 years after relocation.	Active
Wildlife – low mobility species	The Project considers monitoring in the areas where low mobility species will be relocated, as reported during the implementation of the Low Mobility Species Rescue and Relocation Plan, prior to ground disturbance of the site. Likewise, monitoring will be carried out in the areas to which the low mobility species detected in the intervention area will be driven away, in order to verify the effectiveness of the Controlled Disturbance of Low Mobility Fauna Species measure, for which there is a periodic monitoring plan for up to 2 years.	Active
Cultural heritage	The Project will implement a monitoring of the archaeological sites that are close to the Project works (Unimpacted Archaeological Sites Close to the Project Intervention Area for a buffer of 50 m). At the same time, the identified sites that are close to the Intervention Area will be monitored, considering in a similar way, the 50 m buffer. The monitoring will be carried out by a professional certified by the National Monuments Council (CMN). An on-site inspection will be carried out to verify that the fences and signage are not affected by the construction activities.	Active
Non-Significant Impacts		
Wildlife - Birds	For bird monitoring, the anti-collision and anti-electrocution devices will be monitored to verify their functionality and condition and to monitor the number of birds killed by collision or electrocution. The parameters to be monitored will be nesting on pylons, number of eggs and live species, and cleanliness of the area. These parameters will be monitored along the entire power transmission line, from its connection with the Totoralillo substation to the Santo Domingo port. Boat and pedestrian surveys will be conducted to identify the presence of sites with evidence of breeding birds registered in the area with threat categories, such as the guanay (*Phalacrocorax bougainvillii*), yunco (*Pelecanoides garnotii*), and Humboldt penguin *(Spheniscushumboldti*). Three monitoring campaigns will be conducted during the early, middle and late stages of reproduction to evaluate the use of the territory during the nesting period. Censuses will be taken at these sites to determine the number of individuals and the age structure of the population during each of the visits. In the case of nest monitoring, it is necessary to conduct monthly campaigns between September and March, corresponding to the bird breeding season.	Once the construction stage commences
Wildlife - *Lama guanicoe*	Monitoring will be conducted from the beginning of the construction stage and during the first five years of the operation stage. The monitoring stations will correspond to the same ones in which indirect evidence of Lama guanicoe was recorded in the 2012, 2013 and 2014 campaigns. The direction of travel of the transects, extent of transects, sampling effort, duration of monitoring will be identified. The monitoring will be conducted by means of transects at each of these points, in such a way as to cover the area of influence and buffer.	Once the construction stage has started
Marine Wildlife - Cetaceans, marine	Although the bibliographic background confirms the potential existence of cetaceans in the study area (12 species), specifically the Obispito cove sector does not correspond to an area of cetacean concentration. However, a monitoring program for these species is committed.	Once the construction stage has started

Item	Monitoring Description	Status
mammals and sea turtles	This program will have two annual campaigns, to be carried out preferably in late spring and mid-autumn, times that coincide with the migration of large cetaceans along the Chilean coast. The surveys will include the port's maneuvering area and will extend to 25 nautical miles around the port, due to the high mobility of the large cetaceans. SERNAPESCA and the Maritime Authority will be notified if strandings or sightings of disoriented, injured, or dead individuals (cetaceans, sea lions, chungungos, sea turtles) are recorded during both the construction and operation stages. In addition, in the event that dead specimens of cetaceans and mustelids are found during the construction period, the Contractor agrees to pay for a necrological examination to determine the causes of death at a veterinary medical school specializing in wildlife. An annual report with the partial results of the monitoring will be reported to the authority. A final report compiling the information from the three years of monitoring of cetaceans, marine mammals and sea turtles will be issued to the authority upon completion of the monitoring, and a scientific publication will be prepared to report on the observations made in the Obispito Cove area.	
Marine environment - Marine column and marine sediments	To meet the objectives of the plan, monitoring of the marine environment will be implemented, covering the different components: physical, chemical and biological The monitoring will be carried out semi-annually, once in winter and once in summer, during the operation stage of the Project and statistical analysis will be used to establish the spatio-temporal variability, through the use of historical data, provided by the baseline and by the semi-annual monitoring that will be carried out for the different components, for the water column, sediments and benthic organism. The monitoring plan will include filming the evolution of the seabed and biological communities in a transect that will extend along the route of the quay, the ship loading area, and the seawater collection and brine discharge points, also consideration will be given to making films in a controlled transect.	Once the operation stage has started
Hydrobiological resources	The owner commits to monitor during the operation stage, the populations of hydrobiological species of commercial interest, through larval monitoring, in order to verify the effects of seawater suction.	Once the operation stage has started
Hydrology, hydrogeology and water quality	Monitoring will be implemented for all stages of the project at different sampling points, each monitoring well will be drilled until it is verified that the rock basement has been reached, these wells do not require maintenance for their operation, except for visual inspections. The following parameters will be measured monthly. • Flows • Water quality • Groundwater level • pH	Active
Human environment	The objective of the monitoring is to ensure compliance with the proposed environmental management measures associated with non-significant impacts on the human environment. The monitoring sites will apply to the full extent of the human environment study area during the Construction, Operation and Closure Stages. The sites to be monitored are City of Diego de Almagro;access routes used by the Project. Mine-Plant Area: • Office and Information System for Complaints and Claims. • First Aid Attention Centre Construction Camp Operation Camp. • Training rooms. • Port Area. • First Aid Attention Centre.	Once the construction stage has started

Item	Monitoring Description	Status
Noise and vibrations	The parameters to be monitored will be Sound Pressure Level (SPL) for noise and Vertical Particle Velocity (VVP) for vibrations. The sound pressure levels, generated by fixed sources, at the points of interest will be analysed with respect to the Supreme Decree Nº 38/2011, and those generated by vehicular flow, both for the daytime and night-time periods, will use the provisions of the Swiss Confederation Standard OPB 814.14. For the vertical particle velocity, the German standard DIN 4150:1979 of the German Institute for Standardisation (Deutsches Institut für Normung - DIN) shall be used. The maximum permissible value shall be the maximum value recommended for historical monuments (2.4 mm/s). The Environmental Monitoring Plan for the Noise and Vibration component, associated with the construction, operation and closure of the Project, considers in the first instance the points for which it is foreseen that the standard will be exceeded and for which the implementation of mitigation measures are contemplated, also adding the sensitive points from the point of view of populated areas and areas of fauna interest, in order to verify the prediction that these points will not be affected.	Once the construction stage has started

In addition to the monitoring indicated above, Capstone has committed, as part of the environmental permits, to mitigation and compensation measures for flora and vegetation, wildlife and cultural heritage, among other environmental components, to address the impacts on especially relevant environmental elements. Measures include the demarcation and protection or rescue and relocation of plant species under conservation and xerophytic categories and archaeological sites, rescue and relocation of low-mobility wildlife species, and the training of staff to correctly protect this flora and fauna. These measures, along with all other environmental commitments included within the environmental permits, will be stated and controlled through appropriate environmental management plans once the project has started its execution phase.

20.4 Water Management

20.4.1 Construction Water

The industrial water requirement during the construction phase will be provided by an authorized water supplier. For industrial water at the mine and plant site, a 152 mm diameter water delivery pipeline has already been constructed to the process plant site from the Water Treatment Plant near Diego de Almagro, to supply industrial water for road dust control, addition to bulk earthworks for moisture content adjustment at the plant site and TSF, and for dust control at the mine during the pre-stripping activities. For construction at the port, potable water will be used, provided by an authorised water supplier through a tie-in to an existing potable water pipeline located adjacent to the port area.

The potable water requirements during the construction phase at the mine – plant site will be provided by an authorized water supplier. A tie into the existing municipal potable water pipeline that feeds Diego de Almagro will provide a potable water supply point near the planned construction camp. This water will be used at the camp and at the concrete batch plant.

20.4.2 Process Water

The mineral processing facility will use desalinated sea water and recycled water. The current plan is to produce the desalinated water under a build, own, operate, transfer (BOOT) or build, own, operate (BOO) contract with a third party. The forecast desalinated water requirement is 366 L/s.

Two HDPE-lined open ponds for process water will be located near the process plant and will have a total capacity of 80,000 m^3. These ponds will store water reclaimed from the copper and magnetite concentrate thickeners and tailings thickener overflows and make-up from the plant desalinated water storage ponds. Recycled water from the magnetite concentrate filtration at the port will also be pumped to site after treatment to improve its quality.

20.4.3 Potable Water

During the operational phase of the project Capstone will operate potable water treatment plants that will provide desalinated water that can be used for human consumption in the mine and port areas, as well as supplementing water resources in the town of Diego de Almagro.

20.4.4 Hydraulic infrastructure

The project considered the construction of a series of natural water diversion works that will allow the diversion and return of rainwater downstream from the project facilities. These works would include:

- Surface water contour channel not contacted in tailings deposit
- Surface water diversion channel to the North of the Iris Norte pit
- Surface water diversion channels in the landfill

Hydraulic works have been designed for a return period of 50 years and verified for a return period of 100 years, which is more conservative than requirements in Chilean DS50 legislation.

20.4.5 Water Balance

A monthly water balance for the TSF that estimated the quantities of water entering, existing and being stored in the TSF was developed. The inflows included water in the tailings stream and precipitation; the outflows included recycle water to the process, evaporation from the supernatant pond and the active beach as well as seepage from the tailings and the pond; storage included water losses to the pores in the tailings and the varying volumes of surface water in the supernatant pond.

Water from the supernatant water pond will be recovered and recycled to the process throughout the operating life of the TSF. This is necessary for efficient water use and to control the size of the supernatant pond to limit evaporation and potential seepage losses. The general operating principle will be to keep the supernatant pond as small as practicable so that under a range of normal operating conditions it will overlie the geomembrane lined area upstream of the dam. The water balance has been used to estimate the limits of this pond and to establish the limits of the liner.

The rate of water recovery from the TSF to the process is predicted to be restricted by the size of the supernatant pond.

20.4.6 Water Recovery and Transport System

The water recovery and transport system from the supernatant pond includes pumping and piping to the process plant. The pond would form close to the main embankment at the northern end of the TSF. The water recovery system would consist of the following:

- A barge-mounted pump station with two Class #150 vertical pumps (one operating, one stand-by), 100 HP (75 kW) each, located at the supernatant pond. This system would be separated from the geomembrane lined surface to avoid damaging the geomembrane.
- Pumping the recovered water from the supernatant pond to a transfer tank, that would be located above the left abutment of the main embankment.
- Gravity flow from the transfer tank to the process water pond at the processing plant through pipelines.

20.5 Emissions and Waste

20.5.1 Atmospheric Emissions

From the results obtained from the emission inventories for the construction stage, it was identified that the main emission sources within Mine-Plant Area would correspond to pre-stripping activities. For the operation stage, it was determined that the main source of emission in the Mine-Plant area, both for MP10, MP2.5 and MP, will correspond to the transport of waste rock (sterile), equivalent to 51%, 39% and 57% of the total emissions, respectively.

Based on the results obtained from the emission inventories for the construction stage it was possible to identify that the main sources of emission in the Pipeline Area, Port Area and LTE Area. These sources would correspond to the activities of construction of facilities, transport of vehicles on unpaved roads and the operation of machinery and equipment. For the operation phase, the main sources of emissions are associated with vehicular activities.

20.5.2 Greenhouse Gases

According to the update of the Environmental Law (February 2024), it is necessary to include information corresponding to the carbon footprint of the Project. According to the IPPC guidelines for GHG inventories, the emissions of carbon dioxide (CO_2), methane (CH_4) and nitrous oxide (N_2O) were based on the estimated amount of fuel required for the Project approved in the base RCA.

GHG emissions were estimated for the previous project, the optimised project will generate similar levels of emissions. For the construction phase, a total GHG emission of 136,388 t of CO_2 equivalent were estimated for a period of 27 months, while for the operation phase, a total GHG emission of 3.1 Mt of CO_2 equivalent were estimated for the 18 years of operation. Estimated emissions would originate from the combustion of fuel that powers Project machinery and vehicles.

Estimation of HFCs, PFCs and SF6 were not considered as these correspond to fugitive emissions from the use of aerosols, refrigerants or other chemicals for which no data is available.

Finally, according to the update of the Environmental Law, for the future DIA, mitigation measures for GHGs should be specified in accordance with the proposed long-term climate change strategy defined for Chile.

20.5.3 Waste

The Project contemplated the construction of facilities for the management of waste (generation, storage and disposal) generated during the construction, operation and closure stages of the Project.

Regarding infrastructure, facilities for the management of domestic and assimilable solid waste, non-hazardous and hazardous industrial solid waste, and for the treatment of wastewater are considered. The facilities would include landfill, non-Hazardous Industrial waste yard, hazardous waste yard and sludge drying field, these will be utilized for treating waste from the mine-plant and pipeline areas. The project also included the construction and operation of a plant for the treatment of activated sewage sludge, the waste will remain there until it meets specified moisture criteria and then transferred to the Project's landfill. Table 20-6 shows in detail the waste management details for each kind of waste generated by Project activities.

Table 20-6: Waste of Santo Domingo Project

Type of Waste	Management
Waste rock from mining	Pit mining activities involve the generation of approximately 1.2 billion tonnes of waste rock material during the operation of the Project. The material will be transported in 290 tonne trucks and disposed of by hopper tipping, approximately 587 million tonnes will be disposed of in the South-West Dump, 69 million tonnes in the South-East Dump, 545 million tonnes in the North Dump and 9.5 million tonnes of inert waste rock will be used for TSF construction.
Household and Household assimilable waste	Domestic and household waste generated in the Mine-Plant Area and the Pipeline Area will be collected from the areas of generation in specially marked containers set up at each of the work fronts, and then taken to the Project's Landfill for final disposal. Domestic waste generated in the Power Transmission Line Area and Port Area will be stored in specially marked containers placed in work areas from where they will be collected and transported for final disposal at the Caldera landfill (RCA Nº128/2012), which is expected to be operational in the construction stage of the Project, otherwise, this waste will be disposed of at the nearest disposal site that has the corresponding sanitary authorization.
Hazardous industrial waste	Industrial waste will be classified as hazardous according to the provisions of D.S. 148/2004 of the Ministry of Health. Hazardous industrial solid and liquid waste will be stored in appropriate drums and / or containers located in work areas, from where it will be collected and taken for temporary storage to the Hazardous Waste yard, from where it will be removed, within 6 months, and transported for final disposal at authorized facilities.
Non-hazardous industrial waste	Non-hazardous industrial solid waste from the construction and operation stages corresponds to waste such as paper, cardboard, plastics, drums, etc. This waste will be generated in the mine-plant, LTE and pipeline areas of the project and stored in labelled containers placed in work areas from where they will be collected and transferred for temporary storage to the yard areas designated for the temporary storage of non-hazardous industrial waste. From these yard areas, waste that presents commercial value, such as scrap, will be sorted and removed for marketing or delivered to recycling companies. Those that have no commercial value will be withdrawn periodically and made

Type of Waste	Management
	available to the public at authorized locations with remaining materials appropriately disposed of at authorized facilities. Regarding non-hazardous industrial waste within the Port area, an area will be set up near by the Maintenance Workshop, in compliance with current regulations, for the disposal of this waste, which will be periodically removed to authorized sites within the region.
Wastewater	For the management of wastewater during the construction and operation phases, the project will have three wastewater treatment plant (PTAS) of aerated activated sludge, located in the different sectors of the project (2 mine area plant and 1 port area). The effluents from the treatment plants will be stored in containment ponds pools for later use in the wetting of roads for dust control. The effluent quality at the PTAS will be in accordance with NCh. 1.333 on irrigation, the sludge generated at the plant will be taken to the sludge drying field and then transferred to the landfill.

Note: Table information from Project Santo Domingo Environmental Impact Study, 2013.

20.6 Closure and Reclamation Planning

The closure plans for mining sites must be included in the environmental assessment through PAS 137, which must include closure and post-closure activities for all project works. After the environmental approval an additional sectoral permit for the Mine Closure Plan is required, which includes closure and post-closure activities as well as an estimate of their associated costs.

20.6.1 Closure and Reclamation Plans

As general criteria, closure measures will prioritize the dismantling of facilities. Recoverable equipment will be disposed of in the event of posing a risk to people, and foundations will be demolished or covered. Closure measures for remaining facilities will focus on slope stabilization, surface compaction, leveling, land profiling, and rainwater management.

Santo Domingo Project has a current Mine Closure Plan, approved by Resolution N°1910/2019. Due to the optimisations made to the project, the project closure plan must be updated considering the changes made to the facilities.

20.6.2 Overall Closure Measures

The closure measures approved in the current Mine Closure Plan are indicated in the Table 20-7.

Table 20-7: Overall Closure Measures

<table>
<tr><th>Area</th><th>Closure Measure</th></tr>
<tr><td>Santo Domingo</td><td rowspan="2">Perimeter berms will be constructed around the pit perimeters utilizing sterile waste rock material, which will restrict the access by people. The location of the berms will be defined according to the risk associated with each sector of the pits, i.e., considering the exclusion zone defined by the Maximum Credible Earthquake defined specifically for the project.</td></tr>
<tr><td>Iris Norte</td></tr>
<tr><td>Santo Domingo</td><td rowspan="2">Warning signs and signs prohibiting access to unauthorised persons and vehicles are to be installed around the perimeter limiting access to the pits. In addition, information on the risk to people is considered.</td></tr>
<tr><td>Iris Norte</td></tr>
<tr><td>Santo Domingo</td><td rowspan="2">To show the evolution of the ARD generation potential of the pit walls, the analysis of samples from the development phases of the pit is considered.</td></tr>
<tr><td>Iris Norte</td></tr>
</table>

Area	Closure Measure
Iris Norte	The contour channel in the Iris Norte pit will be kept in place for water diversion. This channel will be readapted so that its carrying capacity considers the CMP of the contributing basin in post-operation condition.
Mine equipment	Dismantling and removal of mine equipment: haul trucks, shovels, FELs, drills, bulldozers, wheel dozers, among others.
Mine infrastructure	The dismantling of structures and their final disposal at an authorized site is considered. Demolition of the above-ground concrete elements and final disposal at an approved site is considered. The above-ground concrete elements will be left in place, taking into account a cover.

Source: Table information from Approved Mine closure plan, 2019.

To limit the risk of unauthorized access to the site by the public, access road controls and signage will be established.

A general cleaning of the facilities will be carried out, most of the facilities will be dismantled, possible contaminated sites will be assessed and depending on the results, remediation measures will be proposed to ensure chemical stability.

20.6.3 Monitoring and Maintenance Activities

Once the closure stage is completed, groundwater monitoring is planned for the remaining site components such as the TSF, pit, and waste rock storage areas. This monitoring will be conducted semi-annually for a period of 5 years.

For the landfill in particular, general monitoring is envisaged in addition to groundwater monitoring, on a quarterly basis for a period of 20 years.

20.6.4 Closure Cost Estimate

Similar to the current Mine Closure Plan (2019), the closure costs related to the execution of the measures described in the section above are estimated at US$124M. This closure cost must be reviewed in the next stage of engineering.

Chilean law states that a bond, surety bond or insurance must be provided to the State to cover this cost.

20.7 Permitting Considerations

Permits required by any project are classified in two categories: Environmental Permits and Sectoral Permits. The Environmental Permits are granted for any project approved within the SEIA and results in an Environmental Licence called Environmental Qualification Resolution (RCA). The RCA for a project can only be granted through an Environmental Impact Study (EIA) or an Environmental Impact Declaration (DIA). Additionally, a document called Pertinence Consultation can be submitted to the environmental authority to inquire about the need of entering a particular project (or modification of a project) to the SEIA for approval.

In accordance with the requirements of Article 18 letter l) of the SEIA Regulation (Supreme Decree No. 40/2012), an EIA or DIA must contain the list of Sectoral Environmental Permits (PAS) and pronouncements applicable to the project or activity, as well as the technical and formal contents to comply with the requirements for each permit. PAS are part of the environmental evaluation process and cover the environmental aspects of matters such as water discharge, waste storage facilities, relevant mining and hydraulic infrastructure, forest management plans, among others.

On the other hand, Sectoral Permits (PS) cover non-environmental topics and need to be applied for separately with the corresponding government authority. Some of these PSs are an extension of a PAS and need to be applied for after the RCA and the PAS have been granted.

The Santo Domingo Project EIA was presented to the authorities in October 2013. The environmental assessment process took 22 months, and the RCA was obtained in July 2015 (RCA No. 119/2015). A project owner has up to 5 years after the RCA is awarded to initiate construction. Capstone started early works and notified the authority in April 2020.

The 2015 RCA considered the use of sea water for the process and the construction and operation of two small desalination plants, one at the mine and plant area and other at the port area. Later modifications were made to use desalinated water for the process and the construction of one large desalination plant in the port area, as part of a BOOT contract (the desalination plant and the desalinated water pipeline will be constructed and operated by a third party, who will deliver the water to the port and the mine and plant areas). Based on this modification, Capstone submitted a DIA in 2019 for approval of a new desalination plant in the port area, an increase of water capture flow and water treatment and modifications of auxiliary installations within the port. The DIA was approved in 2020 (RCA No. 122/2020).

