Tetra Tech Sudamérica S.A. | Complex World, Clear Solutions
Evaristo Lillo N° 78, 5th floor, Las Condes | Santiago, Chile
www.tetratech.com | www.tetratech.cl

TETRA TECH

VIZCACHITAS PROJECT PRE-FEASIBILITY STUDY

VALPARAÍSO REGION, CHILE

NI 43-101 TECHNICAL REPORT

Effective Date: February 20, 2023

Report Date: March 30, 2023

Prepared for: Los Andes Copper Ltd.

Prepared by: Tetra Tech Sudamérica S.A.

Severino Modena, BSc, Mining Engineer, MAusIMM, Member of Chilean Mining Commission
Sergio Alvarado, BSc, Geologist, Member of Chilean Mining Commission
Mario Riveros, BSc, Chemical and Industrial Engineer, Member of Chilean Mining Commission
Ricardo Muñoz, BSc, Mining Engineer, MAusIMM, Member of Chilean Mining Commission
María Loreto Romo, BSc, Mining Engineer, Member of Chilean Mining Commission

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TETRA TECH

1. SUMMARY

The Vizcachitas Project (Vizcachitas Project, Vizcachitas or the Project) is a copper-molybdenum porphyry deposit located in central Chile. Tetra Tech Sudamérica S.A. (Tetra Tech) was commissioned by Los Andes Copper Ltd. (Los Andes Copper or the Company) to prepare this Technical Report (TR) at the Pre-Feasibility Study (PFS) level.

1.1 Key Outcomes

The PFS for the Vizcachitas Project considered copper, molybdenum and silver prices of US$3.68/lb Cu, US$12.90/lb Mo and US$21.79/troy ounce Ag, resulting in the Mineral Resources presented in Table 1.1, Mineral Reserves presented in Table 1.2, key Project financial metrics presented in Table 1.3 and summarized cash flows presented in Table 1.4. Copper contributes 88% to the net revenue followed by molybdenum with 10% and the balance being silver credits in copper concentrate.

Table 1.1: Mineral Resources

Resource Classification @ 0.25% Cu Cut-Off Grade	Tonnage (Mt)	Cu (%)	Mo (ppm)	Ag (g/t)	CuEq (%)	Cu (Mlb)	Mo (Mlb)	Ag (Moz)	CuEq (Mlb)
Measured Resources	273	0.433	139	1.3	0.482	2,605	84	11.0	2,900
Indicated Resources	1,268	0.373	158	1.0	0.426	10,416	442	42.8	11,901
Measured & Indicated Resources	1,541	0.383	155	1.1	0.436	13,021	526	53.8	14,801
Inferred Resources	1,823	0.342	123	0.9	0.384	13,747	495	55.3	15,444

Table 1.2: Mineral Reserves

Category	Tonnage (Mt)	Grade				Contained Metal			
		Cu (%)	Mo (ppm)	Ag (g/t)	CuEq (%)	Cu (Mlb)	Mo (Mlb)	Ag (Moz)	CuEq (Mlb)
Proven	302	0.41	135	1.2	0.45	2,714	89.8	11.9	3,031
Probable	918	0.34	136	1.1	0.39	6,908	275.3	31.8	7,858
Proven & Probable	1,220	0.36	136	1.1	0.40	9,623	365.0	43.6	10,889

Table 1.3: Key Project Financial Metrics

Metric	UoM	Pre-Tax	After-Tax
Undiscounted Cash Flow (LOM)	US$ million	9,484	13,128
Net Present Value (8%, 2023_{real})	US$ million	3,999	2,776
Internal Rate of Return	%	28.5%	24.2%
Payback Period from First Production	Years	2.3	2.5

Table 1.4: Cash Flow Highlights (US$ million)

Category	First 8 years	Life of mine
Copper-Silver Sales Income	11,879	32,249
Molybdenum Sales Income	1,097	3,526
Silver Sales Income	251	713
Selling Expenses and Deductions	(1,943)	(5,363)
Total Revenue	**11,284**	**31,125**
Operating Costs	(3,669)	(12,921)
Operating Profit	**7,615**	**18,204**
Taxes and Royalty	(1,446)	(4,521)
Capital costs	(3,428)	(4,199)
Cash Flow (post-tax)	**2,741**	**9,484**

The NPV is most sensitive to the copper price, followed by the discount rate applied and the total operating costs. Table 1.5 shows the Project sensitivities to variations in copper price and discount rate.

Table 1.5: Sensitivity Analysis – Copper Price and Discount Rate

NPV (After-Tax) US$ Million		Copper Price (US$/lb)				
		2.75	3.22	3.68	4.34	5.00
Discount Rate (%)	5.0%	1,604	2,978	4,332	6,216	8,076
	6.5%	1,155	2,319	3,463	5,056	6,628
	8.0%	800	1,796	2,776	4,137	5,480
	9.5%	516	1,378	2,224	3,399	4,559
	11.0%	288	1,039	1,778	2,801	3,811

1.2 Introduction

Tetra Tech was commissioned by Los Andes Copper to prepare a TR for the Vizcachitas Project the standards required by Canadian National Instrument 43-101 (NI 43-101). The capital cost estimates in this report are in line with Association for Advancement of Cost Engineering International (AACEI) guidelines for a Class 4 study, with an accuracy range of +/-25%. This TR, with an Effective Date of February 20, 2023, reports the first estimate of Mineral Reserves.

A previous Preliminary Economic Assessment (PEA) was completed in June 2019. The most significant changes in this report compared to the 2019 PEA are:

- The initial Proven and Probable Reserves of 1.22 billion tonnes at 0.36% copper, 136 ppm molybdenum, 1.1 g/t silver, which equates to a copper equivalent (CuEq) grade of 0.40% (Proven Reserves of 302 million tonnes at 0.41% copper, 135 ppm molybdenum, 1.2 g/t silver; and Probable Reserves of 917 million tonnes at 0.34% copper, 136 ppm molybdenum, 1.1 g/t silver).
- Increase in Measured and Indicated Resources by 16% to 14.8 billion lb CuEq (Measured Resources of 2.605 billion lb copper, 84 million lb molybdenum and 11 million oz silver, and Indicated Resources of 10.416 billion lb of copper, 442 million lb of molybdenum, and 43 million oz of silver) and increase of Inferred Resources by 130% to 15.4 billion lb CuEq (13.747 billion lb copper, 495 million lb molybdenum, 55 million oz silver).
- Initial Life of Mine (LOM) of 26 years producing 8.763 billion lb copper, 273.3 million lb molybdenum and 32.7 million oz silver, based on a new plant design with a mill throughput of 136,000 tpd and a LOM annual average production of 152,883 t of Cu.
- A US$2.776 billion after-tax net present value (NPV) using an 8% discount rate and an internal rate of return (IRR) of 24.2% at US$ 3.68/lb copper, US$12.9/lb molybdenum and US$21.79/oz silver, with an estimated initial capital cost of US$2.44 billion, a construction period of 3.25 years and a payback period of 2.5 years from initial production.
- Use of desalinated water, eliminating the need to draw on continental water. Using dry stacked filtered tailings reduces water consumption by approximately 50% and eliminates the need for a tailing dam, minimizing seismic and environmental risks.
- Use of high pressure grinding roll (HPGR) technology, reducing power consumption by approximately 25% from the previous design. Scope 1 CO_2 emissions are projected to be 1.02 h CO_2e/t CuEq, and Scope 2 emissions are projected to be 0.

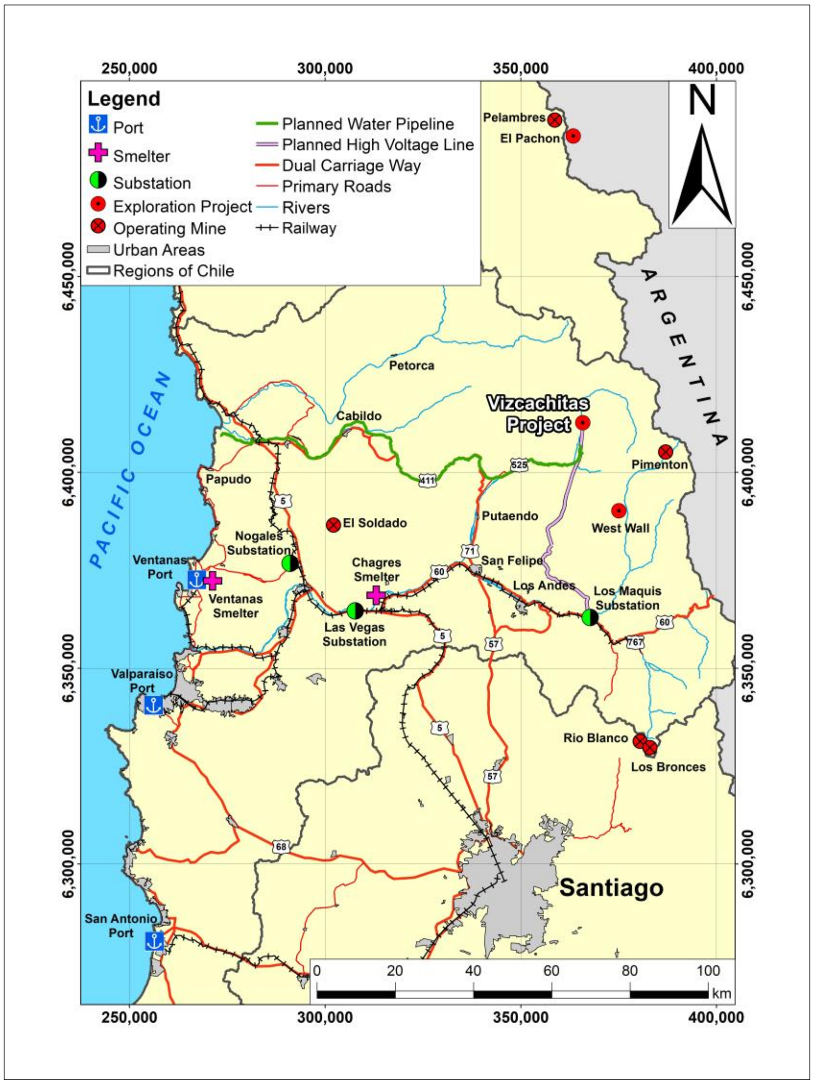
1.3 Terms of References

On February 23, 2023, Los Andes Copper issued the news release, Los Andes Copper Announces Positive PFS for Vizcachitas with a US$2.77 Billion Post-Tax NPV and 24% IRR, announcing for the first time Mineral Reserves on the Vizcachitas Property. This Technical Report is being filed to support the news release above. Currency in this TR is expressed in US dollars (US$ or USD) and units are metric units, unless otherwise indicated.

1.4 Project Description and Location

Vizcachitas is located at 32° 24' 27'' S and 70° 25' 30'' W in the Andes Mountains, Chile. The UTM coordinates at the centre of the property are 366.000mE 6.413.500mN. (Datum WGS84). The Project is located approximately 150 km northeast of Santiago, Chile and 46 km north-east of Putaendo, San Felipe Province (Figure 1.1). Of the total distance between the Project and Santiago, approximately 125 km is paved, and 25 km is unimproved dirt and gravel roads.

Figure 1.1: Vizcachitas Project Location



Source: Los Andes Copper, 2023.

The Project includes 52 mining exploitation concessions covering a surface area of 10,771 ha and 175 exploration concessions covering a surface area of 48,600 ha. All concessions have been granted or are in the process of being granted by the court of Putaendo. Compañía Minera Vizcachitas Holding (CMVH) and Sociedad Legal Minera San José Uno de Lo Vicuña, El Tártaro y Piguchén de Putaendo (SLM San José), both wholly-owned subsidiaries of Los Andes Copper, hold favourable and valid title deeds to the mining concessions above.

1.5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

There is year-round access to the Vizcachitas Project using a four-wheel drive vehicle, currently subject to sporadic interruptions following spring storms or run-off when excessive flow in the Rocín River prevents crossing the river.

The weather is warm and temperate with 6 dry months from late spring to fall (November to April). Average precipitation is about 300 mm per year and falls mostly as rain or snow between April and October. Summer temperatures vary from a few degrees above zero at night to 35°C during the day. Winter temperatures vary between 0°C and 15°C. The relatively low elevation and favourable climate allow year-round exploration, drilling and operation.

The Project is in the Andes Mountains, with elevations ranging from less than 1,800 masl to more than 3,400 masl, with an average elevation of around 1,950 masl. Vegetation consists of shrubs and trees of low to moderate height, which mainly grow in the bottom of the valley near the river.

1.6 History

The central mining claim, San José 1/3000 (San José), was claimed in the 1970s. Placer Dome Sudamerica Limited (Placer) reviewed the Project in 1992 and signed an option agreement in 1993. In 1995 General Mineral Corporation (GMC) acquired 51% of the San José claim and entered into an option agreement for other mining claims for the Project.

Lumina Copper Corp. purchased GMC's subsidiary Vizcachitas Limited in late 2003, including the shares of CMVH. In May 2005 Vizcachitas Limited was transferred to Global Copper Corporation (Global), one of four successor companies of Lumina Copper Corp. In November 2006 GHG Resources Limited (GHG) entered into an agreement with Global to acquire all Global's interests in the Vizcachitas property. After the purchase GHG was renamed Los Andes Copper Ltd.

In 1992 Placer carried out mapping and sampling programmes followed by six diamond drill holes totalling 1,953 m. Between 1996 and 1998 GMC conducted detailed mapping, sampling, geophysical studies and drilling of 61 diamond drill holes totalling 15,815 m. Los Andes Copper completed the following drilling campaigns:

- Between 2007 and 2008, 79 drill holes for a total of 22,616 m
- Between 2015 and 2017, 19 drill holes for a total of 11,872 m
- Between 2021 and 2022, 17 drill holes for a total of 8,668 m.

1.7 Geological Setting and Mineralization

The Vizcachitas Project is located within the Neogene (23 to 2.5 million years ago (Ma)) metallogenic belt that extends along the slopes of the Andes Mountains in Chile and Argentina.

In central Chile this metallogenic belt includes world-class copper-molybdenum porphyries such as Los Pelambres-El Pachón located 75 km north of the Vizcachitas Project, Río Blanco-Los Bronces located 80 km to the south, and El Teniente located 180 km to the south.

The Project is a complex set of porphyries and hydrothermal breccias intruded into a sequence of andesitic and daciandesitic volcanic rocks of the Abanico Formation that have been dated between 34 and 20 Ma.

The productive complex starts with a diorite with early porphyry character, compositionally fine to medium-grained quartz diorites dominate and are dated between 12.5 to 12.58 Ma. The inter-mineral tonalitic and granodioritic phases are dated between 11.8 and 12.08 Ma. Cutting through the volcanic and intrusive units are hydrothermal and magmatic-hydrothermal breccias. The final post-mineral phase of the intrusive complex is a phreatomagmatic breccia (or diatreme) that intruded into the central part of the Project.

The mineralization shows classic zonation with a leached zone above the secondary enrichment zone (or supergene zone) of weak to moderate intensity with chalcocite and covellite. The supergene thickness varies between 2 m and 100 m and the average drilled grade is 0.47% Cu. The hypogene (or primary mineralization) is mainly chalcopyrite with pyrite. Bornite occurs in several of the drill holes below 800 m.

Veinlets are estimated to control 70% of the copper mineralization and 90% of the molybdenum mineralization. The copper mineralization correlates mainly with the intensity of the EDM-type, A-type and C-type veinlets. The B-type veinlets are the main contributor to molybdenum mineralization.

The Vizcachitas Project remains open to the east, west, and at depth.

1.8 Deposit Type

The Vizcachitas deposit is a classic Andean-style porphyry copper-molybdenum deposit. These deposits contain large masses of hydrothermally altered rocks, sulphide-bearing small veinlets, disseminated sulphides and stockworks that may cover several square kilometres. These altered areas commonly coincide with shallow intrusives, and hydrothermal and intrusive breccias.

1.9 Exploration

From the beginning of the 1990s to date three companies have explored the Property: Placer Dome, General Minerals Corporation and Los Andes Copper. This work has included surface mapping, geochemical surface sampling, age dating and an Induced Polarization/Resistivity and Magneto-Telluric survey.

The exploration confirmed that Vizcachitas is a zoned porphyry copper system with early central potassic alteration zoned out to distal propylitic alteration and up into a halo of phyllic alteration. The phyllic zone dominates at the surface, it is of medium size and intensity and measures about

1.5 km in diameter. The advanced argillic alteration is preserved at high elevations and along structures. The potassic zone contains the strongest mineralization.

The IP/Resistivity survey correlated well with the copper mineralization. The lower grade diatreme and inter-mineral intrusive were also reflected in the resistivity data. The survey shows an untested conductive zone extending 750 m north from the northernmost drill hole and conductive zones to the east of the diatreme and along the Campamento fault to the east. Along with the surface mapping a zone of 1,000 m x 500 m of potentially mineralized rock has not been conclusively drill tested north of the phyllic alteration.

The exploration has also identified areas that warrant further work, such as the Breccia Sericita area at the northern end of the Rocín valley with sericite-altered and tourmaline-cemented hydrothermal breccias on the surface, similar to that found on the Vizcachitas Project. There is also a large colour anomaly located 8 km south-west of Vizcachitas covering an area of approximately 2 km x 5 km.

1.10 Drilling

Since 1993, 182 diamond drill holes have been drilled on the Property with a total of 60,924 m. The total metres drilled by each company are summarized in Table 1.6.

Table 1.6: Summary of Drilling Campaigns

Company	Period	Drill Hole Code	N° of Drill Holes	Total Metres
Placer Dome	1993	VP-1 to VP-6	6	1,953
General Minerals	1996 - 1997	V-01 to V-63	61	15,815
Los Andes Copper	2007 - 2008	LAV-064 to LAV-142	79	22,616
Los Andes Copper	2015 - 2017	V2015-01 to V2017-11	19	11,872
Los Andes Copper	2021 - 2022	CMV-001B to CMV-018	17	8,668
Total			182	60,924

1.11 Sample Preparation, Analysis and Security

A complete QA/QC system and protocols were implemented covering the sampling and assaying procedures for the drilling programmes carried out by Los Andes Copper and General Minerals Corporation at the Vizcachitas Project.

These QA/QC programmes were designed to ensure that the sampling, mechanical preparation and analysis of the samples obtained during drilling were performed with acceptable quality. In addition, accuracy, precision and contamination were evaluated using control materials such as certified reference materials (CRM), twin samples, coarse duplicates, pulp duplicates and blanks.

The logging, survey, sampling and assay data were captured, managed and stored in an acQuire Technology Solutions Pty. Ltd, GIM Suite geological data management software running on a Microsoft SQL database. This database has ensured consistency in data handling and storage.

The QA/QC system implemented for the sampling, mechanical preparation and analytical results meets or exceeds mining industry best practices.

1.12 Data Verification

The Qualified Person (QP), Sergio Alvarado, visited the Project site, core cutting facility and warehouse on March 22 to 24, 2022. During these visits he reviewed the drill rigs at the Project site and the core processing from the drill rig to storage in the Quilicura warehouse.

In the opinion of the QP, the geological and geochemical data reviewed are an adequate and accurate reflection of the geology of the Vizcachitas Project. The data reviewed meets the standards of an NI 43-101 Technical Report for use in the Mineral Resource estimation.

1.13 Mineral Processing and Metallurgical Testing

The Vizcachitas Project has been studied since 1998 over several metallurgical testing programmes. The objective of these programmes has been to characterize the plant feed (sulphide or oxide) and define the process recovery method (froth flotation). The testwork programmes were performed by SGS Minerals Chile and Lakefield Research.

Vizcachitas is a primary sulphide that is composed mainly of chalcopyrite and pyrite. Chalcocite and covellite are observed only as a superficial oxidation process typical of zones above the water table, as thin coatings ("patinas") over the primary sulphides. Copper sulphides are mainly liberated and the main association is with hard silicates and phyllosilicates.

The Vizcachitas material has less than 1.3% of clays (kaolinite and montmorillonite). Minerals that contain deleterious elements such as Pb, As and Sb, are present at less than 1% and are mainly associated with pyrite.

Comminution parameters show that Vizcachitas is medium hard; Axb and the Bond Ball Work Index (BWi) are 38.0 and 11.38 kWh/t, respectively. It is planned to use desalinated water in the process plant.

A strong frother must be used in the reagent formula to ensure stable bubble swarms and a steady froth that can carry coarser particles (larger than 240 µm). Operating the rougher stage with pH between 7.0 and 7.5 is recommended to enhance molybdenum recoveries.

The PFS Vizcachitas reagent formula increases rougher copper and molybdenum recoveries by 3.1% and 8.6%, respectively, compared with the reagent formula used in the PEA.

The main association of copper sulphides is with silicates, lower regrinding target sizes must be achieved (25 µm) to liberate the sulphides and produce a clean copper concentrate.

A single stage of cleaner/scavenger flotation is planned. Expected recoveries for this stage are 22.2% for mass, 98.5% for Cu and 95.8% for Mo. No pyrite activation was observed in the locked cycle tests (LCT) at the laboratory scale.

Vizcachitas can produce clean concentrates with approximately 24% Cu, 0.54% Mo, 24.9% Fe and 30.3% S. Silver credits are identified, but no gold credits are expected. No deleterious elements are present in amounts that could lead to penalties.

The effects of sample ageing on recoveries were observed. The rougher recovery decay rate is 0.5%/month for Cu and 0.9%/month for Mo. This should be considered in future sample preparation, stockpile management design and in the mine plan.

Vizcachitas tailings are suitable to be filtered and dry stacked. SGS and Takraf tests showed excellent filtration rates compared with other operations worldwide. This is favoured by the coarser grind size (240 µm) and the lack of clays and ultra-fine particles. The expected cake moisture is 15%.

1.14 Mineral Resource Estimate

For the purpose of Mineral Resource estimation, a Whittle pit shell was prepared to constrain the estimated resource blocks using the general technical and financial assumptions listed below:

- Plant cost: US$5.4/t
- Energy cost: US$65/MWh
- Mine cost: US$1.58/t
- Cu selling cost: US$0.48/lb
- Mo selling cost: US$1.680/lb
- Ag selling cost: US$2.5/oz
- Cu recovery: Variable by lithology and section, averaging 91.1%
- Mo recovery: Variable by lithology, averaging 74.8%
- Ag recovery: 75.0%
- Material to concentrate: Supergene + Hypogene
- Cu price: US$3.68/lb
- Mo price: US$12.90/lb
- Ag Price: US$22.00/oz
- Pit sloped angles: 44º to 52º

The Mineral Resources are contained within an open pit shell to demonstrate the prospects of eventual economic extraction. Only blocks within the Whittle pit shell are included in the Mineral Resources.

Table 1.7, Table 1.8, Table 1.9 and Table 1.10 present the Mineral Resources for different copper cut-off grades.

Table 1.7: Measured Resources In-Pit, Cut-Off Cu

Measured Resources									
Cut-Off (Cu %)	Tonnage (Mt)	Cu (%)	Mo (ppm)	Ag (g/t)	CuEq (%)	Cu (Mlb)	Mo (Mlb)	Ag (Moz)	CuEq (Mlb)
0.18	317	0.403	135	1.2	0.451	2,821	95	12.3	3,154
0.20	308	0.409	136	1.2	0.457	2,784	93	12.1	3,110
0.25	273	0.433	139	1.3	0.482	2,605	84	11.0	2,900
0.30	226	0.466	138	1.3	0.515	2,320	69	9.3	2,564
0.35	180	0.502	137	1.3	0.551	1,991	54	7.6	2,186

Table 1.8: Indicated Resources In-Pit, Cut-Off Cu

Indicated Resource									
Cut-Off (Cu %)	Tonnage (Mt)	Cu (%)	Mo (ppm)	Ag (g/t)	CuEq (%)	Cu (Mlb)	Mo (Mlb)	Ag (Moz)	CuEq (Mlb)
0.18	1,606	0.340	150	1.0	0.390	12,036	530	51.6	13,815
0.20	1,525	0.348	151	1.0	0.399	11,697	509	49.7	13,408
0.25	1,268	0.373	158	1.0	0.426	10,416	442	42.8	11,901
0.30	951	0.405	164	1.1	0.460	8,492	343	33.3	9,643
0.35	644	0.444	171	1.1	0.501	6,298	243	23.6	7,113

Table 1.9: Measured and Indicated Resources In-Pit, Cut-Off Cu

Measured and Indicated Resources									
Cut-Off (Cu %)	Tonnage (Mt)	Cu (%)	Mo (ppm)	Ag (g/t)	CuEq (%)	Cu (Mlb)	Mo (Mlb)	Ag (Moz)	CuEq (Mlb)
0.18	1,923	0.351	147	1.0	0.400	14,859	624	63.9	16,972
0.20	1,833	0.358	149	1.0	0.409	14,480	602	61.8	16,517
0.25	1,541	0.383	155	1.1	0.436	13,021	526	53.8	14,801
0.30	1,176	0.417	159	1.1	0.471	10,812	412	42.6	12,208
0.35	824	0.457	164	1.2	0.512	8,288	297	31.2	9,298

Table 1.10: Inferred Resources In-Pit, Cut-Off Cu

Inferred Resource									
Cut-Off (Cu %)	Tonnage (Mt)	Cu (%)	Mo (ppm)	Ag (g/t)	CuEq (%)	Cu (Mlb)	Mo (Mlb)	Ag (Moz)	CuEq (Mlb)
0.18	2,956	0.294	112	0.8	0.333	19,187	729	80.2	21,683
0.20	2,764	0.302	114	0.9	0.340	18,376	693	76.3	20,748
0.25	1,823	0.342	123	0.9	0.384	13,747	495	55.3	15,444
0.30	1,180	0.379	129	1.0	0.423	9,853	336	38.4	11,009
0.35	655	0.423	142	1.1	0.472	6,117	205	23.3	6,824

Notes:
1. Mineral Resources were classified using CIM Definition Standards (2014), and CIM Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines (2019).
2. The Mineral Resources effective date is February 7, 2023
3. Mineral Resources are inclusive of Mineral Reserves.
4. Copper equivalent grade has been calculated using the following formula: CuEq (%) = Cu (%) 0.000288 x Mo (ppm) + 0.00711 x Ag (g/t), using the metal prices: US$3.68/lb Cu, US$12.9/lb Mo and US$21.79/oz Ag, with metallurgical recoveries of 91.1% for copper, 74.8% for molybdenum and 75% for silver based on the PFS metallurgical testwork.
5. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
6. The quantities and grades of reported Inferred Mineral Resources are uncertain in nature, and further exploration may not result in their upgrading to Indicated or Measured status.
7. Mineral Resources were prepared by María Loreto Romo and Severino Modena both full-time employees of Tetra Tech Sudamérica, and Ricardo Muñoz, a consultant part of the Tetra Tech Sudamérica team, all are Qualified Persons as defined by National Instrument 43-101.
8. Due to rounding, numbers may not add precisely to the totals.
9. All Mineral Resources are assessed for reasonable prospects for eventual economic extraction (RPEEE).

Therefore, using a cut-off base case of 0.25% Cu, the in-pit Mineral Resources are:

- Measured Mineral Resources: 273 million tonnes grading 0.433% Cu, 139 ppm Mo and 1.3 g/t Ag for a 0.482% Cu equivalent (CuEq)
- Indicated Mineral Resources: 1,268 million tonnes grading 0.373% Cu, 158 ppm Mo and 1.0 g/t Ag for a 0.426% CuEq
- Measured and Indicated Mineral Resources: 1,541 million tonnes grading 0.383% Cu, 155 ppm Mo and 1.1 g/t Ag for a 0.436% CuEq
- Inferred Mineral Resources: 1,823 million tonnes grading 0.342% Cu, 123 ppm Mo and 0.9 g/t Ag for a 0.384% CuEq.

1.15 Mineral Reserves Estimate

Mineral Reserves are reported under the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), *Definition Standards* (2014) and *Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines* (2019). Table 1.11 summarizes the Mineral Reserves as of December 2, 2022.

Table 1.11: Mineral Reserve Statement

Category	Tonnage (Mt)	Grade				Contained Metal			
		Cu (%)	Mo (ppm)	Ag (g/t)	CuEq (%)	Cu (Mlb)	Mo (Mlb)	Ag (Moz)	CuEq (Mlb)
Proven	302	0.41	135	1.2	0.45	2,714	89.8	11.9	3,031
Probable	918	0.34	136	1.1	0.39	6,908	275.3	31.8	7,858
Proven & Probable	1,220	0.36	136	1.1	0.40	9,623	365.0	43.6	10,889

Notes:

1. Mineral Reserves were classified using CIM Definition Standards (2014), and CIM Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines (2019).
2. Mineral Reserves have an effective date of December 2, 2022
3. Mineral Reserves are included within the Mineral Resources
4. The Qualified Person for the estimate is Mr. Severino Modena, BSc, Mining Engineer, MAusIMM, Member of the Chilean Mining Commission, and Tetra Tech Sudamérica General Manager
5. The Mineral Reserves have a metallurgical cut-off based on the process plant design of 0.18% Cu for direct mill feed
6. Due to rounding, numbers may not add precisely to the totals
7. The Mineral Reserves estimate uses a marginal phase analysis through a cut-off grade optimization software (COMET)
8. The Mineral Reserves are contained within operational phases defined using a COMET optimized mining schedule, which includes a stockpiling strategy. Key inputs for this process are:
   i. Metal prices of US$3.5/lb copper and US$12/lb molybdenum
   ii. Mining Cost of US$1.59/t at a reference elevation of 1,990 masl, plus costs adjustments of US$0.014/t per bench above reference and US$0.032/t per bench below reference
   iii. Process cost of US$5.7/t milled (inclusive of general and administrative costs of US$0.30/t milled)
   iv. Overall pit slopes angles varying from 44° to 52°
9. Process recoveries are based on lithology for both copper and molybdenum, except for one sector with a fixed copper recovery
10. Cu grades are reported as percentages, Mo and Ag grades are reported as parts per million (ppm)
11. The strip ratio (waste:ore) is 2.33. There are 2,855 Mt of waste in the ultimate pit
12. The Mineral Reserve statement considers the mill feed at the primary crusher as a reference point.

In Tetra Tech's opinion, there are no modifying factors – including environmental, permitting, legal, title, tax, socioeconomic, market, or political – that affect the Mineral Reserve estimate, other than the analysis and considerations presented in this TR. The Mineral Reserve estimate could be materially affected by future changes in the modifying factors.

1.16 Mining Methods

An open pit mining method has been selected for the Vizcachitas deposit mainly based on the copper and molybdenum grades and the continuity of the mineralization occurring near the surface which results in a low strip ratio for open pit mining.

The mine has been scheduled to operate 365 days per year. The plan is based on two 12 hour shifts per day. Mining operations include in-house drilling, blasting, loading, hauling and earthworks, as well as some outsourced services.

The annual operating period for the mine considers negligible snow related downtime. The mining operation will be located at an average elevation of 1,950 masl, with potentially minor heavy rainfall and snow events. The roads and working areas can be promptly cleared using the support equipment included for the operation.

The final pit was optimized using the existing resource model following the economic and geotechnical parameters defined for the Project. This exercise was performed using the Lerchs-Grossman algorithm applying the Whittle4X software. The mine plan schedule was prepared using the COMET strategic planning software. The strategic mine plans considered the following conditions:

- Bench height: 15 m (the same block height as the PFS resource model)
- Maximum vertical head by phase: 10 benches per year, equivalent to a maximum vertical development of 150 m per year
- Minimum number of active phases by period: two, except for the last years of the mine plan, when only one phase is operated
- Number of stockpiles: three; high-grade (HG), medium-grade (MG) and low-grade (LG)
- The mill begins production in the first quarter of the fourth year after construction starts
- Metallurgical cut-off grade: 0.18% Cu
- Maximum number of active phases per period: five
- Some benches at the top of each phase will be mined by contractors; this is not included in the strategic mine plans. The main reason for using contractors is to remove waste on narrow mining widths at the start of each phase with smaller equipment.

The mine plan options were assessed at a detailed level, determining the hauling distances for mineralized material and waste, and then estimating the equipment fleet and the purchasing requirements over time.

The study considered the best set of mine phase designs suitable for 136,000 tpd with an autonomous haulage operation (AHS). Each phase was designed with its own access roads to enable haulage from the loading area to the destination (primary crusher or waste dump). The access roads for the top benches were designed with a width of 25 m to operate small trucks in narrow areas. Earthmoving contractors will mine those areas. Other roads were designed with a width of 38 m to operate high capacity AHS mining trucks.

To guarantee slope stability the mine design includes catch berms of 25 m along the wall slope every eight benches in poor quality rock zones and every 15 benches in good quality rock zones.