All corresponding PAS for the Santo Domingo Project have been granted with the environmental licenses, RCA No. 119/2015 and RCA No. 122/2020.

As a result of changes in applicable law, Capstone's adoption of GISTM standards, improved geotechnical information and modelling, process plant optimization work, and the recent merger of Capstone Mining and Mantos Copper, various optimizations and modifications have been made to the approved 2020 Project scope. Capstone currently has an environmental license that allows it to start construction and operation under the terms stated in that permit, but some of the engineering optimizations currently being developed will require Capstone to enter the Environmental Impact Assessment System (SEIA) through an Environmental Impact Declaration (DIA). Once the favourable Environmental Qualification Resolution (RCA) of the optimized Project is obtained, Capstone will proceed to update and obtain all the necessary sectoral permits required for the construction, operation and closure stages of the project. The relevant optimizations referenced above include:

- A new connection to an alternative existing electrical substation, located adjacent to the original substation. This required a modification to the substation, which is already implemented by a third party.
- An updated tailings storage facility design within the area previously approved for the facility, reflecting GISTM standards.
- Simplified tailings thickening system (one stage thickening instead of two as considered in the 2020 Project).
- Minor optimizations in the rest of the areas (mine, process plant, port, pipelines and tailings deposit). These changes would be made within the areas already assessed in the 2015 EIA; however, in the port area may require a new access road and a small effluent treatment plant for iron concentrate filtrate; this is being evaluating.

Environmental Permits and Sectoral Permits are required for executing the Project, in its construction, operation and closure phases. The identification of the applicable permits is part of the Permit Master Plan, containing the technical requirements and schedule for each permit, that will need to be updated with the additional and updated permits as a result of the next RCA to be obtained by the Project.

The Environmental Licenses and other Resolutions that apply to the Santo Domingo Mining Project are presented in Table 20-8.

Table 20-8: Environmental License for the Santo Domingo Project

Legal Document	Grant date	Description
Environmental License RCA N°119/2015 (Santo Domingo Project)	July 2015	Originally, the Santo Domingo Mining Project was submitted through an EIA and approved by RCA N°119/2015. The original project considered a production total of 3,6 Mt of copper concentrate and 75 Mt of magnetite concentrate, for a project lifespan of 25 years.
Environmental License RCA N°122/2020 (Desalination Plant Project)	November 2020	Favourably qualify the Environmental Impact Declaration of the project “Cambio Planta Desalinizadora” by Minera Santo Domingo, which uses a flow equal to 813 L/s of seawater to generate a flow of 366 L/s of desalinated water to be used in the different activities of the project. The rejection flow is equal to 447 L/s corresponding to brine, which will be disposed of in the marine environment through a discharge system outside the coastal protection zone.
Resolution N°20200310131 (Pertinence Consultation)	June 2020	Resolves the Pertinence Consultation for the project "Clarification and supplementation on temporary and permanent installations of the Santo Domingo project", stating that the modification does not require mandatory entry into the environmental assessment system prior to its implementation.
Resolution N°202103101117 (Pertinence Consultation)	July 2021	Resolves the Pertinence Consultation for the project “Optimisation of the wetting of mine-plant area roads”, stating that the modification does not require mandatory entry into the environmental assessment system prior to its implementation.
Resolution N°202103101211 (Pertinence Consultation)	October 2021	Resolves the Pertinence Consultation for the project "Change in connection point LT 220 kV", stating that the modification does not require mandatory entry into the environmental assessment system prior to its implementation.
Resolution N°20220310170 (Pertinence Consultation)	March 2022	Resolves the Pertinence Consultation for the project "Change in coordinate point of work of art (B-&) at C-17 Bypass channel crossing", stating that the modification does not require mandatory entry into the environmental assessment system prior to its implementation.

Capstone has identified that the installations approved in the 2015 RCA may overlap with new installations belonging to third parties. Modifications may be required to the approved locations for linear works such as the pipeline route.

Regarding Sectoral Permits (PS), about 140 works and facilities were identified, distributed between the four areas of study. The number of permits required for all facilities is estimated to be about 700 in total, with the majority (about 60%) related to the mine and plant area.

Permits that have been classified as critical to ensure they do not impact construction milestones, and their current status are summarized in Table 20-9.

Table 20-9: Critical Sectoral Permits for Santo Domingo Project

Permit Name	Status
Mine Closure Plan	Approved
Exploitation Method Authorization (Open Pit)	Approved
Authorization for a Stockpile or Waste Dump	Approved
Process Plant Operating Permit	Approved
Thickened Tailings Deposit (SERNAGEOMIN permit)	Approved
Thickened Tailings Deposit (DGA permit)	In process
Sanitary Landfill Permit	Approved
Santo Domingo Port Maritime Concession	Approved
Port Infrastructure Permit	Approved
Public Road C-17 By-Pass	Approved
Public Road C-13 By-Pass	Pending
Mine Area Access Road (from C-17 By-Pass)	Pending
Port Area Access Road (from Route 5)	Pending
Authorization for Works in a Water Course (DGA Art.294 letter c)	Pending
Authorization for Works Modifying a Water Course – C-17 Bypass crossings (DGA)	In progress
Construction Permit (IFC (approved for early works December 2019))	Pending
Building Permits	Pending
Hazardous Waste Area	Pending

Capstone started the sectoral permitting process and has received approvals from Sernageomin for the Process Plant (December 2018), Stockpile or Waste Dump (September 2018), the Mine Closure Plan (July 2019), Tailing Storage Facility (February 2019) and the exploitation method permit (May 2018). For the port area, Capstone has received approval for the Maritime Concession permit (September 2015) (its renovation is in progress), and preliminary approval for the port infrastructure (September 2018). The processing of the approval of these permits can be lengthy and can therefore affect construction schedule if they are not approved.

Other permits have been presented or will be presented in the short term, such as the watercourse modification, Paleontological Rescue permit, construction of the water pipeline crossing with Route C-239, and renovation of the Maritime Concession.

20.8 Social Considerations

The area of influence of the Santo Domingo property includes the Provinces of Chañaral and Copiapó in the Atacama Region and particularly the communities and towns of Diego de Almagro, Chañaral, and Caldera.

There are no Indigenous lands or territories of any kind being claimed on the Property. Although the lands of the Colla Community of Diego de Almagro are not within the direct area of influence, Capstone has committed to keeping lines of communication open for possible approaches or inquiries from this community.

A stakeholder plan was developed to implement an early community participation program that allowed the local communities involved to understand the major project components and for MSD to gather community opinions, comments, and feedback. Semi-structured interviews with people in the communities within the area of influence were conducted and supplemented by background information provided by social sources in each community. Meetings were held in Diego de Almagro, Chañaral and Copiapó (March 2012), Diego de Almagro and Chañaral (August and September 2012), Community of Caldera (June 2013) and Diego de Almagro, Chañaral and Caldera (September 2013), Diego de Almagro and Caldera (January 2020). Consultations included open houses, open meetings, meetings for special interest groups such as fishermen and meetings with authorities, regional and community services as well as with professional organizations. Capstone has contacted authorities from government, municipality, business and trade associations and other non-government organizations (NGOs) in the region.

As a result of the early community participation process, changes in the design of the Project were made to minimize impacts to the environment and surrounding communities as follows:

- New location and technology for the TSF. The TSF was relocated 8.5 km southeast of the town of Diego de Almagro and thickened tailings technology to be used.
- The building of a by-pass road for the town of Diego de Almagro, which will reduce traffic congestion and will avoid the transit of heavy equipment vehicles through the town.
- Plan for hiring local workers. This entails re-training programs and strategic partnerships with technical schools in Chañaral and Diego de Almagro.
- Defining guidelines for a community relations plan to contribute to sustainable development in Diego de Almagro, Chañaral and Caldera, according to the real needs of the community in the area of influence.

Additionally, Capstone has committed to support the community of Diego de Almagro with 10 l/s of potable water coming from the desalination plant and potable water system that will supply the Santo Domingo Project.

With regard to archaeological resources, Capstone with develop a cultural resource management plan for use by Capstone employees and contractors prior to land disturbance to prevent harmful interaction with known archaeological sites. In addition, an archaeological chance find procedure will be in place.

As the project approaches the construction phase, Capstone communications strategy will continue to focus on building a positive reputation and supportive environment for the development of a mining operation in the Atacama Region. Specific development strategies are directed to the communities of Diego de Almagro and Caldera. A communications plan, communications committee and crisis response management plan are being developed. As part of the strategy and commitments already defined in the EIA, a Community Office will be established in Diego de Almagro to support the deployment of the communication strategy to stakeholders.

A health and safety management system has been developed to meet local legal requirements and industry best practices. Capstone will implement policies, standards, plans and security procedures and will use facilities, equipment and personnel required to provide adequate security levels for its staff and facilities.

20.9 Comment on Environmental Studies, Permitting and Social or Community Impact

The Santo Domingo Project Environmental Impact Study (EIA) included an environmental baseline conducted between 2010 and 2013 for the project areas defined, which extends from Santo Domingo and the city of Diego de Almagro to the Punta Roca Blanca sector in Caldera. Baseline studies showed that the project is set in arid conditions, where the lack of rainfall is the dominant feature and soils have very limited value for agricultural and livestock use. At the mine area, the El Salado River, approximately 8 km north of the open pit and the TSF, is the only surface waterbody with permanent pluvial runoff in the area, with other occasional streams running east and south of the project. Vegetation and fauna are scarce around the mine area and concentrated along the El Salado River and other water courses but increasing towards the coast with only one species of flora classified as being of relevant conservation concern.

Based on the above, it is estimated that significant environmental impacts will be limited; apart from a reduced amount of highly sensitive environmental components, soils indicated non-acid drainage potential, and the lithology of the waste rock and tailings material have been reported to have appreciable neutralising potential (to be confirmed with additional test work) l. Environmental impacts will be managed through the mitigation measures defined as a result of the environmental assessments and monitoring activities for air and water quality, flora, fauna, among others, that have been carried out and will continue during the upcoming construction and operation phases of the project. Capstone has committed, as part of the environmental permits, to mitigation and compensation measures for flora and vegetation, wildlife and cultural heritage, among other environmental components, to address the impacts on especially relevant environmental elements. Measures include the demarcation and protection or rescue and relocation of plant species under conservation and xerophytic categories and archaeological sites, rescue and relocation of low-mobility wildlife species, and the training of staff to correctly protect this flora and fauna. These measures, along with all other environmental commitments included within the environmental permits, will be stated and controlled through appropriate environmental management plans once the project has started its execution phase.

Santo Domingo Project obtained its environmental licence in 2015 and later, in 2020, another one for the Desalination Plant. A project owner has up to 5 years after the RCA is awarded to initiate construction, so Capstone started early works April 2020 mainly for the construction of access roads and clearing areas for construction facilities. Capstone currently has an environmental licence that allows to start construction and operation under the terms stated in that permit, but some engineering optimizations currently being developed, will require Capstone to enter the Environmental Impact Assessment System (SEIA) through an Environmental Impact Declaration (DIA). Once the favourable Environmental Qualification Resolution (RCA) of the optimized Project is obtained, Capstone will proceed to update and obtain all the necessary sectoral permits required for the construction, operation and closure stages of the project. About 700 sectoral permits are required, with the most critical (for the TSF, process plant, WRFs, Mine Closure Plan, exploitation method and landfill), already approved or currently being processed. Some of these permits might need to be updated based on the modifications due to engineering optimizations currently in progress.

21 CAPITAL AND OPERATING COSTS

21.1 Introduction

The capital and operating cost estimates were developed by Ausenco, and other consulting firms engaged by Capstone. The estimates conform to class 3 guidelines according to the Association for the Advancement of Cost Engineering International (AACE International), with an accuracy of -10% to +15% at the 85% confidence level.

21.2 Capital Cost Estimate

21.2.1 Basis of Estimate

All capital costs are in Q2 2023 US$. A foreign exchange rate of 800 CLP to US$1 was used for the detailed estimate.

Direct costs include direct construction labour, equipment, materials, consumables, and miscellaneous items that form the permanent facilities.

Indirect costs include costs associated with engineering, procurement and construction management (EPCM), in addition to costs of temporary facilities, catering and camp services, freight and spare for acquisitions and construction support services, among others.

Owner costs are Capstone's costs for the period, from project full sanctioning and approval of the project by the Board of Directors to move into execution, to the start process plant ramp-up.

Sustaining capital costs include time-deferred costs of facilities and equipment necessary to maintain or increase production.

The contingency estimate covers uncertainties that could affect the cost of the project. It excludes any changes in the project scope of work.

21.2.2 Summary of Capital Cost Estimate

The capital cost estimate includes the initial capital costs and sustaining costs. The initial capital cost was estimated to be $2,315M. The estimated sustaining capital cost, excluding deferred stripping, totals approximately $441M. The combined initial and sustaining capital costs for the LOM were estimated to be approximately $2,756M.

The initial capital cost estimate by area is presented in Table 21-1. Table 21-2 summarizes the sustaining capital costs by year totalling $1,329M, including deferred stripping.

Table 21-1: Initial Capital Cost Estimate (by Area)

Area	WBS Level 2 included in Area	Cost ($M)	% of Total
Mine	1100	370	10.3
Process plant	2100	486	22.5
Tailings and water reclaim	3100	67	3.1
Plant infrastructure (on site)	4100 and 6100 (Plant Facilities)	144	6.7
Port & port infrastructure (on site)	5100 & 5200	283	13.2
External infrastructure (off site)	4100 and 6100 (External Facilities)	151	7.0
Indirect costs	-	414	19.2
Owner costs		109	5.1
Contingency		291	12.9
Total		**2,315**	**100**

Notes: Costs in this table are distributed and summarized by major area and include costs from consultants, Ausenco, Capstone, or other parties. Values may not sum due to rounding.

Table 21-2: Summary of Sustaining Capital by Year

Description	Amount	
	Excluding deferred stripping ($M)	Including deferred stripping ($M)
Year 1	30.0	84.8
Year 2	73.4	131.5
Year 3	15.1	96.5
Year 4	34.3	73.6
Year 5	42.8	97.5
Year 6	11.3	54.6
Year 7	9.7	77.9
Year 8	19.4	77.9
Year 9	23.8	72.1
Year 10	20.1	28.8
Year 11	10.4	56.5
Year 12	44.8	101.3
Year 13	24.3	65.7
Year 14	37.4	109.7
Year 15	28.7	80.4
Year 16	8.7	64.1
Year 17	2.9	52.3
Year 18	2.4	2.4
Year 19	1.2	1.2
Total Sustaining Capital	**441**	**1,329**

Note: Values may not sum due to rounding.

21.2.3 Mine Capital Costs

The total estimated mining initial capital costs are summarized in Table 21-3.

Table 21-3: Mine Initial Capital Cost Estimate Summary

Cost Area	Initial Capital $M
Mine development	70.1
Equipment purchase	249.4
Other Costs	12.6
Dispatch	3.7
Mine Infrastructure	9.9
Major Components (PCR)	0.8
Truck Shop & Auxiliary Mine Facilities	23.9
Total	**370**

Notes: Values may not sum due to rounding.

The total estimated mining sustaining capital costs are summarized in Table 21-4. A portion of Major Component (PCR) costs are included as Sustaining Capital in Table 21-4, with the remainder being allocated to OPEX described in Section 21.3.

Table 21-4: Mine Sustaining Capital Cost Estimate Summary

Cost Area	Initial Capital $M
Equipment purchase	74.8
Equipment rebuild	93.6
Other Costs	1.6
Dispatch	0.3
Major Components (PCR)	127.8
Total	**298**

Notes: Values may not sum due to rounding.

21.2.4 Process Plant Capital Costs

The total estimated process plant capital costs are summarized in Table 21-5.

Table 21-5: Process Plant Capital Cost Estimate Summary

Description	Cost ($M)
Process Plant General	7.0
Ore Handling	64.1
Grinding	197
Copper Flotation and Regrind	59.5
Magnetic Separation and Regrind	81.3
Tailings Thickening	44.3
Reagent Plant	9.7
Copper Concentrate Filtration	18.2
Tail and Water Recovery	4.6
Total	**486**

Notes: Values may not sum due to rounding.

21.2.5 Tailings Storage Facility Capital Costs

The TSF design and capital cost estimate were prepared by Knight Piésold. The TSF design is for a final capacity of 361 Mt of tailings, equivalent to a total volume of 245 Mm^3, which will be deposited during Life of Mine. The costs include all earthworks and supply and installation of materials for the underdrains, liner system, tailings distribution system and tailings water reclaim system. Costs are also included for contractor mobilization and demobilization, overhead and profit.

It was assumed that waste rock will be supplied from the mine by Capstone's mine fleet. The waste rock will be delivered to the embankment with the additional haul cost included in the mine cost estimate.

Table 21-6 summarizes the cost estimate for Stage 1 of the TSF. It does not include any engineering and construction management costs, other indirect costs or contingency, which are accounted for in project indirect costs and contingency.

Table 21-6: TSF Initial Capital Cost Estimate Summary

TSF Stage1	Cost ($M)
Stage 1: Tail Transport System - HD	19.4
Stage 1: Surface Water Management	0.6
Stage 1: Reclaim Water	1.6
Stage 1: Drain System	1.5
Stage 1: Foundation Treatment	3.9
Stage 1: Instrumentation	1.0
Stage 1: Liner Installation	15.1
Stage 1: Transitional Material	2.4
Stage 1: Starter Dam (Waste rock)	5.3
Stage 1: Indirect Construction Cost	15.8
Total Stage 1 Capital Cost	**66.7**

Notes: Values may not sum due to rounding.

Stages 2 to 6 of the TSF development are assigned to Sustaining costs, as shown in Table 21-7.

Table 21-7: TSF Sustaining Capital Cost Estimate Summary

TSF Stages 2-6	Type of Cost and When Applied	Cost ($M)
Stage 2	Sustaining capital year 2	20.9
Stage 3	Sustaining capital year 5	17.7
Stage 4	Sustaining capital year 8	6.2
Stage 5	Sustaining capital year 12	0.3
Stage 6	Sustaining capital year 16	0.9
Total Stages 2 - 6	**Sustaining capital**	**45.9**

Notes: Values may not sum due to rounding.

21.2.6 Plant Infrastructure Capital Costs (on Site)

The total estimated Process Plant Infrastructure capital costs are summarized in Table 21-8.

Table 21-8: Process Plant Infrastructure (On Site) Capital Cost Estimate Summary

Description	Cost ($M)
Mass Earthworks	9.1
Water Distribution and Reverse Osmosis Plant	46.3
Administrative Buildings	26.6
Electrical Power, Supply & Distribution (Plant)	33.2
Auxiliary Plant Facilities	4.8
Industrial Area Contractors	1.0
Roads And Relocations	1.5
Concentrate Transport System	5.7
Control, Communications and Security System	16.3
Total	**144**

Notes: Values may not sum due to rounding.

21.2.7 Port Facility Capital Costs

The port facility cost estimate is provided in Table 21-9 and totals $283.4M.

Table 21-9: Port Facility Capital Cost Estimate Summary

Facility	Total Cost ($M)
Cleaning And Storage of Magnetite Concentrate	4.0
Concentrated Magnetite Filtering	26.0
Transport Copper Concentrate & Storage	51.4
Transport & Storage of Magnetite Concentrate	40.6
Maritime Works	88.0
Mass Earthworks	34.2
Water Handling & Storage	8.3
Port Administrative Buildings	3.8
Supply And Distribution Electrical Energy Port	25.9
Port Auxiliary Facilities	1.2
Total Port Cost	**283**

Notes: Values may not sum due to rounding.

21.2.8 External Infrastructure Capital Costs (Off Site)

The external (off site) infrastructure capital costs are summarised in Table 21-10.

Table 21-10: External Infrastructure (Off Site) Capital Cost Summary

Description	Cost ($M)
Roads And Relocations (external roads)	14.4
Magnetite Concentrate Pipeline	108.0
Control, Communications and Security System	1.2
Supply And Transmission of Electric Power	20.0
Recovery and Transport System for Water	7.0
Total	**151**

Notes: Values may not sum due to rounding.

21.2.9 Indirect Capital Costs

The estimated project indirect costs for the execution phase are summarized in Table 21-11. Indirect costs total $414M.

Table 21-11: Indirect Capital Cost Estimate Summary

Cost Item	Cost ($M)
Engineering & Procurement	47.8
Construction Management and Field Staff	108.3
Temporary Construction Facilities	35.6
Construction Camp	56.9
Catering	30.4
Freight and Duties	34.3
Spare Parts	13.7
First Fill	3.1
Third Party Engineering Services	0.3
Third Party Field Services	30.4
Vendor Representatives	9.9
Pre-commissioning Craft Assistance	3.4
Commissioning Assistance	1.7
Magnetite Concentrate Pipeline Indirect Cost	12.1
Port Facilities Indirect Cost	18.7
Tailings Storage Facilities Indirect Cost	5.0
Power Lines Indirect Cost	2.3
Total Indirect Costs	**414**

Notes: Values may not sum due to rounding.