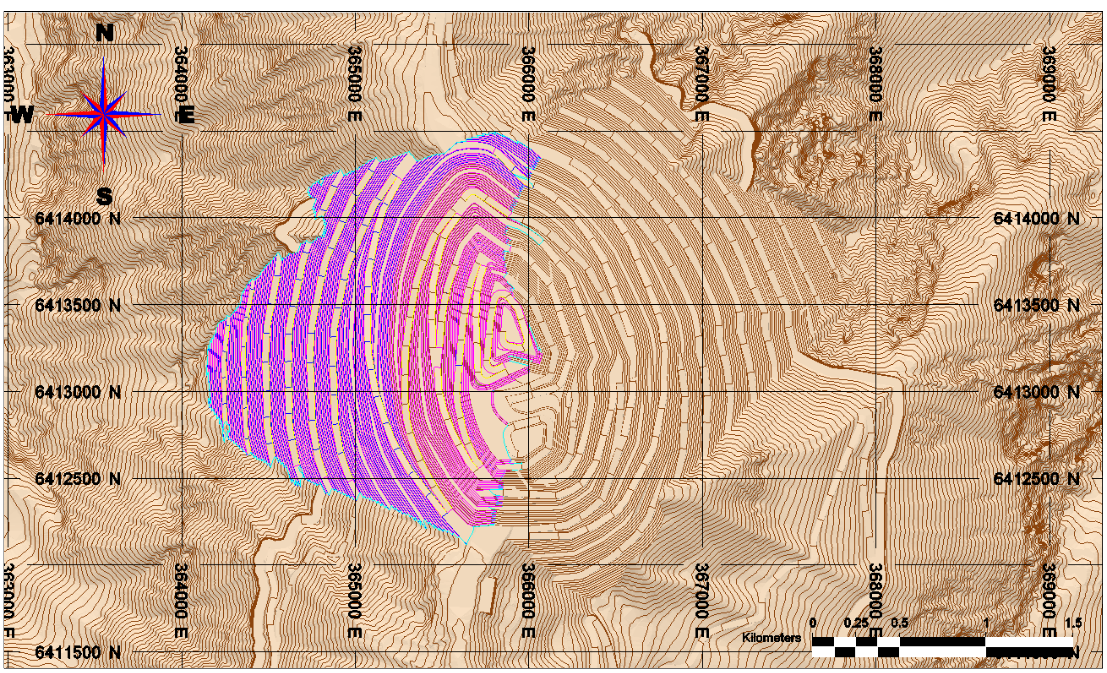
The operating mine design includes 12 phases totalling 1,421 Mt of mineral with an average Cu grade of 0.35% and 4,325 Mt of waste (including waste movement by contractors), using a cut-off grade of ≥ 0.18% of Cu (Table 1.12).

Table 1.12: Mine Design Summary by Operating Phase

Phase	Mineral Feed Measured and Indicated Resource (Cu Cut-Off Grade ≥ 0.18%)				Waste (kt)	Total (kt)	Strip Ratio	Global Index	Mineral Feed to plant (years)
	Tonnage (kt)	Cu Grade (%)	Mo Grade (ppm)	Ag Grade (g/t)					
Ph.1	139,260	0.47	133.8	1.3	136,401	275,662	0.98	0.24	2.8
Ph.2	129,441	0.33	143.3	1.2	89,296	218,737	0.69	0.20	2.6
Ph.3	117,187	0.37	124.7	1.1	151,293	268,480	1.29	0.16	2.4
Ph.4	131,484	0.35	95.3	1.2	158,607	290,091	1.21	0.16	2.6
Ph.5	131,533	0.33	139.3	1.2	253,747	385,280	1.93	0.11	2.6
Ph.6	137,915	0.35	131.7	1.0	320,298	458,212	2.32	0.11	2.8
Ph.7	101,585	0.31	141.9	1.0	352,298	453,883	3.47	0.07	2.0
Ph.8	100,837	0.34	97.9	1.2	383,938	484,775	3.81	0.07	2.0
Ph.9	133,012	0.35	186.2	0.8	470,197	603,208	3.54	0.08	2.7
Ph.10	95,430	0.32	138.4	1.1	527,159	622,588	5.52	0.05	1.9
Ph.11	83,310	0.36	211.8	0.8	590,524	673,834	7.09	0.04	1.7
Ph.12	119,565	0.32	92.1	0.9	616,144	735,709	5.15	0.05	2.4
All Phases	1,420,557	0.353	134.98	1.08	4,049,902	5,470,459	2.85	0.092	28.6

The final condition of the mine is shown in Figure 1.2 for the ultimate pit design, phase 10 (phases 11 and 12 are not mined, due to the high strip ratio they are not economic). This ultimate pit results in 1,220 Mt of feed to the concentrator with an average Cu grade of 0.36% and 2,855 Mt of waste (including waste movement by contractors), using variable cut-off grades.

Figure 1.2: Plan View of Ultimate Pit



Source: Tetra Tech, 2023

The main mine equipment to meet the production schedule is listed below:

- Production drill (10-5/8" – 6-7/8'') total of 8 units at peak
- Electric shovel (73 yd^3) 4 units at peak
- Hydraulic shovel (55 yd^3) 2 units at peak
- Front-end loader (50 yd^3) 2 units at peak
- Mine haul trucks (300 t) 76 units at peak.

Support equipment is listed below:

- Bulldozer
- Wheeldozer
- Motor grader
- Water truck.

The production schedule is shown in Table 1.13.

Table 1.13: Production Schedule

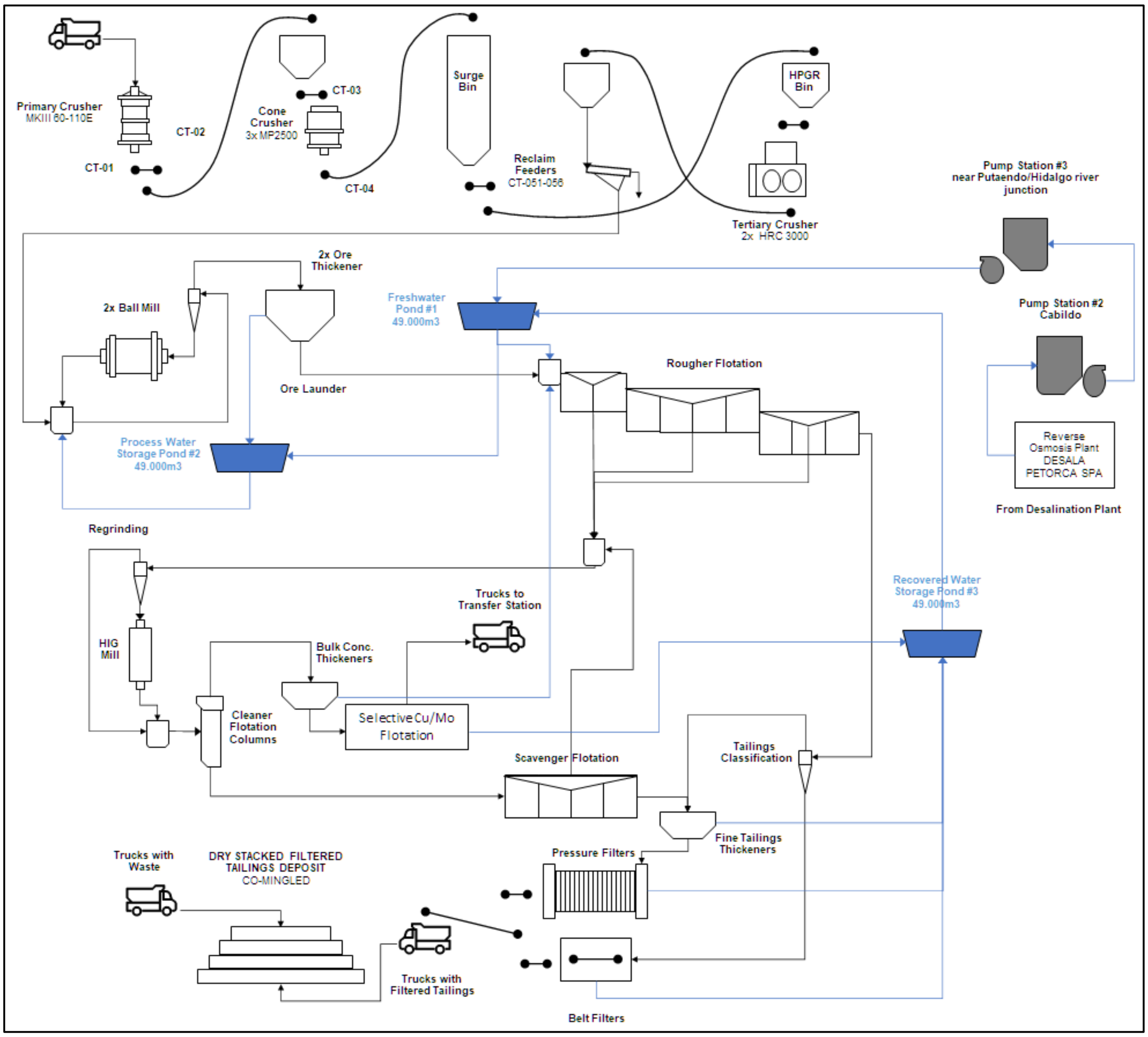
Period (years)	Days	COG	Mineral to Mill					Waste (kt) (including contractors)	Strip ratio (w:o)	Total			ktpd	To Mill: Contained		
			kt	CuEq (%)	Cu (%)	Mo (ppm)	Ag (g/t)			Onsite (kt)	Rehandling (kt)	W/Rehandling (kt)		Cu (t)	Mo (t)	Ag (koz)
- 3	365							15,947				15,947				
- 2	365							46,986		48,514		48,514	133			
- 1	365							68,611		78,881		78,881	216			
1	365	0.25%	33,921	0.53%	0.49%	82.33	1.26	84,561	2.49	115,187	4,827	120,014	329	166,823	2,793	1,378
2	365	0.36%	49,640	0.59%	0.53%	145.35	1.38	74,362	1.50	131,580		131,580	360	262,844	7,215	2,200
3	365	0.37%	49,640	0.52%	0.46%	152.04	1.42	48,832	0.98	120,428		120,428	330	228,443	7,547	2,273
4	365	0.33%	49,640	0.45%	0.40%	138.30	1.20	65,400	1.32	133,864	699	134,562	369	196,723	6,865	1,921
5	365	0.32%	49,640	0.48%	0.42%	167.73	1.21	96,192	1.94	151,190		151,190	414	206,254	8,326	1,930
6	365	0.31%	49,640	0.45%	0.39%	165.46	1.20	86,200	1.74	150,536		150,536	412	191,660	8,214	1,912
7	365	0.26%	49,640	0.42%	0.37%	117.42	1.21	79,526	1.60	129,392	5,737	135,129	370	184,164	5,829	1,928
8	365	0.25%	49,640	0.40%	0.36%	108.67	1.16	146,276	2.95	191,120	8,824	199,944	548	176,272	5,394	1,847
9	365	0.25%	49,640	0.39%	0.35%	97.51	1.22	156,117	3.14	209,627	4,655	214,282	587	174,584	4,840	1,947
10	365	0.26%	49,640	0.38%	0.34%	101.18	1.12	152,757	3.08	212,919		212,919	583	168,577	5,023	1,785
11	365	0.28%	49,640	0.42%	0.35%	186.22	1.23	160,661	3.24	214,269		214,269	587	176,172	9,244	1,956
12	365	0.24%	49,640	0.38%	0.33%	130.22	1.14	148,081	2.98	198,047	2,180	200,227	549	162,918	6,464	1,818
13	365	0.25%	49,640	0.39%	0.35%	114.96	1.06	163,023	3.28	212,653	3,824	216,476	593	172,648	5,706	1,691
14	365	0.25%	49,640	0.44%	0.39%	153.65	1.05	159,259	3.21	215,570		215,570	591	192,951	7,627	1,670
15	365	0.23%	49,640	0.39%	0.34%	127.88	1.08	147,483	2.97	200,307	295	200,602	550	171,010	6,348	1,721
16	365	0.22%	49,640	0.37%	0.31%	186.82	1.07	150,534	3.03	198,950	1,662	200,612	550	151,998	9,274	1,702
17	365	0.23%	49,640	0.32%	0.28%	101.18	0.96	149,486	3.01	177,023	23,694	200,718	550	138,198	5,023	1,529
18	365	0.24%	49,640	0.39%	0.35%	111.77	1.06	137,828	2.78	183,416	6,706	190,122	521	173,889	5,548	1,698
19	365	0.20%	49,640	0.38%	0.34%	118.76	1.19	138,567	2.79	179,473	8,734	188,207	516	166,890	5,895	1,902
20	365	0.18%	49,620	0.32%	0.29%	85.32	0.91	119,952	2.42	148,387	21,185	169,571	465	142,210	4,233	1,449
21	365	0.18%	47,514	0.42%	0.36%	164.60	0.89	107,793	2.27	151,199	4,108	155,307	425	171,240	7,821	1,358
22	365	0.18%	49,372	0.40%	0.33%	206.60	0.85	94,388	1.91	140,844	2,916	143,760	394	161,347	10,200	1,349
23	365	0.19%	49,640	0.39%	0.34%	140.46	1.09	36,775	0.74	86,415		86,415	237	168,230	6,972	1,740
24	365	0.18%	49,498	0.36%	0.31%	158.67	1.04	10,907	0.22	55,357	5,047	60,405	165	151,116	7,854	1,656
25	365	0.19%	45,024	0.27%	0.23%	107.16	0.84	8,162	0.18	21,677	31,508	53,185	146	102,969	4,825	1,221
26	81	0.20%	1,825	0.35%	0.26%	274.36	0.70	706	0.39	2,530		2,530	31	4,744	501	41
Total			**1,219,932**	**0.41%**	**0.36%**	**135.73**	**1.11**	**2,855,370**	**2.33**	**4,059,355**	**136,602**	**4,211,904**				

1.17 Recovery Methods

The concentrator is designed for a capacity of 136,000 t/d and is located in two areas within the Rocín River valley: the crushing and grinding circuits are located on a platform of compacted waste rock obtained from the pre-stripping operation, the wet area is located down the valley on a natural plateau next to the access road on the east side of the valley.

The process flowsheet is shown in Figure 1.3.

Figure 1.3: Process Flowsheet



Source: Los Andes Copper, 2022.

The run of mine material is fed to a primary crusher (MK60-110E) where it is reduced to 7 inches, and then to three secondary cone crushers (2,500 HP) in open circuit. The crushed product of

2 inches is stored in a surge bin that feeds the tertiary crushing circuit that includes two HRC3000 high pressure grinding roll (HPGR) crushers arranged in a closed circuit with banana screens.

The undersize from the crushing plant (P80 < 3.5 mm) is fed to two 23 MW ball mills arranged in a reverse closed circuit with hydrocyclones to achieve a P80 of 240 µm. The overflow from the hydrocyclones is thickened to 50% solids %wt and then transported via a concrete launder to the flotation area.

The flotation circuit has two stages: a Cu-Mo bulk flotation and a Cu-Mo selective flotation. Both circuits use mechanically agitated tanks and column cells to produce concentrates. The concentrates are filtered and then loaded and stored in Rotainers (copper) and bags (molybdenum) to be hauled by trucks to the port.

The tailings from flotation are classified using hydrocyclones. The coarse fraction is fed to belt filters and the fine fraction is thickened and then filtered in pressure filters. The filtered cake contains an average moisture of 15% and is disposed of in 2 m layers co-mingled with mine waste rock, in the bottom and across the valley to shape the dry stacked filtered tailings deposit.

The concentrator produces clean concentrates of approximately 24% Cu. The make-up water consumption is around 270 L/s and the power consumption is 16.6 kWh/t.

1.18 Project Infrastructure

The key drivers of the Project are aimed at reducing the environmental impacts, including reduction of water and energy consumption and minimizing the footprint. The Project design has also looked at reducing earthworks by installing part of the infrastructure on natural plateaus and on platforms using compacted mine pre-stripping material, by avoiding the construction of a tailings impoundment, and by eliminating on-site camps. The major infrastructure elements considered in this TR are:

Access Roads. The main access road from north of Putaendo to the flotation plant is 36 km long and will provide access to the mine site for operating personnel, supplies and concentrate hauling for the life of the operation. The road design speed is 80 km/h; a 30 km/h speed limit is considered in areas with a tight turn radius or limited visibility.

River Diversion. The Rocín River must be diverted to start the operation of the mine. The river diversion works include a coffer dam, a dam, an access road, a river intake infrastructure and a 5 m diameter, 15.965 km long diversion tunnel. The tunnel will be built in two stages, first during construction (10.581 km) and later extended by 5.447 km in the first and second years of production. During the construction phase a culvert will be built at the centre of the valley to allow for the development of the crushing and grinding circuits prior to finishing the construction of the diversion tunnel.

On-Site Facilities. These include mine service facilities (warehouses, storage, wash bay, storage tanks, lubricant-related infrastructure, welding and repair shop, tyre bay, water tanks), plant infrastructure (for the crushing plant, feed launder, flotation plant, molybdenum plant), buildings (offices, control room, clinic, parking areas), internal roads, fire protection systems.

Water Management. Water captured above 1,935 masl whose course is not affected by the Project (non-contact water) will be collected through contour channels which will carry the water downstream to be discharged at the Rocín and Hidalgo River junction. The rain water (contact water) that falls on the infrastructure and operating areas below the contour channels (such as roads, buildings, platforms and concentrator areas) will flow through channels to a central collector leading to the contact water treatment plant. This plant is located downstream of the mine area near the Rocín and Hidalgo River junction. Water captured from the mine dewatering system will be channelled through the same collector to the contact water treatment plant.

Water Supply. Vizcachitas water needs of approximately 280 L/s will be supplied using desalinated water. Los Andes Copper has entered into a non-binding letter of intent with Desala Petorca SpA, a water supply company, for an offtake agreement for 500 L/s desalinated water. The delivery point would be near the town of Cabildo. Vizcachitas plans to build and operate a pipeline with two pumping stations (EB2+EB3) from Cabildo to the Project site. A 500 L/s company-owned desalination plant located near Longotoma, Valparaíso Region, has also been considered as an alternative.

Tailings Disposal. Tailings from the flotation process will be classified into fine and coarse fractions then filtered using belt and press filters to obtain a final cake with 15% moisture (2.17 t/m^3 density). This filtered cake will be stacked in layers with the mine waste rock in a co-mingled configuration across the valley below 1,935 masl to allow the mine trucks loaded with waste to operate down the valley.

1.19 Marketing Studies and Contracts

Three market studies were conducted for the TR:

- CRU+ – Molybdenum Concentrates Market, February 10, 2023
- Synex Consulting Engineers – Projection of Prices for the Power Supply of the Vizcachitas Project, December 1, 2022
- Logsys SpA – Calculation of a Transfer Fee Port for Copper Concentrate through the use of Rotating Container Technology, February 18, 2023.

Copper and silver contained in copper concentrate, the main product of the Vizcachitas project, are openly traded commodities.

Following consensus long term copper and silver prices calculated by a leading Canadian bank, and molybdenum long-term forecast price from CRU, this TR uses the following prices:

- Copper : US$3.68/lb
- Molybdenum : US$12.9/lb

- Silver : US$21.79/oz.

Los Andes Copper has no contracts in place for power, water purchases or for concentrate sales.

1.20 Environmental Studies, Permitting and Social or Community Impact

Los Andes Copper has conducted several environmental studies in the Project area over the years, including two environmental impact declarations that led to the current permits for drilling.

Mine development will require conducting a full environmental impact assessment process, including baseline studies, impact assessment studies, public services reviews and public participation. This is a well-established process under Chilean Environmental Law. There are no environmental issues that are anticipated to have the potential to materially impact the ability of Los Andes Copper to develop the Vizcachitas Project.

Constant communication and engagement with the community, private landowners and other stakeholders is ongoing. There are no identified Indigenous individuals or communities within the Project area.

1.21 Capital and Operating Costs

The capital cost estimate has been developed in accordance with AACEI Class 4. After incorporating the recommended contingency, the capital cost estimate is considered to have a level of accuracy of ±25%. The capital cost estimate in Table 1.14 includes the following:

- Direct Costs of construction and assembly: Acquisition of equipment, labour, auxiliary equipment for construction and building materials are considered.
- Indirect Costs: Transportation and equipment insurance, general spare parts, vendor representatives, engineering, EPCM, start-up and owner costs are considered.
- Contingency, estimated based on Direct Costs plus Indirect Costs.
- Sustaining capital, defined as that required to maintain operations and may include capital spent on expansion or new infrastructure items.
- Deferred capital, the investment required to complete an expansion in the mine facilities and process plant during the life of the Project.

Table 1.14: Capital Cost Estimate (US$ million)

Category	Direct Initial Capital	Sustaining and Deferred
Fleet Capex	127,258	495,941
Pre-stripping, Dewatering and Early Works	308,792	405,651
Total Mine and Fleet Capex	**436,050**	**901,592**
Dry Area - Crushing	245,116	
Wet Area - Mill-float	254,139	
Tailings Filtration/ Reclaim & Water Treatment	180,038	
On-site Infrastructure	280,956	52,972
Off-site Infrastructure	244,104	3,803
Total Plant and Infrastructure Capex	**1,204,353**	**56,775**
Total Direct	**1,640,403**	**958,367**
Indirect	**454,104**	**311,315**
Contingency	**346,449**	**224,009**
Total Capital	**2,440,955**	**1,493,691**

Operating costs have been estimated to an accuracy of ±25% for the operating areas of Mining, Process, Infrastructure and General and Administration. The C1 cash costs for the first 8 years of operation is US$0.93/lb Cu, and US$1.25/lb Cu over the LOM. Unit costs per tonne of feed processed are presented in Table 1.15.

Table 1.15: Operating Cost Unit Rates by Activity

Description	UoM	First 8 years	LOM
Mining	US$/$t_{proc}$	3.98	5.02
Processing	US$/$t_{proc}$	3.85	3.90
Surface Infrastructure	US$/$t_{proc}$	1.18	1.20
Indirects	US$/$t_{proc}$	0.30	0.30
Total Operating Costs	**US$/$t_{proc}$**	**9.32**	**10.41**

1.22 Economic Analysis

The Project Net Present Value (NPV) after-tax at an applied discount rate of 8% is US$2,776 million (US$3,999 million pre-tax$_{8\%}$). The Project Internal Rate of Return (IRR) after-tax is 24.2% (28.5% pre-tax). The Payback Period, from the start of production, is 2.5 years after-tax (2.3 years pre-tax). The Project financial metrics and associated physical parameters are listed in Table 1.16.

Table 1.16: Project Financial Metrics

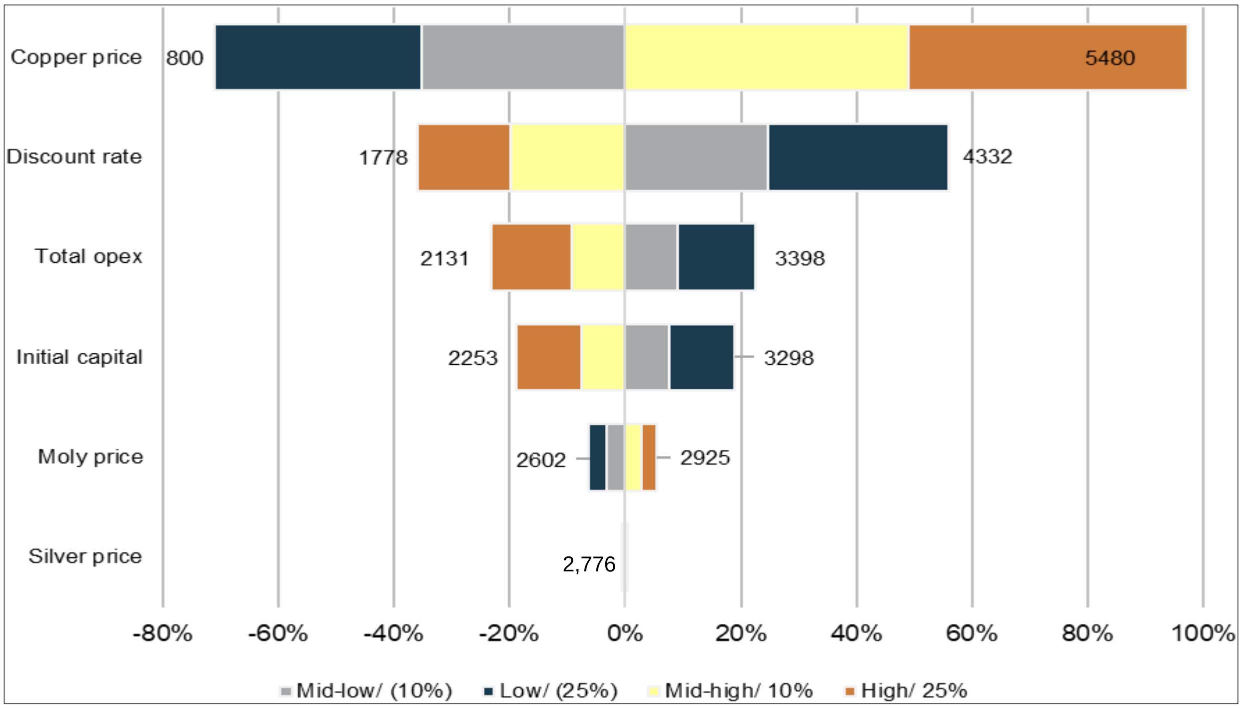
Metric	UoM	LOM
Pre-Tax NPV (8%, Real 2023)	US$ Million	3,999
After-Tax NPV (8%, Real 2023)	US$ Million	2,776
IRR Pre-Tax	%	28.5%
IRR After-Tax	%	24.2%
Undiscounted Post-tax Cash Flow (LOM)	US$ Million	9,484
Payback Period from First Production	Years	2.5
Initial CAPEX	US$ Million	2,441
LOM Sustaining CAPEX (Excluding Closure)	US$ Million	1,494
LOM C1 Cash Costs	US$/lb Cu	1.25
Nominal Process Capacity (Annual)	ktpa	49,640
Nominal Process Capacity (Daily)	tpd	136,000
Mine Life	Years	26
First Concentrate Production	Years	Year 4, Q2
Ore Grade		
Cu Grade	%	0.36
Mo Grade	g/t	136
Ag Grade	g/t	1.1
Cu Equivalent Grade	%	0.41
Metal Production		
Cu in Concentrate	kt	3,975
Mo in Concentrate	kt	124
Ag in Concentrate	koz	32,712
Average Process Recovery		
Cu Recovery	%	91.1%
Mo Recovery	%	74.8%
Ag Recovery	%	75.0%
Physicals		
Total In-Situ Rock	kt	4,075,302
Waste Rock	kt	2,855,370
Ore Mined (all Grades)	kt	1,219,932
Strip Ratio	w:o	2.33
Annual Average Production		
Copper	t Cu	152,883

The sensitivity analyses performed are summarized in the tornado diagram in Figure 1.4. The Project is most sensitive to changes in the copper price and discount rate, followed by Opex and Capex.

- The after-tax NPV ranges from US$820 million to US$5.480 billion as the copper price is varied between US$2.75/lb Cu and US$5.00/lb Cu. For the same input range, the IRR ranges between 13% and 36%. This indicates that the Project can generate positive returns even in low-price environments.

- The after-tax NPV ranges from US$1.788 billion to US$4.332 billion when the discount rate varies between 5% and 11%.
- The after-tax NPV ranges between US$2.131 billion and US$3.398 billion when the operating cost ranges between US$13.5/t feed and US$8.7/t feed. The IRR ranges between 21% and 27%.
- The after-tax NPV ranges between US$1.831 billion and US$3.051 billion with the initial capital range of US$3.298 billion and US$2.253 billion. The IRR ranges between 19% and 33%.

Figure 1.4: NPV (After-tax) Tornado Sensitivity Diagram



Source: Fraser McGill, 2023

1.23 Interpretations and Conclusions

Based on the information, interpretations and conclusions contained in this TR, Tetra Tech confirms that the Vizcachitas Project has technical and economic merit. The Vizcachitas Project has the potential to return a significant net present value and internal rate of return. There are no fatal flaws that could put the Project at risk.

1.24 Recommendations

This TR has demonstrated strong technical and economic foundations for the Vizcachitas Project. It is recommended to continue with optimization studies and then to proceed with feasibility studies.

2. INTRODUCTION AND TERMS OF REFERENCE

2.1 Purpose of the Technical Report

Los Andes Copper commissioned Tetra Tech to prepare a Technical Report (TR) for the Vizcachitas Project in compliance with the standards required by Canadian National Instrument (NI) 43-101 at the Pre-Feasibility Study (PFS) level. The cost estimate inputs meet AACE International recommendations for a Class 4 study. The operating and capital costs for the project were estimated with an accuracy of ±25%.

The scope of the work included the optimization of the recommended case in the PEA for 110,000 tpd mill throughput. Work included debottlenecking the primary crusher operation; this allowed an increase in throughput to 136,000 tpd. Other changes were incorporated, with a focus on adopting leading practices for sustainable mining.

The Vizcachitas copper and molybdenum porphyry deposit is located in the Valparaíso Region of Chile, in San Felipe Province. It is owned by Los Andes Copper Ltd., a Vancouver, B.C. company listed on the TSX Venture Exchange. This TR has been prepared for Los Andes Copper by or under the supervision of Qualified Persons within the purview of NI 43-101 regulation.

The main consultant involved in the preparation of this TR was Tetra Tech who was responsible for the resource estimates, pit design, mine planning, geotechnical review, capital and operating cost estimates and financial models. Certain activities were executed by Los Andes Copper or other consultants and reviewed by Tetra Tech. These included:

- Environmental studies, environmental liabilities, permitting and community matters
- Mining properties, land tenure, legal access, operational permits, adjacent properties
- Metallurgical testwork.

2.2 Frequently Used Acronyms, Abbreviations, Definitions, and Units of Measurement

Units in this TR are based on the International System of Units (IS), except for units that are industry standards, such as troy ounces for the mass of precious metals. The currency used is United States Dollars (US$ or USD), unless specified otherwise.

This report uses abbreviations and acronyms commonly used in the mining industry.

Table 2.1 identifies the terms and abbreviations used in this report.

Table 2.1: Units

Unit	Abbreviation or Symbol	Unit	Abbreviation or Symbol
Abrasion Index	Ai	Maximum	max
American Dollar	US$	Mega Volt Ampere	Mva
American Dollar Cent	cUS$	Megawatt	MW
Centigrade	ºC	Megawatt - Hour	MW-h
Centimetre	cm	Metre	m
Chilean Peso	CLP	Metre per Hour	m/h
Copper	Cu	Metre per Second	m/s
Copper Cyanide	CuCN	Metres above Sea Level	masl
Copper Equivalent	CuEq	Metric Tonne	t
Cubic Foot/Feet	ft^3	Metric Tonne per Day	tpd
Cubic Metre	m^3	Metric Tonne per Hour	tph
Cubic Metre per Hour	m^3/h	Microns	µm
Day	d	Milligram per Litre	mg/L
Foot/Feet	ft	Millimetre	mm
Gram/Litre	g/L	Million	M
Hectares	ha	Million Tonnes per Annum	Mpa
Horse Power	HP	Minutes	min
Hour	h	Molybdenum	Mo
Insoluble Copper	Icu	Part per Million	ppm
Kilo Tonne	kt	Percent	%
Kilo Tonne per Day	ktpd	Pounds	lb
Kilogram	kg	Run of Mine	ROM
Kilogram per Tonne	kg/t	Short Ton	st
Kilometre	km	Specific Gravity	SG
Kilovolt	kV	Square Metres	m^2
Kilovolt Amp	kVA	Square Metres per Tonne per Day	m^2/tpd
Kilowatt	kW	Tonnes per Day	tpd
Kilowatt Hour	kWh	Tonnes per Hour	tph
Kilowatt Hour per Cubic Metre	kWh/m^3	Troy Ounces	oz
Kilowatt Hour per Metric Tonne	kWh/t	Weight (Mass)	wt
Kilowatt Hour per Short Tonne	kWh/st	Weight (Mass) per Cent	%w/w
Life of Mine	LOM	Wet Metric Tonnes	wmt
Litre	L	Work Index	Wi
Litre per Second	L/s	Year	y

2.3 Effective Dates

The overall Effective Date of this report is February 20, 2023.

The Mineral Resource estimate was completed on February 7, 2023 and the Mineral Reserve estimate was completed on December 2, 2022.

There were no material changes to the scientific and technical information in relation to the Project between the Effective Date and the signature date of the report.

3. RELIANCE ON OTHER EXPERTS

This Technical Report has relied on the documentation generated by Los Andes Copper and Tetra Tech. It also includes documents within the public domain and private information provided by Los Andes Copper and information from the documents listed in Section 27 of this report.

The authors consider that the information provided and used for this Technical Report is accurate and the interpretation and opinions expressed herein are reasonable, based on the current understanding of the mining and processing techniques, costs, economics, mineralization processes and geological environment. The authors have made reasonable efforts to verify the accuracy of the data within this TR.

The results and opinions expressed in this Technical Report are accurate and complete as of the issue date, and no information that may affect the conclusions herein has been withheld. The authors have the right to review this report and the conclusions if they become aware of additional material information after the date of this report.