21.2.10 Owner's Capital Costs

The Owner's costs were estimated by Capstone and were provided to Ausenco to incorporate into the overall capital cost estimate. The Owner's costs total an estimated $109M (Table 21-12).

Table 21-12: Owner's Capital Cost Estimate Summary

Description	Total Cost ($M)
Project Cost	2.3
General Management	1.0
Administration and Finance	8.1
Legal, Mining Property and Permits	6.4
Health, Safety, Environment and Com	21.7
Labor Cost -Project	25.6
Labor Cost-Service Areas	5.2
Recruitment and Selection - Milestone Bonuses	2.3
Mine Training (remunerations)	2.2

Description	Total Cost ($M)
Processes Training (remunerations)	16.0
On Going Training	9.1
Catering (includes camp service) Transportation	7.7
Labor Accreditation	0.3
Consultants	0.4
Celebrations	0.3
Consolidated Minor Items: Management annual exams, Vehicles, Air ticket, National Travel, Hotel, Transfers	0.4
Total	**109**

Notes: Values may not sum due to rounding.

21.2.11 Capital Cost Estimate Contingency

Ausenco's contingency calculation is based on the Monte Carlo method and simulates the probability distribution curve of the overall estimated cost. A confidence interval of 85% was used as the basis for calculating the contingency. The total amount of contingency, including the recommended management reserve to achieve the expected confidence level is $279M. This value includes the cost and schedule contingency estimated by Ausenco and the cost contingency provided by third parties.

The contingency costs are summarised in Table 21-13.

Table 21-13: Capital Cost Estimate Contingency

Description	Total Cost ($M)
Process Plant, Plant Infrastructure (on site), Indirect Costs, Owners Costs	152.2
Schedule	70.8
Port Facilities	26.2
Mine	17.3
Access Roads	1.4
Magnetite Concentrate Pipeline	11.5
Tailings Storage Facilities	7.9
Power Lines	3.6
Total Contingency	**291**

Notes: Values may not sum due to rounding.

21.2.12 Tax Considerations

Local taxes on contractor-supplied materials and installation labour are included in the direct cost estimate.

Value-added tax (VAT or IVA in the Spanish acronym) on process equipment, contractor-supplied material and contractors' profit are not included in the estimates of indirect nor direct costs, as any VAT paid on these items are considered VAT credits, which will ultimately be recovered from the Chilean Internal Revenue Service by Capstone.

No escalation has been applied.

21.2.13 Closure Cost

US$124M is allowed for closure cost. This closure cost figure is incorporated into the Economic Analysis in Section 22.

21.3 Operating Costs

21.3.1 Overview

The operating costs basis of estimate was developed to reflect the latest changes in the process plant design and the updated key OPEX cost drivers (e.g. labor rates, cost of water). The operating cost estimate is at feasibility-study level with an accuracy of -10% to +15% and expressed in Q4 2023 US dollars.

The operating costs are reported in the following areas:

- Mine
- Process Plan
- Tailings and Water Recovery
- Plant Infrastructure (On Site)
- Port
- Port Infrastructure (on Site)
- External Infrastructure (Off Site)
- General & Administration (G&A)
- General (Plant Management).

Process plant and infrastructure costs were developed for cost disciplines, typically: energy, maintenance materials, operational supplies, labour, and other costs (mainly contracts and/or area specific costs classes).

Ausenco developed the process plant area and the TSF and water management costs in conjunction with Capstone. Capstone's consultants provided cost estimates for other areas. Ausenco integrated the operating costs for areas out of Ausenco's scope into the overall operating cost estimate with only editorial review.

21.3.2 Basis of Operating Cost Estimate

The overall assumptions for operating costs that apply to all areas are:

- Costs are presented in Q4 2023 US dollars, unless stated otherwise
- Costs are based on an exchange rate of CLP800 to $1.00
- The average concentrate grade for LOM is 26.4% for copper and 65.4% for magnetite
- For the copper equivalent estimate, average life-of-mine prices of $4.10/lb copper, $1,800/oz gold, $60/t for 62% grade magnetite concentrate, $85/t for 65% grade magnetite concentrate and $95/t for 67% grade magnetite concentrate were used
- Operating costs were based on the Mine Plan June 2024 (issued by Capstone on 24 Jun 2024) using a maximum throughput of 72,000 t/d between years 2 and 5 and 65,000 t/d for subsequent years
- Energy consumption for grinding area (AG and ball mills) is based on predicted variation in ore hardness.

21.3.3 Summary of Operating Cost Estimate

The operating cost estimate by area is shown in Table 21-14.

The LOM copper concentrate land transport costs are estimated to be $47.3M and are not included in the total OPEX presented below as they are included in the concentrate treatment costs in Section 22.

Table 21-14: Operating Cost Estimate by Area

Cost Centre	LOM Total ($M)	LOM Average ($/t Treated)	LOM Average ($/lb CuEq)	LOM Average ($/lb Cu Payable)
Mining	1,588	3.64	0.36	0.58
Process Plant	1,944	4.46	0.44	0.71
Tailings and Water Recovery	57.0	0.13	0.01	0.02
Plant Infrastructure	200	0.46	0.05	0.07
Port	195	0.45	0.04	0.07
Port Infrastructure	94.6	0.22	0.02	0.03
External Infrastructure	902	2.07	0.20	0.33
G&A	549	1.26	0.12	0.20
General	315	0.72	0.07	0.12
Total	**5,846**	**13.41**	**1.32**	**2.13**

Note: Values may not sum due to rounding. Cu Eq considers the gold and iron recovered in the process converted to equivalent copper based on a copper price of US$4.10 / lb.

21.3.4 Mining Costs

The mining operating cost estimate was developed by Capstone for an Owner-operated mine open pit operation and the costs are summarised in Table 21-15.

Table 21-15: Mine Operating Costs

Item	LOM Total ($M)	LOM Average ($/t Material Moved) (*)	LOM Average ($/t Treated)	LOM Average ($/lb CuEq)
Drilling	173	0.17	0.40	0.04
Blasting	310	0.30	0.71	0.07
Loading	260	0.25	0.60	0.06
Hauling	1,163	1.14	2.67	0.26
Ancillary	152	0.15	0.35	0.03
Support equipment	34.7	0.03	0.08	0.01
Engineering and administration	95.3	0.09	0.22	0.02
Labour	290	0.28	0.67	0.07
Deferred Stripping (**)	-888	-0.87	-2.04	-0.20
Total	**1,588**	**1.56**	**3.64**	**0.36**

Notes: Values may not sum due to rounding. (*) Total material moved considers the discount associated to pre-stripping and deferred stripping. (**) Deferred Stripping LOM Total $ is shown as a deduction in Mining OPEX as this amount is accounted for in Sustaining Capital. The Deferred Stripping OPEX deduction applies when the partial stripping ratio of a mine phase (period to period) is greater than its global stripping ratio. Cu Eq considers the gold and iron recovered in the process converted to equivalent copper based on a copper price of US$4.10/lb.

21.3.5 Processing and Infrastructure Costs Summary

A summary of the processing and infrastructure operating cost estimate is provided in Table 21-16.

Table 21-16: Processing and Infrastructure Operating Costs Summary

Area	LOM Total ($M)	LOM Average ($/t)	LOM Average ($/lb CuEq)
Process Operating/Plant	**1,944**	**4.46**	**0.44**
Labour	98.7	0.23	0.022
Power	1,023	2.34	0.232
Reagents	325	0.75	0.074
Operating supplies	368	0.84	0.083
Maintenance materials	77.1	0.18	0.017
Other costs	53.4	0.12	0.012
Tailings and Water Recovery	**57.0**	**0.13**	**0.013**
Labour	6.8	0.016	0.002
Power	46.8	0.11	0.011
Reagents	0.0	0.00	0.000
Operating supplies	0.11	0.0002	0.00002

Area	LOM Total ($M)	LOM Average ($/t)	LOM Average ($/lb CuEq)
Maintenance materials	0.04	0.0001	0.00001
Other costs	3.3	0.008	0.001
Plant Infrastructure (on-site)	**200**	**0.46**	**0.045**
Labour	0.0	0.00	0.000
Power	120	0.28	0.027
Operating Supplies	15.0	0.03	0.003
Maintenance materials	0.0	0.00	0.000
Other costs	56.4	0.13	0.013
Truck loading cost	8.0	0.02	0.002
Port	**195**	**0.45**	**0.044**
Labour	52.5	0.12	0.012
Power	25.7	0.06	0.006
Operating Supplies	11.0	0.03	0.002
Maintenance Materials	41.9	0.10	0.009
Maritime Structures	13.8	0.03	0.003
Other Costs	50.4	0.12	0.011
Port Infrastructure	**94.6**	**0.22**	**0.021**
Labour	0.0	0.00	0.000
Power	3.8	0.01	0.001
Operating supplies	19.6	0.04	0.004
Maintenance materials	0.0	0.00	0.000
Other costs	71.3	0.16	0.016
External Infrastructure (Off Site)	**902**	**2.07**	**0.20**
Labour	4.1	0.01	0.001
Power	10.5	0.02	0.002
Operating supplies	0.0	0.00	0.000
Maintenance materials	34.7	0.08	0.008
Other costs	4.3	0.01	0.001
Water supply	849	1.95	0.192
General (Plant Management)	**315**	**0.72**	**0.07**
Grand Total	**3,708**	**8.50**	**0.84**

Notes: Values may not sum due to rounding. Cu Eq considers the gold and iron recovered in the process converted to equivalent copper based on a copper price of US$4.10/lb.

21.3.6 Labour

The process, mining and overhead labour costs were based on shifts with 7 days on x 7 days off and 4 days on x 3 days off, for the operations and administration areas, respectively.

Labour costs are summarized in Table 21-17and total $506M over the LOM.

Table 21-17: Labour Cost Breakdown

Area	LOM Total ($M)	LOM Average ($/t)	LOM Average ($/lb CuEq)
Mine	290	0.67	0.066
Process Plant	98.7	0.23	0.022
Tailing & Water Recovery	6.8	0.02	0.002
Plant Infrastructure	0.0	0.00	0.000
Port	52.5	0.12	0.012
Port Infrastructure	0.0	0.00	0.000
External Infrastructure	4.1	0.01	0.001
General	53.8	0.12	0.012
Total	**506**	**1.16**	**0.115**

Notes: Values may not sum due to rounding. Cu Eq considers the gold and iron recovered in the process converted to equivalent copper based on a copper price of US$4.10/lb.

21.3.7 Reagents and Consumables

Reagents required for processing include lime, flotation reagents (primary collector and frother) and flocculants. Reagents were estimated to total $325M over the LOM. This equates to a LOM average of $0.75/t ore and a LOM average of $0.074/lb CuEq.

Steel and ceramics include liners and ball requirements for crushers and mills. Steel and ceramics requirements are estimated to total $360M over the LOM. This equates to a LOM average of $0.83/t ore and a LOM average of $0.082/lb CuEq.

The operating cost estimate for supply of desalinated water to the mine and plant site is based on a third-party company build, own, operate, transfer (BOOT or BOO) contract for the desalination plant and water pipeline from the port to the mine. Water will be purchased and delivered at a price of $3.97/m^3.

Operating supplies include wear items costs (e.g. for hydrocyclones and screens), fuel costs for the process plant and port, filter cloth costs and operating supplies costs for the tailings and water reclaim. The estimates are summarized in Table 21-18.

Table 21-18: Operating Supplies Estimate

Description	LOM Total ($M)	LOM Average ($/t ore)	LOM Average ($/lb CuEq)
Wear items	6,35	0.015	0.001
Fuel (process plant)	15.0	0.03	0.003
Fuel (Port)	19.6	0.04	0.004
Filter (copper & magnetite)	12.6	0.03	0.003
Operating supplies tailings and water reclaim	0.11	0.00	0.000

21.3.8 Power

The unit cost of electricity (in $/MWh) supplied to the nearest electrical substations (the San Lorenzo substation in the Diego del Almagro commune for the Plant site, and the Totoralillo substation for the port and desalination plant site) was estimated at $92.00/MWh, including all system related charges. Power costs are summarized in Table 21-19. The LOM total is $1,229M or $2.82/t of ore processed.

Table 21-19: Power Consumption and Costs

Description	LOM Total (MWh)	LOM Average (kWh/t to plant)	Total LOM Cost ($M)	LOM Average ($/t to plant)
Process Plant	**11,114,492**	**25.5**	**1.023**	**2.34**
Ore Handling	223,985	0.51	20.6	0.05
Grinding	7,540,044	17.3	694	1.59
Flotation and Copper Regrinding	1,322,107	3.03	122	0.28
Magnetic Separation and Regrinding	1,608,462	3.69	148	0.34
Thickening and Tailings Pumping System	160,743	0.37	14.8	0.03
Reagents Plant	55,496	0.13	5.11	0.01
Copper Concentrate Filtration	203,656	0.47	18.7	0.04
Tailings and Water Recovery	**508,210**	**1.17**	**46.8**	**0.11**
Tailings Transport	504,871	1.16	46.5	0.11
Transport System Recovered Water	3,340	0.01	0.31	0.001
Plant Infrastructure (on-site)	**1,306,111**	**3.00**	**120**	**0.28**
Water distribution and Osmosis Plant	930,614	2.13	85.6	0.20
Administrative Buildings Plant	211,893	0.49	19.5	0.04
Supply and Distribution Electric Power Plant	39,842	0.09	3.67	0.01
Auxiliary Plant Facilities	123,762	0.28	11.4	0.03
Port	**279,555**	**0.64**	**25.7**	**0.06**
General	279,555	0.64	25.7	0.06
Port Infrastructure (on site)	**40,939**	**0.09**	**3.77**	**0.009**
Concentrate Storage and Water Management	14,689	0.03	1.35	0.003
Port Electric Power Supply & Distribution	17,188	0.04	1.58	0.004
Port Auxiliary Facilities	9,062	0.02	0.83	0.002
External Infrastructure (Off Site)	**114,285**	**0.26**	**10.51**	**0.02**
Concentrate Transport System	114,285	0.26	10.51	0.02
Totals	**13,363,593**	**30.7**	**1,229**	**2.82**

Notes: Values may not sum due to rounding.

21.3.9 Maintenance Materials Cost

The estimated maintenance materials cost is $0.35/t treated. Over the LOM, the maintenance costs are estimated to total $153.7M, which equates to a LOM average cost of $0.03/lb CuEq.

21.3.10 Other Costs

Other costs (Table 21-20) include third party contracts considering desalinated water supply, major and minor maintenance, mobile equipment rental, laboratory operation, potable water reverse osmosis plant operation, port operation and maintenance, water treatment plant (magnetite filtration) operation and maintenance at port. The total estimate over the LOM is $1,371M. This equates to a LOM average of $3.14/t ore and a LOM average of $0.31/lb CuEq.

Table 21-20: Other Costs per Area

Description	LOM Total ($M)	LOM Average ($/t ore)	LOM Average ($/lb CuEq)
0000 - General	262	0.60	0.06
2100 - Process Plant	53.4	0.12	0.012
3100 - Tailings & Water Recovery	3.3	0.008	0.001
4100 - Plant Infrastructure (on-site)	64.4	0.15	0.015
5100 - Port	64.3	0.15	0.015
5200 - Port Infrastructure (on site)	71.3	0.16	0.016
6100 - External Infrastructure (off site)	853	1.96	0.193

21.3.11 General and Administration Costs

The G&A costs (Table 21-21) were estimated by Capstone and were provided to Ausenco to incorporate into the overall operating cost estimate. The G&A cost is $549M over the LOM. This equates to a LOM average of $1.26/t ore and a LOM average of $0.12/lb CuEq.

Table 21-21: General and Administration Costs

Description	Total Cost ($M)
General management	10.7
Administration and finance	42.8
Legal, Mining Property and Permits	19.6
Health, Safety, Environment and Com	130
Labor Cost -Project	1.5
Labor Cost-Service Areas	91.5
Recruitment and Selection - Milestone Bonuses	0.1
On Going Training	0.9
Catering (includes camp service & transport)	243
Labor Accreditation	0.7
Consultants	1.1
Annual exams	0.2
Celebrations	6.5
Vehicles	0.3
Total	**549**

21.3.12 General Costs

The general costs (Table 21-22) consider general administration labour and other costs: laboratory contracts, major maintenance, maintenance allowance and power maintenance contracts. Maintenance allowance was updated to align with current maintenance contract costs based on benchmarked project of similar scale. An allowance for labour was aligned with benchmarked project of similar scale.

The general cost is $315M over the LOM. This equates to a LOM average of $0.72/t ore and a LOM average of $0.07/lb CuEq.

Table 21-22: General Costs

Description	Total Cost ($M)
General Administration Labour	**53.8**
Management labour	4.0
Process Control Labour	21.0
Labour Allowance	28.8
Other Costs	**261.5**
Laboratory Contract	19.4
Major Maintenance	26.1
Maintenance Allowance	213
Power Line Maintenance	2.5
Total	**315**

22 ECONOMIC ANALYSIS

22.1 Cautionary Statements

The results of the economic analysis to support Mineral Reserves represent forward-looking information that is subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here. Forward-looking information includes:

- Mineral Reserve estimates
- Assumed copper, gold and magnetite prices, exchange rates, and concentrate sales contracts
- The proposed mine production plan
- Projected mining and process recovery rates
- Assumptions as to mining dilution and ability to mine using open-pit mining methods as envisaged
- Sustaining costs, closure costs, and proposed operating costs
- Salvage value is not included
- Assumptions as to environmental, permitting, and social risks.
- Additional risks to the forward-looking information include:
- Changes to costs of production from what was assumed
- Unrecognized environmental risks
- Unanticipated reclamation expenses
- Unexpected variations in the quantity of mineralized material, grade, or recovery rates
- Geotechnical or hydrogeological considerations during mining being different from what was assumed
- Failure of mining methods to operate as anticipated
- Failure of plant, equipment, or processes to operate as anticipated
- Changes to assumptions in the availability of electrical power, and the power rates used in the operating cost estimates and financial analysis
- Ability to maintain the social licence to operate
- Accidents, labour disputes, and other mining industry risks
- Changes to interest rates

- Changes to tax rates.

22.2 Methodology Used

The Project was evaluated using an 8% discounted cash flow (DCF) analysis on a non-inflated, after-tax basis. The cash flows consist of approximately three years of pre-production costs and 19 years of operations. Cash inflows consist of annual revenue projections for the mine. Cash outflows include capital costs, operating costs, royalties, and taxes, which are subtracted from the inflows to arrive at the annual cash flow projections.

To reflect the time value of money, annual net cash flow (NCF) projections are discounted back to the beginning of the project execution, using an 8% discount rate. The discount rate was determined using several factors, including the type of commodity and the level of risks (market risk, technical risk and political risk). The discounted present values of the cash flows are summed to obtain the net present value (NPV) of the project.

An NPV sensitivity analysis to discount rates was completed using discount rates of 6%, 7%, 8% (selected rate), 9% and 10%. In addition to the NPV, the internal rate of return (IRR) and payback period were also calculated. The IRR is defined as the discount rate that results in an NPV equal to zero. Cash flows are assumed to occur on an average mid-year basis of each annual period.

The capital and operating cost estimates are presented in Section 21 of this Report in Q1 2024 American dollars. The economic analysis was run on a constant dollar basis with no inflation.

22.3 Financial Model Parameters

22.3.1 Assumptions

The financial model is based on the Mineral Reserves outlined in Section 15, the mining rates and assumptions discussed in Section 16 and the recovery and processing rates and assumptions discussed in Section 13 and Section 17, respectively.

Initial capital costs are estimated to be $2,315M. LOM sustaining capital and deferred stripping are estimated to be $441M and $888M respectively.

LOM operating costs are estimated to be $5,846M. Total and net LOM operating costs.

Closure and reclamation costs have been estimated to be $124M.

Smelting and refining terms considered in the evaluation are listed in Table 22-1.

Transport and insurance charges for copper concentrate are provided in Table 22-2. Life of mine refining treatment and transport costs are estimated to be $884M.

Table 22-1: Smelter Terms

Item	Unit	Value
Concentrate Cu grade	%	26.4
Cu concentrate moisture	%	8.0
Cu concentrate losses	%	0.10
Cu treatment charge	$/dmt	70.0
Cu payable factor	%	96.55
Cu refining charge	$/lb Cu	0.07
Magnetite concentrate moisture	%	8.0
62% Grade Magnetite Conc. Price (CFR China)	$/t	85.0
65% Grade Magnetite Conc. Price (CFR China)	$/t	110.0
67% Grade Magnetite Conc. Price (CFR China)	$/t	120.0
Freight Charge (Chile-to-China)	$/t	25.0
Au payable factor	%	90.0
Au refining charge	$/oz	5.0

Table 22-2: Copper and Magnetite Concentrate Transport and Insurance Charges

Item	Unit	Value
Cu concentrate land freight	$/wmt	11.58
Cu concentrate ocean freight	$/wmt	50.00
Cu concentrate insurance	%	0.0495
Cu concentrate marketing and umpiring	$/dmt	3.0
Magnetite concentrate land freight	N/A	In operating costs as pumping costs
Magnetite concentrate ocean freight	N/A	FOB Santo Domingo port
Magnetite concentrate insurance	N/A	FOB Santo Domingo port
Magnetite concentrate marketing and umpiring	N/A	0

Note: N/A = not applicable.

For the transport and insurance charges for the magnetite concentrate, the following are included in the operating costs and the financial model:

- The magnetite concentrate is transferred to the port via pipeline with the costs included in operating costs.
- The realized price for its ore magnetite product is assumed to be free on board (FOB) Santo Domingo port.

As such, no additional transport or insurance charges are required to be included in the financial model for iron concentrate.

22.3.2 Taxes

The project has been evaluated on an after-tax basis to provide an approximate value of the potential economics. The tax model was compiled by Ausenco with assistance from a third-party retained by Capstone. The calculations are based on the tax regime as of the date of the feasibility study. At the effective date of the cashflow, the project was assumed to be subject to the following tax regime:

- The Chilean corporate income tax is 27%.
- The average mining tax rate applicable for large scale mining is 6.4%.
- The Ad Valorem tax from year 16 and onwards totaling US$6.0M over the LOM.
- The total undiscounted tax payments which were estimated to be US$2,020M over the LOM.
- Foreign investment protections and tax stability rights provided in the DL 600 Agreement are all available, in good standing and in effect.

22.3.3 Working Capital

An estimation of working capital has been incorporated into the economic analysis, and includes the following assumptions:

- Accounts Receivable: 0 days
- Inventories: 90 days
- Accounts Payable: 0 days.

22.3.4 Closure Costs

Closure costs are applied at the end of the life of mine. Closure costs were estimated to be US$124M at the end of the LOM as detailed in Section 20.

22.3.5 Royalties

Royalties of 2% NSR are payable to third parties on the production of 87% total reserves. The NSR is charged on all metals (copper, iron and gold) recovered. The LOM royalty payments are estimated to be $288M.

22.3.6 Financing Considerations

Ausenco economic analysis is based on 100% owner equity financing. The reason for this is that a project with a return that exceeds the cost of borrowing tends to show increasingly improved results as more leverage is applied and more of the risk is transferred to a third party such as a bank.

22.4 Results of the Economic Analysis

The pre-tax NPV discounted at 8% is US$2.67B, the IRR is 29.7%, and the payback period is 2.9 years. On a post-tax basis, the NPV discounted at 8% is US$1.64B, the IRR is 23.9%, and payback period is 3.0 years.

A summary of Project economics is shown in Table 22-3 and shown graphically in Figure 22-1.

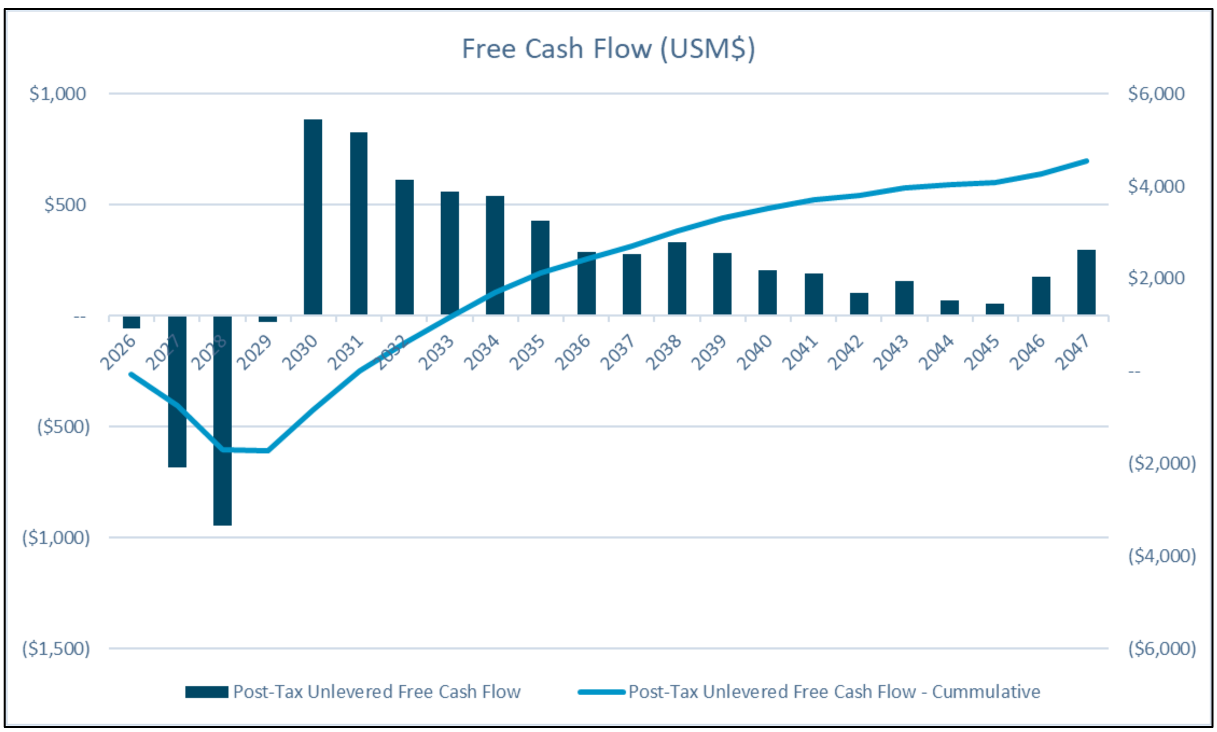
Summary of cash flow is shown in Table 22-4. The cash flow on annualized basis is shown in Table 22-5.

Table 22-3: Results of Economic Analysis

Summary of Cash Flow	Unit	Pre-tax	After Tax
Undiscounted cumulative net cash flow	$M	6,579	4,559
Net present value	-	-	-
Discounted at 4%	$M	4,160	2,802
Discounted at 6%	$M	3,330	2,199
Discounted at 8%	**$M**	**2,670**	**1,720**
Discounted at 10%	$M	2,139	1,335
Discounted at 12%	$M	1,708	1,022
Internal rate of return	%	29.7	24.1
Payback period	Years	2.9	3.0

Note: Base case is bold.

Figure 22-1: After Tax Cash Flow Summary



Source: Ausenco 2024.

Table 22-4: Summary of Cash Flow

General	Unit	Value
Copper Price	US$/lb	4.1
Gold Price	US$/oz	1,800
62% Grade Magnetite Conc. Price (CFR China)	US$/t	85
65% Grade Magnetite Conc. Price (CFR China)	US$/t	110
67% Grade Magnetite Conc. Price (CFR China)	US$/t	120
Freight Charge (Chile-to-China)	US$/t	25
Mine Life	years	19
Production	**LOM Total / Avg.**	
Total Mill Feed Tonnes	kt	436,056
Mill Head Grade Cu	%	0.33
Mill Head Grade Au	g/t	0.045
Mill Head Grade Fe	%	26.48
Mill Recovery Rate Cu	%	90.13
Mill Recovery Rate Au	%	64.74
Mill Recovery Rate Fe	%	38.76
Total Recovered Copper	kt	1,292
Total Gold Ounces Recovered	koz	411.6
Total Iron 62% Grade Tonnes Recovered	Mt	13.7
Total Iron 65% Grade Tonnes Recovered	Mt	35.38
Total Iron 67% Grade Tonnes Recovered	Mt	19.29
Total Average Annual Copper Production	Mlb/y	144
Average Year 1 to 5 Annual Copper Production	Mlb/y	240
Total Average Annual Gold Production	koz/y	19.5
Average Year 1 to 5 Annual Gold Production	koz/y	33.6
Operating Costs	**LOM Total / Avg.**	
Mining Operating Cost	USD$/t moved	1.57
Processing Cost	US$/t Milled	8.50
G&A Cost	US$/t Milled	1.26
Refining & Transport Cost	US$/lb Cu Eq.	0.21
Total Operating Costs	US$/t Milled	13.41
C1 (co-product basis)*	US$/lb Cu Eq.	1.59
C1 (by-product basis)*	US$/lb Cu	0.33
Capital Costs	**LOM Total / Avg.**	
Initial Capital	US$B	2.32
Sustaining Capital (includes deferred stripping)	US$B	1.33
Closure Costs	US$B	0.12
Financials - Pre-Tax	**LOM Total / Avg.**	
Revenue	US$B	17.1
Royalty	US$B	0.29
NPV (8%)	US$B	2.6
IRR	%	29.7
Payback	years	2.9
Capital Intensity	US$/t Cu Eq.	21,860
NPV/Inital CAPEX	-	1.15
Financials - Post Tax	**LOM Total / Avg.**	
Tax	US$B	2.02
NPV (8%)	US$B	1.72
IRR	%	24.1
Payback	years	3.0
NPV / Initial Capex	-	0.74

Note: Totals may not sum due to rounding.

Table 22-5: Cash Flow on Annualized Basis

	Unit	LOM	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19
Production Summary																								
Mineral Resource Mined	kt	377,303	--	--	--	19,939	26,279	21,319	25,086	26,278	23,726	23,722	16,311	18,838	23,726	18,234	17,929	21,884	11,486	15,140	18,352	16,641	17,588	14,826
Mineral Resource Mined to Stockpile	kt	58,753	--	1,149	5,126	1,106	989	74	9,251	6,623	13,522	6,634	1,928	2,104	8,559	692	995	--	--	--	--	--	--	--
Waste Mined	kt	1,074,757	--	8,859	26,321	66,716	62,260	62,080	50,408	43,729	51,137	58,265	62,652	62,960	57,480	66,275	60,415	61,860	56,808	62,241	66,139	54,252	24,250	9,652
Mineral Resource Rehandled	kt	58,753	--	--	--	--	--	4,962	1,267	--	--	--	7,480	4,887	--	5,491	5,862	1,841	12,240	8,586	--	2,320	1,400	2,418
Throughput	ktpd	n/a				54.6	72.0	72.0	72.2	72.0	65.0	65.0	65.2	65.0	65.0	65.0	65.2	65.0	65.0	65.0	50.3	51.9	52.0	47.2
Resource Sent to Mill	kt	436,056	--	--	--	19,939	26,279	26,281	26,353	26,278	23,726	23,722	23,791	23,725	23,726	23,725	23,791	23,725	23,726	23,725	18,352	18,961	18,988	17,243
Cu Head Grade	%	0.33%	--	--	--	0.50	0.59	0.48	0.49	0.43	0.45	0.39	0.26	0.22	0.30	0.33	0.31	0.22	0.20	0.29	0.20	0.12	0.11	0.13
Au Head Grade	g/t	0.05	--	--	--	0.07	0.08	0.07	0.07	0.06	0.07	0.06	0.04	0.03	0.04	0.05	0.05	0.03	0.03	0.04	0.03	0.02	0.01	0.01
Fe Head Grade	%	26.5%	--	--	--	25.05	29.00	26.40	29.36	29.99	32.04	30.66	26.62	26.69	28.38	25.82	23.25	21.36	21.93	21.21	24.11	25.83	26.36	27.52
Cu Recovery	%	90.1%	--	--	--	91	92	91	91	89	89	90	90	89	90	90	90	89	89	90	90	90	89	89
Au Recovery	%	64.7%	--	--	--	70	71	69	68	66	66	65	61	59	61	62	62	59	58	63	59	57	57	53
Fe Recovery	%	39%	--	--	--	17	31	40	38	38	37	34	45	47	41	29	30	44	35	25	41	57	57	61
Cu Recovered	kt	1,293	--	--	--	91	142	115	118	99	95	84	56	47	65	71	67	47	42	62	33	21	19	19
Au Recovered	koz	412	--	--	--	31	47	38	39	32	33	27	18	13	21	23	22	14	12	18	10	6	5	4
Cu Payable	kt	1,243	--	--	--	88	137	111	114	95	92	81	54	45	62	68	64	45	40	59	32	20	18	19
Cu concentrate grade	%	26.4%	--	--	--	28.3	28.4	27.6	27.1	26.5	26.1	26.0	25.4	25.1	25.2	25.2	25.6	25.5	25.2	26.1	25.6	26.4	25.9	23.7
Au Payable	koz	370	--	--	--	28	42	34	35	29	29	25	16	12	19	20	19	13	11	16	9	5	4	4
Fe conc. Production	Mt	68.4	--	--	--	1	4	4	4	4	4	4	4	4	4	3	3	3	3	2	3	4	4	4
Fe conc. Grade	%	65.4%	--	--	--	62.5	65.0	66.2	66.6	66.4	65.6	65.6	65.6	66.2	66.2	65.4	64.4	65.4	65.0	63.5	64.2	64.6	65.0	65.6
Revenues																								
Copper Revenue	US$/mm	11,235	--	--	--	791	1,237	1,000	1,027	859	829	733	486	408	561	612	582	411	365	535	286	180	167	167
Gold Revenue	US$/mm	171	--	--	--	8.9	13.6	11.1	11.5	9.4	9.5	8.3	6.5	4.6	7.4	9.9	16.7	11.0	9.3	13.9	7.3	4.7	3.7	3.3
Magnetite concentrate revenue	US$M	5,664	--	--	--	84.4	311.3	366.4	396.3	405.5	373.3	328.6	370.9	408.0	373.0	232.9	168.8	275.4	214.3	136.9	177.9	278.2	373.9	387.4
Gross Revenue	US$M	17,069	--	--	--	884.4	1,561.9	1,377.9	1,434.7	1,273.9	1,211.4	1,069.6	862.9	821.0	941.0	854.6	767.5	697.1	589.1	685.9	471.2	462.4	544.4	558.1
Operating Costs																								
Mine Operating Costs	US$M	(1,588)	--	--	--	(75)	(101)	(73)	(134)	(79)	(88)	(69)	(73)	(88)	(131)	(91)	(75)	(88)	(42)	(73)	(81)	(90)	(80)	(56)
Mill Processing Costs	US$M	(3,708)	--	--	--	(193)	(216)	(211)	(217)	(214)	(208)	(202)	(207)	(198)	(214)	(204)	(191)	(189)	(203)	(196)	(160)	(168)	(166)	(152)
G&A Costs	US$M	(549)	--	--	--	(29)	(28)	(27)	(27)	(29)	(29)	(30)	(30)	(29)	(29)	(30)	(29)	(29)	(29)	(30)	(29)	(28)	(28)	(28)
Refining Charges, Treatment Charges, Transportation Cost & Royalties																								
Refining	US$M	(194)	--	--	--	(13.6)	(21.3)	(17.3)	(17.7)	(14.8)	(14.3)	(12.6)	(8.4)	(7.0)	(9.7)	(10.5)	(10.0)	(7.1)	(6.3)	(9.2)	(4.9)	(3.1)	(2.9)	(2.9)
Treatment Costs	US$M	(342)	--	--	--	(22)	(35)	(29)	(30)	(26)	(26)	(23)	(15)	(13)	(18)	(20)	(18)	(13)	(12)	(16)	(9)	(5)	(5)	(6)
Transport Costs	US$M	(328)	--	--	--	(21)	(33)	(28)	(29)	(25)	(24)	(22)	(15)	(13)	(17)	(19)	(18)	(12)	(11)	(16)	(9)	(5)	(5)	(5)
Storage & Marketing	US$M	(15)	--	--	--	(1)	(2)	(1)	(1)	(1)	(1)	(1)	(1)	(1)	(1)	(1)	(1)	(1)	(1)	(1)	(0)	(0)	(0)	(0)
Insurance Charges	US$M	(5)	--	--	--	(0)	(1)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)
Royalties	US$M	(288)	--	--	--	(17)	(30)	(27)	(28)	(18)	(15)	(15)	(16)	(14)	(11)	(12)	(11)	(13)	(10)	(13)	(9)	(9)	(10)	(11)
Cost Guarantee																								
Cost Guarantee	US$M	(11)	--	--	--	(0)	(0)	(0)	(0)	(0)	(1)	(1)	(1)	(1)	(1)	(1)	(1)	(1)	(1)	(1)	(1)	(1)	--	--
Other Income																								
Other Income	US$M	47	--	--	--	5	4	4	4	4	5	3	3	4	2	2	2	2	2	1	--	--	--	--
EBITDA																								
EBITDA	US$M	10,086				516	1,099	967	952	870	810	698	501	462	511	469	415	346	276	332	167	152	246	297
Gold Streams Proceeds																								
Remaining Gold Streams Proceeds	US$M	260	28	58	174	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Capital Expenditures																								
Initial Capital	US$M	(2,315)	(85)	(744)	(1,119)	(366)	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Sustaining Capital	US$M	(441)	--	--	--	(30)	(73)	(15)	(34)	(43)	(11)	(10)	(19)	(24)	(20)	(10)	(45)	(24)	(37)	(29)	(9)	(3)	(2)	(1)

	Unit	LOM	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19
Deferred Stripping	US$M	(888)	--	--	--	(55)	(58)	(81)	(39)	(55)	(43)	(68)	(59)	(48)	(9)	(46)	(57)	(41)	(72)	(52)	(55)	(49)	--	--
Closure Cost	US$M	(124)	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	(18)	(18)	(18)	(18)	(18)	(18)	(18)
Change in Working Capital																								
Change in Working Capital	US$M	--	--	--	--	(80)	(15)	10	(17)	15	(1)	7	(0)	(1)	(16)	12	8	(1)	8	(7)	9	(3)	3	70
Pre-Tax Unlevered Free Cash Flow																								
Pre-Tax Unlevered Free Cash Flow	US$M	6,579	(57)	(686)	(945)	(15)	952	881	861	788	755	627	423	389	466	424	322	261	157	226	95	79	228	349
Pre-Tax Cumulative Unlevered Free Cash Flow	US$M		(57)	(743)	(1,688)	(1,703)	(751)	130	991	1,779	2,534	3,161	3,583	3,972	4,438	4,863	5,184	5,446	5,602	5,829	5,923	6,002	6,231	6,579
Taxes																								
Unlevered Cash Taxes	US$M	(2,020)	--	--	--	(13)	(68)	(51)	(247)	(230)	(216)	(199)	(137)	(113)	(135)	(140)	(119)	(72)	(56)	(72)	(25)	(24)	(52)	(50)
Post-Tax Unlevered Free Cash Flow																								
Post-Tax Unlevered Free Cash Flow	US$M	4,559	(57)	(686)	(945)	(28)	884	829	615	558	540	427	286	277	331	284	203	189	101	154	70	55	176	298
Post-Tax Cumulative Unlevered Free Cash Flow	US$M	-	(57)	(743)	(1,688)	(1,717)	(832)	(3)	611	1,169	1,708	2,136	2,421	2,698	3,029	3,313	3,516	3,705	3,806	3,960	4,030	4,085	4,261	4,559
Cost KPI's																								
C1 (by-product basis)**	USD$/lb Cu	0.33	--	--	--	1.36	0.37	0.04	0.20	(0.12)	0.04	0.13	(0.24)	(0.65)	0.29	0.89	1.11	0.53	0.90	1.46	1.56	0.40	(2.19)	(3.45)
C1 (co-product basis)*	USD$/lb Cu Eq.	1.59	--	--	--	1.62	1.12	1.13	1.29	1.23	1.30	1.36	1.64	1.72	1.81	1.77	1.78	1.96	2.08	2.00	2.53	2.65	2.16	1.82

Note: Totals may not sum due to rounding.
*C1 consist of mining costs, processing costs, mine-level G&A, gold revenue credit, and refining charges over payable copper equivalent pounds (copper plus magnetite)
**C1 consist of mining costs, processing costs, mine-level G&A, gold revenue credit, magnetite revenue credit, and refining charges over payable copper pounds

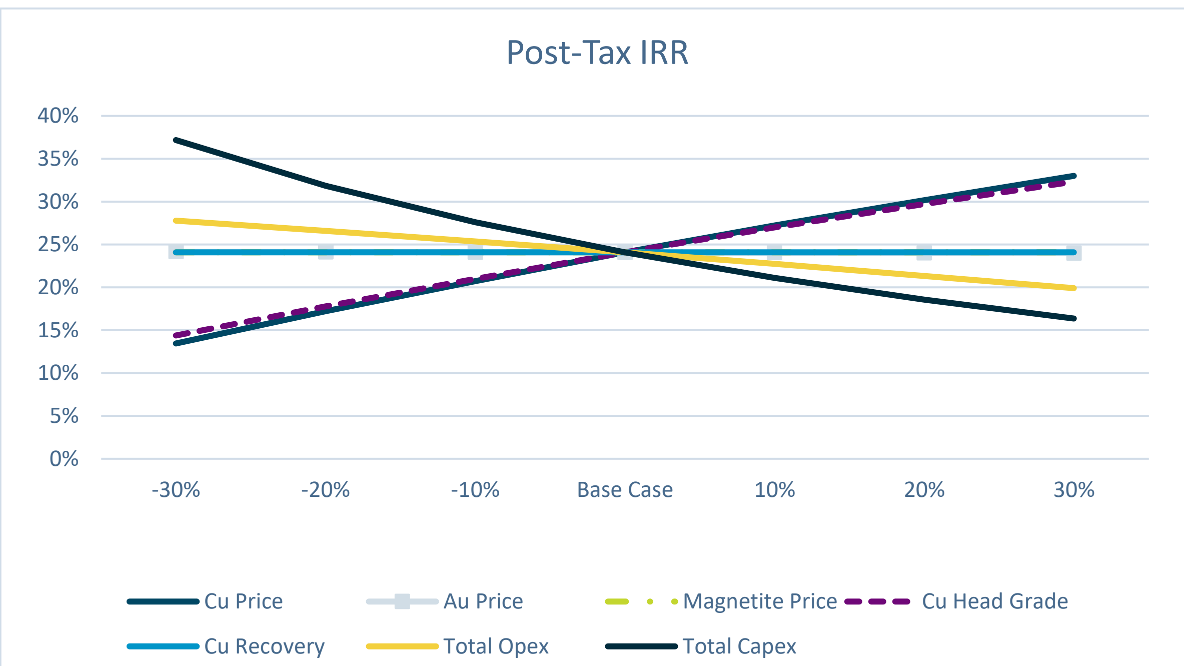
22.5 Sensitivity Analysis

A sensitivity analysis was conducted on the base case pre-tax and post-tax NPV, IRR, and payback of the Project, using the following variables:

- Metal price (copper, iron and gold)
- Operating costs (including power)
- Capital costs
- Cu Recovery
- Cu Head Grade.