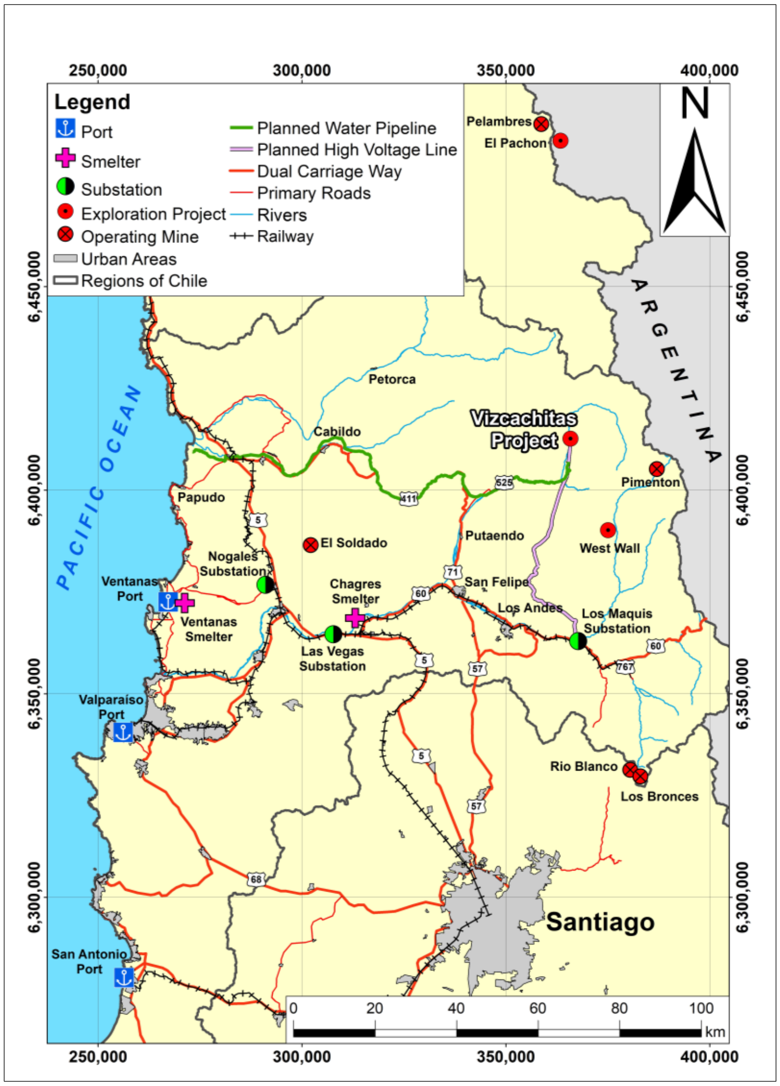
The authors of this TR are not qualified to provide comment on legal issues associated with Legal Title and Taxation. For portions of Section 4 dealing with the mining concessions, the nature and extent of Los Andes Copper's title and interest in the Property, the terms of royalties, back-in rights, payments or other agreements and encumbrances to which the Project is subject, Tetra Tech has fully relied on the legal opinion of Ossa Alessandri Abogados, lawyers for land tenure, and Los Andes Copper for information related to the Net Smelter Returns (NSR) applicable to the Project. For portions of Section 22 dealing with legal aspects of taxation, Tetra Tech has fully relied on the legal opinion of Fischer y Cia. Abogados.

4. PROPERTY DESCRIPTION AND LOCATION

4.1 Summary

The Vizcachitas project is located at 32° 24' 27" S and 70° 25' 30" W at an average altitude of 2,050 masl in the Andes Mountains of Chile in the Rocín River valley. The location is in San Felipe Province in the Valparaíso Region 150 km north of Santiago, Chile's capital. The Company owns 52 mining exploitation concessions (mining claims) covering a surface area of 10,771 ha and 175 mining exploration concessions covering an surface of 48,600 ha. Figure 4.1 shows the location of the Project.

Figure 4.1: Vizcachitas Project Area



Source: Los Andes Copper, 2022

4.2 Mineral Tenure

4.2.1 Chilean Mining Laws

The regulatory framework for the mining industry in Chile is contained in two main statutes: the Organic Mining Act (Law N° 18.097), and the Mining Code (Law N° 18.248). The framework provides a strong legal protection for mining activities.

In Chile there are two kinds of mining concessions: exploration concessions and exploitation concessions. Each one grants different types of rights. The exploration concession grants rights to study and search for the existence of minerals in a certain area. The exploitation concession grants rights to obtain and commercialize the minerals found in the area covered by the concession. Although it is not necessary to obtain an exploration concession prior to obtaining an exploitation concession, the Mining Code establishes a preference for exploitation of an area to the holder of an exploration concession.

Exploration concessions grant rights to search and explore for minerals within a specified area. It is forbidden for the holder of an exploration permit to operate a mine in the area of the concession (without first obtaining an exploitation concession). Exploration concessions last 2 years and can be extended for 2 additional years if the area of the concession is reduced in at least 50%. After that period the holder of the title can choose between applying for an exploitation concession or simply abandoning the area.

An amendment to the Mining Code (passed in February 2022 with initial effect considered for February 4, 2023, subsequently extended to January 1, 2024) increased the term of exploration concessions from 2 years to 4 years, excluding the possibility of renewal (among other changes).

Exploitation concessions grant the holder the exclusive right to operate a mine in the area within the boundaries of the concession. The Mining Code establishes the right for the exploitation concession holder to access the surface land and the surface owner is legally required to allow the holder of the concession to operate a mine in the area. Mining concessions are unlimited in their duration. The principal obligation of the concession holder is to pay an annual licence to keep the claim.

4.2.2 Vizcachitas Mineral Tenure

The project includes 52 exploitation concessions covering a surface area of 10,771 ha, and 175 exploration claims covering a surface area of 48,600 ha. All the exploitation concessions and exploration concessions (together the Mining Properties) are 100% owned by CMVH or SLM San José and have been granted or are in the process of being granted (18 exploration concessions in progress) by the court of Putaendo. Part of the exploration concessions overlap the exploitation concessions, a practice commonly used in Chile to create an additional layer of protection (over that already granted by law) to the underlying properties.

Eighteen of the 175 exploration concessions are *Pedimentos* (exploration concession submissions) (lines 1 to 18 in Table 4.2). Los Andes Copper has recently filed applications with the Putaendo Court for exploration rights and preference over the requested area. These 18 *Pedimentos* replace expired exploration concessions that Los Andes Copper owned over the same area.

Table 4.1 and Table 4.2 list the exploration concessions and the exploitation concessions currently held (including the 18 *Pedimentos*). Figure 4.2 and Figure 4.3 show the locations of the concessions.

Los Andes Copper has expanded its exploration concession area during the last decade, limited overlaps with third parties have been created at the boundaries, totalling 643.91 ha where Los Andes Copper does not hold the preferential right. Figure 4.3 does not show these overlapping areas.

TETRA TECH

Table 4.1: Exploitation Concessions as of February 20, 2023

N°	Mining Claim Name	Owner	ROL NACIONAL	Hectares	Validity
1	SANTA TERESA 1 AL 60	CIA. MRA VIZCAHITAS HOLDING	056040216-3	271	Indefinite
2	SANTA MARIA 1 AL 60	CIA. MRA VIZCAHITAS HOLDING	056040214-7	236	Indefinite
3	SAN CAYETANO 1 AL 20	CIA. MRA VIZCAHITAS HOLDING	056040215-5	100	Indefinite
4	TIGRE TRES 1-30	CIA. MRA VIZCAHITAS HOLDING	056040301-1	300	Indefinite
5	HUEMUL 1-40	CIA. MRA VIZCAHITAS HOLDING	056040336-4	200	Indefinite
6	SAN JOSE 1/3000	SLM SAN JOSE	055040138-K	70	Indefinite
7	LEON II 1/30	CIA. MRA VIZCAHITAS HOLDING	056040289-9	20	Indefinite
8	LEON III 1/30	CIA. MRA VIZCAHITAS HOLDING	056040290-2	20	Indefinite
9	LEON IV 1/30	CIA. MRA VIZCAHITAS HOLDING	056040291-0	20	Indefinite
10	LEON V 1/30	CIA. MRA VIZCAHITAS HOLDING	056040292-9	10	Indefinite
11	TIGRE UNO 1/30	CIA. MRA VIZCAHITAS HOLDING	056040284-8	20	Indefinite
12	TIGRE DOS 1/20	CIA. MRA VIZCAHITAS HOLDING	056040285-6	10	Indefinite
13	TIGRE CUATRO 1/20	CIA. MRA VIZCAHITAS HOLDING	056040286-4	10	Indefinite
14	TIGRE CINCO 1/60	CIA. MRA VIZCAHITAS HOLDING	056040287-2	104	Indefinite
15	LOMA UNO 1 AL 31	CIA. MRA VIZCAHITAS HOLDING	056040352-6	155	Indefinite
16	LOMA UNO 46 AL 52	CIA. MRA VIZCAHITAS HOLDING	056040353-4	35	Indefinite
17	LOMA DOS 1 AL 50	CIA. MRA VIZCAHITAS HOLDING	056040337-2	250	Indefinite
18	LOMA TRES 1 AL 18	CIA. MRA VIZCAHITAS HOLDING	056040354-2	90	Indefinite
19	LOMA CUATRO 1/56	CIA. MRA VIZCAHITAS HOLDING	056040355-0	280	Indefinite
20	LOMA CINCO 1/20	CIA. MRA VIZCAHITAS HOLDING	056040356-9	100	Indefinite
21	LOMA SEIS 1/60	CIA. MRA VIZCAHITAS HOLDING	056040357-7	300	Indefinite
22	LOMA SIETE 1/60	CIA. MRA VIZCAHITAS HOLDING	056040358-5	300	Indefinite
23	LOMA OCHO 1/60	CIA. MRA VIZCAHITAS HOLDING	056040359-3	300	Indefinite
24	LOMA NUEVE 1/60	CIA. MRA VIZCAHITAS HOLDING	056040360-7	300	Indefinite
25	LOMA DIEZ 1/60	CIA. MRA VIZCAHITAS HOLDING	056040361-5	300	Indefinite
26	LOMA ONCE 1/60	CIA. MRA VIZCAHITAS HOLDING	056040362-3	300	Indefinite
27	LOMA DOCE 1/40	CIA. MRA VIZCAHITAS HOLDING	056040363-1	200	Indefinite
28	LOMA TRECE 1/40	CIA. MRA VIZCAHITAS HOLDING	056040364-K	200	Indefinite
29	LOMA CATORCE 1/60	CIA. MRA VIZCAHITAS HOLDING	056040365-8	300	Indefinite
30	LOMA QUINCE 1/60	CIA. MRA VIZCAHITAS HOLDING	056040366-6	300	Indefinite
31	LOMA DIECISEIS 1/18	CIA. MRA VIZCAHITAS HOLDING	056040367-4	90	Indefinite
32	LOMA DIECISIETE 1/56	CIA. MRA VIZCAHITAS HOLDING	056040368-2	280	Indefinite
33	LOMA DIECIOCHO 1/60	CIA. MRA VIZCAHITAS HOLDING	056040369-0	300	Indefinite
34	ROMA 24 1 AL 100	CIA. MRA VIZCAHITAS HOLDING	056040508-1	100	Indefinite
35	ROMA 25 1 AL 300	CIA. MRA VIZCAHITAS HOLDING	056040532-4	300	Indefinite
36	ROMINA 8 1 AL 300	CIA. MRA VIZCAHITAS HOLDING	056040554-5	300	Indefinite
37	ROMINA 9 1 AL 300	CIA. MRA VIZCAHITAS HOLDING	056040555-3	300	Indefinite
38	ISIDRO 8 1 AL 200	CIA. MRA VIZCAHITAS HOLDING	056040553-7	200	Indefinite
39	PAYACAN 1 1 AL 20	CIA. MRA VIZCAHITAS HOLDING	056040684-3	200	Indefinite
40	PAYACAN 2 1 AL 20	CIA. MRA VIZCAHITAS HOLDING	056040685-1	200	Indefinite
41	VALLE 1 1 AL 20	CIA. MRA VIZCAHITAS HOLDING	056040688-6	200	Indefinite
42	VALLE 2 1 AL 20	CIA. MRA VIZCAHITAS HOLDING	056040689-4	200	Indefinite
43	CHINCOL 1 1 AL 30	CIA. MRA VIZCAHITAS HOLDING	056040690-8	300	Indefinite
44	CHINCOL 2 1 AL 30	CIA. MRA VIZCAHITAS HOLDING	056040691-6	300	Indefinite
45	CHINCOL 3 1 AL 30	CIA. MRA VIZCAHITAS HOLDING	056040692-4	300	Indefinite
46	CHINCOL 4 1 AL 30	CIA. MRA VIZCAHITAS HOLDING	056040693-2	300	Indefinite
47	CHINCOL 5 1 AL 30	CIA. MRA VIZCAHITAS HOLDING	056040694-0	300	Indefinite
48	CHINCOL 6 1 AL 30	CIA. MRA VIZCAHITAS HOLDING	056040695-9	300	Indefinite
49	CHINCOL 7 1 AL 30	CIA. MRA VIZCAHITAS HOLDING	056040696-7	300	Indefinite
50	CHINCOL 8 1 AL 30	CIA. MRA VIZCAHITAS HOLDING	056040697-5	300	Indefinite
51	ROJO 8 1 AL 30	CIA. MRA VIZCAHITAS HOLDING	056040716-5	300	Indefinite
52	ROJO 9 1 AL 30	CIA. MRA VIZCAHITAS HOLDING	056040717-3	300	Indefinite

Note: ROL Nacional is a unique identifying number for property in Chile

Table 4.2: Exploration Concessions as of February 20, 2023

N°	Exploration Claim Name	Owner	ROL NACIONAL	Hectares	Validity
1	CAMILA 1	CIA. MRA VIZCACHITAS HOLDING	056042235-0	300	01-02-2023
2	CAMILA 2	CIA. MRA VIZCACHITAS HOLDING	056042236-9	300	01-02-2023
3	CAMILA 3	CIA. MRA VIZCACHITAS HOLDING	056042237-7	300	01-02-2023
4	CAMILA 4	CIA. MRA VIZCACHITAS HOLDING	056042238-5	300	01-02-2023
5	CAMILA 5	CIA. MRA VIZCACHITAS HOLDING	056042239-3	300	01-02-2023
6	CAMILA 6	CIA. MRA VIZCACHITAS HOLDING	056042240-7	300	01-02-2023
7	CAMILA 7	CIA. MRA VIZCACHITAS HOLDING	056042241-5	300	01-02-2023
8	CAMILA 8	CIA. MRA VIZCACHITAS HOLDING	056042242-3	300	01-02-2023
9	CAMILA 9	CIA. MRA VIZCACHITAS HOLDING	056042243-1	300	01-02-2023
10	CAMILA 10	CIA. MRA VIZCACHITAS HOLDING	056042244-K	300	01-02-2023
11	CAMILA 11	CIA. MRA VIZCACHITAS HOLDING	056042245-8	300	01-02-2023
12	CAMILA 12	CIA. MRA VIZCACHITAS HOLDING	056042246-6	300	01-02-2023
13	CAMILA 13	CIA. MRA VIZCACHITAS HOLDING	056042247-4	300	01-02-2023
14	CAMILA 14	CIA. MRA VIZCACHITAS HOLDING	056042248-2	300	01-02-2023
15	CAMILA 15	CIA. MRA VIZCACHITAS HOLDING	056042249-0	300	01-02-2023
16	CAMILA 16	CIA. MRA VIZCACHITAS HOLDING	056042250-4	300	01-02-2023
17	CAMILA 17	CIA. MRA VIZCACHITAS HOLDING	056042251-2	300	01-02-2023
18	CAMILA 18	CIA. MRA VIZCACHITAS HOLDING	056042252-0	300	01-02-2023
19	MAGNOLIA 1	CIA. MRA VIZCACHITAS HOLDING	056042262-8	300	20-04-2023
20	MAGNOLIA 2	CIA. MRA VIZCACHITAS HOLDING	056042263-6	300	20-04-2023
21	MAGNOLIA 3	CIA. MRA VIZCACHITAS HOLDING	056042264-4	300	20-04-2023
22	MAGNOLIA 4	CIA. MRA VIZCACHITAS HOLDING	056042265-2	300	20-04-2023
23	MAGNOLIA 5	CIA. MRA VIZCACHITAS HOLDING	056042266-0	300	20-04-2023
24	MAGNOLIA 6	CIA. MRA VIZCACHITAS HOLDING	056042267-9	300	20-04-2023
25	MAGNOLIA 7	CIA. MRA VIZCACHITAS HOLDING	056042268-7	300	20-04-2023
26	MAGNOLIA 8	CIA. MRA VIZCACHITAS HOLDING	056042269-5	300	20-04-2023
27	MAGNOLIA 9	CIA. MRA VIZCACHITAS HOLDING	056042270-9	300	20-04-2023
28	MAGNOLIA 10	CIA. MRA VIZCACHITAS HOLDING	056042271-7	300	20-04-2023
29	MAGNOLIA 11	CIA. MRA VIZCACHITAS HOLDING	056042272-5	300	20-04-2023
30	MAGNOLIA 12	CIA. MRA VIZCACHITAS HOLDING	056042273-3	300	20-04-2023
31	MAGNOLIA 13	CIA. MRA VIZCACHITAS HOLDING	056042274-1	300	20-04-2023
32	MAGNOLIA 14	CIA. MRA VIZCACHITAS HOLDING	056042275-K	300	20-04-2023
33	MAGNOLIA 15	CIA. MRA VIZCACHITAS HOLDING	056042276-8	200	20-04-2023
34	MAGNOLIA 16	CIA. MRA VIZCACHITAS HOLDING	056042277-6	200	20-04-2023
35	MAGNOLIA 18	CIA. MRA VIZCACHITAS HOLDING	056042279-2	300	20-04-2023
36	COIRON 1	CIA. MRA VIZCACHITAS HOLDING	056042297-0	300	20-04-2023
37	COIRON 2	CIA. MRA VIZCACHITAS HOLDING	056042298-9	300	20-04-2023
38	COIRON 3	CIA. MRA VIZCACHITAS HOLDING	056042299-7	300	20-04-2023
39	COIRON 4	CIA. MRA VIZCACHITAS HOLDING	056042300-4	300	20-04-2023
40	COIRON 5	CIA. MRA VIZCACHITAS HOLDING	056042301-2	300	20-04-2023
41	COIRON 6	CIA. MRA VIZCACHITAS HOLDING	056042302-0	300	20-04-2023
42	AMARILLO 1	CIA. MRA VIZCACHITAS HOLDING	056042280-6	300	07-06-2023
43	AMARILLO 2	CIA. MRA VIZCACHITAS HOLDING	056042281-4	300	07-06-2023
44	AMARILLO 3	CIA. MRA VIZCACHITAS HOLDING	056042282-2	300	07-06-2023
45	AMARILLO 4	CIA. MRA VIZCACHITAS HOLDING	056042283-0	200	07-06-2023
46	AMARILLO 5	CIA. MRA VIZCACHITAS HOLDING	056042284-9	200	07-06-2023
47	AMARILLO 6	CIA. MRA VIZCACHITAS HOLDING	056042285-7	300	07-06-2023
48	AMARILLO 7	CIA. MRA VIZCACHITAS HOLDING	056042286-5	300	07-06-2023
49	AMARILLO 8	CIA. MRA VIZCACHITAS HOLDING	056042287-3	300	07-06-2023
50	AMARILLO 9	CIA. MRA VIZCACHITAS HOLDING	056042288-1	300	07-06-2023
51	AMARILLO 10	CIA. MRA VIZCACHITAS HOLDING	056042289-K	300	07-06-2023

N°	Exploration Claim Name	Owner	ROL NACIONAL	Hectares	Validity
52	AMARILLO 11	CIA. MRA VIZCACHITAS HOLDING	056042290-3	300	07-06-2023
53	AMARILLO 12	CIA. MRA VIZCACHITAS HOLDING	056042291-1	300	07-06-2023
54	AMARILLO 13	CIA. MRA VIZCACHITAS HOLDING	056042292-K	300	07-06-2023
55	AMARILLO 14	CIA. MRA VIZCACHITAS HOLDING	056042293-8	300	07-06-2023
56	AMARILLO 15	CIA. MRA VIZCACHITAS HOLDING	056042294-6	200	07-06-2023
57	AMARILLO 16	CIA. MRA VIZCACHITAS HOLDING	056042295-4	200	07-06-2023
58	AMARILLO 17	CIA. MRA VIZCACHITAS HOLDING	056042296-2	200	07-06-2023
59	MAGNOLIA 17	CIA. MRA VIZCACHITAS HOLDING	056042278-4	300	30-06-2023
60	LAUREL 1	CIA. MRA VIZCACHITAS HOLDING	056042334-9	200	14-09-2023
61	LAUREL 2	CIA. MRA VIZCACHITAS HOLDING	056042333-0	200	14-09-2023
62	LAUREL 3	CIA. MRA VIZCACHITAS HOLDING	056042332-2	300	14-09-2023
63	LAUREL 5	CIA. MRA VIZCACHITAS HOLDING	056042330-6	300	14-09-2023
64	LAUREL 6	CIA. MRA VIZCACHITAS HOLDING	056042329-2	300	14-09-2023
65	LAUREL 7	CIA. MRA VIZCACHITAS HOLDING	056042327-6	300	14-09-2023
66	LAUREL 8	CIA. MRA VIZCACHITAS HOLDING	056042328-4	300	14-09-2023
67	LAUREL 9	CIA. MRA VIZCACHITAS HOLDING	056042337-3	300	14-09-2023
68	LAUREL 10	CIA. MRA VIZCACHITAS HOLDING	056042336-5	300	14-09-2023
69	LAUREL 11	CIA. MRA VIZCACHITAS HOLDING	056042335-7	300	14-09-2023
70	LAUREL 12	CIA. MRA VIZCACHITAS HOLDING	056042326-8	300	14-09-2023
71	LAUREL 13	CIA. MRA VIZCACHITAS HOLDING	056042325-K	300	14-09-2023
72	ALI 16	CIA. MRA VIZCACHITAS HOLDING	052031594-2	300	27-09-2023
73	ALI 17	CIA. MRA VIZCACHITAS HOLDING	052031595-0	300	27-09-2023
74	ALI 20 A	CIA. MRA VIZCACHITAS HOLDING	052031597-7	100	27-09-2023
75	ARRAYAN 2	CIA. MRA VIZCACHITAS HOLDING	052031599-3	200	27-09-2023
76	ARRAYAN 7	CIA. MRA VIZCACHITAS HOLDING	052031600-0	300	27-09-2023
77	LAUREL 4	CIA. MRA VIZCACHITAS HOLDING	056042331-4	300	29-09-2023
78	ALI 14	CIA. MRA VIZCACHITAS HOLDING	052031592-6	300	04-10-2023
79	ALI 15	CIA. MRA VIZCACHITAS HOLDING	052031593-4	300	04-10-2023
80	ALI 8	CIA. MRA VIZCACHITAS HOLDING	052031586-1	200	12-10-2023
81	ALI 9	CIA. MRA VIZCACHITAS HOLDING	052031587-K	300	12-10-2023
82	ALI 10	CIA. MRA VIZCACHITAS HOLDING	052031588-8	300	12-10-2023
83	ALI 12	CIA. MRA VIZCACHITAS HOLDING	052031590-K	300	12-10-2023
84	ALI 13	CIA. MRA VIZCACHITAS HOLDING	052031591-8	300	12-10-2023
85	ALI 1	CIA. MRA VIZCACHITAS HOLDING	052031579-9	200	20-10-2023
86	ALI 2	CIA. MRA VIZCACHITAS HOLDING	052031580-2	200	20-10-2023
87	ALI 3	CIA. MRA VIZCACHITAS HOLDING	052031581-0	300	20-10-2023
88	ALI 4	CIA. MRA VIZCACHITAS HOLDING	052031582-9	300	20-10-2023
89	ALI 5	CIA. MRA VIZCACHITAS HOLDING	052031583-7	300	20-10-2023
90	ALI 7	CIA. MRA VIZCACHITAS HOLDING	052031585-3	200	20-10-2023
91	ARRAYAN 18	CIA. MRA VIZCACHITAS HOLDING	052031601-9	200	27-10-2023
92	ALI 20 B	CIA. MRA VIZCACHITAS HOLDING	056042350-0	200	09-11-2023
93	ALI 21	CIA. MRA VIZCACHITAS HOLDING	056042351-9	300	09-11-2023
94	ALI 22	CIA. MRA VIZCACHITAS HOLDING	056042352-7	300	09-11-2023
95	ALI 23	CIA. MRA VIZCACHITAS HOLDING	056042370-5	300	09-11-2023
96	ALI 24	CIA. MRA VIZCACHITAS HOLDING	056042369-1	300	09-11-2023
97	ALI 25	CIA. MRA VIZCACHITAS HOLDING	056042368-3	300	09-11-2023
98	ALI 26	CIA. MRA VIZCACHITAS HOLDING	056042367-5	300	09-11-2023
99	ARRAYAN 3	CIA. MRA VIZCACHITAS HOLDING	056042366-7	300	09-11-2023
100	ARRAYAN 4	CIA. MRA VIZCACHITAS HOLDING	056042365-9	300	09-11-2023

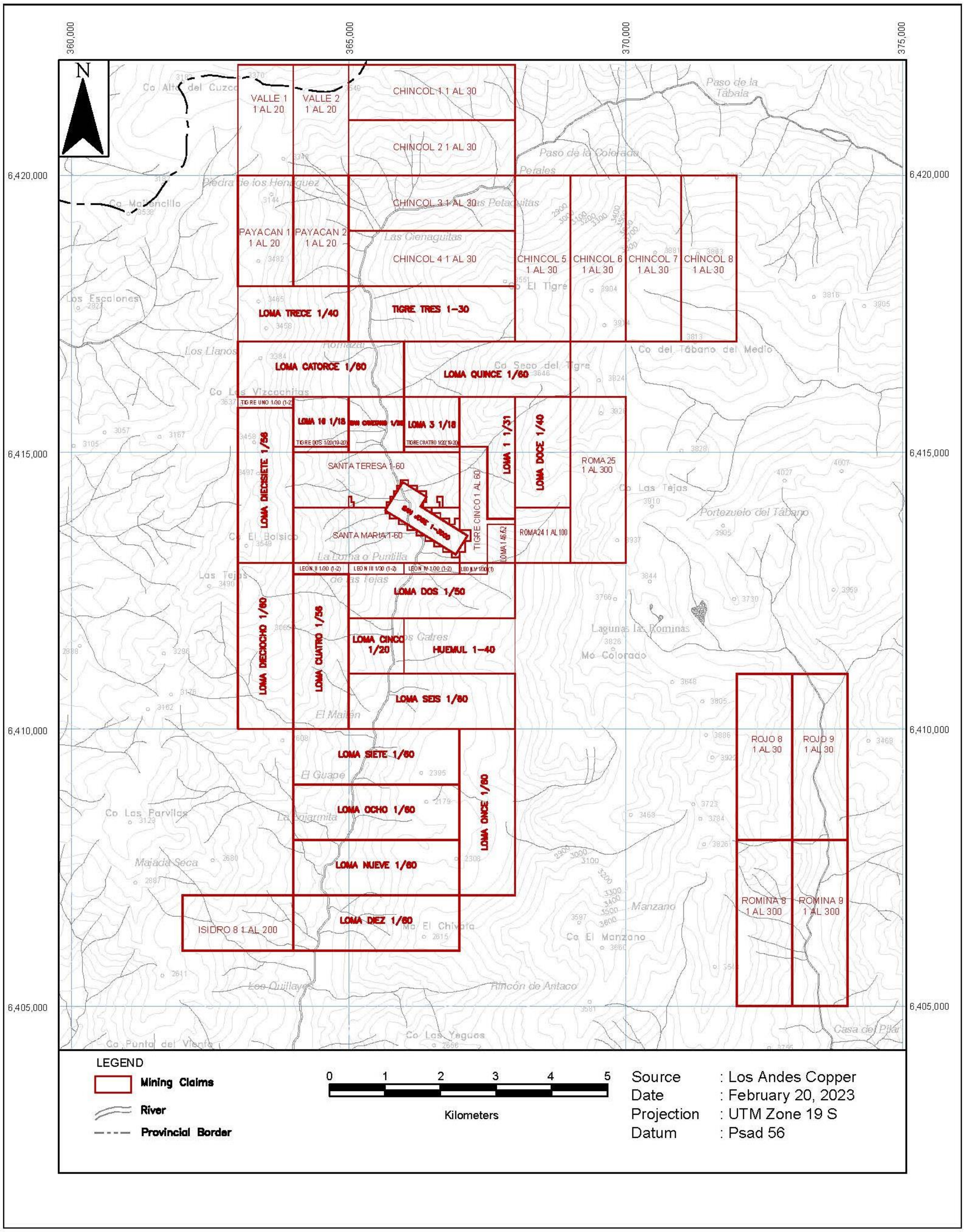
N°	Exploration Claim Name	Owner	ROL NACIONAL	Hectares	Validity
101	ARRAYAN 5	CIA. MRA VIZCACHITAS HOLDING	056042364-0	300	09-11-2023
102	ARRAYAN 6	CIA. MRA VIZCACHITAS HOLDING	056042340-3	300	09-11-2023
103	ARRAYAN 8	CIA. MRA VIZCACHITAS HOLDING	056042341-1	300	09-11-2023
104	ARRAYAN 9	CIA. MRA VIZCACHITAS HOLDING	056042342-K	300	09-11-2023
105	ARRAYAN 10	CIA. MRA VIZCACHITAS HOLDING	056042343-8	300	09-11-2023
106	ARRAYAN 11	CIA. MRA VIZCACHITAS HOLDING	056042344-6	300	09-11-2023
107	ARRAYAN 12	CIA. MRA VIZCACHITAS HOLDING	056042345-4	300	09-11-2023
108	ARRAYAN 14	CIA. MRA VIZCACHITAS HOLDING	056042347-0	300	09-11-2023
109	ARRAYAN 15	CIA. MRA VIZCACHITAS HOLDING	056042348-9	300	09-11-2023
110	ARRAYAN 16	CIA. MRA VIZCACHITAS HOLDING	056042349-7	300	09-11-2023
111	ARRAYAN 17	CIA. MRA VIZCACHITAS HOLDING	056042363-2	300	09-11-2023
112	PERALES 1	CIA. MRA VIZCACHITAS HOLDING	056042362-4	300	09-11-2023
113	PERALES 2	CIA. MRA VIZCACHITAS HOLDING	056042361-6	300	09-11-2023
114	PERALES 3	CIA. MRA VIZCACHITAS HOLDING	056042360-8	300	09-11-2023
115	PERALES 4	CIA. MRA VIZCACHITAS HOLDING	056042359-4	300	09-11-2023
116	PERALES 5	CIA. MRA VIZCACHITAS HOLDING	056042358-6	300	09-11-2023
117	PERALES 6	CIA. MRA VIZCACHITAS HOLDING	056042357-8	300	09-11-2023
118	PERALES 7	CIA. MRA VIZCACHITAS HOLDING	056042356-K	300	09-11-2023
119	PERALES 8	CIA. MRA VIZCACHITAS HOLDING	056042355-1	300	09-11-2023
120	ALI 19	CIA. MRA VIZCACHITAS HOLDING	052031602-7	300	10-11-2023
121	ALI 6	CIA. MRA VIZCACHITAS HOLDING	052031584-5	300	03-12-2023
122	ALI 11	CIA. MRA VIZCACHITAS HOLDING	052031589-6	300	03-12-2023
123	ALI 18	CIA. MRA VIZCACHITAS HOLDING	052031596-9	300	03-12-2023
124	ARRAYAN 1	CIA. MRA VIZCACHITAS HOLDING	052031598-5	200	03-12-2023
125	ESTERO 1	CIA. MRA VIZCACHITAS HOLDING	052031644-2	200	11-01-2024
126	ESTERO 2	CIA. MRA VIZCACHITAS HOLDING	052031645-0	200	11-01-2024
127	ESTERO 3	CIA. MRA VIZCACHITAS HOLDING	052031646-9	300	11-01-2024
128	ESTERO 4	CIA. MRA VIZCACHITAS HOLDING	052031647-7	200	11-01-2024
129	ESTERO 5	CIA. MRA VIZCACHITAS HOLDING	052031648-5	200	11-01-2024
130	ESTERO 6	CIA. MRA VIZCACHITAS HOLDING	052031649-3	300	11-01-2024
131	ESTERO 7	CIA. MRA VIZCACHITAS HOLDING	052031650-7	300	11-01-2024
132	ESTERO 8	CIA. MRA VIZCACHITAS HOLDING	052031651-5	300	11-01-2024
133	ESTERO 9	CIA. MRA VIZCACHITAS HOLDING	052031652-3	200	11-01-2024
134	ESTERO 10	CIA. MRA VIZCACHITAS HOLDING	052031653-1	200	11-01-2024
135	ESTERO 11	CIA. MRA VIZCACHITAS HOLDING	052031654-K	300	11-01-2024
136	ESTERO 12	CIA. MRA VIZCACHITAS HOLDING	052031655-8	200	11-01-2024
137	ARRAYAN 13	CIA. MRA VIZCACHITAS HOLDING	056042346-2	300	21-01-2024
138	ROMERO 1	CIA. MRA VIZCACHITAS HOLDING	056042397-7	300	13-04-2024
139	ROMERO 2	CIA. MRA VIZCACHITAS HOLDING	056042398-5	300	13-04-2024
140	ROMERO 3	CIA. MRA VIZCACHITAS HOLDING	056042399-3	300	13-04-2024
141	ROMERO 4	CIA. MRA VIZCACHITAS HOLDING	056042400-0	300	13-04-2024
142	ROMERO 5	CIA. MRA VIZCACHITAS HOLDING	056042404-3	300	13-04-2024
143	ROMERO 6	CIA. MRA VIZCACHITAS HOLDING	056042403-5	300	13-04-2024
144	ROMERO 7	CIA. MRA VIZCACHITAS HOLDING	056042402-7	300	13-04-2024
145	ROMERO 8	CIA. MRA VIZCACHITAS HOLDING	056042401-9	300	13-04-2024
146	ROMERO 9	CIA. MRA VIZCACHITAS HOLDING	056042396-9	300	13-04-2024
147	ROMERO 10	CIA. MRA VIZCACHITAS HOLDING	056042395-0	300	13-04-2024
148	ROMERO 11	CIA. MRA VIZCACHITAS HOLDING	056042394-2	300	13-04-2024
149	ROMERO 12	CIA. MRA VIZCACHITAS HOLDING	056042393-4	100	13-04-2024
150	ROMERO 13	CIA. MRA VIZCACHITAS HOLDING	056042392-6	300	13-04-2024

N°	Exploration Claim Name	Owner	ROL NACIONAL	Hectares	Validity
151	ROMERO 14	CIA. MRA VIZCACHITAS HOLDING	056042391-8	200	13-04-2024
152	ROMERO 15	CIA. MRA VIZCACHITAS HOLDING	056042390-K	300	13-04-2024
153	ROMERO 16	CIA. MRA VIZCACHITAS HOLDING	056042389-6	300	13-04-2024
154	ROMERO 17	CIA. MRA VIZCACHITAS HOLDING	056042405-1	300	13-04-2024
155	ROMERO 18	CIA. MRA VIZCACHITAS HOLDING	056042406-K	300	13-04-2024
156	ROMERO 19	CIA. MRA VIZCACHITAS HOLDING	056042407-8	300	13-04-2024
157	ROMERO 20	CIA. MRA VIZCACHITAS HOLDING	056042408-6	300	13-04-2024
158	ROMERO 21	CIA. MRA VIZCACHITAS HOLDING	056042409-4	300	13-04-2024
159	ROMERO 22	CIA. MRA VIZCACHITAS HOLDING	056042410-8	200	13-04-2024
160	ROMERO 23	CIA. MRA VIZCACHITAS HOLDING	056042411-6	200	13-04-2024
161	ROMERO 24	CIA. MRA VIZCACHITAS HOLDING	056042412-4	300	13-04-2024
162	ROMERO 25	CIA. MRA VIZCACHITAS HOLDING	056042413-2	300	13-04-2024
163	ARRAYAN 19	CIA. MRA VIZCACHITAS HOLDING	056042385-3	300	13-05-2024
164	ARENA B1	CIA. MRA VIZCACHITAS HOLDING	056042415-9	300	28-07-2024
165	ARENA B2	CIA. MRA VIZCACHITAS HOLDING	056042416-7	300	28-07-2024
166	ARENA B3	CIA. MRA VIZCACHITAS HOLDING	056042417-5	300	28-07-2024
167	ARENA B4	CIA. MRA VIZCACHITAS HOLDING	056042418-3	300	28-07-2024
168	ARENA B5	CIA. MRA VIZCACHITAS HOLDING	056042419-1	300	28-07-2024
169	ARENA B6	CIA. MRA VIZCACHITAS HOLDING	056042420-5	100	28-07-2024
170	CAROLA B1	CIA. MRA VIZCACHITAS HOLDING	056042421-3	100	28-07-2024
171	CAROLA B2	CIA. MRA VIZCACHITAS HOLDING	056042422-1	300	28-07-2024
172	CAROLA B3	CIA. MRA VIZCACHITAS HOLDING	056042423-K	300	28-07-2024
173	CAROLA B4	CIA. MRA VIZCACHITAS HOLDING	056042424-8	100	28-07-2024
174	CAROLA B5	CIA. MRA VIZCACHITAS HOLDING	056042425-6	200	28-07-2024
175	CAROLA B6	CIA. MRA VIZCACHITAS HOLDING	056042426-4	200	28-07-2024

Note: ROL Nacional is a unique identifying number for property in Chile

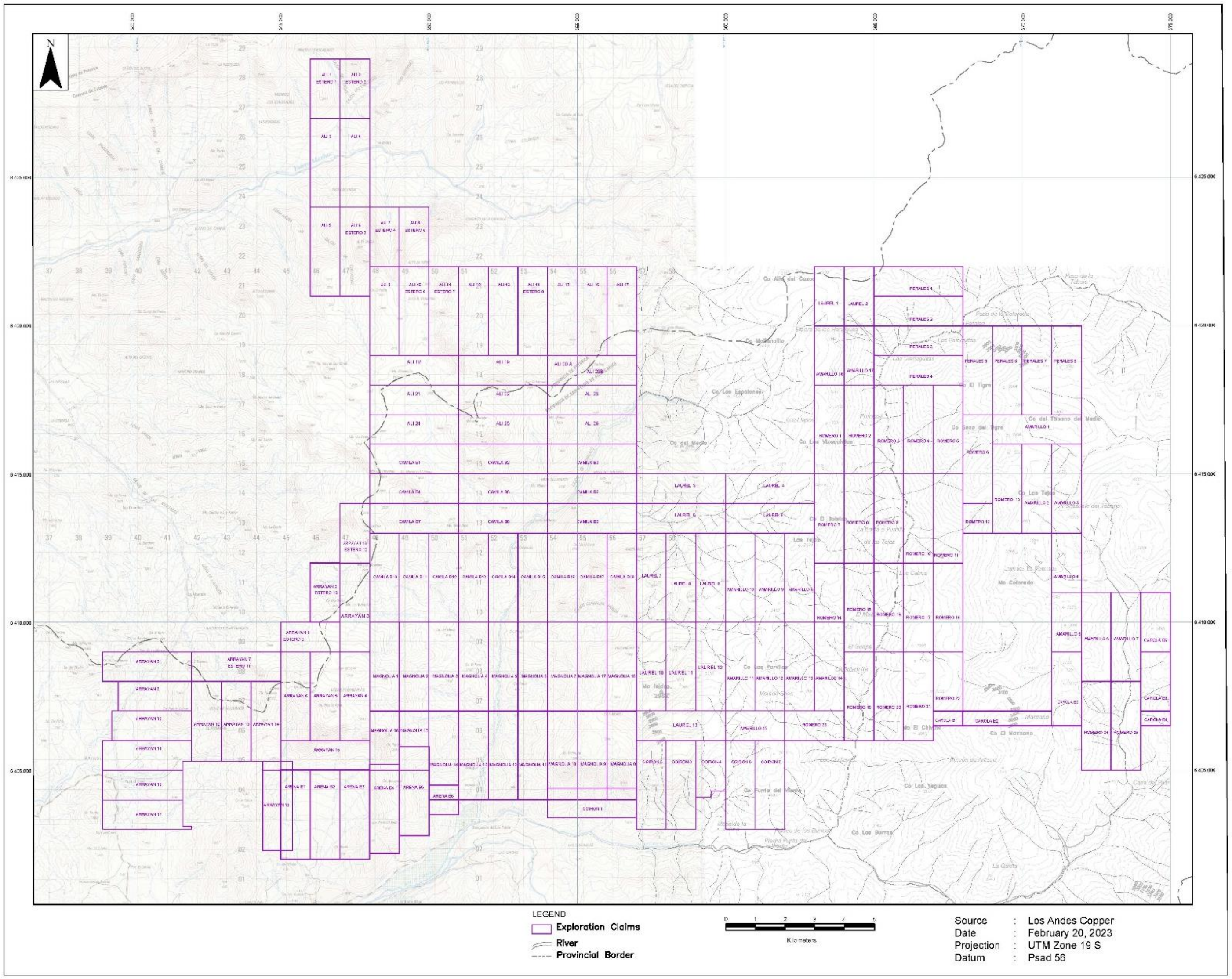
Tetra Tech is not qualified to issue a legal opinion on the mining property and has relied on the letter dated February 20, 2023 provided by Ossa Alessandri Abogados, the mining attorneys for Los Andes Copper in Chile.

Figure 4.2: Exploitation Concessions as of February 20, 2023



Source: Los Andes Copper, 2023

Figure 4.3: Exploration Concessions as of February 20, 2023



Source: Los Andes Copper, 2023

4.2.3 Surface Rights and Legal Access

CMVH has signed a legal agreement with the owner of the land granting access to the Vizcachitas Project, in order to carry out exploration and drilling activities. This easement agreement lasts for a period of 5 years (starting in June 2022) with automatic renewals for additional periods of 2 years. The Company has not yet entered into easements or rights of way for the development of the Project.

According to the Chilean Mining Code, the holder of an exploration or exploitation concession has the right to establish easements, including rights of way, over the land and other mining concessions as required for the adequate exploration or exploitation of the claim. If the owner of the surface property or mining concessions does not grant such easement voluntarily, the holder of the exploration or exploitation concession may apply for the right of way to the Courts of Justice. According to the Mining Code the courts must grant the easement after determining appropriate compensation for potential loss to the landowner.

4.3 Underlying Agreements

A Net Smelter Return (NSR) of 2% for open pit operations and 1% for underground operations is in place over the area where the Mineral Resources will be mined. The economic model includes the third party NSR at the 2% level.

Tetra Tech is not qualified to provide a legal opinion on NSRs related to the mining properties and has relied upon a letter provided by the Company, dated January 31, 2023, confirming the status of the NSRs.

4.4 Taxation & Royalties

Companies and individuals with residence or domicile in Chile are subject to income tax on their worldwide income. Non-resident entities and individuals are taxed only on their Chilean source income. Companies organized under Chilean Law are deemed residents of Chile.

The Corporate Income Tax (First Category Tax, *Impuesto de Primera Categoría* in Spanish) is imposed on income derived from investments, commercial, industrial and mining activities, among others. As a general rule, the corporate tax applies at a rate of 27%. In addition, all mining properties are subject to statutory obligations to the Chilean Government in the form of a Mining Royalty Tax (*Impuesto Específico a la Minería* in Spanish, IEM). This tax was introduced in 2006 and amended in 2010 and is applied to the operating (mining) profits of all operating units. More details on taxation are provided in Chapter 22.

4.5 Environmental Permits

Under Chilean Environmental Law, the Environmental Impact Assessment System (SEIA) reviews and manages the environmental impacts of projects, defining the process for obtaining environmental licences. The Environmental Assessment Service (SEA) is the legal entity in charge of managing the SEIA. Law N° 19.300 defines the types of projects that must be submitted to the SEIA. Mining projects are among those that must be submitted to the SEIA.

There are two different proceedings for environmental assessment. For projects that do not generate “significant environmental effects”, the applicant must initiate the process with an Environmental Impact Declaration (DIA), which consists of an affidavit describing the project. For projects that generate “significant environmental effects”, a complete assessment must be conducted, followed by the presentation of an Environmental Impact Assessment (EIA). The EIA must include a complete description and analysis of the project and its impacts, and the proposed mitigation, reparation and compensation measures. All major mining projects enter the SEIA through the submission of an EIA.

Currently, the Company holds environmental licence RCA N° 14, 2021. This authorization was issued following the presentation of a DIA and authorized the pre-feasibility exploration campaign (350 drill pads over a 48-month period). In March 2022 the Environmental Court issued a preliminary injunction suspending RCA N° 14, 2021, this decision was later revoked in July 2022. According to the July 2022 decision, the Company was authorized to resume drilling subject to meeting certain conditions.

The Company has not yet submitted an EIA for the Vizcachitas Project.

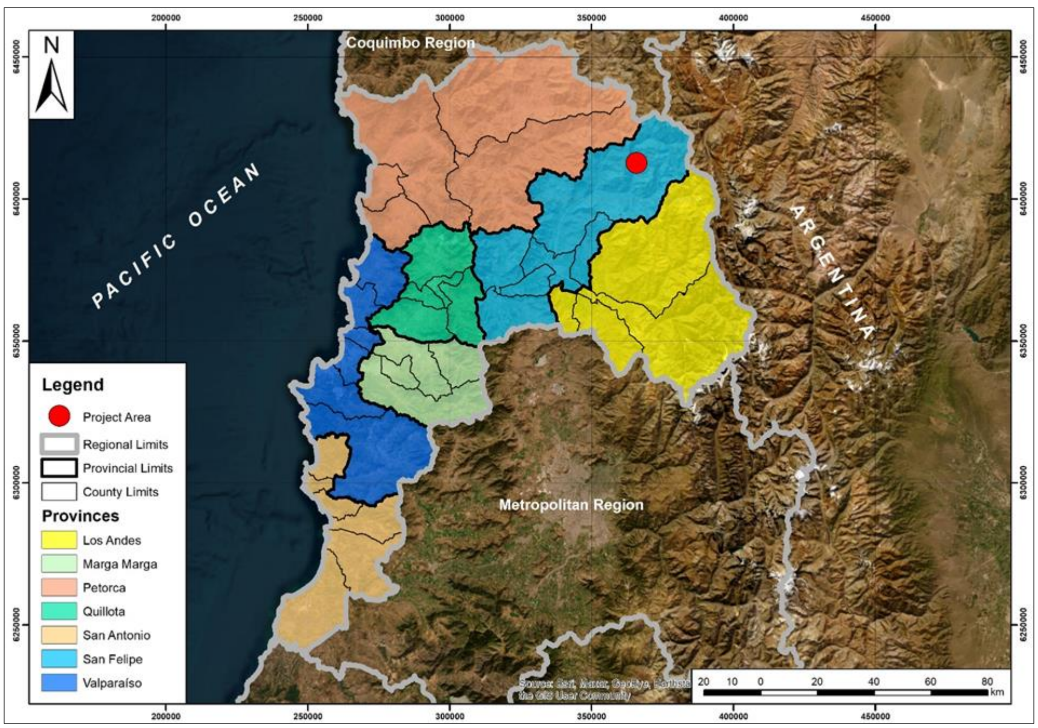
5. ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

The Vizcachitas Project site is located about 150 km north-east of Santiago, Chile and 46 km northeast of Putaendo, San Felipe Province. The closest village to the Project is Resguardo Los Patos, which is 25 km southwest of the site. The road from Santiago to Resguardo Los Patos is 125 km and fully paved, from there to the Project site is by dirt and gravel roads. The travel time from Santiago to the site by the above route is approximately 3 hours. Figure 5.1 shows the Provinces in the Valparaíso Region and Figure 5.2 shows the roads between San Felipe and the Project site.

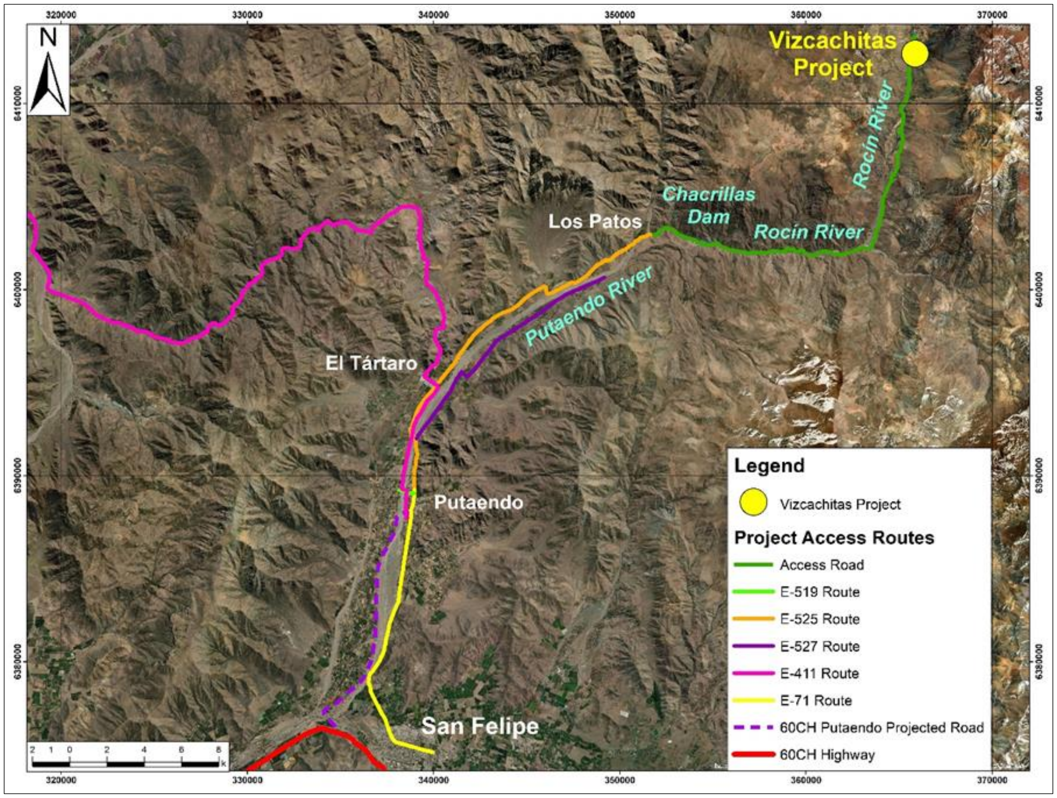
The Vizcachitas Project has year-round access using a four-wheel drive vehicle, currently subject to occasional interruptions following spring storms or run-off when excessive flow in the Rocín River prevents crossing the river.

Figure 5.1: Regional Map



Source: Los Andes Copper, 2022

Figure 5.2: Project Area Map



Source: Los Andes Copper, 2022

5.2 Climate

The Property is located on the western side of the Andes Mountains at an average elevation of 1,950 masl.

The weather is warm and temperate with 6 dry months from late spring to fall (November to April). Average precipitation is about 300 mm per year and falls as rain or snow between April and October. Summer temperatures vary from a few degrees above zero at night to 35°C during the day. Winter temperatures vary between 0°C and 15°C. The relatively low elevation and favourable climate allow year-round exploration and drilling.

A future mining operation should be able to run on a year-round basis, since it will have comparable conditions to other operations in the Valparaíso Region such as Los Bronces and Andina, but at a much lower elevation.

5.3 Local Resources and Infrastructure

The area where the mineral resources are located is called Las Tejas. All exploration works are currently based in this area.

There are no major population centres near the site. Resguardo Los Patos, the nearest village, is 25 km away from the Project site, and has a population of less than 400. The town of Putaendo, 46 km away, has a population of 17,768 (Figure 5.3). In San Felipe and Los Andes, 72 km and 78 km from the Project, and in Putaendo and other surrounding towns, there is a significant skilled and semi-skilled labour force. This labour force works for Los Bronces, Andina and other smaller mining operations (such as Anglo American's El Soldado mine and Chagres smelter) and there are also several suppliers for the Central Chile mining district.

Figure 5.3: Arturo Prat Square, Putaendo

Source: Los Andes Copper, 2022

The Rocín River is dammed downstream of the Project by the Chacrillas dam to provide a stable source of fresh water for agriculture in the Putaendo Valley. The Rocín River and the Chalaco Stream join downstream of the Chacrillas Dam to form the Putaendo River. All water consumers in the Rocín River valley are located downstream of the Chacrillas dam. The Project considers the diversion of the Rocín River through a tunnel that will return the full flow to the river, upriver of the Chacrillas dam.

Vizcachitas is a greenfield site, and the Project will benefit from substantial regional infrastructure including the national power grid and an extensive road network leading to existing port facilities. The relatively low altitude at the site will also be beneficial for operations.

The Property is large enough to accommodate the planned open pit mining operation, although the optimal locations for infrastructure may overlap with mining concessions owned by third parties.

This TR considers shipping the copper concentrate by truck to Ventanas. However, the ports of Valparaíso and San Antonio are possible alternatives to be explored in future studies. There is an operating railway line in San Felipe with connections to the Chagres smelter, and the ports of Ventanas, Valparaíso and San Antonio.

There are several large electrical substations near the Project site. This TR considers that the power for the Project will be provided from the national power grid through a connection to the Los Maquis substation, 61 km from the Project site. Other potential alternatives exist, such as connections to the substations at Nogales or Las Vegas, 105 km and 74 km respectively, from the Project.

Los Andes Copper has declared that it will not use continental water resources and has entered into a non-binding letter of intent with a private desalination plant developer (DESALA) to provide desalinated water from the sea to the future mine. Although the Project considers the use of desalinated water, Los Andes Copper currently owns sufficient water rights for all the anticipated water requirements. The permit has an extraction point on the Aconcagua River approximately 80 km from the Project site.

Other infrastructure for the Project will be new and upgraded access roads.

5.4 Physiography

The Project area is located in the western ridges of the Andes Mountains. The Rocín River valley runs through the property between steep mountain slopes. Elevations on the property range from less than 1,800 masl to more than 3,400 masl, with an average elevation around 1,950 masl. The vegetation consists of shrubs and trees of low to moderate height, which mainly grow at the bottom of valleys near the river and streams. Figure 5.4 shows the topography in the Project area.

Figure 5.4: Project Area Topography

Source: Los Andes Copper, 2022

6. HISTORY

6.1 Prior Ownership of the Property and Ownership Changes

The central mining claim, San José 1/3000 (San José), was claimed in the 1970s. There is no documentary evidence showing that the original owners worked on the Project.

Placer Dome Sudamerica Limited (Placer) reviewed the Project in 1992 and signed an option agreement in 1993. Placer completed mapping and sampling programmes followed by six diamond drill holes totalling 1,953 m.

In 1995 General Mineral Corporation (GMC) acquired 51% of the San José claim and entered into an option agreement for the Santa Teresa, Santa Maria, San Cayetano, and Tigre 1 to Tigre 3 claims. GMC independently claimed the León 1 to León 16 claims, and the total area of this land package was 3,788 ha (Osterman, 1997). In 1997 GMC entered into a joint venture agreement with Westmin Resource Limited (Westmin). Boliden acquired Westmin during the period of the joint venture and terminated the joint venture in 1998.

In 1995 GMC started detailed mapping, sampling, geophysics and drilling (61 diamond drill holes totalling 15,815 m through to 1998). There is no comprehensive written summary of this work. Based on the information gathered, GMC calculated a non-NI 43-101-compliant Measured and Indicated Mineral Resource of 645 million tonnes at an average copper grade of 0.45% and an average molybdenum grade of 0.014% at a 0.3% Cu cut-off.

In 1998 GMC commissioned Kilborn International (Kilborn) to complete an initial feasibility study on the Vizcachitas Project. Kilborn audited the historical GMC resource and concluded that at a copper price of US$1.00/lb, the Net Present Value of the project was US$201 million at a discount rate of 8% with a 20% Internal Rate of Return after-tax (Kilborn, 1998). Having completed the report, GMC put the Project on a care-and-maintenance basis, dropping most of the claims, except the central core of concessions, shortly after completing the initial feasibility study.

Lumina Copper Corp. purchased GMC's subsidiary Vizcachitas Limited in late 2003, including the shares of CMVH, which in turn owned 51% of San José 1/3000 and other surrounding claims constituting the Vizcachitas Project.

In May 2005, under a Plan of Arrangement, Vizcachitas Limited was transferred to Global Copper Corporation (Global), one of four successor companies of Lumina Copper Corp. The camp and core storage were rehabilitated.

In November 2006 GHG Resources Limited (GHG) entered into an agreement with Global to acquire all Global’s interest in the Vizcachitas Property. The acquisition was completed in February 2007. GHG paid US$10,400,000 and issued to Global 6,280,000 shares and 3,900,000 warrants in the capital of GHG. After the purchase GHG focused exclusively on Vizcachitas and was renamed Los Andes Copper Ltd. (Los Andes Copper).

During the period 2007 through 2008 Los Andes Copper drilled a total of 79 drill holes for a total of 22,616 m. The drill hole numbers run from LAV-064 to LAV-142. Towards the end of this period AMEC and SIM Geological Inc. prepared a NI 43-101 Technical Report. The last drill hole included in that report was LAV-124, and its effective date was June 9, 2008 (AMEC, 2008).

The remaining 49% of the San José claim was brought under the control of Los Andes Copper in December 2010, consolidating all the mining property under Los Andes Copper.

6.2 Exploration and Development Work by Previous Operators

6.2.1 Placer Dome Sudamerica S.A.

In 1992 Placer Dome carried out mapping and sampling programmes. These programmes focused on improving the understanding of the geological alteration and mineralization of the deposit to design a drill programme. The mapping defined a semi-circular alteration zone measuring 1.5 km^2 in area (Acosta and Zapata, 1993). Placer identified that the porphyry copper, alteration and mineralization are centred on two complex breccia pipes.

The surface geochemical sampling identified a copper anomaly with values greater than 150 ppm copper and locally of 300 ppm copper related to the breccias. The most consistent anomaly occurred in the stockwork zone in the south-western part of the Project. There are two other anomalies to the north and south of the breccia body.

A 50 ppm molybdenum anomaly is generally coincident with the copper anomaly. Anomalous gold, silver and arsenic were also detected in the same area as the copper and molybdenum anomaly. In 1993 Placer completed six diamond drill holes totalling 1,953 m. The drilling programme is discussed in Chapter 10 of this TR.

6.2.2 General Minerals Corporation

GMC acquired its share in the Vizcachitas Property in 1995 and carried out the following work:

- Surface geological mapping and geochemical sampling
- Trenching with a bulldozer
- Surface geochemical sampling
- 30-line kilometres of induced polarization and resistivity measurements
- Radiometric dating of intrusive rocks associated with the mineralized deposit
- A fluid inclusions study
- Preliminary metallurgical classification
- Laboratory scale metallurgical testing, including detailed flotation and leaching tests

- Collection of hydrological and environmental data for baseline studies
- Resource estimate
- A pre-feasibility study (not NI 43-101 compliant).

Original copies of the drill logs, assay batch dispatch forms and assay certificates are stored in the Los Andes Copper office in Santiago. Analytical reports from Geochron Laboratories with results of the potassium-argon age determinations are included in the 1997 Osterman report.

In 1996 GMC completed 30 line-km of Induced Polarization, Resistivity and Spontaneous Potential measurements on the Vizcachitas Property. Readings were initially made in the south-east part of the main exploration area and then extended 3 km to the north. The survey was systematically expanded to cover 7 km^2 of the Property.

Due to the terrain's rugged nature, survey lines were initially located along drill roads and slopes that geophysical crews could easily and safely negotiate. Most of the area was surveyed with 50 m between stations and lines 300 m to 500 m apart.

Between 1997 and 1998 GMC completed 61 drill holes with a total of 15,815 m drilled. The GMC drilling programme is discussed in Chapter 10 of this TR.

6.2.3 Global Copper Corporation

Global did not undertake any exploration work on the property.

6.3 Historical Resource Estimates

Placer and GMC estimated mineral resources at Vizcachitas as part of the exploration work undertaken on the Property. These resource estimates are not NI 43-101 compliant and should not be relied upon; they are included for historical purposes only.

NI 43-101 compliant resource estimates were prepared by GHG Resources Ltd. in 2007 and Los Andes Copper in 2008, 2013, 2014 and 2019.

6.3.1 Placer Dome

This resource estimate used five of the six diamond drill holes. The results are described in Acosta (1992) and Acosta and Zapata (1993). Placer concluded that Vizcachitas contained an inferred mineral resource of 300 Mt with an average copper grade of 0.42%. This resource estimate is not NI 43 101 compliant (Acosta and Zapata, 1993).

6.3.2 General Minerals Corporation

In 1997, using a cut-off grade of 0.3% Cu, GMC estimated a non-NI 43-101-compliant measured and indicated resource of 645 Mt with an average grade of 0.45% Cu and 0.014% Mo and an inferred resource of 496 Mt with an average grade of 0.38% Cu and 0.014% Mo. The resource estimate was calculated from 14,370 assays from 68 drill holes (Kilborn, 1998).

There is no report on this resource, but an audit of the resource estimate was completed in early 1998 by Mine Reserve Associates Incorporated (MRA) as part of an initial pre-feasibility study on the Vizcachitas Project carried out by Kilborn. According to MRA, all resource estimation parameters met or exceeded industry standards and met the reporting requirements for Canadian securities commissions at that time (Kilborn, 1998).

The Kilborn Initial Feasibility Study on the Vizcachitas Project (Kilborn, 1998) is no longer valid and was not NI 43-101 compliant. The study envisaged an open pit mine, conventional crushing, grinding and flotation for the recovery of copper and molybdenum concentrates from primary sulphide minerals.

This study detailed the previous work on the Project describing the geology, surface sampling, drilling, resource estimate and metallurgical studies. The report concluded that using a copper price of US$1.0/lb, the Project had an IRR before tax of 22% and an NPV at an 8% discount rate of US$201 million (Kilborn, 1998).

6.3.3 GHG Resources Ltd.

In 2007 GHG commissioned A.C.A. Howe International Limited (ACA) to prepare a NI 43-101 resource estimate. The resource estimate was based on 68 diamond drill holes for a total of 18,300 m. Using a cut-off grade of 0.3% Cu, ACA reported 232 Mt with an average grade of 0.46% Cu, 0.014% Mo and 8 ppb Au as an indicated resource and 619 Mt at an average grade of 0.38% Cu, 0.013% Mo and 7 ppb Au as an inferred resource (Priesmeyer and Sim, 2007).

6.3.4 Los Andes Copper Ltd.

In 2008, before the completion of the drilling programme, Los Andes Copper commissioned AMEC and SIM Geological Inc. to prepare a NI 43-101 compliant Mineral Resource estimate from a total of 130 drill holes with a cumulative length of 35,255 m.

Los Andes Copper reported an indicated mineral resource of 515 Mt with an average grade of 0.39% Cu and 0.011% Mo and an inferred mineral resource of 572 Mt with an average grade of 0.34% Cu and 0.012% Mo in the sulphide area using a cut-off grade of 0.30% CuEq.

Los Andes Copper also reported an oxide resource with an Indicated Mineral Resource of 55 Mt with an average grade of 0.38% Cu and 0.01% Mo and an Inferred Mineral Resource of 33 Mt with an average grade of 0.28% Cu and 0.007% Mo using a cut-off grade of 0.20% Cu. Estimates of sulphide and oxide resources are summarized in Table 6.1 and Table 6.2, respectively.

Due to changes in metal prices during this period and the relatively high molybdenum content in the deposit, Los Andes Copper reported the Mineral Resources in the sulphide area based on the copper equivalent grades. The copper equivalent grades in AMEC's report of 2008 were calculated using the following formula:

$$CuEq\ (\%) = Cu\% + (Mo\% * 6.67)$$

The formula assumed a metal price of US$1.50/lb Cu and US$10.00/lb Mo. The formula does not account for metallurgical recoveries.

Table 6.1: Sulphide Mineral Resources Estimate, AMEC 2008

AMEC 2008 Sulphide Resources										
Cut-off CuEq (%)	Tonnage (Mt)	Cu Grade (%)	Mo Grade (%)	CuEq Grade (%)	Cu (Mlb)	Mo (Mlb)	Cu (kt)	Mo (kt)	CuEq (kt)	CuEq (Mlb)
Indicated										
0.20	597	0.36	0.010	0.43	4,738	132	2,149	60	2,567	5,659
0.25	563	0.37	0.011	0.44	4,592	137	2,083	62	2,477	5,461
0.30	515	0.39	0.011	0.46	4,428	125	2,009	57	2,369	5,223
0.35	442	0.41	0.012	0.48	3,995	117	1,812	53	2,122	4,677
0.40	351	0.43	0.012	0.51	3,327	93	1,509	42	1,790	3,946
0.45	252	0.47	0.013	0.55	2,611	72	1,184	33	1,386	3,056
0.50	160	0.51	0.013	0.60	1,799	46	816	21	960	2,116
Inferred										
0.20	798	0.30	0.010	0.36	5,278	176	2,394	80	2,873	6,333
0.25	685	0.32	0.011	0.39	4,833	166	2,192	75	2,672	5,890
0.30	572	0.34	0.012	0.41	4,288	151	1,945	69	2,345	5,170
0.35	420	0.36	0.013	0.44	3,333	120	1,512	55	1,848	4,074
0.40	280	0.39	0.013	0.48	2,407	80	1,092	36	1,344	2,963
0.45	176	0.43	0.014	0.52	1,668	54	757	25	915	2,018
0.50	92	0.46	0.016	0.57	933	32	423	15	524	1,156

Notes: CuEq=Cu%+(Mo%*6.67). Metal Price US$1.50/lb Cu, US$10.00/lb Mo. Assuming a 100% mining and metallurgical recovery.