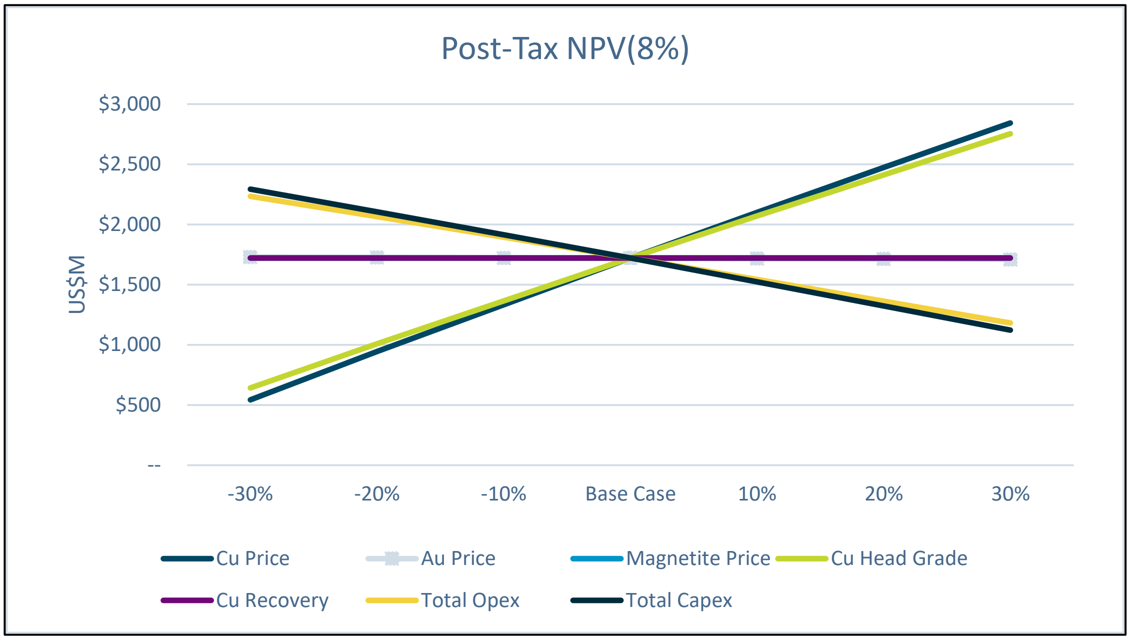
The analysis shows that the Santo Domingo NPV8% is most sensitive to changes in the copper price (copper grade) and capital expenditures. The sensitivity analysis showed that Santo Domingo is less sensitive to changes in the gold and iron price.

Figure 22-2: Sensitivity of IRR Post Tax



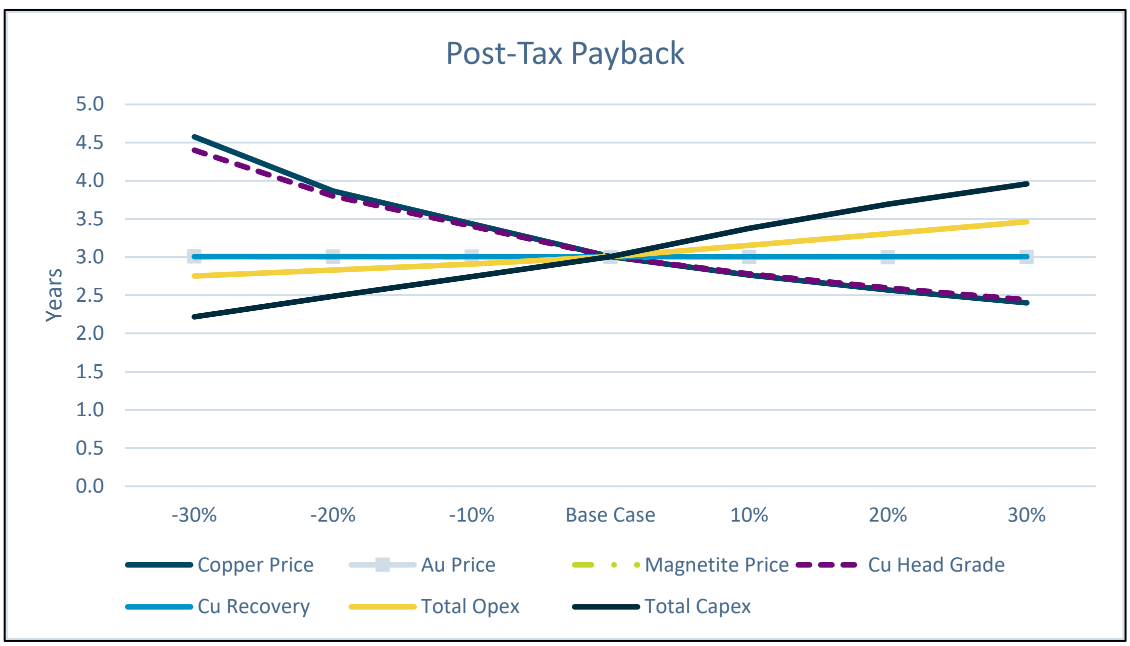
Source: Ausenco, 2024.

Figure 22-3: Sensitivity of NPV$_8$% Post Tax



Source: Ausenco, 2024.

Figure 22-4: Sensitivity of Payback Post Tax



Source: Ausenco, 2024.

Table 22-6: Sensitivity Post Tax

Post-Tax NPV Sensitivity to Discount Rate

Discount Rate	Copper Price (25%)	(10%)	--	10%	25%
6%	$1,073	$1,753	$2,199	$2,639	$3,287
7%	$900	$1,531	$1,946	$2,355	$2,955
8%	$745	$1,334	$1,720	$2,100	$2,658
9%	$607	$1,157	$1,517	$1,872	$2,391
10%	$483	$998	$1,335	$1,666	$2,151

Post-Tax IRR Sensitivity to Discount Rate (%)

Discount Rate	Copper Price (25)	(10)	--	10	25
6	15.4	20.7	24.1	27.2	31.6
7	15.4	20.7	24.1	27.2	31.6
8	15.4	20.7	24.1	27.2	31.6
9.0	15.4	20.7	24.1	27.2	31.6
10.0	15.4	20.7	24.1	27.2	31.6

Post-Tax Payback Sensitivity to Discount Rate (%)

Discount Rate	Copper Price (25)	(10)	--	10	25
6	4.2	3.4	3.0	2.8	2.5
7	4.2	3.4	3.0	2.8	2.5
8	4.2	3.4	3.0	2.8	2.5
9	4.2	3.4	3.0	2.8	2.5
10	4.2	3.4	3.0	2.8	2.5

Post-Tax NPV Sensitivity to OPEX

OPEX	Copper Price (25%)	(10%)	--	10%	25%
(20.0 %)		$1,687	$2,066	$2,437	$2,986
(10.0 %)	$930	$1,512	$1,895	$2,270	$2,823
--	$745	$1,334	$1,720	$2,100	$2,658
10.0%	$558	$1,154	$1,543	$1,926	$2,490
20.0%	$369	$971	$1,363	$1,751	$2,319

Post-Tax IRR Sensitivity to OPEX (%)

OPEX	Copper Price (25)	(10)	--	10	25
(20.0)	18.4	23.4	26.6	29.5	33.7
(10.0)	17.0	22.1	25.4	28.4	32.7
--	15.4	20.7	24.1	27.2	31.6
10.0	13.7	19.3	22.7	26.0	30.5
20.0	11.9	17.8	21.3	24.7	29.3

Post-Tax Payback Sensitivity to OPEX (%)

OPEX	Copper Price (25)	(10)	--	10	25
(20.0)	3.8	3.1	2.8	2.6	2.4
(10.0)	3.9	3.3	2.9	2.7	2.4
--	4.2	3.4	3.0	2.8	2.5
10.0	4.5	3.6	3.2	2.8	2.5
20.0	4.7	3.7	3.3	2.9	2.6

Post-Tax NPV Sensitivity To CAPEX

Initial CAPEX	Copper Price (25%)	(10%)	--	10%	25%
(20.0 %)	$1,150	$1,729	$2,106	$2,478	$3,029
(10.0 %)	$949	$1,533	$1,915	$2,290	$2,845
--	$745	$1,334	$1,720	$2,100	$2,658
10.0%	$539	$1,134	$1,522	$1,907	$2,470
20.0%	$331	$932	$1,323	$1,710	$2,280

Post-Tax IRR Sensitivity to CAPEX (%)

Total CAPEX	Copper Price (25)	(10)	--	10	25
(20.0)	21.9	28.1	31.9	35.4	40.5
(10.0)	18.3	24.1	27.6	30.9	35.6
--	15.4	20.7	24.1	27.2	31.6
10.0	12.9	18.0	21.1	24.1	28.3
20.0	10.8	15.6	18.6	21.4	25.4

Post-Tax Payback Sensitivity to CAPEX (%)

Total CAPEX	Copper Price (25)	(10)	--	10	25
(20.0)	3.3	2.7	2.5	2.3	2.1
(10.0)	3.7	3.0	2.7	2.5	2.3
--	4.2	3.4	3.0	2.8	2.5
10.0	4.7	3.8	3.4	3.0	2.7
20.0	5.2	4.1	3.7	3.3	2.9

Post-Tax NPV Sensitivity to Discount Rate					
	Magnetite Price				
Discount Rate	(25%)	(10%)	--	10%	25%
6.0%	$2,199	$2,199	$2,199	$2,199	$2,199
7.0%	$1,946	$1,946	$1,946	$1,946	$1,946
8.0%	$1,720	$1,720	$1,720	$1,720	$1,720
9.0%	$1,517	$1,517	$1,517	$1,517	$1,517
10.0%	$1,335	$1,335	$1,335	$1,335	$1,335

Post-Tax IRR Sensitivity to Discount Rate (%)					
	Magnetite Price				
Discount Rate	(25)	(10)	--	10	25
6.0	24.1	24.1	24.1	24.1	24.1
7.0	24.1	24.1	24.1	24.1	24.1
8.0	24.1	24.1	24.1	24.1	24.1
9.0	24.1	24.1	24.1	24.1	24.1
10.0	24.1	24.1	24.1	24.1	24.1

Post-Tax Payback Sensitivity to Discount Rate (%)					
	Magnetite Price				
Discount Rate	(25)	(10)	--	10	25
6.0	3.0	3.0	3.0	3.0	3.0
7.0	3.0	3.0	3.0	3.0	3.0
8.0	3.0	3.0	3.0	3.0	3.0
9.0	3.0	3.0	3.0	3.0	3.0
10.0	3.0	3.0	3.0	3.0	3.0

Post-Tax NPV Sensitivity To OPEX					
	Magnetite Price				
OPEX	(25%)	(10%)	--	10%	25%
(20.0 %)	$2,066	$2,066	$2,066	$2,066	$2,066
(10.0 %)	$1,895	$1,895	$1,895	$1,895	$1,895
--	$1,720	$1,720	$1,720	$1,720	$1,720
10.0%	$1,543	$1,543	$1,543	$1,543	$1,543
20.0%	$1,363	$1,363	$1,363	$1,363	$1,363

Post-Tax IRR Sensitivity to OPEX (%)					
	Magnetite Price				
OPEX	(25)	(10)	--	10	25
(20.0)	26.6	26.6	26.6	26.6	26.6
(10.0)	25.4	25.4	25.4	25.4	25.4
--	24.1	24.1	24.1	24.1	24.1
10.0	22.7	22.7	22.7	22.7	22.7
20.0	21.3	21.3	21.3	21.3	21.3

Post-Tax Payback Sensitivity to OPEX (%)					
	Magnetite Price				
OPEX	(25)	(10)	--	10	25
(20.0)	2.8	2.8	2.8	2.8	2.8
(10.0)	2.9	2.9	2.9	2.9	2.9
--	3.0	3.0	3.0	3.0	3.0
10.0	3.2	3.2	3.2	3.2	3.2
20.0	3.3	3.3	3.3	3.3	3.3

Post-Tax NPV Sensitivity to CAPEX					
	Magnetite Price				
Initial CAPEX	(25%)	(10%)	--	10%	25%
(20.0 %)	$2,106	$2,106	$2,106	$2,106	$2,106
(10.0 %)	$1,915	$1,915	$1,915	$1,915	$1,915
--	$1,720	$1,720	$1,720	$1,720	$1,720
10.0%	$1,522	$1,522	$1,522	$1,522	$1,522
20.0%	$1,323	$1,323	$1,323	$1,323	$1,323

Post-Tax IRR Sensitivity to CAPEX (%)					
	Magnetite Price				
Total CAPEX	(25)	(10)	--	10	25
(20.0)	31.9	31.9	31.9	31.9	31.9
(10.0)	27.6	27.6	27.6	27.6	27.6
--	24.1	24.1	24.1	24.1	24.1
10.0	21.1	21.1	21.1	21.1	21.1
20.0	18.6	18.6	18.6	18.6	18.6

Post-Tax Payback Sensitivity to CAPEX (%)					
	Magnetite Price				
Total CAPEX	(25)	(10)	--	10	25
(20.0)	2.5	2.5	2.5	2.5	2.5
(10.0)	2.7	2.7	2.7	2.7	2.7
--	3.0	3.0	3.0	3.0	3.0
10.0	3.4	3.4	3.4	3.4	3.4
20.0	3.7	3.7	3.7	3.7	3.7

23 ADJACENT PROPERTIES

This section is not relevant to this Report.

24 OTHER RELEVANT DATA AND INFORMATION

24.1 Project Execution Plan

At the effective date of this report Capstone has a team of personnel that directly manages any consultants, site campaign contractors and early infrastructure construction contractors for the project. This direct management approach will transition to an EPCM (Engineering, Procurement and Construction Management) approach for project execution.

24.2 Engineering, Procurement and Construction Management Approach

An EPCM (Engineering, Procurement, Construction Management) strategy has been developed for the design, procurement and construction of the Santo Domingo Project.

This EPCM strategy includes an Interim Engineering phase prior to the execution of the project, to achieve the following:

- Level the engineering of all packages, improving the Project definition level
- Develop basic engineering for major equipment procurement, following LNTP (limited Notice to Proceed) and FNTP (Full Notice to Proceed) strategy
- Develop detailed engineering necessary to feed early works processes.

The EPCM contractor will carry out detailed design, procurement, and construction management for on-site and off-site infrastructure scopes, and administer the interface between contractors under its management.

The EPCM contractor will manage all assigned scope responsibilities within the parameters of the EPCM agreement scope, the approved Project Schedule, and the approved Project Budget. The EPCM contractor will be responsible for managing all off-site and on-site design, procurement and construction management activities and services.

An EPCM bid process will be prepared and issued for competitive tender by Capstone Copper. The EPCM contractor will be appointed prior to any of the major works for the commencing of the project.

The EPCM contractor will provide all project support services such as cost reporting and management, procurement of equipment and materials needed for the project, planning, health, and safety, and environmental.

The EPCM contractor will integrate the contributions to the project made by other consultants and contractors.

24.3 Engineering, Procurement and Construction Management Duration

The Project schedule includes interim engineering in the year prior to Capstone providing the investment approval for the execution of the project. Detail engineering and procurement activities commences upon obtaining Capstone's execution investment approval. Construction will be completed and ramp up will commence approximately 3.3 years after Capstone's execution investment approval. Production ramp up will be complete approximately 4 years after obtaining Capstone's execution investment approval.

Table 24-1 shows the year that key milestones are scheduled to be achieved.

Table 24-1: Key Milestones

Description	Year
Commence Interim Engineering	Yr -1
Project Investment Approval	Yr 0
Commence detailed Engineering	Yr 0
Finish Construction Administration Buildings Plant	Yr 1
Finish Construction Process Plant	Yr 3
Finish Mine Prestripping	Yr 3
Finish TSF construction	Yr 3
Finish Commissioning Marine Works Port	Yr 3
Finish Commissioning Process Plant	Yr 3
Finish Commissioning TSF	Yr 3
Start Ramp Up	Yr 3
Ramp up complete	Yr 4

24.4 Optimization Opportunities

A number of optimization opportunities exist for the project, and these are listed below in order of potential project NPV impact. Each of these opportunities requires significant further work to develop the opportunity to a level where the realisable value is at Feasibility Study confidence.

- Flotation of a pyrite concentrate containing cobalt and copper values that could be recovered by hydrometallurgical processing
- Copper oxide hydrometallurgical processing
- Review of regrind technology and installed power requirements
- Optimization of the magnetic circuit water balance and reduce water flows reporting to the tailings thickener
- Jameson Cell cleaner circuit layout detail in association with Glencore
- Integrated tailings thickeners-TSF assessment
- Mantoverde-Santo Domingo Mining District Integration

24.4.1 Pyrite Concentrate Processing for Cobalt Recovery

The pyrite recovery opportunity is described in Section 13 (metallurgy) and Section 17 (processing). Pyrite concentrate containing cobalt, and copper can be recovered and pumped to the Mantoverde operation where it can be processed for copper and cobalt recovery and acid generation in heap leaching.

Capstone is completing pilot tests to evaluate this opportunity. These tests are likely to be completed in H2 2024, and if successful, will result in a substantial increase in revenue for the Santo Domingo project.

24.4.2 Acid Leaching of Oxide Ore for Copper Recovery

The mine plan described in Section 16 (mining methods) considers the extraction and stockpiling of approximately 48.9 Mt of oxide ore with a cut off grade of 0.2% copper. Copper recovery from the hydrometallurgical processing of this mineral potential through oxide acid leaching is an opportunity described in Section 13 (metallurgy). Pregnant leach solution (PLS) can be recovered from ROM leaching at Santo Domingo site and pumped to Mantoverde operation where it can be processed taking advantage of the existing SX-EW infrastructure to produce copper cathodes. Rafinate can be pumped from Mantoverde operation to Santo Domingo for ROM pad irrigation, returning water and transporting sulphuric acid for leaching.

Capstone has planned a drilling campaign to be able to estimate mineral resources and mineral reserves of the oxide ore. Also, to complete additional test work to support these opportunities. If there is a confirmation of the positive business case, this opportunity would allow for doing circular economy, by processing waste material to the Santo Domingo concentrator, and would maximize utilization of Mantoverde asset, Capstone’s adjacent property.

24.4.3 Review Regrind Mill Technology and Installed Power Requirements

There is an opportunity to significantly reduce the installed cost and installed power of the regrind mills. At present, the magnetite circuit uses two horizontal regrind mills in parallel. These mills could be replaced by one variable speed vertical mill. The variable speed capability of the vertical mill would be used to optimise the regrind power draw based on magnetite concentrate feed rate and required product size. Preliminary concept investigation indicates that a reduction in total capital cost of between $5M and $10M is possible.

24.4.4 Optimisation of the Water Balance Around the Magnetite and Tailings Thickening Areas

The present water balance around the magnetite circuit results in a large volume of water reporting to the tailings thickeners. This could result in operational issues for tailings discharge management, sampling, and tailings launders flow management due to the very high volumetric flows. There is an opportunity to reduce the flow to the tailings thickeners by improving the management of water inside the magnetite plant.

24.4.5 Improve the Design of the Jameson Cells

The current Jameson Cell design can be further optimised. Ausenco is discussing this opportunity with Glencore. The intent is to rationalise and optimise the relationship between the tank containing the Jameson downcomers and the hopper used to feed and recycle the Jameson downcomers. The optimised design is expected to reduce the height of the Jameson Cell installation and reduce the associated bulk material quantities and capital cost.