Table 6.2: Oxide Mineral Resources Estimate, AMEC 2008

AMEC 2008 Oxide Resources							
Cut-off CuEq (%)	Tonnage (Mt)	Cu Grade (%)	Mo Grade (%)	Cu (Mlb)	Mo (Mlb)	Cu (kt)	Mo (kt)
Indicated							
0.10	69	0.33	0.009	502	14	228	6.21
0.15	63	0.35	0.010	486	14	221	6.30
0.20	55	0.38	0.010	461	12	209	5.50
0.25	47	0.4	0.010	414	10	188	4.70
0.30	38	0.44	0.010	369	8	167	3.80
0.35	29	0.47	0.010	300	6	136	2.90
0.40	21	0.51	0.010	236	5	107	2.10
Inferred							
0.10	67	0.21	0.005	310	7	141	3.35
0.15	51	0.24	0.005	270	6	122	2.55
0.20	33	0.28	0.007	204	5	92	2.31
0.25	22	0.31	0.007	150	3	68	1.54
0.30	8	0.37	0.006	65	1	30	0.48
0.35	5	0.42	0.007	46	1	21	0.35
0.40	3	0.46	0.008	30	1	14	0.24

6.3.4.1 Historical Preliminary Economic Assessment 2013

Los Andes Copper published a Preliminary Economic Assessment (PEA) and an updated NI 43-101 resource estimate on December 12, 2013 (Coffey et al., 2013). An updated Preliminary Economic Assessment and an updated NI 43-101 resource estimate were published on February 18, 2014.

The 2014 PEA mineral resource estimate is shown in Table 6.3.

Table 6.3: Mineral Resources at Selected Cut-Off Grades, 2014 PEA

2014 PEA: Indicated Resources						
Cut-off (CuEq %)	Tonnage (Mt)	CuEq Grade (%)	Cu Grade (%)	Mo Grade (%)	Cu (Mlb)	Mo (Mlb)
0.20	1,317	0.396	0.341	0.011	9,913	318
0.25	1,191	0.414	0.356	0.012	9,353	305
0.30	1,038	0.434	0.373	0.012	8,539	281
0.35	824	0.462	0.396	0.013	7,201	240
0.40	566	0.501	0.431	0.014	5,374	179
0.45	368	0.543	0.467	0.015	3,788	125
0.50	244	0.588	0.509	0.016	2,515	79

2014 PEA: Inferred Resources						
Cut-off (CuEq %)	Tonnage (Mt)	CuEq Grade (%)	Cu Grade (%)	Mo Grade (%)	Cu (Mlb)	Mo (Mlb)
0.20	521	0.343	0.296	0.010	3,407	111
0.25	404	0.376	0.322	0.011	2,873	101
0.30	318	0.405	0.345	0.013	2,415	88
0.35	212	0.443	0.372	0.015	1,734	70
0.40	130	0.488	0.402	0.018	1,152	51
0.45	76	0.533	0.428	0.022	714	36
0.50	40	0.584	0.466	0.024	415	22

Notes: Copper equivalent grades (CuEq) were calculated using the following expression: CuEq (%) = Cu (%) + 4.95 x Mo (%), where 4.95 reflects the Mo/Cu price ratio of: US$2.75/lb Cu, US$13.6/lb Mo.

6.3.4.2 Historical Preliminary Economic Assessment 2019

A PEA dated June 13, 2019, with an Effective Date of May 10, 2019, titled Preliminary Economic Assessment of the Vizcachitas Project was prepared under NI 43-101 rules by the independent consulting firm, Tetra Tech Chile.

The report presented the Project with:

- After-Tax NPV @ 8% US$1.8 billion and IRR of 20.77% at US$3.00/lb copper
- Payback period of 3.4 years from initial operations; 5.4 years from initial construction
- 45 year mine life
- 0.53% CuEq average head grade to mill over first 5 years of operation

- C1 Cash Cost (net of by-product credits) US$1.36/lb for first 8 years of operation; US$1.58/lb for LOM
- 1,284.06 million tonnes of measured and indicated resources with a 0.451% CuEq grade and 0.396% Cu grade (at 0.25% Cu cut-off grade), and 788.82 million tonnes of inferred resources with a 0.386% CuEq grade and 0.337% Cu grade (at 0.25% Cu cut-off grade).

To assess reasonable prospects for eventual economic extraction, a Whittle pit shell was prepared using the technical and financial assumptions listed below to constrain the estimated resource blocks:

- Plant cost : US$4.9/t
- Energy cost : US$45/MWh
- Mine cost : US$2.2/t
- Cu selling cost : US$0.5/lb
- Mo selling cost : US$1.4/lb
- Cu recovery : 90%
- Mo recovery : 75%
- Material to concentrate : Supergene + Hypogene
- Cu price : US$3.75/lb
- Mo price : US$10.00/lb.

The mineral resources were contained within an open pit shell to demonstrate the prospects of eventual economic extraction. Only blocks within the Whittle pit shell were included in the mineral resources.

The in-pit mineral resources below were reported using a 0.25% Cu cut-off:

- Measured mineral resources: 254.4 million tonnes grading 0.439% Cu, 119.2 ppm Mo and 1.26 g/t Ag giving a 0.489% CuEq
- Indicated mineral resources: 1,029.67 million tonnes grading 0.385% Cu, 146.9 ppm Mo and 1.00 g/t Ag giving a 0.442% CuEq
- Measured and indicated mineral resources: 1,284.06 million tonnes grading 0.396% Cu, 141.4 ppm Mo and 1.05 g/t Ag giving a 0.451% CuEq
- Inferred mineral resources: 788.82 million tonnes grading 0.337% Cu, 127.0 ppm Mo and 0.88 g/t Ag giving a 0.386% CuEq.

The copper equivalent grade was calculated using the following expression: *CuEq (%) = Cu (%) + 3.33 x Mo (%) + 82.6389 x Ag (%)*, using the following metal prices: US$3.00/lb Cu, US$10.00/lb Mo and US$17.00/oz Ag. No allowance for metallurgical recoveries was considered.

Table 6.4 to Table 6.7 present the mineral resources using different cut-off grades. The base case for the estimation of resources was 0.25% Cu.

Table 6.4: Measured Resources In-Pit, 2019 PEA

Measured Resources									
Cut-Off (Cu %)	Tonnage (Mt)	Cu (%)	Mo (ppm)	Ag (ppm)	CuEq (%)	Cu (Mlb)	Mo (Mlb)	Ag (Moz)	CuEq (Mlb)
0.15	282.33	0.415	116.3	1.22	0.464	2,583	72	11.1	2,888
0.20	270.80	0.426	118.4	1.24	0.475	2,543	71	10.8	2,836
0.25	254.40	0.439	119.2	1.26	0.489	2,462	67	10.3	2,743
0.30	221.85	0.463	118.2	1.30	0.513	2,264	58	9.3	2,509
0.35	180.95	0.495	117.4	1.35	0.546	1,975	47	7.9	2,178
0.40	140.40	0.531	117.0	1.42	0.582	1,644	36	6.4	1,801
0.45	101.73	0.574	115.9	1.50	0.625	1,287	26	4.9	1,402

Table 6.5: Indicated Resources In-Pit, 2019 PEA

Indicated Resource									
Cut-Off (Cu %)	Tonnage (Mt)	Cu (%)	Mo (ppm)	Ag (ppm)	CuEq (%)	Cu (Mlb)	Mo (Mlb)	Ag (Moz)	CuEq (Mlb)
0.15	1,430.59	0.332	133.4	0.91	0.384	10,471	421	41.9	12,111
0.20	1,239.16	0.357	140.6	0.96	0.412	9,753	384	38.2	11,255
0.25	1,029.67	0.385	146.9	1.00	0.442	8,740	333	33.1	10,034
0.30	784.35	0.421	154.5	1.04	0.481	7,280	267	26.2	8,317
0.35	549.21	0.463	159.9	1.09	0.526	5,606	194	19.2	6,369
0.40	359.56	0.513	159.3	1.14	0.575	4,066	126	13.2	4,558
0.45	249.22	0.555	156.5	1.20	0.617	3,049	86	9.6	3,390

Table 6.6: Measured and Indicated Resources In-Pit, 2019 PEA

Measured and Indicated Resources									
Cut-Off (Cu %)	Tonnage (Mt)	Cu (%)	Mo (ppm)	Ag (ppm)	CuEq (%)	Cu (Mlb)	Mo (Mlb)	Ag (Moz)	CuEq (Mlb)
0.15	1,712.92	0.346	130.6	0.96	0.397	13,054	493	53.0	14,999
0.20	1,509.96	0.369	136.6	1.01	0.423	12,296	455	49.0	14,091
0.25	1,284.06	0.396	141.4	1.05	0.451	11,202	400	43.4	12,777
0.30	1,006.20	0.430	146.5	1.10	0.488	9,544	325	35.5	10,826
0.35	730.16	0.471	149.4	1.15	0.531	7,581	241	27.1	8,547
0.40	499.96	0.518	147.4	1.22	0.577	5,710	162	19.6	6,359
0.45	350.95	0.561	144.7	1.29	0.619	4,336	112	14.5	4,792

Table 6.7: Inferred Resources In-Pit, 2019 PEA

Inferred Resource									
Cut-Off (Cu %)	Tonnage (Mt)	Cu (%)	Mo (ppm)	Ag (ppm)	CuEq (%)	Cu (Mlb)	Mo (Mlb)	Ag (Moz)	CuEq (Mlb)
0.15	1,635.15	0.264	111.4	0.76	0.308	9,517	402	40.0	11,103
0.20	1,252.87	0.294	118.3	0.82	0.340	8,121	327	33.0	9,391
0.25	788.82	0.337	127.0	0.88	0.386	5,861	221	22.3	6,713
0.30	486.94	0.381	135.6	0.96	0.434	4,090	146	15.0	4,659
0.35	255.39	0.436	144.1	1.03	0.493	2,455	81	8.5	2,776
0.40	135.60	0.497	138.5	1.11	0.553	1,486	41	4.8	1,653
0.45	70.89	0.567	140.6	1.31	0.625	886	22	3.0	977

Notes: Copper equivalent grade has been calculated using the following expression: CuEq (%) = Cu (%) + 3.33 x Mo (%) + 82.6389 x Ag (%), using the metal prices: US$3.00/lb Cu, US$10.00/lb Mo and US$17.00/oz Ag. No allowance for metallurgical recoveries were considered.

7. GEOLOGICAL SETTING AND MINERALIZATION

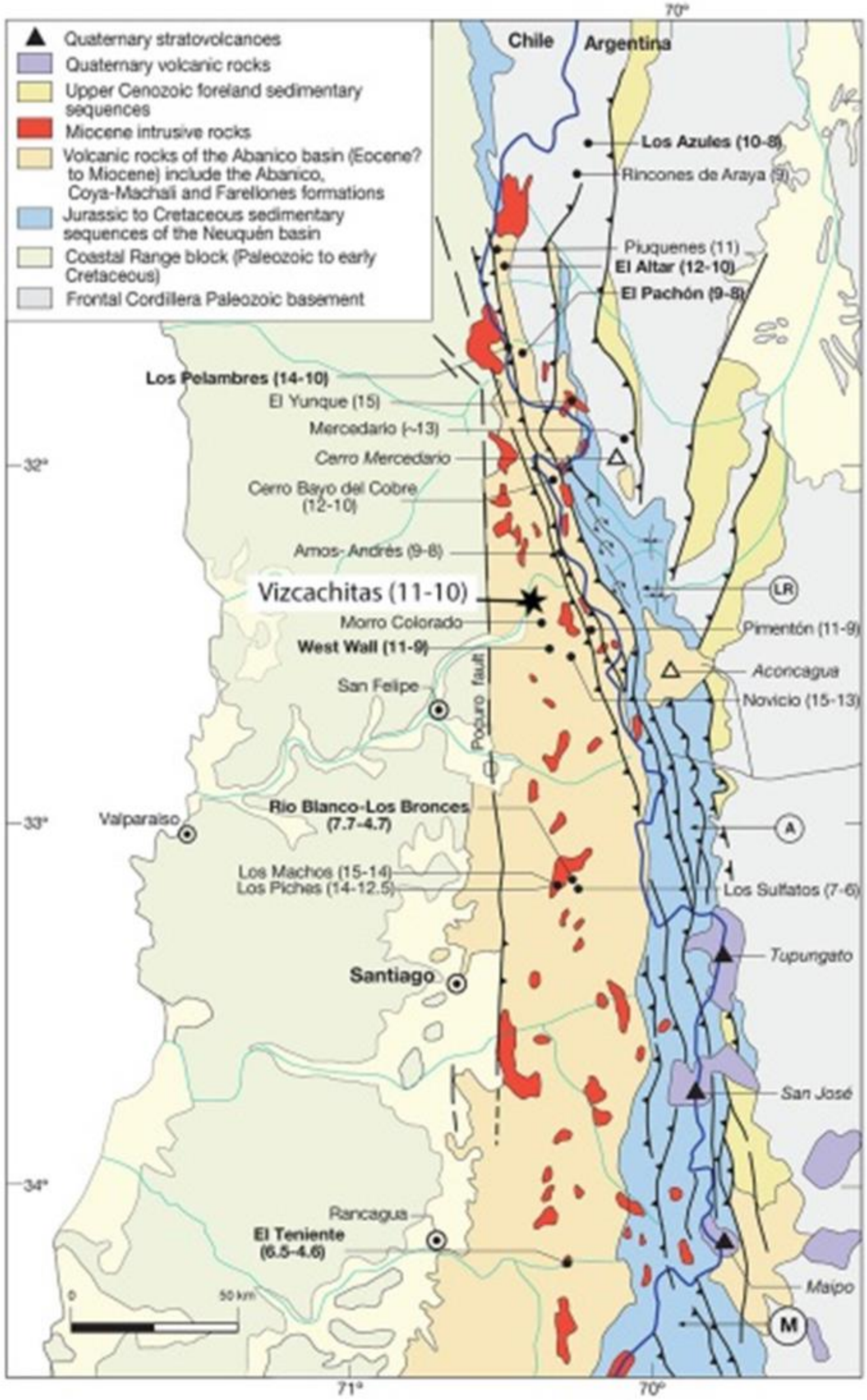
7.1 Regional Geology

The dominant geological feature of this region is the north-south trending Neogene (23 to 2.5 million years ago (Ma)) metallogenic belt that extends along the slopes of the Andes Mountains in Chile and Argentina (Figure 7.1).

In central Chile this metallogenic belt includes world-class copper-molybdenum porphyries, such as Los Pelambres-El Pachón, located 75 km north of the Vizcachitas Project; Río Blanco-Los Bronces, 80 km to the south; and El Teniente, 180 km to the south. Further north the Neogene metallogenic belt includes world-class Miocene (23.03 to 5.332 Ma) epithermal precious metal deposits and other gold, copper and copper-gold porphyries in the El Indio-Maricunga belt (Davidson and Mpodozis, 1991; Sillitoe, 1991) (Figure 7.1).

The Neogene metallogenic belt in central Chile coincides with the position of Miocene volcanic centres and associated flat-lying volcanic rocks, sills and dikes. The Miocene volcanic sequence, with an average thickness of 2,500 m, consists of andesite, basalt (lavas and sills), dacite and intercalations of rhyolitic tuff. They constitute a north-south belt approximately 20 km wide (Farellones Formation; Thiele, 1980; Rivano et al., 1990). Eruption of these volcanic rocks occurred at several volcanic centres, possibly localized by intersections of regional structures. These volcanic rocks overlie folded Oligocene to Early Miocene (34 to 23 Ma) andesitic volcanic and continental sedimentary rocks (Abanico and Coya-Machalí Formations; Thiele, 1980; Charrier et al., 2002) in a non-conforming manner.

Figure 7.1: Tectonic Sketch of the Northern Part of the Abanico Intra-Arc Basin (31°−34° S)



Source: Mpodozis & Cornejo, 2012

The Neogene porphyry copper-molybdenum deposits occur within hydrothermal alteration zones related to multi-phase porphyritic stocks with compositions ranging from quartz diorite to granodiorite. These intrusions and the country rocks host dense networks of sulphide-bearing veins and associated hydrothermal breccia complexes. The country rocks are late Miocene basaltic and andesitic volcanic rocks, diabase sills and gabbro at El Teniente; Miocene andesite and a Middle Miocene granodioritic batholith at Río Blanco-Los Bronces (Serrano et al., 1996), and folded Lower Cretaceous (145 to 100 Ma) volcanic and sedimentary rocks at Los Pelambres (Atkinson et al., 1996).

7.2 Local and Property Geology

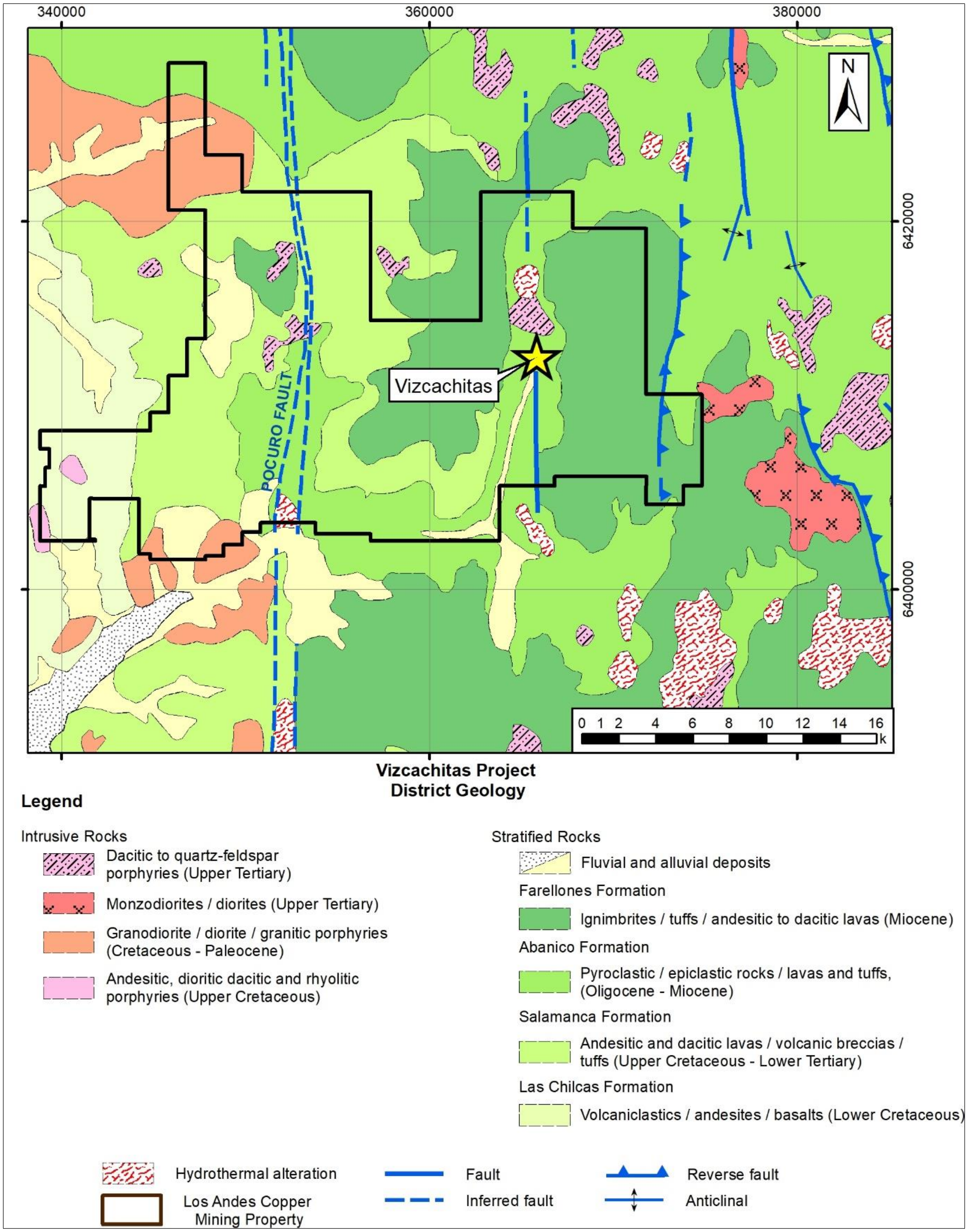
As shown in Figure 7.2 the oldest rocks within the Project area correspond to the Abanico Formation, a sequence of andesitic to basaltic-andesite volcanic flows and volcanic flow breccias that outcrop along the lower slopes of the Rocín River valley. The Farellones Formation unconformably overlies a bedded sequence of conglomerates, andesite flows and breccias that outcrop in the lower half of the valley.

The National Geological and Mining Service (*Servicio Nacional de Geología y Minería*, SERNAGEOMIN) initially mapped the lower sequence as part of the Cretaceous Los Pelambres Formation (Hoja Geológica Quillota y Portillo, 1:250.000, 1993) and the higher sequence as Farellones Formation (Early to Mid-Miocene in age). However, a more recent Ph.D. thesis from Universidad de Chile (Jara, 2013) proposed that the lower sequence is part of the Late-Eocene or Early Miocene Abanico Formation. The regional mapping is supported by U/Pb zircon dating that confirmed these findings.

Irrespective of their exact age and stratigraphic correlation, the lower volcanic and the upper sedimentary-volcanic sequences are the host rocks for the mineralized porphyries and intrusives of the late Middle Miocene and early Upper Miocene age.

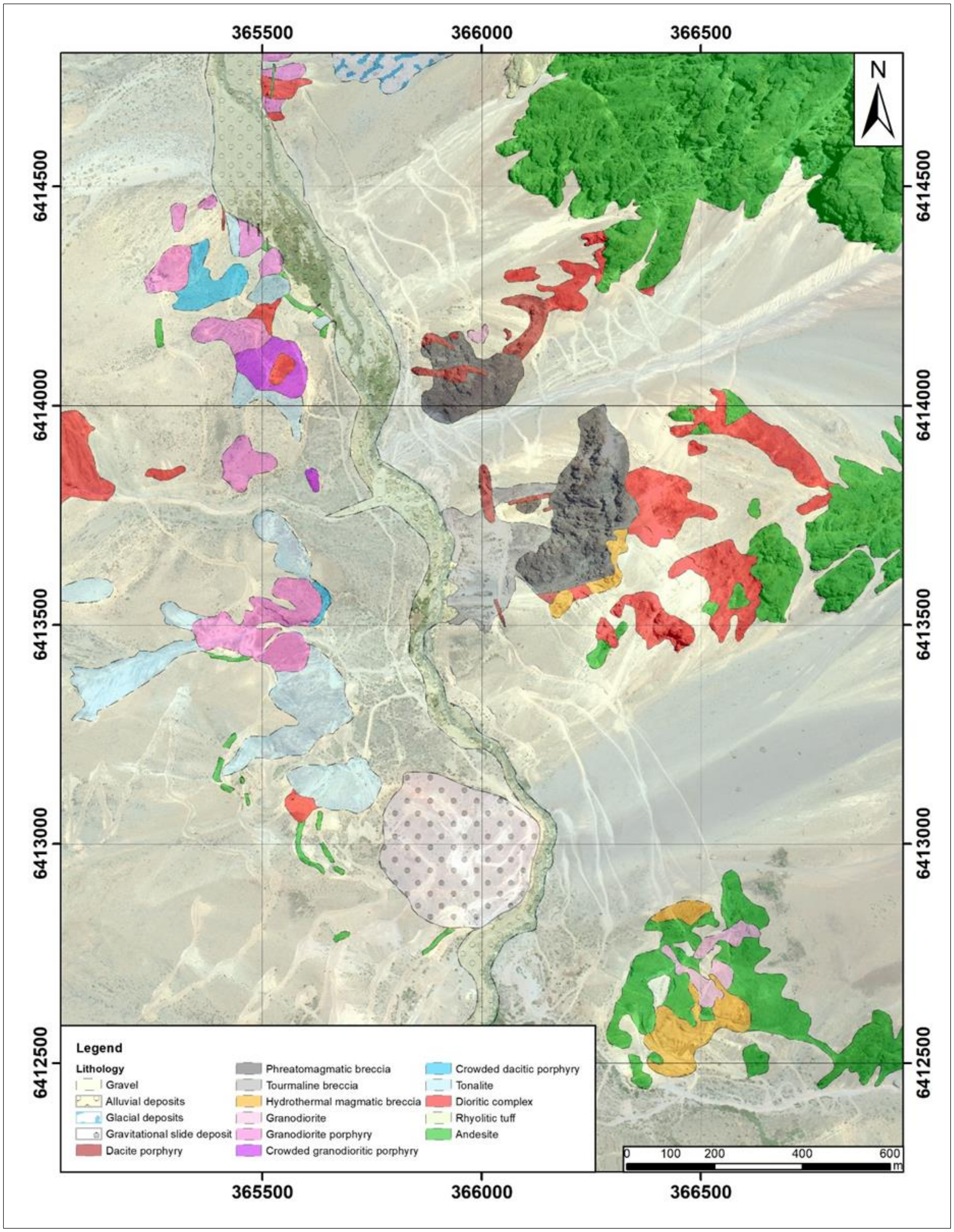
Figure 7.3 shows an integrated surface geological map approximately centred on the Vizcachitas Project.

Figure 7.2: District Geology



Source: Adapted from Rivano et al., 1993

Figure 7.3: Integrated Surface Geological Map



Source: Los Andes Copper, 2022

The Vizcachitas Project is a complex set of porphyries and hydrothermal breccias intruded into a sequence of andesitic and daciandesitic volcanic rocks of the Abanico Formation of the Eocene-Oligocene age.

The geological model was prepared using new data from the drill core, thin sections and radiometric dating. Twenty vertical sections oriented N110°, separated every 100 m, and two horizontal sections formed the basis of the geological model. The same drill holes as used in the Resource Estimate plus the 6 Placer Dome drill holes were also used for the geological model. A total of 174 drill holes, 60,518 m drilled between 1993 and 2022 From this sectional model, a 3D model was developed in Leapfrog software. The Leapfrog model was used to generate the .dxf files used to prepare the estimation domains.

The activation of the magmatic chamber began with the injection of the precursor granodiorite. The barren inter-mineral granodiorite outcrops on the western margin of the system. The precursor granodiorite is a granodiorite biotite-hornblende porphyry, with age varying from 12.7 to 12.4 Ma. It is challenging to identify because it is texturally similar to the inter-mineral granodiorite, especially with the overprinting alteration.

The productive complex starts with a diorite with early porphyry character, compositionally fine to medium-grained quartz diorites dominate along with crowded dioritic porphyry segments (Figure 7.4). The diorites occur mainly north of the phreatomagmatic breccia body and on the western edge of the system. They may also extend north-west, but this area has not been adequately drilled.

Figure 7.4: Left, V2015-05 (562 m) Quartz Diorites
Right, LAV-066 (126.6 m) Dioritic Crowded Porphyry

Source: Los Andes Copper, 2022

A crowded porphyry is a porphyry or segments of a porphyry rich in phenocrysts with high granulometric contrast. They have a fine quartz-feldspathic fundamental mass, unlike other rocks of the intrusive complex, where the interstitial phases have a coarser grain size and their textures are more cohesive. A crowded porphyry is typically associated with magmas having a greater capacity to contain volatiles and, consequently, more likely to transport or contain a higher level of copper mineralization. The crowded porphyry is important in directing drilling programmes to target the most prospective areas of the Project. The early intrusive dioritic complex consisting mainly of quartz diorites but also including crowded porphyry segments is dated between 12.5 Ma and 12.58 Ma.

The next phases of intrusives are inter-mineral tonalitic and granodioritic phases. The age dating indicates little difference between these units, and they are probably from the same intrusive magma. The early inter-mineral tonalitic complex consists of crowded tonalites and dacitic porphyries and is dated between 11.8 and 12.08 Ma. The granodiorites, granodioritic porphyries and crowded granodioritic porphyries within the modelled area are part of the inter-mineral productive porphyries with an age of 11.9 Ma.

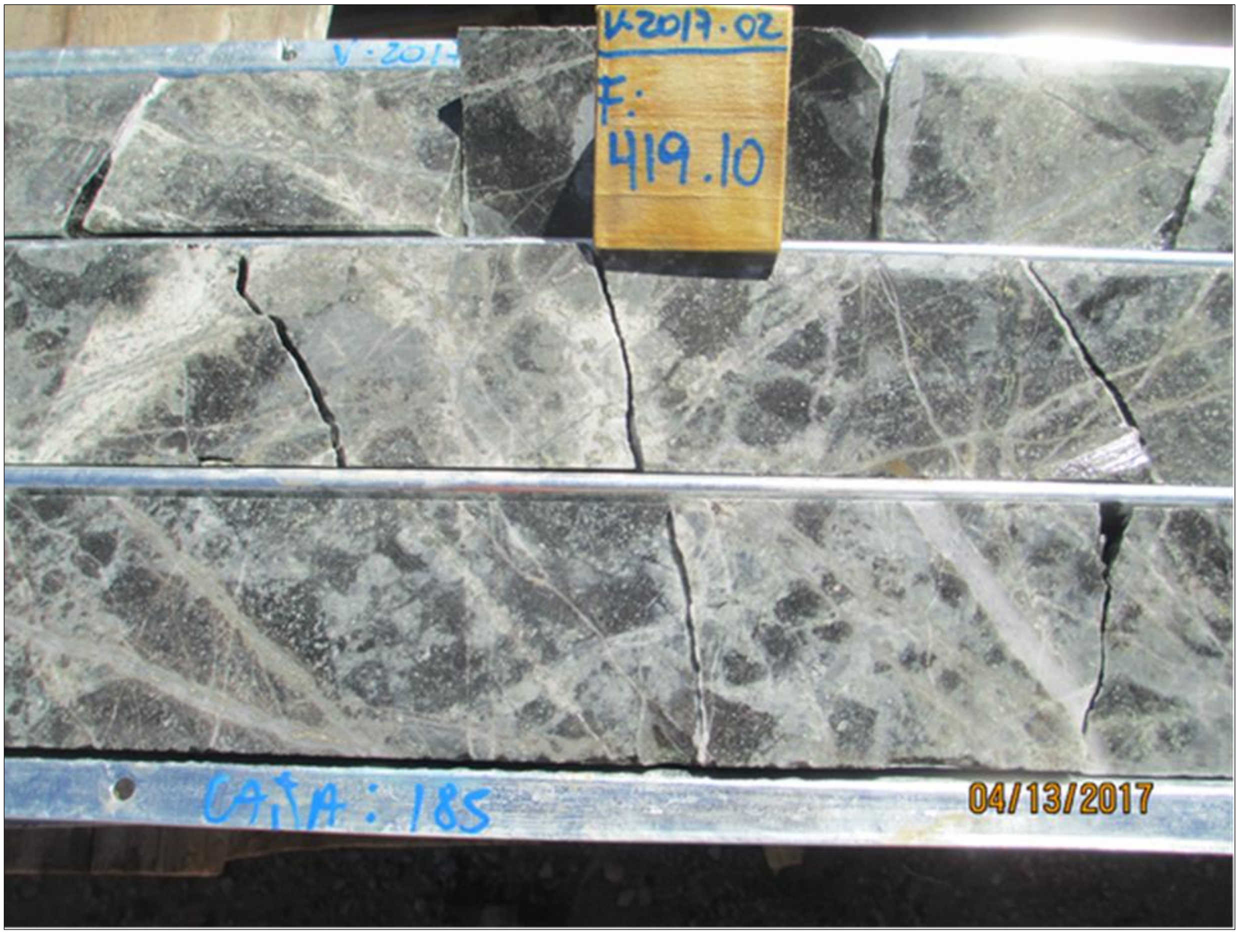
Mineral mapping using the Inductively Coupled Plasma-Mass Spectrometry assay data could not separate the two, also suggesting that they may be from the same intrusive complex (Halley 2018, Halley 2022). The earliest unit is of tonalitic character and crowded dacitic porphyries, which make up a sizable volume of rock of the Vizcachitas igneous complex (Figure 7.5). The late inter-mineral units of a granodioritic character are inequigranular granodiorites, granodioritic porphyries and crowded granodioritic porphyries. The tonalitic and granodioritic units are located on the southern edge and in a continuous corridor in the western part of the system.

Cutting through the volcanic and intrusive units are a set of hydrothermal and magmatic-hydrothermal breccias with a quartz-potassium feldspar-anhydrite matrix, local presence of biotite and associated chlorite (Figure 7.6). Two breccia bodies have been identified: the northern and southern breccias.

**Figure 7.5: Top Left, LAV-064 (400 m) - Crowded Porphyritic Dacite
Top Right, LAV-117 (218 m) - Granodiorite, Bottom
Bottom Left, LAV-114 (40.5 m) Tonalite
Bottom Right, V2017-04 (442.5 m) Granodiorite Porphyry**

Source: Los Andes Copper, 2022

Figure 7.6: V2017-02 (419 m) – Hydrothermal Magmatic Breccia



Source: Los Andes Copper, 2017

The final post-mineral phase of the intrusive complex is phreatomagmatic breccias or diatremes that intruded into the central part of the Project (Figure 7.7). Followed by a later set of dacitic composition dikes and plugs with preferential orientations north-south to NNE. The diatreme comprises polymictic breccias with a rock dust-clay-silica matrix or quartz-tourmaline matrix. Mineralized fragments are found at the margins of the diatreme breccia.

Figure 7.7: V2017-05 (88 m) – Phreatomagmatic Breccia

Source: Los Andes Copper, 2022

7.3 Mineralization

The latest mapping carried out by Los Andes Copper shows vertical zoning that is typical of porphyry-type systems. The first 10 m to 70 m thick upper zone is partially leached with some copper sulphides remaining, iron oxide mineralization, jarosite, goethite and, to a lesser extent, hematite. Copper oxides, such as chrysocolla, are occasionally observed in fractures.

Below the leached zone is a secondary enrichment zone or supergene zone of weak to moderate intensity, with chalcocite and covellite. The secondary minerals occur in fractures and as fine surface coatings on pyrite and chalcopyrite. The supergene thickness varies between 2 m and 100 m, with a mean thickness of 50 m, copper grades may exceed 1%, and the average grade for the supergene drilling is 0.47% Cu.

The hypogene or primary mineralization is mainly chalcopyrite, with pyrite. Bornite occurs in several of the drill holes below 800 m. In drill hole V2017-10, located in the northern area of the Project, bornite accounts for 15% of the total sulphides below 900 m, indicating that a possible bornite core could be found below the current drilling.

There is no correlation between the molybdenum mineralizing event and the copper mineralization. The molybdenite is associated with small B-type quartz veins and small late hydrothermal D-type veins.

Chalcographic studies conducted by GMC in 1998 and Los Andes Copper in 2020 show the local presence of grey copper sulphides (tennantite, tetrahedrite and enargite) in small amounts.

7.3.1 Veinlet Mapping 2020

During 2020 the consulting geologist Marco Carrasco carried out an 8 month programme to re-log all the core, specifically to record the veinlet distribution and relationships with each other and with each intrusive phase. A total of 44,610 m of core were mapped from a total of 162 drill holes. During the 2021 to 2022 drilling campaign, the core was logged using the same veinlet classification scheme to ensure that all the data were compatible with the previous mapping.

Each lithological unit had a different distribution or morphology for the veinlets, probably due to the different chemical compositions and textures for each unit. For example, in andesites the C-veinlets are straight and thin and have little halo development.

Veinlets are estimated to control 70% of the copper mineralization and 90% of the molybdenum mineralization.

Table 7.1 shows the main characteristics of the veins identified in the Vizcachitas porphyry system and Figure 7.8 to Figure 7.13 show examples of the veins.

Table 7.1: Vizcachitas Vein Types

Veinlet Type	Main Minerals	Phase	Central Vein	Halo	Morphology	Width
EDM - 1		Late Magmatic	Biotite/ quartz / chalcopyrite / pyrite	No	Curved to straight	< 3 mm
EDM - 2		Late Magmatic	Biotite / chlorite / quartz / chalcopyrite / pyrite	quartz / potassium feldspar	Straight to curved	< 1.0 cm
M		Late Magmatic	Biotite / chlorite / quartz / chalcopyrite / pyrite / manganese +Anf	No	Straight	< 1.0 cm
A		Late Magmatic	Granular quartz / chalcopyrite / pyrite	No	Curved to straight	< 1.0 cm
B	Molybdenum	Transitional	Crystalline quartz / molybdenite /- chalcopyrite / pyrite anhydrite	No	molybdenite	< 1.0 cm
C	Copper	Early Hydrothermal	Granular quartz / chalcopyrite - pyrite / anhydrite	No	Straight to curved	< 5 mm
D	Pyrite	Hydrothermal Principal	Crystalline quartz / pyrite / chalcopyrite	Quartz sericite	Straight	<2.0 cm
E	Arsenic	Late Hydrothermal	Crystalline quartz / calcite / pyrite +Ten-chalcopyrite	Quartz sericite - Advanced Argillic	Straight	< 1.0 cm
T		Post mineral Hydrothermal	Crystalline quartz / tourmaline / chlorite	Quartz sericite	Straight	< 1.0 cm
Late Veins	Cu - Mo - As	Post mineral Hydrothermal	Quartz / anhydrite / chalcopyrite / molybdenite / pyrite / tennantite	Quartz sericite	Curved to straight	< 2 cm

Figure 7.8: EDM-1, EDM-2, A, B and C Veinlets (V2015-02 421m)	Figure 7.9: EDM Veinlets in Diorite (V-37 98m)
	
Figure 7.10: A Veinlets in Granodiorite (LAV-139 282m)	**Figure 7.11: B Veinlets in Granodiorite with Quartz and Molybdenite**
	
Figure 7.12: C Veinlets Stockwork in Andesite (LAV-140 342.5m)	**Figure 7.13: D Veinlets with Pyrite and Quartz-Sericite Halo**
	

Source: Los Andes Copper, 2022

The veinlet interactions at different stages of the evolution of the hydrothermal system are:

- Lateral and vertical gradation: Compositional and morphological change laterally and vertically.
- Veinlet paragenesis: The intersection of the different vein types shows the temporal relationship between the different vein events.

- Hypogene enrichment: The mineralization of veins that are either barren or very weekly mineralized. For example, EDM-2 veinlets mineralized during the C-type veinlet event.
- Supergene enrichment: Exogenous processes that generate changes in the primary mineralogy of veins, for example, the coating of pyrite and chalcopyrite with chalcocite.
- Remobilization depletion: Pervasive hydrothermal solutions producing remobilization of pre-existing mineralization.
- Remobilization mutation: Compositional change of veins due to subsequent events, for example, biotites altered to chlorite.

The total volume of veinlets in each metre was calculated by multiplying the number of veinlets per metre by their average width. The model was then generated from that value.

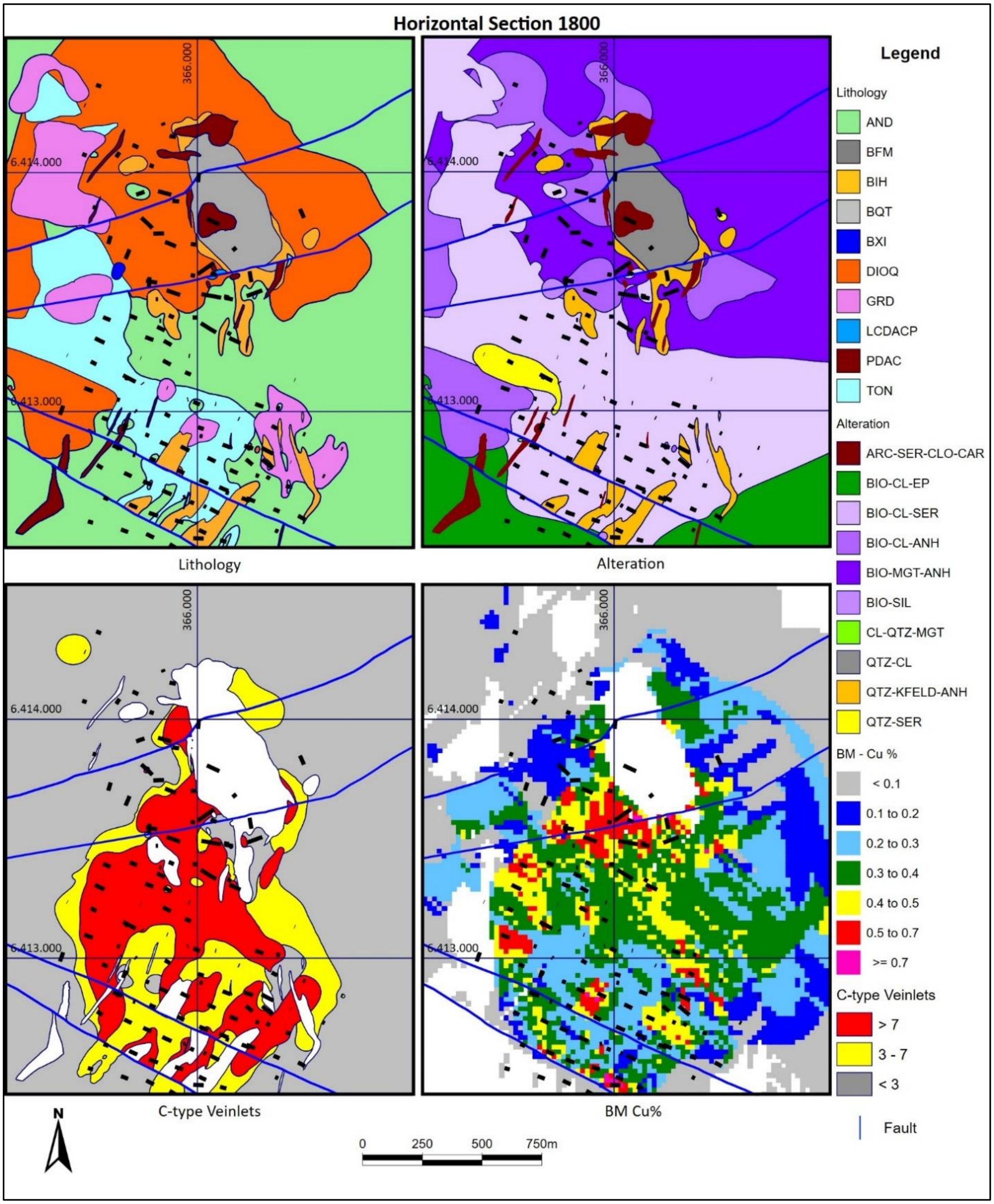
The following models were prepared:

- The sum of EDM-type + A-type + B-type + C-type – main copper veins
- C-type veins – the vein type which contributes the most copper to the system
- B-type – the main contributor of molybdenum to the mineralized system

Figure 7.14 and Figure 7.15 are horizontal sections at 1,800 masl and 1,500 masl showing the lithology, alteration, C-type veinlet intensity and copper distribution within the block model. These clearly show the strong correlation between the C-type veinlet intensity and the copper grade distribution. The blocks with copper grades greater than 0.4% Cu are either associated with the hydrothermal breccias or the quartz diorites and tonalites with high C-type vein intensities. The EDM-type, A-type and B-type veinlets also have copper mineralization, but higher grade areas are nearly always associated with the later C-type veinlets.

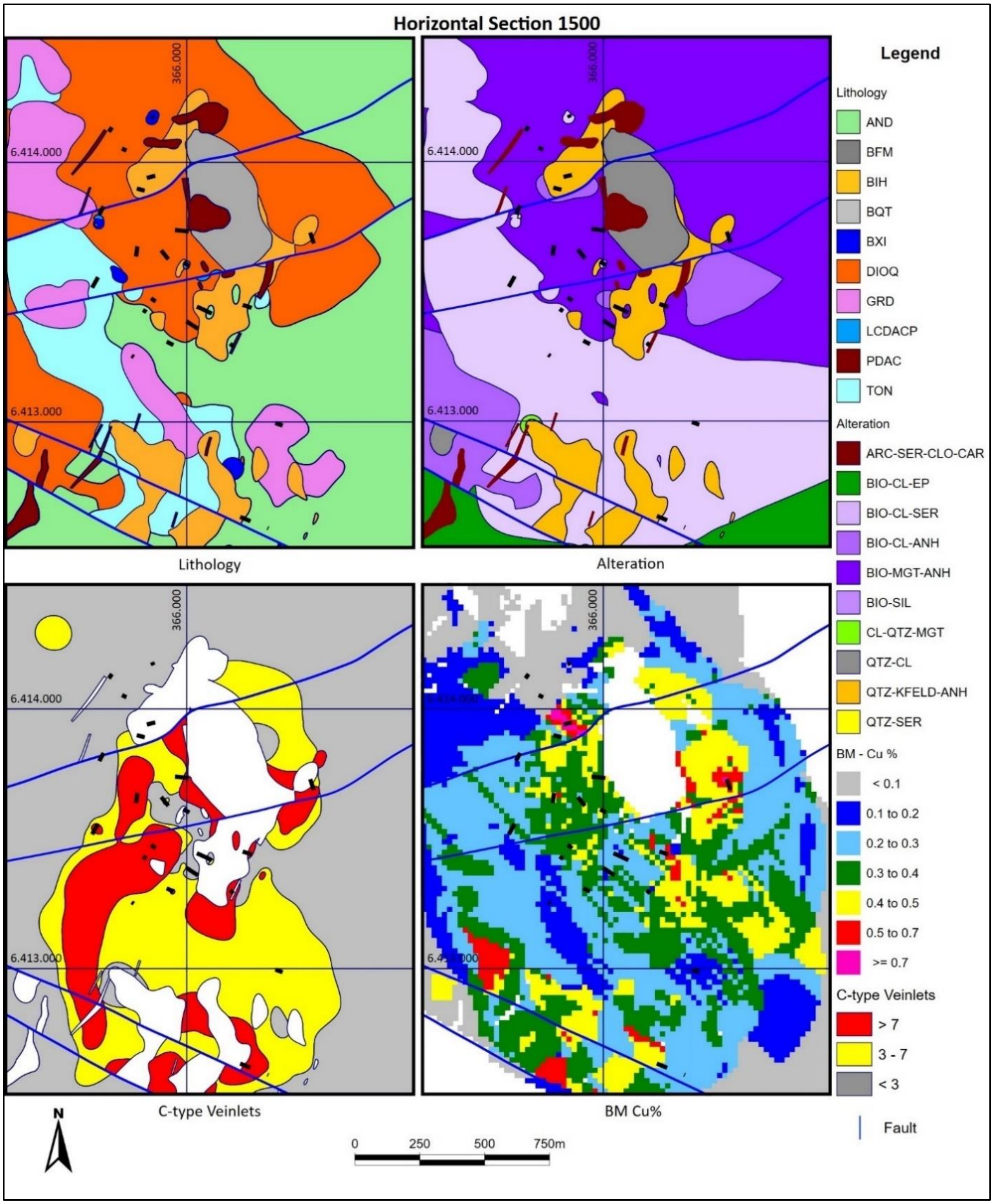
Figure 7.16 and Figure 7.17 are horizontal sections 1,800 masl and 1,500 masl showing the lithology, alteration, B-type veinlets intensity and the molybdenum distribution within the block model. A strong correlation between the B-type veinlets and the molybdenum distribution is unrelated to either the lithology or the alteration. However, the molybdenum mineralization has a north-west trend mirroring the B-type veinlet distribution.

Figure 7.14: Horizontal Section 1,800 masl – Lithology, Alteration, C-type Veinlets and Cu% BM



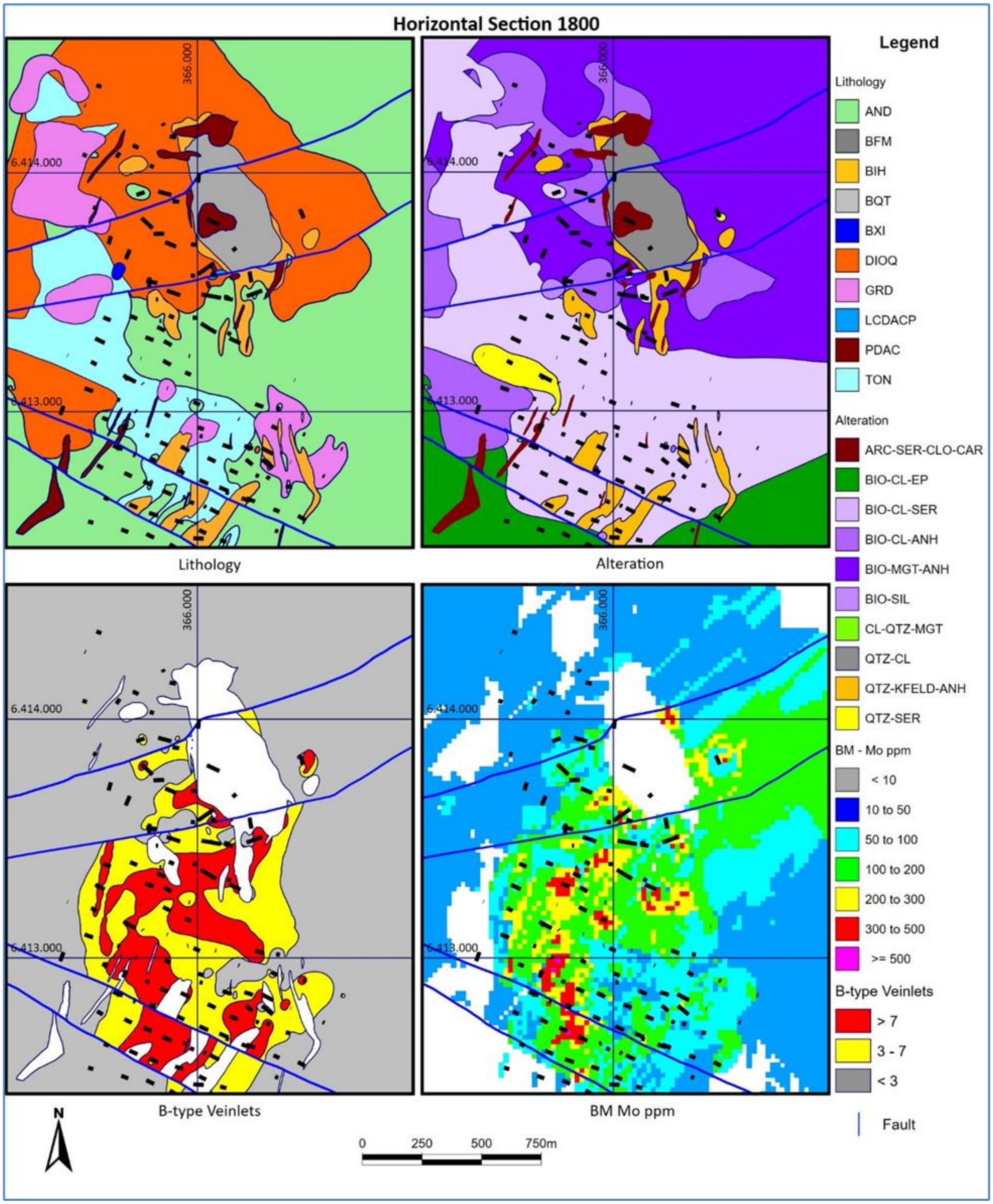
Source: Los Andes Copper, 2022

**Figure 7.15: Horizontal Section 1,500 masl –
Lithology, Alteration, C-type Veinlets and Cu% BM**



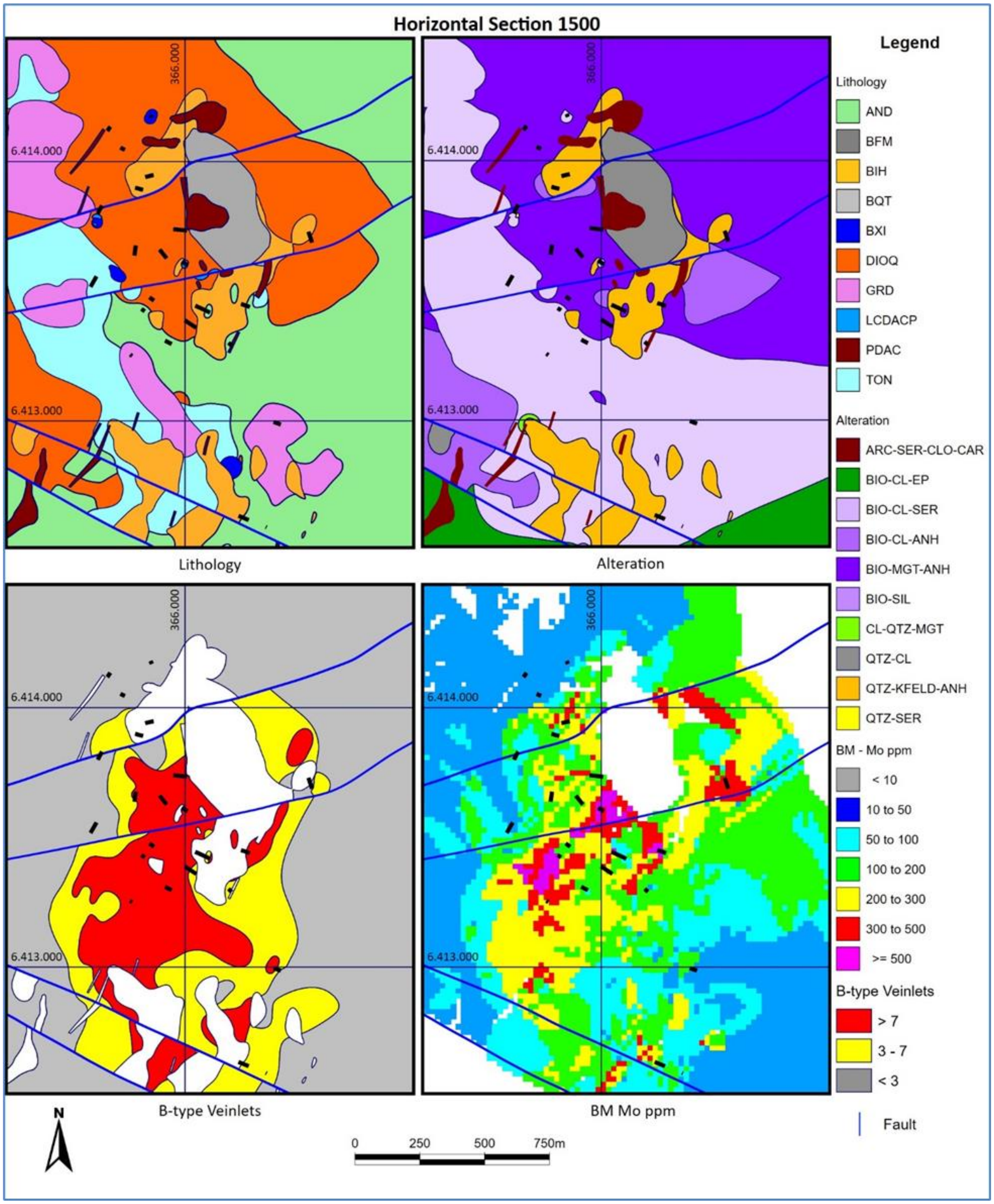
Source: Los Andes Copper, 2022

Figure 7.16: Horizontal Section 1,800 masl –
Lithology, Alteration, B-type Veinlets and Mo ppm BM



Source: Los Andes Copper, 2022

Figure 7.17: Horizontal Section 1,500 masl – Lithology, Alteration, B-type Veinlets and Mo ppm BM



Source: Los Andes Copper, 2022

8. DEPOSIT TYPES

The Vizcachitas mineral deposit has similar characteristics to other Andean-style copper-molybdenum porphyry mineral deposits. These mineralized deposits contain large masses of hydrothermally altered rocks, sulphide-bearing small veins, disseminated sulphides, quartz veins and stockworks that may cover several square kilometres. These altered areas are commonly coincident with shallow intrusives, hydrothermal breccias and intrusion breccias.

The intrusives, hydrothermal breccias and intense fracture zones developed from a set of mineralized fractures commonly coincident with the highest concentrations of metals.

Surface oxidation commonly modifies the distribution of mineralization in degraded environments. The acidic meteoric waters generated by pyrite oxidation leach the copper from the soluble copper minerals. The leached copper is re-deposited as secondary minerals (such as chalcocite and covellite) in a supergene enrichment zone in the form of a mantle. This leaching process produces a copper-poor zone above a relatively thin zone of the supergene enrichment zone. Below the supergene enrichment there is a thicker zone of primary (hypogene) mineralization. At Vizcachitas, the leached and supergene zones are immature, probably due to rapid erosion caused by the Rocín River. The leached zone has some copper sulphides, iron oxide mineralization, jarosite and goethite. Copper oxides, such as chrysocolla, are occasionally observed in fractures. The supergene zone has chalcocite and covellite coated on chalcopyrite and pyrite.

Porphyry systems may also show hypogene enrichment. The hypogene enrichment process may be related to the introduction of late hydrothermal fluids enriched in copper along structural pathways into areas of primary mineralization. Such enrichment processes result in high grades in the hypogene zone. At Vizcachitas, the hypogene mineralization is mainly chalcopyrite, with pyrite, but bornite occurs in several drill holes below 800 m.

9. EXPLORATION

From the beginning of the 1990s to date three companies have explored the Property: Placer Dome, General Minerals Corporation and Los Andes Copper. The Placer Dome and General Minerals Corporation exploration is described in Chapter 6 of this TR. This Chapter only reviews Los Andes Copper exploration.

9.1 2006-2008 Surface Mapping

In 2006 Los Andes Copper re-assessed the geological model prepared by GMC, updating cross-sections and surface mapping. The district mapping was updated and expanded to cover the area to the north of the drilling.

9.2 2012-2014 Geological Data Compilation

From 2012, under a new geology team, Los Andes Copper reviewed and documented the historical data available on the Property. The historical paper-based logging and assaying information was digitally captured and checked using double data entry. The captured data was entered into a Micromine GBIS database. The most relevant paper surface maps and vertical sections were digitized in either ArcMap or Micromine.

Coordinate data was converted from the Provisional South American Datum 1956 to the World Geodetic System 1984 Datum (SIRGAS/WGS84). Converting the topographic data into one projection ensures that there are no errors due to inadvertently using the wrong datum.

9.3 2017 Geological Mapping

In 2017 the geological consultant, Santiago Gigola, conducted 11 days of geological mapping at a scale of 1:10,000. This work consolidated the previous surface mapping and integrated this into the 3D model generated from the diamond drilling (Gigola, 2017).

Mapping around the Vizcachitas deposit and a review of the core allowed the recognition of an alteration zonation (inner potassic to phyllic to outer chlorite-rich propylitic) that shows the northern, western and eastern limits of the porphyry centre. Recent mapping also indicates that the porphyries and breccias that outcrop on both sides of the valley intrude a hypidiomorphic, equigranular to inequigranular composite (multi-phase) pluton, dominated by granodiorite and quartz-diorite phases. This sizeable composite intrusion was interpreted to represent a pre-mineral precursor pluton. A review of previous petrographic studies suggests that several descriptions support this thesis.

The mapping identified the Breccia Sericita area at the northern end of the Rocín valley. This has sericite-altered and tourmaline-cemented hydrothermal breccias on the surface, similar to those found on the Vizcachitas Project. Despite the relatively small hydrothermal breccia (approximately

100 m by 100 m), the area requires further work, especially the colour anomaly extending up to 1.2 km to the west.

Evaluation of a large colour anomaly 8 km south-west of Vizcachitas (3,000 masl) confirmed the favourable location of the area along the Farellones porphyry belt, of considerable size (~5 km x 2 km) and close to Vizcachitas. Evidence of advanced argillic alteration (i.e. Morro Colorado) and sericite-altered hydrothermal breccias are positive indications.

Mapping the NDF prospect, 4 km north of Vizcachitas, showed that it probably represents an intrusive-centred hydrothermal alteration zone measuring ~1 km in diameter. It is spatially and genetically associated with a well-exposed chlorite-tourmaline +/-magnetite +/-pyrite altered equigranular granitic intrusive measuring approximately 400 m x 200 m. The surrounding volcanics host locally intense silicification and jarosite (after pyrite) veining. The exposed granitic intrusive is responsible for the alteration and veining observed here and lacks most of the hallmark characteristics of a mineralized porphyry.

9.4 2020 Surface Mapping

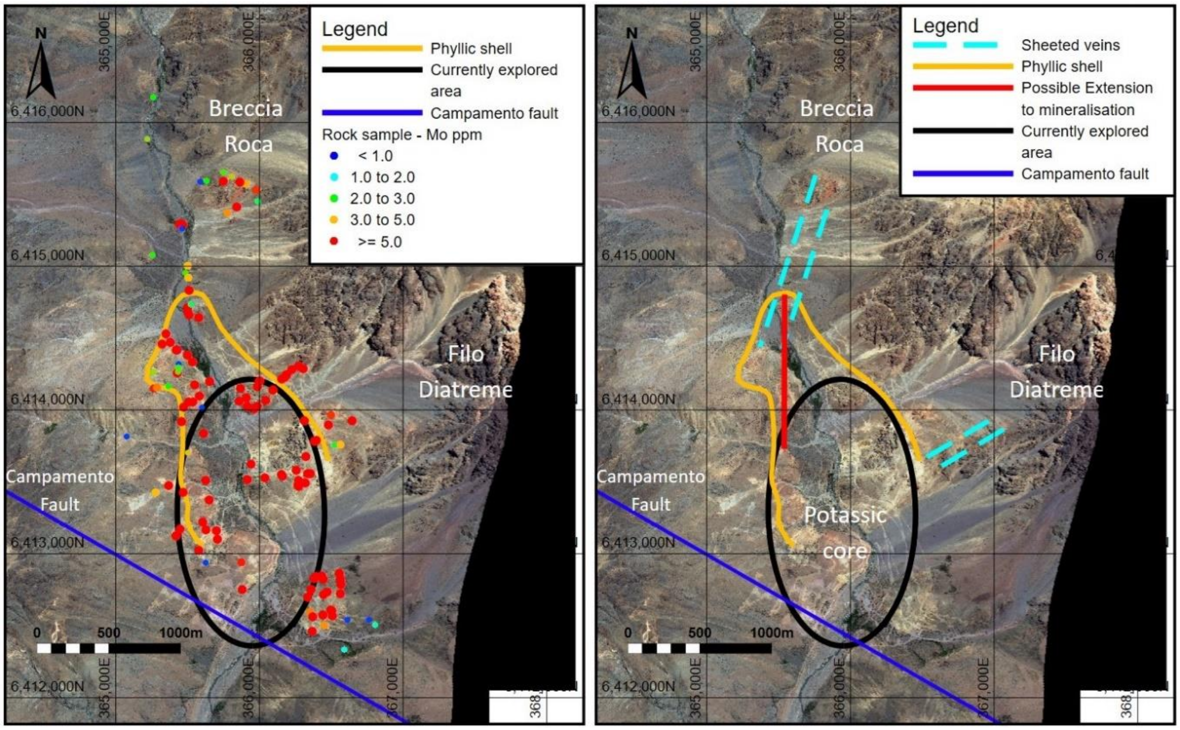
In 2020 the geological consultant David Hopper conducted reconnaissance geological mapping and rock chip sampling of the northern extensions of the Vizcachitas Project area (Hopper, 2020). This work confirmed that Vizcachitas is a zoned porphyry copper system with early central potassic alteration zoned out to distal propylitic alteration and up into a halo of phyllic alteration. The phyllic zone dominates at surface, is of medium size and intensity, and measures about 1.5 km in diameter. The advanced argillic alteration is preserved at high elevations and along structures. The potassic zone contains the strongest mineralization, but the zoning is complicated by multiple intrusives, breccias and faults.

The consultant concluded that within the north-west, north and north-east part of the phyllic alteration, a zone of 1,000 m x 500 m of potentially mineralized rock has not been conclusively drill-tested (Figure 9.1).

The northward projection of the phyllic envelope comprises a corridor of sheeted D-type veinlets trending 010° to 020° extending about 1,000 m towards Breccia Roja (Figure 9.1). Breccias with possible pyrophyllite and alunite cement occur at both ends. The corridor may be a structurally focused fluid flow and developed into stronger alteration and mineralization at depth.

Another corridor of sheeted veins occurs at the top of Filo Diatreme, trending roughly 60° indicating an eastward projection of the Vizcachitas system.

Figure 9.1: Surface Geochemistry and Summary of the Geological Surface Mapping



Source: Los Andes Copper, 2020

Moderately silicified daci-andesitic volcanic rocks are intruded by a 500 m x 400 m granodiorite stock at NDF. The intrusive has a variegated texture with centimetric hornblende and lesser biotite phenocrysts in a typically medium-grained subhedral groundmass. The mafic minerals are altered to shreddy secondary biotite and magnetite which have been retrograded to chlorite. Centimetric veins of coarse-grained quartz, orthoclase, tourmaline and centimetric rosettes of tourmaline are locally observed. Near the top of the intrusive, close to the contact with wall rocks, a local zone of possible sericite-pyrite alteration is seen. Both the intrusive and wall rocks are cut by later millimetric pyrite-quartz veins similar to the distal D-veins seen at Vizcachitas. The pyrite oxidation in these veinlets produces the colour anomaly at NDF.

The NDF intrusive is not a "porphyry" in the ore deposit sense. The igneous texture is not typical of ore-related porphyries, and the alteration and quartz-feldspar veins can also occur in late-magmatic (deuteric) potassic alteration in the upper portions of typical hydrous intrusions. They are not necessarily related to or indicative of porphyry copper-style potassic alteration.

However, the pyrite quartz D-veins cut the intrusive and the wall rocks and indicate a later hydrothermal event. This event could be related to the waning stages of the NDF granodiorite or the Central Vizcachitas complex. Another interpretation is that it could be related to an unseen centre at depth below the NDF colour anomaly or under the extensive talus slopes to the immediate north.

As part of the geological mapping of the Vizcachitas, 11 samples were sent for age dating. Nine of these samples were dated using U-Pb in the zircons of the intrusive rocks to date the age of the intrusion. The results are shown in Table 9.1. Two of the samples were dated using the Re-Os on molybdenite to date the age of the mineralization. The results for these two samples collected and analyzed by BHP are shown in Table 9.2. The sample locations are shown in Figure 9.2.