24.4.6 Integrated Tailings Thickeners-TSF assessment

The current tailings thickening design considers high density thickeners type of equipment. Tailings physical properties and site conditions influence the TSF wall design. There is an opportunity to have a reduction in TSF wall volume, improving water recovery, and reducing sectoral permitting requirements and time by implementing paste thickening technology, which has a higher total installed cost associated. Trade-offs are to be conducted and further developed to support a decision in relation to this.

24.5 Mantoverde-Santo Domingo Mining District Integration

After the ramp-up of MVDP at the end of 2024, Capstone's next phase of transformational growth will be a construction decision and integration of the Project with the expanded Mantoverde Mine. Capstone aims to create a 250 ktpa of copper cathode and concentrate (consolidated Mantoverde and Santo Domingo) world-class mining district in the Atacama region of Chile (MV-SD). The combination of key infrastructure already in place alongside an experienced mine build and operating team supports future development of the MV-SD district.

Capstone's base case plan for MV-SD district integration includes completion and ramp-up of MVDP (Capstone, 2024), a sanctioning decision followed by construction of the Santo Domingo Project's copper-iron project, and completion of upgrades to the existing water and power infrastructure proximal to both projects.

Santo Domingo is located 35 km northeast of Capstone's Mantoverde Mine, approximately 65 km via public roadways, and 16 km southeast of Capstone's Sierra Norte property.

24.5.1 Mantoverde Mine

Mantoverde Mine is a copper-gold producer, with 70% ownership held by Capstone and 30% held by Mitsubishi Materials Corporation, described in the "Mantoverde Mine and Mantoverde Development Project NI 43-101 Technical Report Chañaral / Región de Atacama, Chile", effective November 29, 2021.

The Mantoverde Development Project (MVDP) expansion is expected to increase production at Mantoverde Mine from approximately 35,000 tonnes of copper as cathodes in 2023 to approximately 110,000 tones of copper per year combining cathodes and concentrate at full capacity. MVDP is a conventional open pit operation using truck-and-shovel technology. A concentrator plant was constructed in 2023 to treat sulphide ore. Copper concentrate production is expected to begin in the second half of 2024. Oxide ores will continue to be treated in the existing solvent extraction and electrowinning plant (SX-EW). MVDP is progressing under a lump-sum turn-key EPCM contract with Ausenco Limited. The Mantoverde Mine will be able to use the planned Santo Domingo port to ship concentrate.

24.5.2 Sierra Norte Property

The Sierra Norte property is an IOCG-type copper deposit and covers over 7,000 hectares. Copper mineralization at Sierra Norte occurs in irregular tabular bodies of specularite breccias and stockwork with chalcopyrite and lesser pyrite. Copper oxide mineralization is present above the sulfide orebody. A historical resource estimate, not compliant with NI 43-101, of approximately 100Mt at 0.45% CuT is shown in Table 24-2. Sierra Norte represents an opportunity to potentially become a future sulphide feed source for Santo Domingo, extending its higher-grade copper sulphide life, with additional upside for future exploration for copper oxides and sulfides. Capstone entered into a binding share purchase agreement with Inversiones Alxar S.A. ("Alxar") and Empresas COPEC S.A. ("EC"), collectively the "Sellers" to acquire 100% of Compania Minera Sierra Norte S.A. ("Sierra Norte") on July 17, 2024. Under the terms of the purchase agreement, Capstone paid the Sellers $40 million in share consideration.

Table 24-2: Sierra Norte - Historical Mineral Resource

Category	Tonnage (Mt)	Grade		Contained Metal
		CuT (%)	CuSA (%)	Cu (kt)
Carmen-Paulina				
Measured	8	0.47	0.16	35
Indicated	63	0.46	0.10	292
Total Measured + Indicated	**71**	**0.46**	**0.11**	**327**
Inferred	25	0.40	0.04	102
Esther				
Measured	1	0.42	0.26	3
Indicated	3	0.40	0.24	13
Total Measured + Indicated	**4**	**0.40**	**0.24**	**16**
Inferred	0.1	0.35	0.22	0.3

The Historical Mineral Resource was derived from the report "Actualización del Modelo Geológico y de la Estimación de Recursos Minerales del Proyecto Diego de Almagro" completed by Amec Foster Wheeler with an effective date of April 29, 2016, prepared for Alxar S.A. The historical estimates are strictly historical in nature and are not compliant with NI 43-101 and should not be relied upon. A QP has not done sufficient work to classify the historical estimates as current "mineral resources", as such term is defined in NI 43-101 and it is uncertain whether, following further evaluation or exploration work, the historical estimates will be able to report as mineral resources in accordance with NI 43-101. Capstone has not done sufficient work to classify the historical estimate as current mineral resources and is not treating the historical estimate as current mineral resources. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. The historical estimate is reported using a cut-off grade of 0.20% with further economic extraction parameters outlined below. Categories are based on average spacing of drillholes and levels of confidence in the grade estimation. There are no more recent estimates or data available to Capstone. The Sierra Norte deposit will require further evaluation including drilling to verify the historical estimate as current mineral resources. Readers are cautioned not to rely on the historical estimate in this section. Economic Parameters for the historical estimate include the following: Copper price: $3.00/lb; Mining cost: $1.69/t; Sulphide recovery: 91%; Sulphide processing cost: $7.26/t; Oxide (heap) recovery: 60%; Oxide (heap) processing cost: $8.12/t; Oxide (SX-EW) processing cost: $0.30/lb; Concentrate selling costs: $0.41/lb; and Cathodes selling costs: $0.04/lb.

25 INTERPRETATION AND CONCLUSIONS

25.1 Introduction

The QP notes the following interpretations and conclusions in their respective areas of expertise, based on their review of data available for this technical report.

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

Capstone owns 100% of Minera Santo Domingo.

Mineral tenure documentation supports that Capstone has two groups of concessions with a total of 119 claims which cover a total of 29,221 ha and includes the areas of the planned mine site, plant area, TSF and auxiliary facilities including the port facilities. Concessions are held in the name of Minera Santo Domingo. The information provided supports that the mining tenure held is valid and is sufficient to support declaration of Mineral Resources and Mineral Reserves

Concessions are surveyed as part of the grant process and are protected under Chilean law by payment of the annual mining license fees. All concession fees were current as of 31 March 2024.

Capstone possesses 3 provisional surface rights and 22 definitive surface rights. These easements cover 100% of the facilities and infrastructure area.

The surface land in the districts of Diego de Almagro, Caldera and Chañaral where the Santo Domingo property is located are owned by the State and are managed and represented by the Ministerio de Bienes Nacionales.

Capstone has developed a legal strategy to obtain all necessary surface rights to cover the mine, plant, camp, tailings storage facility, pipelines, port and high voltage transmission lines envisaged in this Report.

There is sufficient suitable land available within the exploitation concessions for the planned tailings disposal, mine waste disposal and mining-related infrastructure such as the open pit, process plant, workshops and offices in this Report.

An application for water rights will not be required for the purpose of raw water supply for the Project. The water for operations in this Report will consist solely of desalinated sea water. A maritime concession has been obtained, which will allow the extraction of sea water. However, Capstone will need to apply for water rights in the Tailings Storage Facility based on its plan to establish a shallow well pumping system to intercept infiltrating process water and transfer that contact water back to the process plant for re-use.

A mining tax will be payable once operations commence and is a sliding-scale tax.

Royalties of 2% NSR are payable to third parties. The NSR is charged on all the metals (copper, iron, gold) recovered. The LOM royalty payments are estimated to be $288M.

Capstone entered into a precious metals purchase agreement with Wheaton Precious Metals (WPM) and received an early deposit of $30 million on April 21, 2021. Additional deposits of $260 million are expected during the Santo Domingo construction period.

Wheaton will receive 100% of Santo Domingo's gold production until 285,000 ounces are delivered, then dropping to 67% of the gold production. As gold is delivered, Capstone receives payments equal to 18% of the spot gold price at the time of delivery for each ounce delivered to Wheaton, until the $290 million deposit is reduced to zero. Payments will then increase to 22% of the spot gold price upon delivery.

To the extent known by Capstone, there are no other significant factors and risks that may affect access, title, or the right or ability to perform work on the property.

25.3 Geology and Mineralization

Mineralization at Santo Domingo occurs primarily in the form of IOCG deposits that form sub-horizontal mantos and lesser veins and breccias hosted by a volcaniclastic and volcanic stratigraphic sequence.

Fault trends at the Santo Domingo property are variably to the north, northwest, northeast and east–west and correspond to high angle faults with limited displacement that generally does not affect continuity of mineralization.

Hydrothermal alteration selectively affects the different lithologies and is characterized by sodic (-calcic), potassic, carbonate and calc-silicate skarn.

Weathering processes are weakly developed with a shallower oxidation zone, which rapidly transitions into a narrow mixed and enrichment zone, and then into the primary sulfide zone.

Drilling at 100 m centres or less at the Santo Domingo Sur deposit has outlined a 150 m to 500 m thick copper bearing, specularite-magnetite-chalcopyrite-pyrite manto sequence hosted within tuffaceous sedimentary rocks and andesitic flows cut by inter-mineral and late dikes, within an area of approximately 1,000 m by 1,200 m. Mineralization occurs in the form of copper-bearing semi-massive to massive specularite and magnetite layers with clots and stringers of chalcopyrite and pyrite. The upper parts of the manto sequence are frequently oxidized and contain various amounts of copper oxides and lesser chalcocite.

The Iris deposit is a narrow zone (100 m to 500 m wide) of copper-bearing iron mantos and breccias extending over 1,200 m that are hosted by andesitic tuffs and andesitic breccias. The dominant iron oxide at Iris is specular hematite and the main copper mineral is chalcopyrite.

Mineralization at Iris Norte is similar to Iris and Santo Domingo but appears to be hosted by altering volcaniclastics and andesitic flows. The deposit has been intruded by significant amounts of diorite dykes and sills and is approximately

1,000 m wide, tested over a strike length of 1,600 m. Mineralization consists of abundant magnetite and lesser hematite-rich mantos. The main sulphides in Iris Norte are pyrite and lesser chalcopyrite.

Drilling at the Estrellita deposit has delineated a tabular body of copper mineralization hosted by breccias, veins and lesser mantos along an east-west fault zone around the Estrellita artisanal mine workings. The east–west extent of the Estrellita deposit along the Santo Domingo fault totals more than 1,000 m and the deposit remains open in both directions. Copper mineralization typically consists of copper oxides such as chrysocolla, malachite, Cu-limonite, black oxides, Cu-sulphates, and Cu-clays, and transitions through a mixed zone of oxides and sulphides that contain chalcocite and then into a primary sulphide zone where the main copper mineral is chalcopyrite.

The QP considers that knowledge of the geology and mineralization is sufficient to support Mineral Resource and Mineral Reserve estimation.

25.4 Exploration, Drilling and Analytical Data Collection in Support of Mineral Resource Estimation

Between 2002 and 2011, work by Far West included a regional airborne geophysical survey, geological mapping, surface, rock and drainage sampling, IP and TEM geophysical surveys, core and RC drilling and resource estimation. In 2013, Capstone Mining flew a ZTEM and VTEM airborne geophysical survey for exploration purposes in Santo Domingo Project area.

Between April and July 2021, a brownfield program of 19 reverse circulation/diamond drill holes totalling 8,546 m was completed mainly for exploration purposes. The brownfield program confirmed the extension of the mineralized sequence at depth between the planned Santo Domingo and Iris Norte pits. A geometallurgy-focussed drill program August through October 2021 involved drilling 8,035 metres in 33 drill holes, mostly twins of previously drilled holes. This campaign mainly collected samples for metallurgical variability testing. Between November 2021 and February 2022, a metallurgical drill program was developed to obtain samples for development of a pilot plant, mainly focused on the first 5 years of production. A total of 8,872 m in 41 drill holes (mainly twin holes) were drilled in 2022.

During 2023, a new mineral resource estimate was carried out. This update considered all the drilling carried out until the end of 2021, which included the brownfield drilling program and part of the drilling from the metallurgical program carried out that same year. Drilling that supports the mineral resource estimate or was used in support of the construction of the geological models, comprises 626 holes (152,785.1 m). The mineral resource estimation excluded various twinned, geotechnical, hydrological, condemnation, and water exploration holes.

Most drill holes are vertical because mineralization is typically horizontal or gently dipping.

Drill cuttings and core were logged using a set of rock type codes. Drill collars were located using a differential GPS. Downhole surveying was conducted using accepted down-hole survey tools.

The QP considers that the drilling has been conducted in a manner consistent with standard industry practices. The spacing and orientation of the holes are appropriate for the deposit geometry and mineralization style.

Reverse circulation drill cuttings were collected at 2 m intervals. Core was nominally sampled at 2 m intervals.

The QP considers that sampling methods are acceptable, meet industry-standard practice, are appropriate for the mineralization style and are acceptable for Mineral Resource estimation. The quality of the analytical data is reliable, and analysis and security are generally performed in accordance with exploration best practices and industry standards.

Logging procedures are well designed and implemented.

Magsus measurements and instrument calibrations are well planned, researched, and understood.

Samples were shipped to an independent analytical and preparation laboratory, ALS Chemex, in Chile and ALS laboratory in Lima, Peru.

Samples were analyzed for 27 elements using ICP. Gold assays were determined using fire assay with an AAS finish. Copper values over 10,000 ppm were re-assayed. All samples over 15% Fe inside the existing block model for which sample material was still available were re-assayed as a check on the Fe analyses as the initial analytical method was considered to have understated the iron content. Soluble copper analysis was conducted on some samples from the 2012 drill program.

Regression formulae based on the specific gravity values reported by Capstone were used to convert volumes to weights, using Fe concentration as the independent variable.

The QA/QC protocols have remained largely consistent throughout all programs conducted by Far West Mining and Capstone.

The database is reasonably free from error and suitable for use in Mineral Resource estimation.

The CRM assays were carried out at an acceptable insertion rate, were reviewed in a timely fashion, and the results triggered reasonable and appropriate responses. The CRM results indicate that the assaying was generally of good quality and acceptable for use in Mineral Resource estimation.

Capstone’s database workflows and controls are both systematic and thorough.

Twin hole drilling has provided a reasonably consistent verification of the earlier drill results, particularly considering the differences in assay protocols and possible survey errors.

The results of the QA/QC programs installed at Santo Domingo support the database used for Mineral Resource estimation.

Regular data verification programs have been undertaken by third-party consultants from 2005 to date on the data collected in support of technical reports on the Property.

The QP considers that, as a result of this work, the data verification findings acceptably support the geological interpretations and the database quality and therefore support the use of the data in Mineral Resource estimation.

25.5 Metallurgical Test Work

Additional metallurgical test work was completed for all main processing areas such as comminution, flotation, iron concentration and tailings, improving general representativeness of the Santo Domingo and Iris Norte orebodies. In detail:

- For the comminution circuit update, a total of 45 additional variability samples were added - this is about 15% increase in SMC compared to previous dataset. This data plus the previous data collected allowed to develop a geometallurgical model that indicated that an increase in throughput was possible during the first 4 years, which was further established within the updated mine plan.
- Flotation test work completed on volcaniclastic units (main units with respect to current reserves), allowed for an optimization of copper flotation reagents which allowed then for a successful pilot plant using conventional cells, reporting 26% Cu with global Cu recovery of 91.1%, and which reported even higher performance for a pneumatic pilot cleaner flotation cell with 29%Cu and global Cu recovery of 93.1% .
- The flotation test work also considered different mineralized geological units and yearly composites covering LOM which demonstrated the feasibility for the ore to produce saleable copper concentrate under the flotation conditions established – copper concentrate grade may also improve when pneumatic technology for cleaner stage is used, as per current flotation circuit design.
- For flotation, 140 variability samples were added to the original 28 variability samples tested with desalinated water during 2019. This allowed for the Cu and Au recovery equation update and the inclusion of acid-soluble copper as an independent input variable.
- From the 140 variability samples, it was also possible to establish a copper concentrate grade model, which has been used to populate the block model.
- With respect to iron concentration, more than 1300 DTT were added to the previous dataset of about 500 DTT, allowing development of block model equations for mass recovery and iron concentrate grade for Santo Domingo and one for Iris Norte.
- Magnetic separation tests were conducted on 39 variability samples following the current processing diagram, demonstrating the feasibility for the Santo Domingo ore to produce iron concentrate >65% Fe. This test also allowed to demonstrate the feasibility for the Santo Domingo ore to produce iron concentrate grade >67% Fe.
- The analysis of the samples with iron concentrate < 62%Fe and their respective geological units, indicated that ore/concentrate blending strategies or operational strategies (such as finer regrind) may be implemented to improve their iron concentrate grade during the plant operation.
- Tailings test work was completed using a representative sample of tailings from copper and iron concentration circuit, which led to a more cost-effective design with only one stage of tailings thickening. Tailings variability samples were tested for FS Optimization in order to identify potential risks and their mitigation strategies.

- Finally, independent and vendor testing was also completed on several areas such as flotation, iron concentration, regrind, concentrate thickening and filtration, as well as tailings thickening which contributed to a more robust design and thus reducing its associated risk.

25.6 Mineral Resource Estimates

The Mineral Resource estimates for Santo Domingo Sur, Iris and Iris Norte were completed in 2023, and the estimates for Estrellita were completed in 2022. The NSR calculations and Mineral Resource tabulations were updated by Capstone in early 2024 using current metal prices, new Cu and Au recovery equations, revised Fe mass recovery, Fe concentrate grade estimates, and NSR calculations, updated cut-off grades, and the application of constraining pit shells constructed using the updated 2024 parameters.

Reasonable prospects for eventual economic extraction were assessed by Capstone for the SD-IN and Estrellita deposits using pit shells constructed based on the 2024 updated parameters.

Regarding the Mineral Resource estimate for Santo Domingo–Iris Norte, the QP makes the following conclusions:

- The new geological model, including updated structure and oxidation modelling, is a significant improvement over the previous model.
- The Mineral Resource database is well constructed and sufficient for Mineral Resource estimation.
- Capping levels for all metals are light, since grade distribution shows long continuous tails at high grades.
- Analysis of metal grades at SD-IN shows correlation between gold and copper, and iron and cobalt, but not iron and copper.
- Dynamic anisotropy was used to interpolate grade along its orientations in the deposit.
- Variograms and visual review show strongest continuity up to 75 m and lesser continuity out to 130 m to 150 m.
- Mass recovery estimates, including the chosen cut-off grades for magnetic susceptibility, are sensible given the intensity and continuity of iron mineralization through the different lithologies.
- Density weighting (DW) produces approximately 2-7% high grade and metal content than non-DW estimation.
- New NSR and class criteria produce more tonnage at slightly higher grade (for Cu and Au) and slightly lower grade for Fe.
- Density sampling is insufficient to split out regression populations by SD and IN, while for the combined SD-IN, three regression populations, for oxide, mixed, and sulphide, have been produced.
- Quadratic curves fit the density information better than linear.
- The oxidation boundaries contain some uncertainty, because the sequential copper dataset contains less complete, irregular coverage - approximately 8% of the samples at SD-IN have soluble copper information - compared to the total copper assays. There are few available soluble copper assays to define a mixed domain.

Regarding Mineral Resource estimate for Estrellita, the QP makes the following conclusions:

- In a comparison of the updated domains, classifications, and resource pit volume versus those from the 2020 resource estimation
- The new Mineral Resource excludes oxide material.
- The inclusion of data from additional drill holes (41 holes for a total of 6,424 m, including 10 twin holes, or 31 holes for a total of 4,908 m without the twins) tends to reduce the average grade.
- The DA interpolation in the block model extrapolates along lithological orientations.
- Grade often truncates against new interpreted fault boundaries, better representing drilled grades.
- The new classification schema encloses larger volumes, especially in Inferred Mineral Resource material.
- Sequential copper analysis suggests that oxide boundaries may be inaccurate in the range of five to fifteen metres in places.
- Mineralization appears to be less restricted by lithology and more restricted by structure than SD-IN but is still banded to some extent.
- Variograms show similar behaviour to IN, but with shorter ranges.
- Updated density regression formulae for copper oxide and sulphide material resulted in small differences from the previous formula.
- There are 130 total drill holes (comprising 25 km of drilling) in the ES Mineral Resource estimate. This includes 41 drill holes drilled in 2007 and 2008 that were not available for the 2007 block model, which was used for the 2018 and 2020 estimates. This includes 27 drill holes that were drilled inside the 2020 modelled pit shell. Inclusion of these 27 additional drill holes results in a potential reduction in average grade of approximately 13%.
- There appears to be little displacement on central fault blocks 32 and 34 and visual validation of the grade estimates supports treating them as one structural domain given the drill density.
- The DA estimation methodology works well to control search orientation while domaining of mineralized material and fault blocks constrains interpolation appropriately.
- Block grade estimation is well controlled overall.

25.7 Mineral Reserve Estimates

Mineral Reserves amenable to open pit mining were constrained by practical pit designs that had been guided by an optimized pit shell (LG). An internal cut-off of $9.77/t milled was applied to all of the Mineral Reserve estimates, as well as during the process of generating the optimized pit shell.