Table 9.1: Geochronology Samples – U-Pb in Zircons

Sample	East	North	Lithology	Date
DH-31	365504	6414918	Pyroxene Quartz Diorite	14.1 ± 0.2
DH-34	365525	6414735	Pyroxene Rich Diorite	14.3 ± 0.2
DH-37	365381	6414470	Biotite-Rich Granodiorite Porphyry	12.7 + 0.2 /- 0.1
DH-50	365869	6414153	Granodiorite Porphyry with Vuggy Breccia	12.4 ± 0.2
VCT-08 (V-39, 431 m)	365743	6413947	Biotitsed Diorite	12.5 ± 0.8
VCT-21 (V2015-08, 868 m)	366159	6413542	Strongly Deformed Biotitized Diorite	12.5 + 0.1 /- 0.2
BHP-01 (V2015-01, 397 m) (*)	365791	6413735	Quartz Diorite	12.58 ± 0.14
BHP-02 (V2015-08, 696.3 m) (*)	366159	6413542	Crowded Granodiorite Porphyry	12.08 ± 0.14
VCT-45 (V2017-04, 375.4 m)	366199	6413048	Crowded Granodiorite Porphyry	11.9 + 0.1 /- 0.2
VCT-32 (LAV-064, 134.6 m)	365973	6412729	Crowded Dacite Porphyry	11.8 ± 0.1
LAV-100 (100 m)	365465	6413438	Granodiorite Porphyry	11.9 + 0.1 /- 0.2

(*) BHP dating

Table 9.2: Molybdenite Geochronology – Re/Os

Sample	East	North	Lithology	Age
V2017-10 (846.9 m)	365679	6413882	Granodiorite Breccia	10.7 ± 0.05
V2015-04 (620.35 m)	365682	6413878	B-Type Quartz Vein in Diorite	11.24 ± 0.05

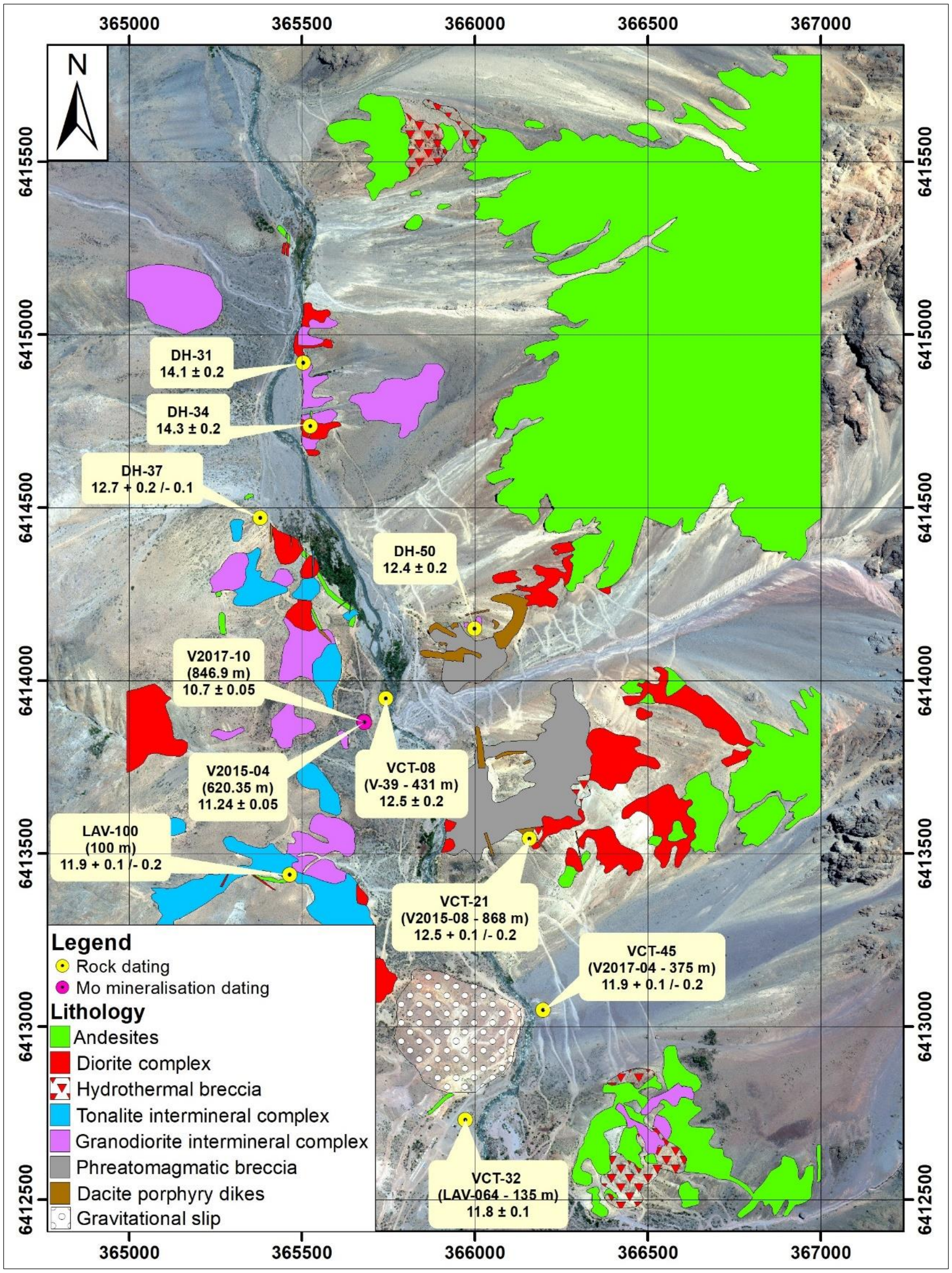
K-Ar radiometric dating on core from drill hole V-03 at 102 m from secondary biotite in porphyry andesite gave an age of 11.36 ±0.13 Ma (J.J. Cortez, personal communication, March 27, 2017).

K-Ar dating of coarse biotite included in the matrix of a polymictic breccia (sulphide breccia) biotite-anhydrite-muscovite-Cpy-Py matrix gave an age of 11.5 ±0.3 Ma (Osterman, 1997).

The conclusions from the geochronology data are that within the primary Vizcachitas system, the intrusions developed over a relatively short period of 1 million years (from 12.7 to 11.8 Ma).

The earliest intrusives are the granodiorites with ages between 12.7 Ma and 12.4 Ma and are considered to be the precursor intrusive. The primary copper mineralizing event is the diorite complex, with dates around 12.5 Ma. The inter-mineral tonalites and granodiorites have ages between 12.08 Ma and 11.8 Ma.

Figure 9.2: Geochronology Sample Locations



Source: Los Andes Copper, 2022

The ages from the molybdenite dating are later than the intrusive complex and indicate two distinct mineralizing events. The first is 11.24 Ma associated with diorite B veinlets, and the latter is

10.7 Ma associated with waning hydrothermal events as the system cooled down. These dates are consistent with the classic porphyry mineralizing model.

These dates confirm the geological model developed from the core logging and surface mapping. The Project's complex intrusive and mineralizing history can be defined more precisely using these dates.

9.4.1 2020 Geophysics

A 3D Induced Polarization (IP)/Resistivity and Magneto-Telluric survey programme was carried out in July 2020. The principal objective was to map the resistivity and induced polarization parameters over the area of interest where anomalous responses might be associated with buried disseminated sulphide mineralization.

Inversion modelling of the IP/Resistivity data reasonably represents the chargeability and resistivity parameters distribution to almost 750 m with an arbitrary volumetric sensitivity cut-off of 0.5.

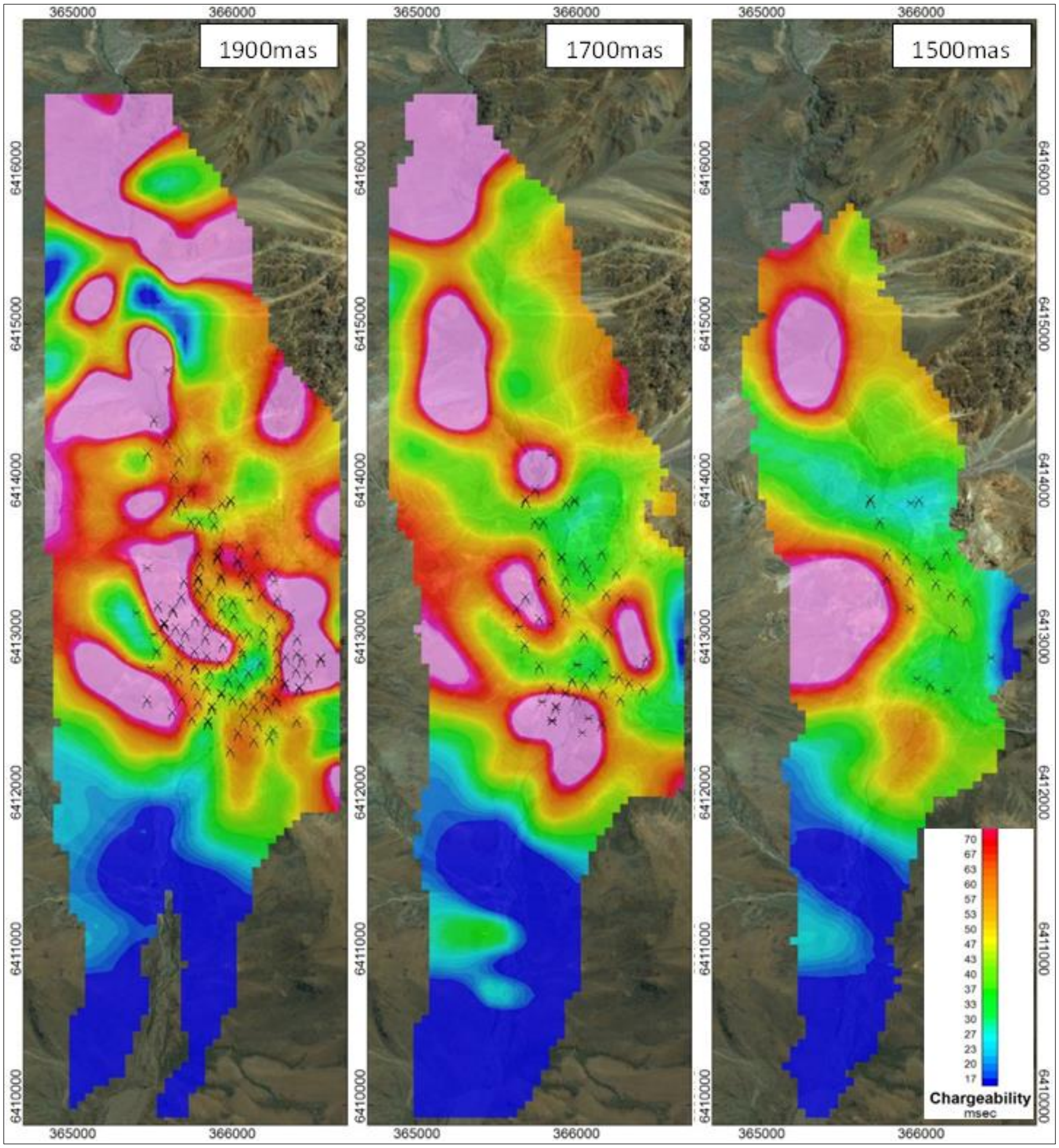
Magneto-Telluric data have provided apparent resistivity and impedance phase information appropriate for inversion modelling sub-surface resistivity to several kilometres in depth. Results are coherent with those obtained from the inversion of resistivity information from the IP/Resistivity survey.

The 3D IP/Resistivity and Magneto-Telluric surveying at the Vizcachitas survey area consisted of a total of 72 Ex- and 22 Ey-field dipoles about 200 m long along three near north-south oriented survey lines, with 33 transmitter bipoles about 600 m long spaced at 200 m intervals along central transmitter lines, for a total of 15 line-km of in-line and offset Bipole-Dipole IP/Resistivity data. Broadband Magneto-Telluric data acquisition was acquired nocturnally using the IP/Resistivity set-up with additional instrumentation acquiring the local and remote magnetic field data.

9.4.1.1 Induced Polarization / Resistivity Data and Inversion Model Results

Generally, IP responses at Vizcachitas are very high, with 80% of the final inversion model space yielding a chargeability from 30 msec to over 100 msec, predominantly located to the north of a north-west trending fault cross-cutting the valley at around 6,412,000 mN. Elevated to very high chargeability responses are associated with the abundant sulphide mineralization reported in exploration drilling, which appears to extend north beyond the current survey coverage. Figure 9.3 summarizes the 3D inversion model chargeability as elevation slices with the corresponding drill hole traces plotted over satellite imagery to reference the location of the current drilling with the geophysical signatures.

Figure 9.3: Vizcachitas, Bipole-Dipole IP / Resistivity Survey



Source: Southernrock Geophysics, 2020.

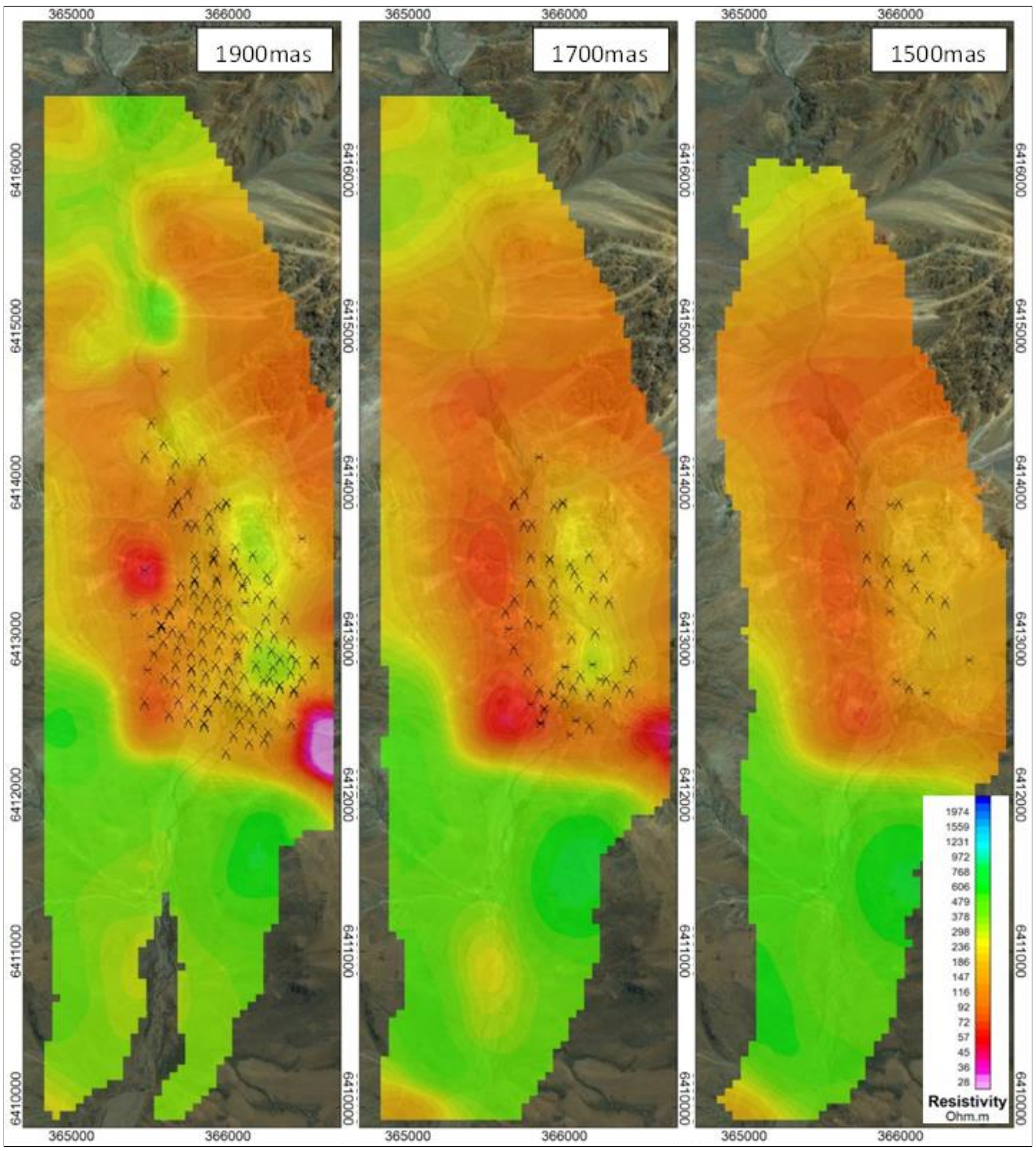
Note: 3D inversion model chargeability elevation plan maps over satellite imagery and showing the drill hole locations.

The 3D inversion model results also demonstrate a distinct change in the sub-surface resistivity north of 6,412,000 mN, with moderately low resistivity responses (<150 Ωm) coincident with the general zone of alteration at Vizcachitas, with other coherent zones of lower resistivity (<50 Ωm) at depth perhaps related to more intense alteration or sulphide mineralization. Figure 9.4

summarizes the 3D inversion model resistivity as elevation slices with the corresponding drill hole traces plotted over satellite imagery.

The main focus of drilling to date appears to fall short of a northerly trending zone of the low resistivity observed at 1,700 masl, only a few drill holes appear to intersect this feature below 1,500 m elevation.

Figure 9.4: Bipole-Dipole IP/Resistivity Survey



Source: Southernrock Geophysics, 2020

Note: 3D inversion model resistivity elevation plan maps over satellite imagery.

9.4.1.2 Magneto-Telluric Data and Inversion Model Results

Interpolation of the 1D inversion model results provides images at constant elevations that describe a similar tenor and distribution of resistivity to that provided by the IP/Resistivity data. However, the MT result provides enhanced imaging of the sub-surface resistivity, particularly at depth, where it has been used in preference to the resistivity parameter derived from the IP/Resistivity survey.

Figure 9.5 shows the north-west trending contact between the moderately resistive environment south of 6,412,000 mN and the more conductive domain to the north. The lower tenor of resistivity (<150 Ωm) is generally coincident with the Vizcachitas alteration zone. In contrast, more coherent zones of lower resistivity (<50 Ωm) may indicate more significant mineralization at depth. Resistivity appears to increase north of the current drilling area except for a low resistivity zone on the eastern side of the valley from around 6,415,000 mN to 6,416,000 mN. A higher resistivity "hole" observed at around 366,000 mE/6,413,200 mN within the general low resistivity response increases in size with depth, which may suggest a more resistive lithology or lesser content of sulphide mineralization.

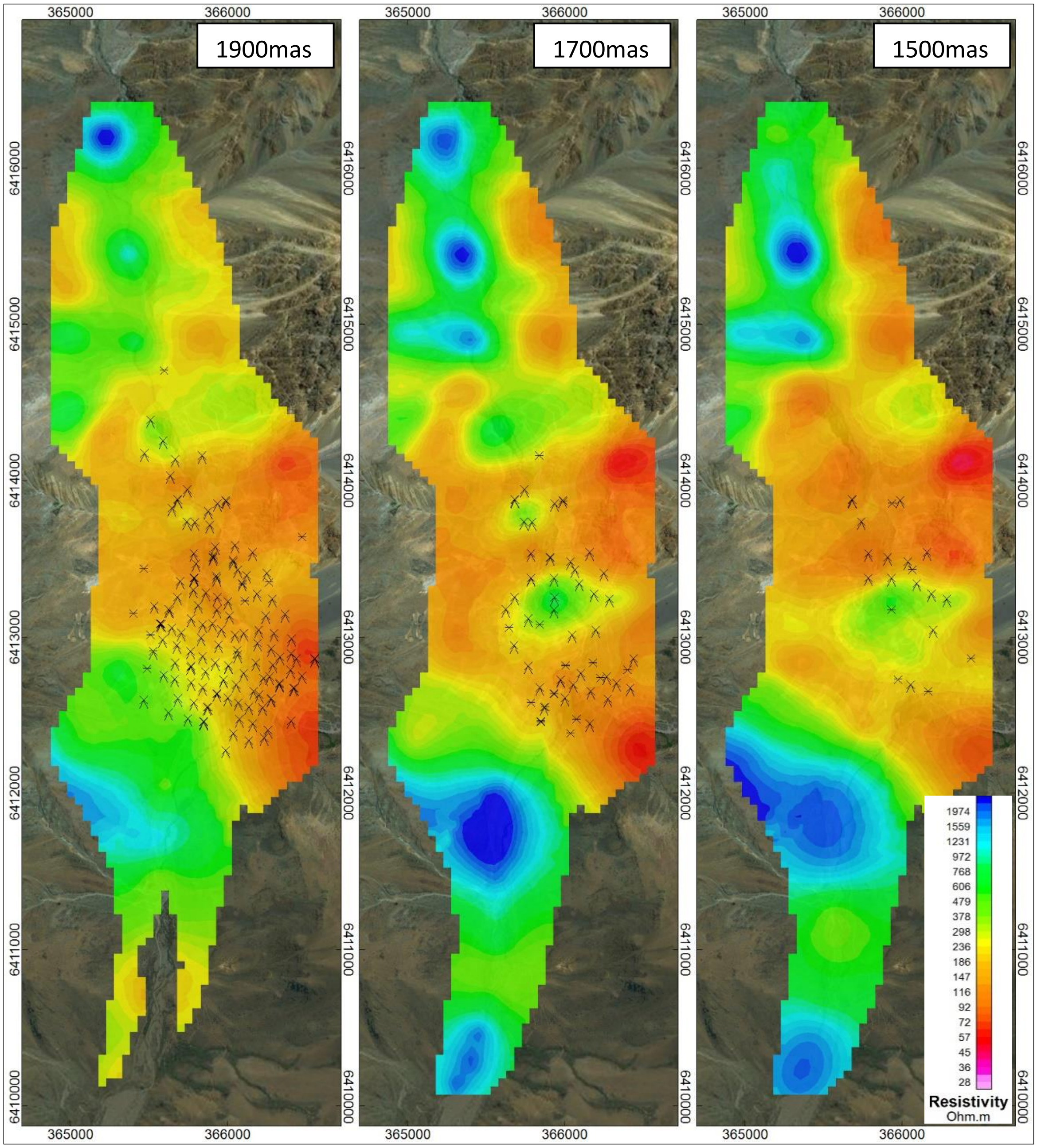
Figure 9.6 summarizes the results of the 1D inversion model for Line 1. The section shows the drill holes and copper assay intervals as coloured bands down each hole, and the elevation slice through the 1D inversion resistivity model is shown for reference.

There is a consistently positive correlation between increased copper assays in drill holes and low resistivity in the 1D inversion model of the MT data, confirmed by shallow drilling from around 6,412,400 mN to 6,413,200 mN and deeper drilling from 6,413,200 mN to 6,414,000 mN. Furthermore, drill holes into the moderately resistive "hole" at depth between these two conductive zones report lower copper assays or drill holes have been stopped as mineralization decreases towards this resistor.

The extension of the conductor at depth towards the north constitutes a significant future drill target.

Figure 9.7 shows the locations of drill holes with copper assays on a north-south vertical section with the IP/Resistivity values as the background.

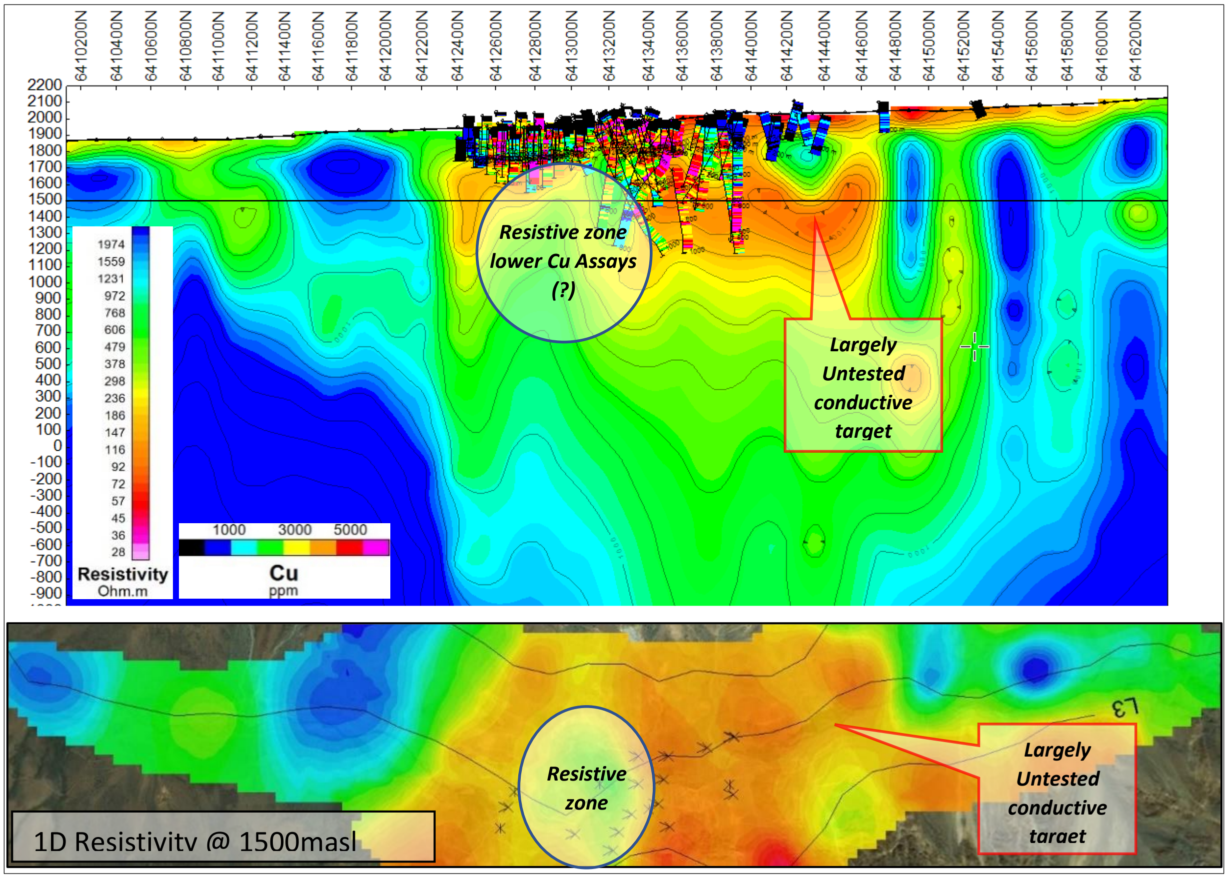
Figure 9.5: Magneto-Telluric Survey



Source: Southernrock Geophysics, 2020

Note: 1D inversion model resistivity elevation plan maps over satellite imagery. Drill holes traces are plotted only where they are within 25 m of each elevation slice.

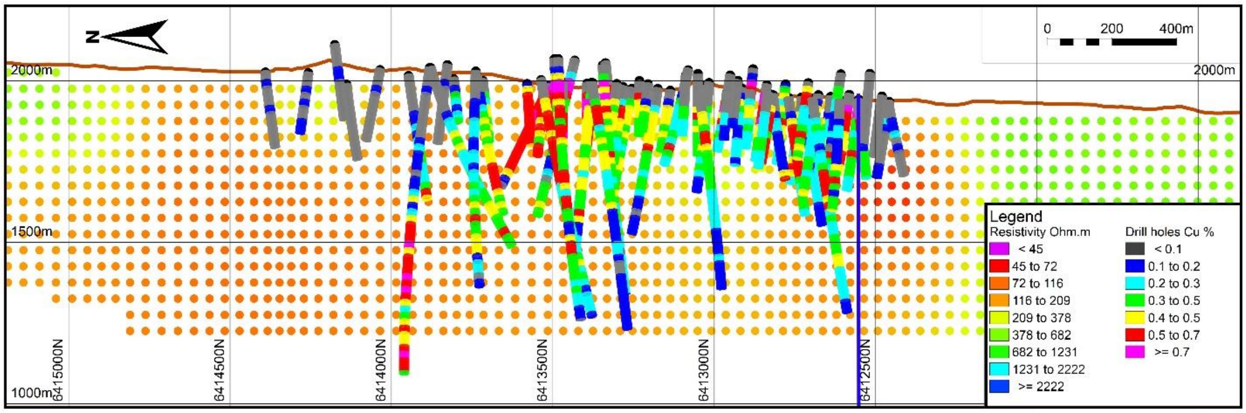
Figure 9.6: Magneto-Telluric Survey, Line 1



Source: Southernrock Geophysics, 2020

Note: 1D inversion model resistivity section with exploration drill holes plotted out to 250 m from the section with Cu-assay coloured bands, viewed from the east. The lower panel summarizes elevation slice at 1,500 masl.

Figure 9.7: North-South Vertical Section with IP Resistivity and Historical Drilling



Source: Southernrock Geophysics, 2020

9.4.1.3 Geophysics Summary

The 3D inversion model results demonstrate a distinct change in the sub-surface resistivity north of 6,412,000 mN, with moderately low resistivity responses (<150 Ωm) coincident with the general zone of alteration at Vizcachitas, with further coherent zones of lower resistivity (<50 Ωm) at depth perhaps related to more intense alteration or sulphide mineralization. The main focus of drilling appears to fall short of a northerly trending zone of low resistivity observed at 1,700 masl.

In summary, the geophysical survey shows good correlation between IP/Resistivity and copper mineralization with the lower grade diatreme and inter-mineral intrusive reflected in resistivity data. The survey shows an untested conductive zone extending 750 m north from the northernmost drill hole and conductive zones to the east of the diatreme and along the Campamento fault to the east.

10. DRILLING

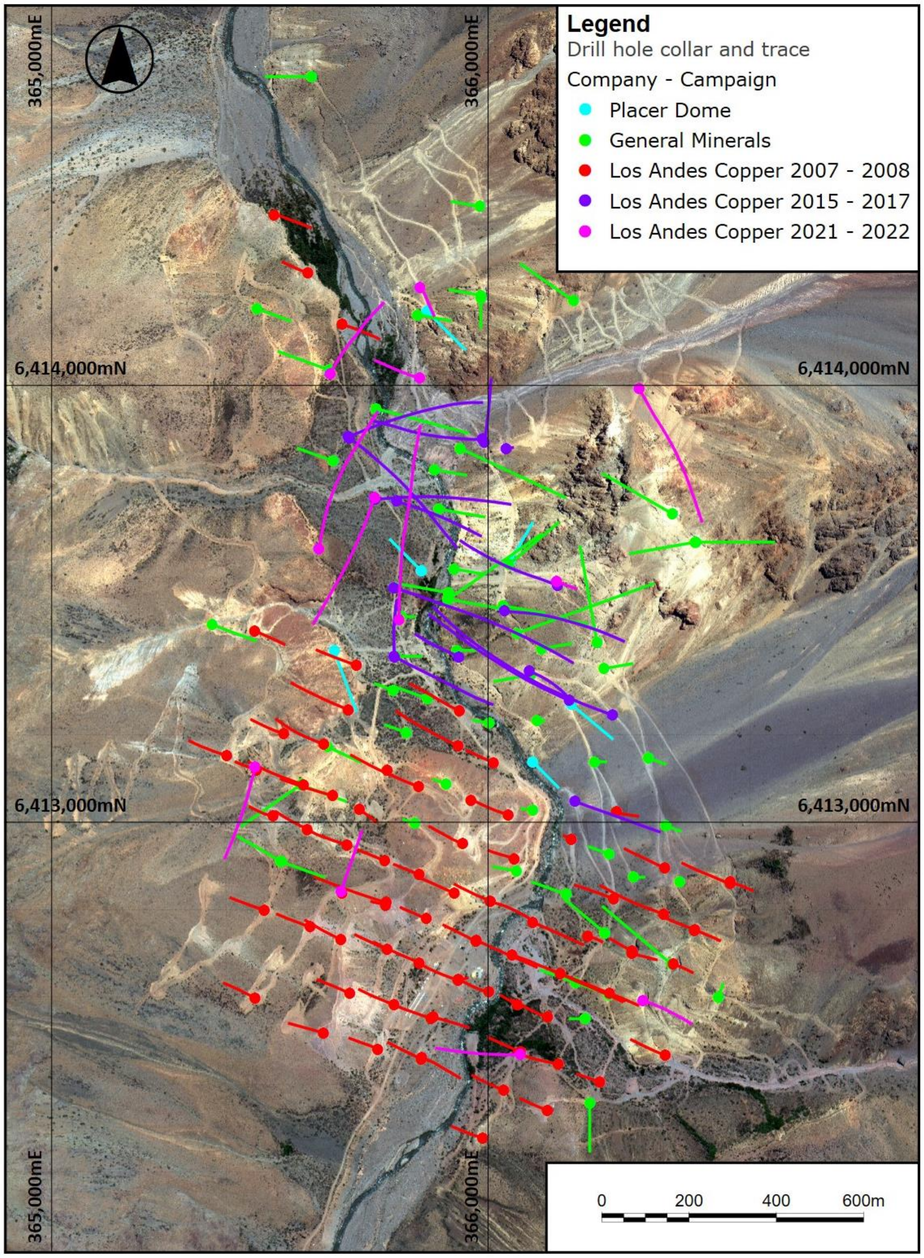
Since 1993, 182 diamond drill holes have been drilled on the Property with a total of 60,924 m. The total metres drilled by each company are summarized in Table 10.1. The locations of the drill holes are shown in Figure 10.1

Table 10.1: Summary of Vizcachitas Drill Holes

Company	Period	Drill Hole Code	N° of Drill Holes	Total Metres
Placer Dome	1993	VP-1 to VP-6	6	1,953
General Minerals	1996 - 1997	V-01 to V-63	61	15,815
Los Andes Copper	2007 - 2008	LAV-064 to Lav-142	79	22,616
Los Andes Copper	2015 - 2017	V2015-01 to V2017-11	19	11,872
Los Andes Copper	2021 - 2022	CMV-001B to CMV-018	17	8,668
Total			182	60,924

The Place Dome drill holes are not included in this PFS resource estimate because Los Andes Copper cannot identify the collar locations. Also, some drill holes from later campaigns were not used for the resource estimate because they did not reach bedrock or were drill holes that failed to reach their objective and were re-drilled.