A Mine plan was developed to process generally 72/65 kt/d (26.3 to 23.7 Mt/y) with a peak total mining rate of 90 Mt/y. Inferred Mineral Resources were treated as waste in the mine plan.

Pit optimization and mine planning were performed without introducing any additional factors to account for dilution. Any potential impacts to ore feed that might arise due to ore loss or dilution were not considered material, and no provisions for ore loss or dilution were included in the mine plane.

Proven and Probable Mineral Reserves total 436 Mt grading 0.33% Cu, 0.05 g/t Au and 26.5% Fe.

The main factors that may affect the Mineral Reserve estimates are metal prices, metallurgical recoveries, operating costs (fuel, energy and labour) and block model accuracy. However, the Mineral Reserves are not considered sensitive to moderate changes in these factors, due in part to the use of a 0.84 Revenue Factor LG shell used to guide pit design. Other factors that may impact the Reserves include land tenure or permitting, water supply and geotechnical stability of pit walls and tailings facilities.

25.8 Mine Plan

Pit designs are based on optimized LG shells at a revenue factor of 0.84 with variable overall slope angles according to geotechnical domains ranging from 36.3° to 47.9°.

Ten pit phases are planned: eight for Santo Domingo and two for Iris Norte.

The mine plan will generally process 72,000 t/d to 65,000 t/d of feed (26.3 to 23.7 Mt/y) with a peak total mining rate of 90 Mt/y.

The 18-month pre-production period requires mining 45 Mt of total material to expose sufficient ore to start commercial production in Year 1. To allow for this, pre-production mining activities (pre-stripping) accesses and platforms are needed ahead of time. This is accounted for as an allowance and will be performed by a contractor.

The total mined waste considers two main destinations for the material, the main waste storage areas and the tailings storage facility for the embankment construction.

The mined ore will be hauled to the primary crusher for direct tipping. The mine plan employed an elevated cut off grade and stockpiling strategy during years of surplus production. Blending material will be mined and hauled to a stockpile located between the Santo Domingo and Iris Norte pits. This material will be re-handled and will become part of ore feed to regulate the iron grade. Oxide material is considered as waste in the mine plan.

The mine is scheduled to operate seven days per week for 365 days per year. Each day will consist of two 12-hour shifts. Four mining crews will rotate, two working and two on time off, to cover the operation.

This Report assumes that the mining operation will use electric shovels, hydraulic excavators, front loaders and trucks with a capacity of 290 t. The fleet will be complemented with blasthole drill rigs that will also be used for ore and waste delineation. Auxiliary equipment will include track dozers, wheel dozers, motor graders and water trucks.

25.9 Process Plant

The process flowsheet for the mined ore is based on a conventional copper and magnetite concentrator design, with several changes since the 2020 feasibility study to improve project net revenue and update the design.

The capacity of the process plant for the first 4 years will be approximately 72,000 t/d or 26.3 Mt/y excluding the ramp up period. The throughput after the first 4 years will be approximately 65,000 t/d or 23.7 Mt/y. Variability in throughput has been modelled based on the operating capacity of each major element of the plant flowsheet.

- The comminution circuit is a twin line circuit with each line comprised of an AG mill and ball mill, augmented by a pebble crusher to manage critical size material in the AG mill. The use of an AG mill reduces steel media costs and indirect greenhouse gas emissions. The design is based on the 75th percentile ore characteristics and this provides adequate margin to manage process risk.
- The flotation circuit is a combination of conventional tank rougher cells and Jameson Cells. Jameson Cells were used due to their capital efficiency and reduced gangue entrainment. There is an allowance in the design for extension of the flotation circuit for the flotation of a separate pyrite concentrate as a future project.
- Copper concentrate is dewatered at the mine site with conventional pressure filters and then trucked to the port for storage and shipment.
- Magnetite is recovered to one of two concentrate quality specifications depending on the concentrate grade. The concentrates are stored as slurry at the plant and transferred, in batches, by pipeline to the filter plant located at the port area.
- A single stage tailings thickening process is used to dewater tailings prior to multistage pumping to the TSF for long term storage. A lower thickened tailings density is allowed for in the first two years to allow for early operator familiarisation and training.
- The process plant water requirements will be provided from desalinated water, pumped from the port to the plant site under a BOOT arrangement. Capstone has received expressions of interest for the BOOT arrangement.
- The highest annual production rate of copper concentrate (500 kt) occurs within the first five years of production. The highest annual production of magnetite concentrate is around 4.5 Mt, which occurs in Year 5.

The process plant has been designed using sound engineering principles and the plant layout has resulted in a cost-effective outcome that should provide ease of access, operation, and maintenance.

25.10 Infrastructure

The main project infrastructure sites are greenfield. Some basic infrastructure early works have been carried out at the mine-plant site, such as earthworks platforms, internal site access roads, partially completed C-17 public highway bypass road, industrial water pipeline to site, and high voltage connection at the San Lorenzo substation.

Road access to the mine-plant site and port site is via paved highways. Logistics access to the project are feasible, as demonstrated by the recent construction of a copper concentrator plant in a Capstone's adjacent Mantoverde property.

Electrical power supply will be provided by high voltage power line connections to the national power grid. Capstone has entered into a long-term power purchase agreement (PPA) with one of the major power companies that operate on the national grid and supply several of Chile's major mining companies. This is consistent with the strategy used by other mining companies to contract for long-term electricity supply in Chile. The terms and conditions of the PPA are considered customary and competitive in the Chilean electricity market.

Negotiations have commenced and are ongoing to reach agreements with third parties for the relocation of powerlines interfering with project planned infrastructure.

An agreement is in place with Aguas Nueva Atacama for the supply of water during construction.

The mineral processing facility will use desalinated sea water and recycled water. The current plan is to produce the desalinated water under a build, own, operate, transfer (BOOT) or build, own, operate (BOO) contract with a third party.

In the QP's opinion, all project infrastructure is either contained in owned concessions, right of ways or there are processes to obtain them.

25.11 Markets and Contracts

Market studies were conducted for copper and magnetite concentrates using Subject Matter Experts David Osachoff and David Trotter, respectively.

Santo Domingo will produce a clean copper concentrate with low deleterious material content and a high quality magnetite iron ore concentrate, suitable for pellet feed (bulk and higher grade).

In the QP's opinion, based on the Santo Domingo copper and iron concentrates characterization (grades and composition) and the market reports developed, it is envisaged there will be demand for them.

A streaming agreement with Wheaton Precious Metals is in place for the gold production contained in the copper concentrate.

Capstone has entered into a long-term power purchase agreement (PPA) with one of the major power companies that operate on the national grid and supply several of Chile's major mining companies.

A Build, Own Operate (BOO) contract exists for the mine-plant electrical power supply connection point at San Lorenzo Substation.

The agreements for the supply of water for construction with Aguas Nueva Atacama have been ratified, confirming their validity. Constant communication is maintained.

No other contracts are in place for any other aspects of the Santo Domingo project.

25.12 Environmental, Permitting and Social Considerations

The Santo Domingo Project obtained its environmental licence in 2015 and later, in 2020, another one for the Desalination Plant. The Santo Domingo Project Environmental Impact Study (EIA) included an environmental baseline conducted between 2010 and 2013 for the four project areas defined (Mine-Plant, Pipeline, Port, Electrical Transmission Line), which extend from Santo Domingo and the city of Diego de Almagro to the Punta Roca Blanca sector in Caldera. Studies conducted for the environmental assessments included weather and meteorology, natural risks, soils, noise and vibrations, air quality, water resources, hydrochemistry, marine environment, flora/vegetation/wildlife, natural hazards, and human environment. No critical environmental elements were found during the baseline studies, however, due to the time elapsed, an update of environmental studies could be required within a new environmental permit process for the engineering optimizations currently in progress. Monitoring activities for air and water quality have been carried out since the environmental approval and monitoring of other environmental components will also occur during the upcoming construction phase.

Based on the 2015 and 2020 approvals, Capstone started early works in April 2020. Capstone currently has an environmental licence that allows starting construction and operation under the previously granted Environmental Qualification Resolutions (RCAs) 119/2015 and 122/2020. Capstone has received sectoral approvals for critical permits including the construction and operation of the tailings storage facility (TSF), tailings deposits, process plant, mining method, mine closure plan, as well as maritime concession and preliminary approval for port infrastructure and will continue to obtain all sectoral authorizations required for the Project, based on environmental approvals.

Some engineering optimizations currently in progress, such as the increased production described in this Report, will require Capstone to enter the Environmental Impact Assessment System (SEIA) through an appropriate mechanism, such as an Environmental Impact Declaration (DIA). Once the favourable Environmental Qualification Resolution (RCA) of the optimized Project is obtained, Capstone will proceed to update and obtain all the necessary sectoral permits required for the construction, operation and closure stages of the optimized project.

Recent optimizations that have been made to the approved project scope include:

- A new connection to an alternative existing electrical substation, located adjacent to the original substation. This required a modification to the substation which is already implemented by a third party.
- An updated tailings storage facility design within the area previously approved for the facility.
- Simplified tailings thickening system (one stage thickening instead of two as considered in the 2020 Project).
- Minor optimizations in the rest of the areas (mine, port, pipelines and tailings deposit) these changes would be made within the areas already assessed in the 2015 EIA. Only in the port area additional information associated with a new access road and a small effluent treatment plant for iron concentrate filtrate is being collected.

About 140 works and facilities were identified, distributed between the four areas of study. The number of sectoral permits required for all facilities is estimated to be about 700 in total some of which are classified as critical since they can impact construction milestones. Capstone has started the sectoral permitting process and has received approvals for several of these critical permits such as for the tailings storage facility, the process plant, waste rock dump, the Mine Closure Plan and the exploitation method permit, among others. For the port area, Capstone has received approval for the Maritime Concession permit and preliminary approval for the port infrastructure. These approvals are critical paths and could have delayed construction if not approved.

Capstone is preparing to start construction at Santo Domingo and is developing the first building permit packages and Environmental Management Plans. Capstone will prepare management tools for environmentally sensitive areas, commitments and permits to facilitate management and ensure compliance with environmental, permitting and social commitments.

Similar to the current Mine Closure Plan for the Santo Domingo Project (2019), closure costs related to the execution of the closure measures are estimated at $US124M, which must be reviewed in the next stage of engineering. According to Chilean law, a bond, surety bond or insurance must be provided to the State to cover this cost.

The project's surrounding communities and towns are Diego de Almagro, Chañaral and Caldera. Although there are no Colla or other indigenous lands or territories of any kind being claimed on the Property, Capstone will keep lines of communication open for possible approaches or inquiries from this community.

A stakeholder plan was developed to implement an early community participation program that allowed the local communities involved to understand the major project components and for MSD to gather community opinions, comments and feedback. Meetings were held between 2012 and 2020 in Diego de Almagro, Chañaral, Copiapó and Caldera, including open houses, open meetings, meetings for special interest groups such as fishermen and meetings with authorities, regional and community services as well as with professional organizations. As a result of this process, changes in the design were made to minimize impacts to the environment and surrounding communities.

As the project approaches the construction phase, Capstone communications strategy will focus on building a positive reputation and supportive environment for the development of a mining operation in the Atacama Region. Specific development strategies are directed to the communities of Diego de Almagro and Caldera. A communications plan, communications committee and crisis response management plan are being developed. As part of the strategy and commitments already defined in the EIA, a Community Office will be implemented in Diego de Almagro to support the deployment of the communication strategy with stakeholders.

A health and safety management system has been developed to meet local legal requirements and industry best practices. Capstone will implement policies, standards, plans and security procedures and will use facilities, equipment and personnel required to provide adequate security levels for its staff and facilities.

25.13 Capital Cost Estimates

The estimate is a Type 3 estimate according to ACCE International standards, with an accuracy of -10% to +15% at the 85% confidence level. All capital cost estimates are in second-quarter 2023 US dollars.

The costs in the estimates are based on a combination of direct quotes, benchmarking, and Capstone-supplied data.

The initial capital cost estimate is $2,315M.

The estimated sustaining capital cost is $441M excluding deferred stripping. The combined initial and sustaining capital costs for the LOM are estimated to be $2,756M, excluding deferred stripping.

The estimated sustaining capital cost is $1,329M including deferred stripping. The combined initial and sustaining capital costs for the LOM are estimated to be $3,644M, including deferred stripping.

Closure costs are estimated at $124M.

25.14 Operating Cost Estimate

All operating cost estimates are in second-quarter 2023 US dollars with an accuracy of -10% to +15%.

The total operating cost over the LOM is estimated to be $5,846M.

The LOM average operating cost is $1.32/lb CuEq.

25.15 Economic Analysis

Based on $2,315M of initial capital costs, sustaining capital costs of $441M excluding deferred stripping, closure costs of $124M, total LOM Operating cost of $5,846M, the pre-tax NPV discounted at 8% is US$2.67B, the IRR is 29.7%, and payback period is 2.9 years. On a post-tax basis, the NPV discounted at 8% is US$1.72B, the IRR is 24.1%, and payback period is 3.0 years.

The NPV8% is most sensitive to changes in the copper price (copper grade) and capital expenditures. The sensitivity analysis showed that Santo Domingo is less sensitive to changes in the gold and iron price.

The economic analysis is considered current and suitable to be included in this Report.

Other Relevant Information: Project Execution Plan, Project Execution Duration and Optimization Opportunities

The EPCM (Engineering, Procurement, Construction Management) strategy has an interim engineering phase to prepare for execution. This interim engineering takes place in the year prior to Capstone providing investment approval to commence. Capstone's provides full investment approval triggers detailed engineering and procurement activities for the construction phase.

Production ramp up is complete approximately 4 years after Capstone providing full execution investment approval.

of the following list of optimisation opportunities exist for the Project:

- Flotation of a pyrite concentrate containing cobalt and copper values that could be recovered by hydrometallurgical processing.

- Copper oxide hydrometallurgical processing
- Review of regrind technology and installed power requirements
- Optimization of the magnetic circuit water balance and reduce water flows reporting to the tailings thickener
- Jameson Cell cleaner circuit layout detail in association with Glencore
- Integrated tailings thickeners-TSF assessment
- Mantoverde-Santo Domingo Mining District Integration.

25.16 Risks and Opportunities

25.16.1 Risks

25.16.1.1 Mineral Tenure

The QP has identified no risks related to mineral tenure specific to the Project, although in general terms changes to mineral tenure requirements or political risk such as changes in government regulation relating to the mining industry, including but not limited to nationalization of mines or expropriation without fair compensation would impact mineral tenure.

25.16.1.2 Metallurgical Test Work

Metallurgical test programs are subject to risk, and those relevant to the Santo Domingo project include:

- The ore samples used for the metallurgical program may be non-representative, aged or oxidized, insufficient or otherwise biased,
- The conditions used for the metallurgical testing may not adequately reflect the expected industrial plant conditions, including particle size distributions, water chemistry and reagent concentrations, laboratory equipment characteristics, and equipment operating conditions,
- The test methodologies used may be uncalibrated or biased; these test methods include sample preparation procedures, assaying methods, hardness tests, settling tests, and pilot plant testing, and
- The results may be subject to data handling and other human error.

25.16.1.3 Mineral Resource Estimate

The risk factors that could potentially affect the Mineral Resources estimates at SD-IN and ES include mostly the assumptions used to generate the conceptual data for consideration of reasonable prospects of eventual economic extraction. Those include:

- Commodity price assumptions

- Exchange rate assumptions
- Density assumptions
- Geotechnical and hydrogeological assumptions
- Operating and capital cost assumptions
- Metal recovery assumptions
- Concentrate grade and smelting/refining terms
- Changes in interpretations of mineralization geometry and continuity of mineralization zones.

There are no environmental, permitting, legal, title, taxation, socio-economic, marketing, political or other relevant factors known to the QP, other than those discussed in this Report, that could affect the Mineral Resource estimates.

25.16.1.4 Mineral Reserve Estimates

As reserve models are an estimate based on certain assumptions and interpretations, they have certain inherent risks. Risks to the MSD Mineral Reserve as outlined in this report include, but may not be limited to:

- Changes to the Mineral Resource estimate, potentially resulting from revised interpretation and/or the results of additional drilling and sampling
- Changes to assumptions in metals prices and operating costs
- Adjustment to the Pit Design due to future reinterpretation of geomechanical parameters in some sectors.

25.16.1.5 Mine Plan

Risks to achieve the mine plan include:

- Availability of skilled labour
- Potential increases to ancillary equipment due to slope cleaning, oversize on blasting and mitigation of environmental commitments
- Potential increases in waste movement due to adjustments to pit design
- The mine plan assumes high productivity from the electric shovels that will require double-sided loading whenever reasonably possible. This will require operational discipline to achieve, otherwise an additional loading unit will be required to reach planned production rates.

25.16.1.6 Process Plant

The process risks are typical of all Cu concentrators and magnetite plants and these have been mitigated through adequate test work and design allowances throughout the flowsheet.

There is a risk of bias inherent in the scale-up models and simulators used to select and size the primary processing equipment.

There is a risk that the ore is harder or more competent than expected and this results in lower than planned throughput or coarser than design grind size, but this has been mitigated by designing the plant for the 75th percentile ore characteristics. There is also a risk that the ore is softer or less competent than expected, resulting in larger equipment than required to meet the permitted design throughput, and therefore the overcapitalisation of the comminution circuit.

The arrangement of the Jameson Cells in the cleaner circuit will require detailed consideration in the design phase of the project and discussions between Capstone, the engineer, and the vendor, Glencore Technologies, are already underway.

Further work is required to optimise the water balance in the magnetite plant to reduce the water reporting to the tailings thickener.

The tailings thickener is designed to produce a final tailings density of 67% solids, and this is pumped to the tailings dam via a multistage pumping system. There is precedent in other projects for both lower than expected solids content and pumping issues at high density. The project has two years to achieve the target operating parameters, and this adequately allows for ramp up based on good design.

Geometallurgical modelling has been completed to map plant throughput against ore characteristics by year. The trends are typical of other projects with softer ore processed in the first 4 years. Capstone should continue to develop the geometallurgical models for both throughput and flotation response to improve net revenue prediction.

25.16.1.7 Infrastructure

25.16.1.7.1 Tailings Beach Slopes

The beach slope profiles used for the TSF design have been interpreted based on state-of-the-art tailings slurry rheology testing and beach slope prediction modelling. However, they are best estimates only. The slope predictions are important because after the first few years of operation a substantial portion of the tailings in storage will be in the tailings deposit that extends upslope to the south and east in the TSF basin beneath the slopes. For design purposes, the following slope gradients have been used at this time:

- Year 0 – 2: 0.0% - 0.1%
- Year 3 – 5: 0.3% - 0.5%
- Year 5 – 10: 0.5% - 1.0%
- Year 10 – 20: 1.0% - 1.5%

In selecting these values, care has been taken to avoid overestimating the slopes and therefore risk overestimating the available storage capacity. However, the use of slopes that are too low would result in the need for a substantially larger Main Dam. A balance between these two factors that has been deemed to be suitably conservative based on the information currently available has been made for this design. Further assessments of this will be made in the next stages of design, and throughout the period of operation of the TSF regular beach slope surveys will be conducted to further assess the risk. Adjustments to the staged planning of the tailings deposit and the dams will then be made.

25.16.1.8 Environmental, Permitting and Social

The main risks associated with the environmental, permitting, and social aspects of the Project include:

- Potential for some of the environmental and socio-economic baseline studies and modelling to be out-of-date due to the lapse in time between submission of the EIA (2013) and the present, and also due to the changes in site infrastructure and updated models and assumptions related to climate change.
- Maintaining support for the Project from local communities and stakeholders. A stakeholder identification study was completed and identified several parties that are in the direct and indirect area of influence of the proposed operation. Based on the information provided, during the environmental approval processes several community consultation meetings were held (open houses, open meetings, themed meetings for specialist interests and meetings with authorities and regional and community groups) but there has been little, or no engagement conducted over the last two to three years.
- Maintaining regulatory compliance with all construction and operating permits, as well as social commitments, especially during the start-up construction and operational phases.
- Potential impacts to listed/threatened species and the requirement under the applicable legislation.
- Potential delays in obtaining approvals for the purpose of construction and operation of key infrastructure.

- Due to long linear project structures that are required (i.e., pipelines and roads), there is the potential that installations approved in the EIA may overlap with third party installations.
- The timely implementation of the recommendations presented in Section 26 will help to quantify, qualify, and mitigate these risks to construction and operations schedules.

25.16.2 Opportunities

25.16.2.1 Drilling

Additional drilling in areas where sulphide mineralization is open at depth following the main stratigraphy and in areas inside or near the "reasonable prospects" resource pit, especially near the limestone-volcaniclastic rock contact where higher grades are generally found, represents a potential opportunity for the Project.

25.16.2.2 Mineral Resource Estimate

Opportunities regarding the Resources at SD-IN:

- Drill new holes between SD and IN to delineate the transitional volume, especially considering that the host rock of the mineralization appears to be shallower in this area
- Explore the northwest area of the Santo Domingo model to determine the depth of mineralization and outline any continuity with the adjacent Estrellita deposit.
- Drill additional holes at SD to upgrade Inferred Mineral Resource material to Indicated Mineral Resources
- Continue detailed correlation work on metals by area and lithology
- Review and refine the oxide model contacts in context of additional cyanide and soluble copper assays
- Incorporate lithogeochemical and sequential copper analyses into the workflow to refine oxidation interpretation and ore characteristics.
- Review and potentially incorporate the complete version of the Santo Domingo and Iris Norte structural model into the next iteration for the mineral resource.

Opportunities regarding the Resources at ES:

- Drill diamond holes in key areas to assess metallurgical recoveries, oxide contacts, and fault locations and displacement, and to obtain additional sequential copper data
- Review the amenability of the oxide material to mineral processing to determine if some of the oxide material could be classified as Mineral Resources
- Perform lithogeochemical studies similar to those performed for SD-IN

- Review sequential copper and metallurgical data and revise the oxidation model to better understand the character of the oxide material
- Analyse metal grades at Estrellita to determine correlations between gold, copper, iron, and cobalt

25.16.2.3 Mineral Reserve Estimates

Blocks interpreted as Oxides are not considered for processing in the Mining Plan presented in this report. However, future technical studies that indicate the economic viability of processing these minerals could represent a potential increase in reserves.

The mineralization within the pit shell that is currently classified as Inferred, and therefore set to waste for this study, may, with infill and blast hole drilling during the mining process, be upgraded to higher confidence categories. These resource blocks represent a future opportunity for potential incorporation into the mine plan.

Future studies may be able to demonstrate positive economics from extraction of cobalt, which could have a positive impact on reserves.

The material falling outside the 0.84 RF pit but within the 1.00RF pit may represent a future opportunity to add reserves, particularly if commodity prices move favorably.

25.16.2.4 Metallurgical Test Work

Metallurgical test work on the oxide zone in conjunction with cobalt opportunity may represent a significant opportunity for the development of the project and for the company.

Additional physical characterization test work on later years of the mine plan may represent an opportunity for the project to maintain throughput established for first 4 years.

Additional test work on copper and on iron concentrator with focus on reducing operational cost may represent an opportunity to improve financial results for the project.

25.16.2.5 Mine Plan

The mine plan includes the following opportunities:

- Continue to review the mining fleet to optimize type and size of equipment used.
- Continue analysing the economic optimum of the rate of total mine movement and its impact on the Plant production plan.
- Share equipment with other local companies/projects.

25.16.2.6 Process Plant

The concentrator design includes some allowances to manage risk. These allowances may allow project throughput and production to be increased, if permitting allows.

Capstone is conducting test work on the potential to recover a pyrite concentrate that contains both copper and cobalt. This concentrate could be used to supply acid and ferric ions to a Cu heap leach at the nearby Mantoverde mine site. In addition, the production of cobalt and copper from the oxidation of the pyrite concentrate could provide additional revenue via a separate hydrometallurgical facility. The feasibility of this process is still being assessed by Capstone. An allowance is made in the plant design for the retrofit of the requisite flotation cells for the production, dewatering and pumping of pyrite concentrate.

25.16.2.7 Infrastructure

25.16.2.7.1 Tailings Deposit Stiffness, Strength, and Storage Efficiency

After the initial few years of operation as the tailings deposit enlarges and expands to the south and east, there will be a reduction in its rate of rise and an increase in the extent of air drying of its tailings surface. Both factors can be expected to stiffen, strengthen and increase the density of the tailings deposit, which will increase its storage efficiency. Key to realizing these opportunities will be re-positioning and regularly alternating the active spigot offtake points to maximize the overall deposit area, reduce the comingling of individual streams on the beach, and form thin, uniform layers. A well-managed tailings deposition system will also increase the beach slope gradient to further improve storage.

25.16.2.8 Environmental, Permitting and Community

Opportunities, as listed below, should be considered as the project continues through the development path:

- Timely resumption of community and regulatory engagement regarding the Project. This engagement should include presentation of anticipated impacts (both positive and adverse) and proposed impact mitigation, including discussions with communities on anticipated benefits of the project.
- Timely baseline gap filling for any areas that require additional information for the purpose of future impact assessment and validation of predictions.
- Regarding the need for any additional hydrological, hydrogeological, and geochemical studies, there are opportunities to work closely and collaborate with the geotechnical, water resources, and mineralized material processing engineering teams and, hence, reduce effort and costs.

25.16.2.9 Mantoverde-Santo Domingo Mining District Integration

Opportunities related to Capstone's Mantoverde-Santo Domingo (MV-SD) district integration potential include:

- Port Infrastructure – Opportunity for Capstone's Mantoverde Mine to use the planned Santo Domingo port to ship concentrate. The planned Santo Domingo Port is located 65 kilometres from Mantoverde Mine versus Puerto Angamos (475 km away). Santo Domingo would charge a reasonable fee for use of its planned port; Mantoverde would reduce trucking costs by $10M per year. The planned Santo Domingo port is expected to have sufficient scale to handle capsize vessels suitable for large cargo, including Santo Domingo copper concentrate, iron ore, district cobalt production, and the potential for sulphuric acid handling.
- Integrated Operations – Potential to lower MV-SD district operating costs by $20-30M by optimizing the workforce across both operations, increasing purchasing power given district scale, and standardizing equipment to promote productivity gains.
- Santo Domingo Oxides – Potential addition of 8,000-10,000 tpa of copper production over the first 10 years of production, after evaluating oxide mineralization primarily in pre-strip material for inclusion in MRMR. The potential plan will consider leaching copper oxides at Santo Domingo then processing the concentrated solutions at Mantoverde's underutilized SX-EW facility.
- Ongoing exploration across the MV-SD district, including Sierra Norte
- Cobalt – A district cobalt plant for MV-SD is designed to unlock cobalt production from Mantoverde and Santo Domingo while reducing sulphuric acid consumption and increasing heap leach copper production. The on-site pilot of the continuous counter-current ion exchange (CCIX) commenced in January 2024 at Mantoverde, with potential cobalt production by 2026 after completion of a feasibility study in late 2024. As currently envisioned, a smaller capacity CCIX plant will initially treat cobalt byproduct streams from Mantoverde, and pending sanctioning of Santo Domingo project, the facility will be expanded to accommodate byproduct streams from Santo Domingo.
- Taxation Advantages – Tax synergies between $150-200M may be realized by re-investing cash flows to support Capstone's growth plan in Chile.

26 RECOMMENDATIONS

26.1 Introduction

The list of recommendations is provided by the QP in his respective areas of competence and can be completed concurrently. Total cost estimate for 3rd party services to complete the proposed work programs is $20.3M. This cost estimate excludes Capstone's in-house costs, which are to be incurred as part of the Capstone team's ongoing work on the project prior to execution approval.

The recommended costs are broken down by the categories in this section.

Table 26-1: Recommended Work Costs

Area	($M)*
Exploration & Drilling	6.20
Sample Preparation, Analyses and Security	0.06
Data Verification	0.16
Metallurgical Testing	0.83
Mineral Resource Estimates	0.20
Recovery Methods	0.80
Infrastructure	2.63
Markets & Contracts	0.02
Environmental, Permitting & Community	0.37
Other Relevant Information: Project Execution Plan (Interim Engineering Phase)	9.00
Total	**20.3**

*3rd Party services cost estimate summary for the recommendations, excluding Capstone's in-house costs.

26.2 Exploration and Drilling

To improve the quality of exploration and, drilling data, the QP recommends the following work to be performed:

- Perform additional twin drilling, especially to add sequential copper data to the model areas at a cost of approximately US$6.2M that would allow an improvement in the definition of the oxide, mixed and sulphide zones and the testing and extension of the mineralized zones.

26.3 Sample Preparation, Analyses and Security

The QP makes the following recommendations with respect to sample preparation, analysis, and security:

- Complete further review of past performance of CRMs recently certified for cobalt and sulphur to evaluate a possible impact of values outside acceptable ranges. If reanalysis of batches with newly recognized QA/QC failures is needed, approximately 3,500 analyses for cobalt and sulphur could be required at a cost of approximately US$45K.

- Continue to closely monitor duplicate performance and investigate solutions to improve homogenization of samples or reduce variability through analysis of larger sub-samples for cobalt and gold. The cost of an external study is estimated at US$8K.
- Continue to perform a check sample program at regular intervals to determine if any significant bias has been introduced into the sample assay programs at the main laboratories. Cost estimated at US$8K, cost will vary with the size of subsequent drill campaigns.
- Fully investigate bias in duplicate magnetic susceptibility readings.
- Implement a stronger sample sealing procedure to reduce potential for tampering after the samples leave the logging facility.
- Ensure that density sampling in oxide and mixed material is carried out at regular intervals in all future drilling.

The work without associated costs can be performed in-house by Capstone.

26.4 Data Verification

Regarding data verification, the QP recommends the following:

- Migrate the database from MS Access to an industry standard geological information management system (GIMS). The estimated cost for the database migration is around $0.15M, plus an annual license fee of around $0.05M for GIMS software. Total estimate $0.16M

Regarding data verification, the QP recommends that Project personnel undertake the following as part of their regular duties:

- Continue to validate the drill collars against topography.
- Continue to monitor magsus instrument performance and calibrate magsus instruments with calibration pads in lieu of reference materials.
- Add drilling company and property owner names to the collar table.
- Add a 'Year' field to the collar table.
- Ensure that the final sampleIDs and certificate IDs match the source material.
- Continue with systematic verification and validation of the database to ensure that sampleID supersedence and assay supersedence are captured from the same and separate certificate IDs in ways that are transparent and facilitate reproduction outside of the database software.
- Ensure that the 'best' fields clearly indicate which assay and certificate ID was chosen, and that a flag field serves to mark superseded assays.
- Store the reasons for re-assaying in separate fields.

- Set the LDL values in the compiled ‘best’ fields to half of the '<’ value in the database and continue to monitor for negative values.
- Set the upper detection level (UDL) values in the compiled ‘best’ fields to the '>’ value in the database + 0.0001.

26.5 Metallurgical Test Work

Respect to Metallurgical Test work, the following recommendations are made:

- Completion of additional hardness physical characterization test work on areas where additional reserves were added. Estimated budget US$80K.
- Completion of additional flotation and LIMS test work in areas where additional reserves were added and mixed zone. Estimated budget US$300K.
- Conduct iron concentrate downstream testing for potential marketing requirements. Estimated budget US$450K.

26.6 Mineral Resource Estimates

Regarding the Mineral Resource estimate for Santo Domingo–Iris Norte, the QP makes the following recommendations:

- Correct minor interpretation inconsistencies in the IN geological model where they do not match SD rock types.
- Review sequential copper and metallurgical data further and refine oxide and mixed boundaries.
- Refine redox, geological, and structural model to unify SD and IN. Create one large model spanning the entire block model volume. Re-assay overlimit assays to avoid artificially reducing grade in the model.

Regarding the Mineral Resource estimate for Estrellita, the QP offers the following recommendations:

- Refine the geological and structural model.
- Update the ES geological model using lithogeochemistry as a reference to refine the contacts between the main geological units, similar to what was done at SD-IN.
- Improve the ES weathering model with emphasis on delimiting the mixed and/or enrichment zone between the sulphide and oxide zone, as has been done in SD-IN.
- The majority of this work can be performed in-house by Capstone. Re-assay of over-limits will require them to be performed at a third-party lab. The budget for this work is between $0.1M and $0.25M depending on the extent of the work required. Total cost estimate adopted $0.2M.

26.7 Mineral Reserves

The QP has the following recommendations related to Mineral Reserves:

- Implement a robust ore control system to ensure minimal dilution of the ore grades, particularly for preserving higher grade of Fe concentrate.

26.8 Mine Plan

The QP has the following recommendations related to the mine plan or mining methods:

- Analyse the economic optimum of the rate of total mine movement and its impact on the plant production plan
- Analyse effect of increasing the pre-stripping to optimize the feed to the process plant, improving the revenue generation
- The majority of this work can be completed in-house by Capstone.

26.9 Recovery Methods

The process flowsheet development plan should include the following:

- Continued development of the geometallurgical models for throughput, based on ore competence and hardness, and recovery based on mineralogy and geochemistry to enable revenue forecasting to be further improved.
- Optimisation of the Jameson Cell component design with Glencore Technologies for incorporation into the flowsheet.
- Continued assessments of tailings dewatering behaviour to minimise risks associated with achieving and maintaining the required tailings % solids.
- Finalisation of work on the pyrite processing option for cobalt recovery at Mantoverde.
- Re-evaluation of the technology selection for magnetite regrind to reduce capital costs and improve flexibility.
- Further review of the plant design to improve accessibility, constructability, maintainability and operability and reduce capital costs associated with earthworks and other bulk commodities so that detailed design commences with a clear scope of work.
- Perform column tests on the Santo Domingo oxide resource to determine amenability to heap leaching.

The majority of this work can be completed in-house by Capstone. The Jameson optimisation and design optimisation will require the services of an engineering consulting company. Total cost of the work is estimated to be $0.8M.

26.10 Infrastructure

The QP recommends the following work be undertaken regarding port and site infrastructure:

- Update the logistic study for the project taking into consideration ports that currently are an alternative to handle project cargo. US$25 to $50K. Cost estimate $30K adopted.
- Assess potential acquisition of small land packages currently owned by third parties and located within the project area which might improve the project infrastructure deployment. Assessment performed in-house by Capstone.
- Progress and execute contract agreements with remaining third parties owning infrastructure that need to be relocated to eliminate interferences with planned project infrastructure. Assess through engineering a potential rerouting of these lines considering a corridor allowing for further growing of the TSF, recognizing current mineral resources potential. Performed in-house by Capstone.
- Complete assessment to the port infrastructure to understand maximum vessel size/type allowed for iron concentrate cargo. Around $25K is estimated to conduct this study.
- Assess power supply to the port and desalination plant area from a dedicated and “behind the metre” infrastructure alternative, 100% from renewable sources, avoiding system costs. Performed in-house by Capstone.
- Conduct a semi-quantitative geohazard assessment by a subject matter expert to the TSF and plant area to understand exposure and to plan responses accordingly. Cost Estimate US$100K.
- Plant area complementary geotechnical assessment for foundation design parameters. This geotechnical assessment would typically involve excavation and laboratory analysis of soil material taken from test pits that would be dug in key foundation locations, along with seismic refraction profiles for linear interpolations between test pits. US$300K.

A summary of the recommendations for the next stages of design and planning of the TSF is as follows:

- Additional geotechnical investigations at the dam sites - to verify or modify the depths of removal of loose or undesirable soils including soils with any concerning soluble salt contents as part of the foundation preparation and sub-excavation plan. The engineering associated with this work is estimated to be $700K. Depending on the results of these tests, additional foundation excavation and backfill may be required for the construction of the dams.
- Additional geotechnical investigations in the bedrock ridge on the west side of the site - to verify or modify the current findings of only very minor karst development with small, non-continuous dissolution features that are noted largely on the surface of the rock. The engineering associated with this work is estimated to be $300K. Depending on the results of these tests, additional foundation excavation and backfill may be required for the construction of the dams.
- Conduct a large size triaxial test on a representative sample of mine waste rock to be used as dam construction material – to add confirmation to the strength and stiffness of the rockfill that will be used in the dams (the Chilean authorities are now asking for this type of test on rockfill in TSF dams). A budget of $150K will be required to support the large triaxial test work planned.

- Integrate the findings from the large triaxial test into the next stage of stability and deformation analyses - the results are not expected to have much effect on the dams due to their relatively small sizes, but a check is supported. The engineering associated with this work is estimated to be $150K.
- Installation of additional groundwater wells - around the perimeter of the TSF in a next stage of project planning for stepped up groundwater depth and water quality monitoring. Between $250K and $1M is estimated for the installation depending on the number of wells.
- Update of the seismic hazard assessment – to be done as a matter of good practice since the current one is over 8 years old. This will likely change some of the ground motions for the design earthquakes applied to the TSF dams and these analyses should be updated accordingly, but the dams are unlikely to be significantly affected because they will be constructed out of compacted dense rockfill using the downstream method of raising to produce inherently stable structures. The engineering associated with this work is estimated to be $25K.
- Review and update as necessary of the tailings deposition model – to include for any changes to the mining plan and thus tailings tonnage, density and surface slope values. Significant changes may affect the staged or final limits of the TSF and/or the geomembrane liner as well as the schedule and timing of the stages. The engineering associated with this work is estimated to be $25K.
- Review and update as necessary of the site hydrology and TSF water balance - to include for any the latest climatic data with any recent significant hydrological events in the area. This may change the design PMP and PMF events, which may have some impact on the crest elevation of the dam at each stage. The engineering associated with this work is estimated to be $50K.
- Conduct a local hydrogeological model of the TSF – to assess the groundwater flow regime, seepage rates and potential impacts. This may change the groundwater monitoring design and the mitigation measures. The engineering associated with this work is estimated to be $125K.
- Complete the engineering work to assess the TSF expansion – to support the potential future conversion of mineral resources into mineral reserves. The engineering work at a feasibility level is estimated to be $400K.

26.11 Markets and Contracts

To reduce uncertainty related to markets and contracts, the QP makes the following recommendations:

- Conduct a dynamic simulation for the copper and iron concentrate handling to confirm the infrastructure defined in the project capital cost estimate (equipment, storage area capacity, etc.) both at the process plant and port area, estimated cost for the port area assessment is US$18K. For the process plant area, this scope will be addressed through an interim engineering scope (estimated cost of US$15 to 20K).
- Consider alternatives to avoid penalties related to the copper concentrate grade when copper content drops below 28.59%. Options might involve a production segregation and/or blending with higher grade copper concentrates produced at Capstone's adjacent properties or by others.
- Assess potential for iron concentrate fines through delivering samples to buyers.

- Assess potential optimization of the iron concentrate selling strategy based on the grades of the products expected, premiums, penalties, etc. in order to maximize revenue/value generated.
- Conduct specific marketing study for lower grade iron concentrate (<63% grade) to assess the pricing and potential buyers. The prices estimated by Mr. Trotter are theoretical and in the actual market further discounts may apply.

26.12 Environmental, Permitting and Community

The following recommendations are made with regard to reducing risk and uncertainty in the areas of environmental studies, permitting schedule, and community. Qualified professionals should be retained to oversee the implementation of each of these studies. Implementation of these recommendations will help to mitigate risk and will optimize detailed design considerations and allow for a smooth transition into construction and operations. The following recommendations do not include the baseline studies and modelling costs required to support a cobalt installation and transportation route for sulphuric acid which are currently considered as future opportunities for the Project.

- Conduct a review of environmental and socio-economic baseline studies for adequacy in consideration of the age of some of the baseline data that dates back over 10 years, changes in infrastructure location, and potential effects of climate change. Estimated cost of review: US$7K.
- A review of available geochemical data base should be conducted to confirm the assessment that the ARD/ML risk for the project is low. The review should utilize the existing geological model for the site and results from previous static and kinetic geochemical test work and modelling. Estimated cost of review: US$15K.
- Conduct a review of surface water (water balance), groundwater, and geochemical modelling to ensure models remain current and valid considering changes in infrastructure locations, climate change considerations, and geochemical static and kinetic test results (geochemical source terms). Based on results of the review develop and implement a program. Estimated cost of review: US$30K.
- Revision of quantitative and qualitative groundwater model, geochemical model, and water balance. Note that this estimate assumes availability of adequate groundwater, surface water, and geochemical monitoring data and test results (the QP was not able to verify the availability of these data based on information reviewed). Estimated cost: US$220K.
- Conduct a comprehensive review of currently approved infrastructure and possible direct (overlap) or indirect (within area of influence) with third party installations. If conflicts are identified, mitigate risk by means of minor relocations and/or direct engagement with potentially affected parties. Estimated cost: US$35K.
- To assist with the commencement of construction and operational stages of the Project, environmental constraints mapping should be developed and continuously updated, based on the results of historical and future baseline environmental and community engagement/land use studies. This mapping should be provided to design and construction personnel so it can be utilized to limit risks at the final design and construction/operations stages. The sensitive receptors that should be considered include, but are not limited to, at risk flora and fauna, archaeological and culturally important sites, potential impacts to water quality including surface water (Angostura Creek) and groundwater resources. Adequate and effective environmental management plans associated SOPs

and training modules should be developed as administrative controls used to limit potential impacts and maintain regulatory compliance. Estimated cost: US$60K.

26.13 Capital Cost Estimate

Further refine accuracy of the Capital cost estimate from the current Class 3 estimate (AACE International classification system). Estimated cost of US$9M for interim engineering phase includes this CAPEX refinement.

26.14 Operating Cost Estimate

Further refine accuracy of the OPEX cost estimate from the current estimate. Estimated cost of US$9M for interim engineering phase includes this OPEX refinement.

26.15 Other Relevant Information: Project Execution Plan (Interim Engineering Phase)

The EPCM strategy includes an Interim Engineering phase prior to the execution of the project, to prepare for execution. Cost Estimate for 3rd Party services in interim engineering is $9M.

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