Figure 10.1: Drill Hole Locations



Source: Los Andes Copper, 2022

10.1 Surveying

10.1.1 Collar Coordinates

During the Placer Dome, GMC and the Los Andes Copper 2007-2008 drilling campaigns, surveying used the Universal Transverse Mercator (UTM) Zone 19H and the Provisional South American Datum 1956 (PSAD56). Topographic measurements were referenced to three triangulation points installed as part of the procedures required when registering mining claims.

After drilling was completed, the collar locations were surveyed using a total station and the surveyor prepared a certificate with the coordinates.

In March 2012 Los Andes Copper carried out a topographic survey of the triangulation points to tie these points to the national survey grid that uses the newer SIRGAS/WGS84 datum maintained by the *Instituto Geográfico Militar* (IGM, Military Geographic Institute) of Chile. The Molodensk transformation parameters were derived from this survey and the historical UTM PSAD56 coordinates were converted to UTM SIRGAS/WGS84 (EGV Geomensura, 2012).

In March 2016 Los Andes Copper conducted a topographic survey of the drill hole collars on the Property; this included historical drill holes and recent drill campaign collars. A total of 115 drill collars were surveyed using the SIRGAS/WGS84 datum and professional-grade GPS survey equipment (EGV Geomensura, 2016). This survey confirmed that historical coordinates can be relied upon and that the historical surveys were accurate.

The Los Andes Copper drill collars from the 2015-2017 and 2021-2022 campaigns have been surveyed using professional-grade GPS survey equipment and the surveyor prepared reports after each campaign.

10.1.2 Downhole Survey

During the Los Andes Copper 2007-2008 and 2015-2017 drilling campaigns the drilling contractor carried out the downhole surveying using the Flexit method, which determines the azimuth by magnetic readings and the inclination of the hole. Readings were taken at 24 m or shorter intervals. During the 2021-2022 drilling campaign the initial orientation of the drilling rigs was determined using a G-Rad Gyroscope Rig Alignment Device to ensure the proper initial inclination and orientation of the drilling rig. Company employees carried out the downhole survey using an Axis Mining Technology gyroscope tool, taking measurements every 10 m.

10.2 Placer Dome

Placer Dome Sudamerica Limited conducted the first drilling on the Project in 1993. It consisted of 6 diamond drill holes located in the central-north part of the Project with variable 250 m to 500 m lengths for a total length of 1,953 m. The drill holes showed the presence of copper-molybdenum porphyry mineralization.

The best intercepts were: VP-01 with 66 m @ 0.74% Cu and 660 ppm Mo in hydrothermal breccia, VP-03 drill hole with 153 m @ 0.48% Cu and 100 ppm Mo in a granodiorite with potassic alteration and VP-04 with 30 m @ 0.92% Cu and 160 ppm Mo in a granodiorite to granodiorite porphyry brecciated with potassium alteration.

The collars for these drill holes have not been located, although Los Andes Copper has the geological logging and assay certificates. These drill holes have not been included in the PFS resource estimation.

10.3 General Minerals Corporation

From 1995 to 1998 GMC carried out a diamond drilling campaign with 61 drill holes and a total of 15,815 m drilled. The length of the drill holes ranged between 114 m and 585 m, with an average of 260 m.

GMC drilled over the whole mineralized corridor, identifying the main lithological types that compromise the Vizcachitas mineralized system. The best drilling results are shown in Table 10.2 and are mainly related to the early diorite complex, magmatic-hydrothermal breccia and andesite country rock. The associated alteration corresponds to hydrothermal alteration overlaid on the late-magmatic alteration, i.e. chlorite-sericite over potassic alteration.

The copper mineralization is chalcopyrite with chalcocite-covellite within the supergene secondary enrichment.

Table 10.2: Highlights of General Minerals Drill Campaign

Hole Number	Depth Downhole From (m)	Depth Downhole To (m)	Length (m)	Cu (%)	Mo (ppm)	Ag (g/t)	CuEq (%)
V-03	42.78	95.63	52.85	0.90	67	1.5	0.93
V-04	6.18	344.73	338.55	0.67	191	1.2	0.73
V-05	6.18	88.53	82.35	0.78	143	0.9	0.83
V-06	74.28	121.08	46.80	1.12	137	1.9	1.17
V-15	54.37	134.67	80.30	0.74	66	0.0	0.76
V-16	96.68	163.78	67.10	0.81	129	1.1	0.86
V-18	48.74	201.38	152.64	0.76	257	1.5	0.84
V-23	11.28	285.73	274.45	0.64	100	0.0	0.67
V-58	96.30	185.85	89.55	0.65	221	1.7	0.72

Note: Copper equivalent grade has been calculated using the following calculation: CuEq (%) = Cu (%) + 0.000288 x Mo (ppm) + 0.00711 x Ag (g/t). Assumptions used for the copper equivalent calculation were metal prices of US$3.68/lb. Copper, US$12.9/lb. Molybdenum, US$21.79/oz Silver, with metallurgical recoveries of 91.1% for copper, 74.8% for molybdenum and 75% for silver based on the PFS metallurgical testwork. All thicknesses from drill holes are downhole drilled thicknesses, and true widths cannot be determined from the information available.

10.4 Los Andes Copper Ltd.

10.4.1 2007-2008 Los Andes Copper Drilling

Between 2007 and 2008 Los Andes Copper carried out an exploration programme at the Project. Los Andes Copper drilled 79 diamond drill holes, ranging from 150 m to 717 m in length, for a total of 22,616 m drilled. The drill holes are in the southern part of the Project, apart from three drill holes in the north. At this time Los Andes Copper did not have 100% of the San José claim area and did not drill within this claim. Figure 10.1 shows the locations of the 2007-2008 Los Andes Copper drill holes.

The best intercepts from this campaign are shown in Table 10.3. The intercepts are within the early diorite complex, magmatic-hydrothermal breccia, andesites and, to a lesser extent, the tonalite intrusive. The alteration has a substantial hydrothermal contribution overlaid on earlier potassic alteration.

Table 10.3: Highlights of Los Andes Copper 2007-2008 Drill Campaign

Hole Number	Depth Downhole From (m)	Depth Downhole To (m)	Length (m)	Cu (%)	Mo (ppm)	Ag (g/t)	CuEq (%)
LAV-068	78.00	166.00	88.00	0.68	105	1.8	0.72
LAV-072	34.00	140.00	106.00	0.69	133	2.3	0.75
LAV-081	82.00	260.00	178.00	0.58	114	1.6	0.62
LAV-085	10.00	144.00	134.00	0.65	29	1.6	0.67
LAV-089	34.00	102.00	68.00	0.78	200	2.7	0.86
LAV-091	68.00	362.00	294.00	0.58	105	1.3	0.62
LAV-108	42.00	254.00	212.00	0.56	89	1.6	0.60
LAV-124	238.00	376.00	138.00	0.60	164	0.2	0.65
LAV-131	56.80	266.00	209.20	0.68	137	1.7	0.73

Note: Copper equivalent grade has been calculated using the following calculation: CuEq (%) = Cu (%) + 0.000288 x Mo (ppm) + 0.00711 x Ag (g/t). Assumptions used for the copper equivalent calculation were metal prices of US$3.68/lb. Copper, US$12.9/lb. Molybdenum, US$21.79/oz Silver, with metallurgical recoveries of 91.1% for copper, 74.8% for molybdenum and 75% for silver based on the PFS metallurgical testwork. All thicknesses from drill holes are downhole drilled thicknesses, and true widths cannot be determined from the information available.

10.4.2 2015-2017 Los Andes Copper Drilling

Between 2015 and 2017 Los Andes Copper conducted a drilling programme at the Project with 19 diamond drill holes for a total length of 11,872 m. The deepest drill hole reached 1,030 m, and the average depth of the drill holes was 695 m.

The 2015 to 2017 drill campaign was carried out in two phases. In 2015-2016 eight diamond drill holes totalling 3,610 m were drilled, and during 2017 eleven drill holes totalling 8,262 m were drilled.

The 2015 to 2017 drill campaigns validated the new geological model confirming the importance of the early diorite porphyry and hydrothermal breccias in controlling the higher grade mineralization of the deposit. The drilling also confirmed the near-surface higher grade supergene-enriched mineralization. In an area of 400 m by 400 m the drill holes had average supergene grades of greater than 0.5% Cu. Some examples of this near-surface supergene mineralization are:

- V2015-03 – 18 m @ 0.78% Cu from 44.1 m downhole
- V2015-08 – 92 m @ 0.71% Cu from 92 m downhole
- V2017-06 – 92 m @ 0.66% Cu from 46 m downhole.

The drilling also demonstrated that the early diorite porphyry and hydrothermal breccias extend 250 m to the north and 1,000 m below the surface. The northernmost drill holes had the following intersections:

- V2017-10 – 506 m @ 0.57% Cu from 486 m downhole
- V2017-05 – 90 m @ 0.49% Cu from 170 m downhole.

In the centre of the Project the drilling intersected good mineralization to the west and south of the diatreme. Drill hole V2015-05 drilled 52 m @ 0.81% Cu from a downhole depth of 492 m. Drill hole V2017-02 was drilled underneath V2015-05 and intersected 88 m @ 0.60% Cu from a downhole depth of 680 m, demonstrating the vertical continuity of this mineralization over a distance of 250 m. Drill hole V2015-08 intersected 502 m @ 0.63% Cu from a downhole depth of 130 m showing high-grade continuity from near surface to depth. Table 10.4 shows a selection of drill hole results from this campaign.

Table 10.4: Highlights of Los Andes Copper 2015-2017 Drilling Campaigns

Hole Number	Depth Downhole From (m)	Depth Downhole To (m)	Length (m)	Cu (%)	Mo (ppm)	Ag (g/t)	CuEq (%)
V2015-01	322.00	386.00	64.00	0.60	258	1.2	0.68
V2015-05	492.20	544.25	52.05	0.81	190	2.0	0.88
V2015-08	130.00	632.00	502.00	0.63	209	1.3	0.70
including	130.00	184.00	54.00	1.02	128	1.4	1.07
V2017-01A	226.00	374.00	148.00	0.59	172	1.5	0.65
V2017-06	64.00	504.00	440.00	0.51	164	1.1	0.56
including	76.00	132.00	56.00	0.81	72	1.6	0.84
V2017-10	486.00	992.00	506.00	0.57	357	1.1	0.68
including	684.00	772.00	88.00	0.70	278	1.4	0.79
including	924.00	984.00	60.00	0.73	341	1.5	0.84

Note: Copper equivalent grade has been calculated using the following calculation: CuEq (%) = Cu (%) + 0.000288 x Mo (ppm) + 0.00711 x Ag (g/t). Assumptions used for the copper equivalent calculation were metal prices of US$3.68/lb. Copper, US$12.9/lb. Molybdenum, US$21.79/oz Silver, with metallurgical recoveries of 91.1% for copper, 74.8% for molybdenum and 75% for silver based on the PFS metallurgical testwork. All thicknesses from drill holes are downhole drilled thicknesses, and true widths cannot be determined from the information available.

10.4.3 2021-2022 Los Andes Copper Drilling

Los Andes Copper drilled 17 drill holes with a total of 8,668 m from November 2021 through March 2022. These drill holes were designed to test the mineralization extent to the north, east and west. Highlights of the drilling campaign are shown in Table 10.5.

Table 10.5: Highlights of the 2021-2022 Drill Programme

Hole Number	Depth Downhole From (m)	Depth Downhole To (m)	Length (m)	Cu (%)	Mo (ppm)	Ag (g/t)	CuEq (%)
CMV-001B	64.00	156.00	92.00	0.57	59	1.5	0.60
	180.00	1,265.15	1,085.15	0.42	206	1.0	0.48
Average of			1,177.15	0.43	194	1.1	0.49
CMV-009	108.00	260.00	152.00	0.94	122	2.6	0.99
CMV-011	802.00	926.00	124.00	0.50	223	1.7	0.58
CMV-012B	560.00	890.70	330.70	0.55	212	1.3	0.62

Note: Copper Equivalent grade has been calculated using the following calculation: CuEq (%) = Cu (%) + 0.000288 x Mo (ppm) + 0.00711 x Ag (g/t). Assumptions used for the copper equivalent calculation were metal prices of US$3.68/lb. Copper, US$12.9/lb. Molybdenum, US$21.79/oz Silver, with metallurgical recoveries of 91.1% for copper, 74.8% for molybdenum and 75% for silver based on the PFS metallurgical testwork. All thicknesses from drill holes are down-hole drilled thicknesses, and true widths cannot be determined from the information available.

Drill hole CMV-01B was drilled northwards from the centre of the Project to test the mineralization to the west of the diatreme. From the top of the bedrock at a depth of 64 m, the drill hole intersected mineralized quartz diorites down to a depth of 1,048 m, continuing into a crowded tonalite. The whole drill hole is mineralized with 1,177 m of core with a grade of 0.49% CuEq. The hole confirmed that the mineralization previously intersected in drill holes V2017-01A, V2015-04 and V2017-05 extended to the east and deeper.

Drill hole CMV-009 drilled eastward from the southern extent of the mineralization was designed to intersect the Campamento fault. From the top of the bedrock the drill intersected good mineralization within hydrothermal breccias and andesites until the Campamento fault at 326 m. The drill hole intersected 152 m of 0.99% CuEq.

Drill hole CMV-012B was drilled southwards from the northeastern part of the Project. CMV012B was the first deep drill hole on the eastern side of the diatreme, previous drill holes V-09 and V-11 were short and only drilled within the phyllic, quartz sericite alteration. The drill hole started in a quartz diorite with few veinlets and weak potassic alteration that increased in intensity with depth. From a depth of 550 m the veinlet and alteration intensity increased along with the accompanying mineralization. From a depth of 560 m until the end of the drill hole at 891 m the drill hole intersected 0.62% CuEq. The drill hole has shown that the mineralization to the east of the diatreme is the same as the mineralization to the south and west of the diatreme.

11. SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 Historical QA/QC Programmes

11.1.1 General Minerals Corporation Drilling 1996 QA/QC Results

There is little documentation for the Quality Assurance/Quality Control (QA/QC) procedures used by General Minerals Corporation (GMC) during their drilling programmes; however, Los Andes Copper has maintained the original documentation of the drill holes, including the original logs, assay batch forms and analysis certificates. The drill hole core is well preserved and stored at the Los Andes Copper Quilicura warehouse.

The original GMC laboratory shipment sheets are handwritten with the number of each drill hole, sample number and sampling interval. The GMC drill cores are stored together with the signed test certificates. Photocopies of GMC analysis certificates are available for the drill holes. Photographs showing uncut cores are available for the drill holes V-47 to V-63.

The cores from the drilling programme were sent to ACME Analytical Laboratories (Chile) Limited (ACME) in Santiago for analysis.

During the GMC drill campaign, a QA/QC programme was implemented with duplicate samples and two different standards (Certified Reference Material, CRMs) for analysis within each batch. These samples were not introduced at regular intervals but were introduced every 20 to 40 samples. At that time no blind sample system was implemented by GMC, the laboratory entered blind samples as duplicates in the batches.

Los Andes Copper compiled and documented data for the sampling and analysis conducted by GMC, including the original test shipment forms and analysis certificates for the surface sampling and drilling. The data are now stored in the Los Andes Copper database, which includes 90% to 95% of test certificates and nearly 100% of the drilling logs.

A significant number of duplicate samples were sent to secondary laboratories (Lakefield Research Chile S.A. (Lakefield), CIMM Tecnologías y Servicios S.A. (CIMM), and ALS Geolab S.A. (ALS)) as summarized in Table 11.1. The samples sent to the three laboratories showed good correlation.

Table 11.1: GMC Secondary Laboratory Duplicates

Laboratory	No. of Samples	Trend Line
Lakefield	957	y=0.9275x
CIMM	497	y=1.0237x
ALS	656	y=0.9661x

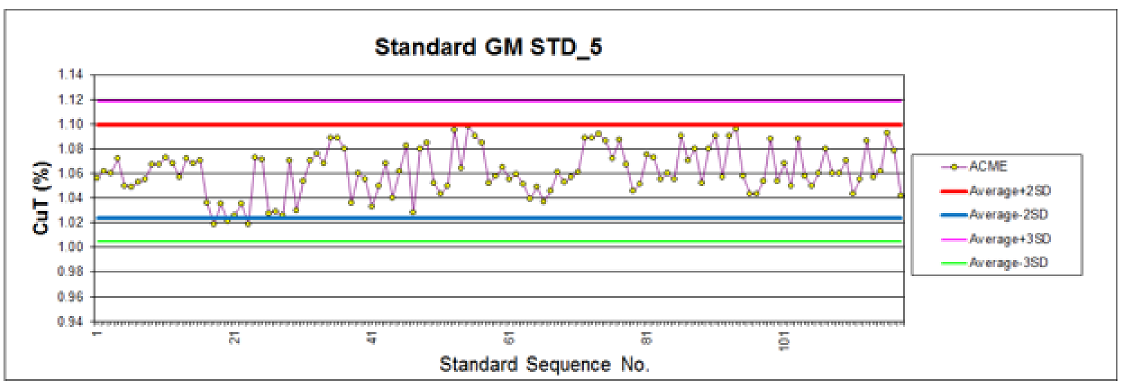
The copper assays from the drill holes V-28 to V-37 were used by MRA as part of the Kilborn study (Kilborn, 1998). According to MRA the primary laboratory was ACME and quality control assays were performed by CIMM. ACME and CIMM were commercial laboratories in Chile and were certified under ISO 9001. A total of 220 verification tests were conducted. MRA determined a slight bias in copper data at the 95% confidence level. The bias represents a difference of 0.005% Cu.

GMC used six Certified Reference Materials (CRM) without an established procedure, ACME also added reference material. The results are consistent with low variance and no change to the precision over time. The results of the ACME reference material are shown in Table 11.2. The results for STD_5 are shown in Figure 11.1.

Table 11.2: ACME Reference Material Results Copper Values

Sample Type	Element	No. of Samples	Average (%)	Standard Deviation	CV (%)	No. of Outliers	Average Less Outliers (%)
GM STD-1	Cu	149	1.066	0.05	4.7	1	1.07
GM STD-2	Cu	90	0.485	0.032	6.6	5	0.492
GM STD-3	Cu	68	0.147	0.003	2.4	3	0.147
GM STD-5	Cu	118	10.62	0.019	1.8	3	10.62

Figure 11.1: Acme Reference Material STD_5 Copper Values



Source: Los Andes Copper, 2022

Following the AMEC QA/QC review as part of the 2008 resource estimate, Los Andes Copper improved the analytical procedures and quality control protocols.

11.1.2 Los Andes Copper 2007-2008 Drilling QA/QC Results

During the 2007-2008 drilling Los Andes Copper implemented a QA/QC protocol that consists of inserting seven control samples into each batch of 50 samples (14% insertion frequency) as follows:

- Coarse Duplicates in a proportion of one per batch (2%), prepared from rejects from previously analyzed batches
- Coarse Blanks in a proportion of two per batch (4%), consisting of coarse fragments of unaltered granodiorite that assayed approximately 100 ppm Cu and 10 ppm Mo
- Samples of four Certified Reference Materials (CRM) in a proportion of four per batch (8%). Two samples are CRM Oreas 43P (for Mo), and two samples are CRM Oreas 92, Oreas 93 or Oreas 94 (for copper), the three copper CRMs were alternated.

Los Andes Copper staff inserted the control samples into the batches on site before sending them to SGS. Los Andes Copper also assigned 5% of the routine samples as verification samples that were sent to the Actlabs laboratory in La Serena. Los Andes Copper quality control programme did not include Twin Samples or Pulp Duplicates; AMEC recommended in 2008 QA/QC review that such control samples be inserted in the same batches as the originals. AMEC also recommended that Pulp Duplicates be sent to a secondary laboratory to verify the analytical precision of the primary laboratory.

During the drilling campaign conducted by Los Andes Copper between 2007 and 2008 SGS analyzed 10,092 copper samples using industry standard procedures. Copper grades ranged from the detection limit of 0.001% to 1.74%. A total of 10,088 samples were analyzed for molybdenum with grades varying from 0.0005% to 0.5%. During the 2007-2008 drilling campaign, Los Andes Copper inserted 159 Coarse Duplicates, 940 CRMs and 469 Blanks into the batches sent for analysis.

11.1.2.1 Los Andes Copper Assay Procedures

SGS was the primary analytical laboratory and assayed the samples from the 2007-2008 drill hole campaign. SGS was certified under ISO 9001-2000.

Tetra Tech did not review the SGS preparation and assaying protocols; according to the documentation provided by Los Andes Copper the samples were prepared as follows:

- Drying at 105°C for three to four hours
- Crushing to 95% passing 2.36 mm (8 mesh Tyler)
- Homogenizing and splitting samples using a rotary splitter to obtain a nominal 1,000 g sub-sample for pulverizing
- Pulverizing the nominal 1,000 g split with an LM-2 pulverizing mill to 90% passing 0.105 mm (150 mesh Tyler)
- Bagging two samples: one 250 g sample for assaying and one 750 g for back-up.

After three-acid digestion, samples were analyzed by Atomic Absorption Spectrometry (AAS) for copper and molybdenum.

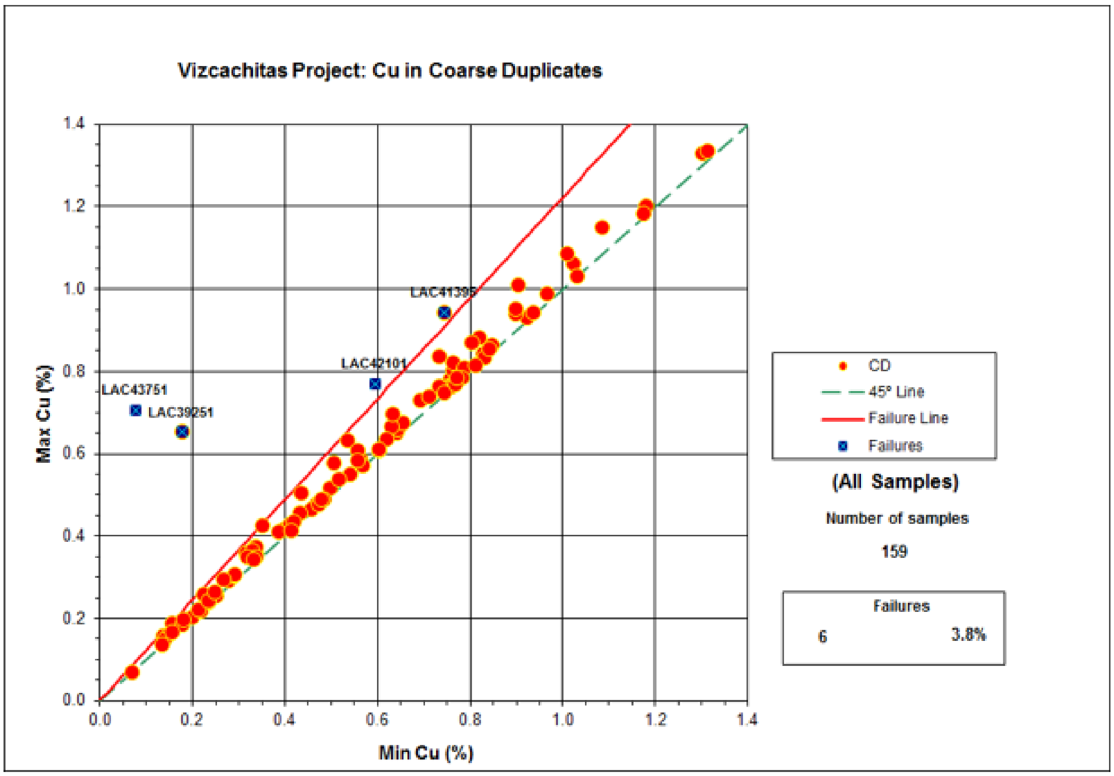
11.1.2.2 Duplicates

During the 2007-2008 drilling campaign 159 Coarse Duplicates were added to the assay sample batches. The copper failure rate was 4%, with six failures. An acceptable level of precision is achieved if the failure rate does not exceed 10% of all pairs.

One coarse duplicate was inserted into each batch (2%). Coarse Duplicates were prepared from the coarse reject sample from a previous sample batch and inserted into the batch. During the QA/QC review for the 2008 Technical Report, AMEC found that the sub-sampling variance was within acceptable limits. However, AMEC suspected some of the failing duplicates might have been mixed up and recommended that these samples should be re-examined. Subsequently, a detailed review and validation of the assay database were completed by Los Andes Copper.

Figure 11.2 presents the error variance of the duplicate samples that Los Andes Copper submitted during the 2007-2008 drill programme. Statistical analysis shows that the results are acceptable considering the average, median, standard deviation and variance.

Figure 11.2: Coarse Duplicates Analysis



Source: Los Andes Copper, 2022

11.1.2.3 Certified Reference Materials (CRM)

Los Andes Copper established a programme to insert systematically four CRMs. Table 11.3 shows the certified grades and results for the CRMs used.

Table 11.3: CRM Sample Summary

Sample Type	Element	No. of Samples	Best Value (%)	Confidence Interval (%)	Average (%)	Bias (%)
Oreas 92	Cu	159	0.229	0.0038	0.2415	5.46
Oreas 93	Cu	153	0.582	0.0072	0.5911	1.56
Oreas 94	Cu	160	1.140	0.0299	1.1527	1.11
Oreas 43P	Mo	468	0.013	0.0005	0.0106	-18.46

Table 11.3 shows that the bias detected was low for Oreas 93, 94 and 95 (copper standards). However, the molybdenum standard (Oreas 43P) shows a high negative bias (-18.46%). The analytical results indicate that the copper values for the standards (CRMs) are slightly higher than those recommended but are within an acceptable range. However, the molybdenum values reported by the laboratory indicate a potential under-estimation of the molybdenum values in the deposit at low grades.

11.1.2.4 Blank Samples

Los Andes Copper inserted quartz rock blank samples into each batch. The results gave an acceptable confidence level in the assay analysis carried out by SGS.

11.1.2.5 Samples to Check Laboratory

Los Andes Copper sent 440 coarse reject check samples to Actlabs for independent validation. The results indicated an acceptable correlation between SGS and Actlabs with error variations no greater than ±20%.

11.1.3 Los Andes Copper 2015-2017 QA/QC Results

11.1.3.1 Core Cutting and Sample Preparation

The following core sampling procedure was implemented for the 2015-2017 drill programmes:

- The drilling contractor laid out the drilled core into core boxes provided by Los Andes Copper. The drillers then inserted wooden tags into the core boxes indicating the end of each run and the depth.
- Los Andes Copper personnel transported the core from the drilling rig in securely covered boxes to the Project camp site.
- The core lengths were checked, geotechnical logging was carried out, including core recovery and RQD.
- The core was photographed.

- The geologist logged the core for lithology, alteration, veining, mineralization and structure.
- The geotechnician marked the 2 m sample intervals.
- The geologist marked the centre line along which the core was to be cut to ensure that the core was cut diagonal to the veining as far as possible.
- Geotechnical and geological data were entered into an Excel spreadsheet.
- The core was cut in half using a diamond saw. One half was sent to the laboratory for analysis and the other half was stored in the core box. When the core was highly fractured and could not be cut, it was separated by hand.
- The samples were stored securely at the Project site until shipment.
- The samples were shipped to the laboratory in a truck accompanied by Los Andes Copper personnel. The sample bags were counted at the laboratory to check for any discrepancies between the samples sent and received.
- Once the samples were assayed, the coarse rejects and the pulp samples were shipped to the Project site for storage.

11.1.3.2 QA/QC Programme

For the 2015-2017 drilling campaign Los Andes Copper implemented a protocol of inserting approximately one control sample for every six routine samples, giving an insertion frequency of 16%. The following control samples were inserted:

- Twin Samples at a proportion of approximately one in every 50 samples. A Twin Sample is where both halves of the cut core are analyzed.
- Coarse Blanks in a proportion of approximately one in every 40 samples. The Coarse Blank was prepared from 2” cubes of quartz provided by a certified assay laboratory.
- Coarse Duplicates at a proportion of approximately one in every 60 samples prepared from the coarse reject from the previous sample.
- Pulp Duplicates at a proportion of approximately one in every 60 samples were prepared from the pulverized sample from the previous sample.
- Pulp Blanks in a proportion of approximately one in every 50 samples. These were prepared from pulverized quartz samples provided by a certified assay laboratory.
- CRMs in a proportion of approximately one in every 15 samples. The CRMs were purchased from ORE Research & Exploration Pty. Ltd., Victoria, Australia.

Los Andes Copper personnel inserted Twin Samples and Coarse Blanks in the sequence of samples that were then sent to the ALS sample preparation laboratory in Santiago, Chile. The samples used for the Coarse Blanks and Coarse Duplicates were identified in each batch. Once ALS had prepared all the samples, including the Coarse Blanks, Coarse Duplicates and Pulp Duplicates, Los Andes Copper personnel inserted the CRMs and the Pulp Blanks into the sequence of pulp samples. The sample numbers were then changed so that the laboratory would not know which samples were QA/QC samples. This work was carried out at the ALS laboratory in Santiago. ALS then sent the re-numbered samples to their laboratory in Lima, Peru for analysis.

ALS Chile and Peru operated according to ISO 9001:2008 quality management systems.

For the re-analysis of historical pulp samples, the insertion of check samples was as follows:

- Pulp Duplicates at a proportion of approximately one in every 60 samples. These were prepared from the pulverized sample from the previous sample.
- Pulp Blanks at a proportion of approximately one in every 50 samples. These were prepared from pulverized quartz sample provided by a certified assay laboratory.
- CRMs at a proportion of approximately one in every 15 samples.

The type and number of control samples used in the QA/QC programme are summarized in Table 11.4.

Table 11.4: Summary of QA/QC Samples

<table>
<tr><th rowspan="2">Sample Type</th><th rowspan="2">QC Type</th><th colspan="2">2015-2017 Drill Program</th><th colspan="2">Re-Analysis of Pulp Samples</th></tr>
<tr><th>No. of Samples</th><th>Ratio of Routine vs Check Samples</th><th>No. of Samples</th><th>Ratio of Routine vs Check Samples</th></tr>
<tr><td>Routine Samples</td><td>Routine</td><td>5,270</td><td></td><td>16,225</td><td></td></tr>
<tr><td rowspan="5">Check Samples</td><td>Coarse Blank</td><td>126</td><td>41</td><td></td><td></td></tr>
<tr><td>Pulp Blank</td><td>100</td><td>52</td><td>470</td><td>34</td></tr>
<tr><td>Coarse Duplicate</td><td>79</td><td>66</td><td></td><td></td></tr>
<tr><td>Pulp Duplicate</td><td>79</td><td>66</td><td>480</td><td>33</td></tr>
<tr><td>Twin Sample</td><td>116</td><td>45</td><td></td><td></td></tr>
<tr><td rowspan="9">Standards</td><td>Oreas_151b</td><td>74</td><td rowspan="9">16</td><td>67</td><td rowspan="9">34</td></tr>
<tr><td>Oreas_152b</td><td>31</td><td>74</td></tr>
<tr><td>Oreas_153b</td><td>19</td><td>69</td></tr>
<tr><td>Oreas_501b</td><td>76</td><td></td></tr>
<tr><td>Oreas_501c</td><td>14</td><td>71</td></tr>
<tr><td>Oreas_502b</td><td>44</td><td>70</td></tr>
<tr><td>Oreas_503b</td><td>60</td><td>70</td></tr>
<tr><td>Oreas_504b</td><td></td><td>49</td></tr>
<tr><td>Total Standards</td><td>318</td><td>470</td></tr>
</table>

11.1.3.3 Mineral Grade Analysis

The samples were analyzed using the ALS geochemical procedure ME-MS61 with ultra-trace level Inductively Coupled Plasma-Mass Spectrometry (ICP-MS) analysis. Copper assays with values greater than 6,000 ppm Cu were re-analyzed using the ALS geochemical procedure ME-OG62, which uses 4-acid near-total digestion and ICP finish. The cut-off was selected from the precision of the analysis versus the copper grade after a discussion between Los Andes Copper and ALS.

The results from the ME-OG62 analysis were the assay results used in the resource estimation.

11.1.3.4 Certified Reference Material

CRMs were inserted into the sample sequence. The CRMs were purchased from ORE Research & Exploration Pty. Ltd. of Australia.

The following CRMs were used during the 2015-2017 drilling campaign and for re-analyzing the historical pulp samples (Table 11.5). The eight standards cover the grade spectrum for the copper, molybdenum and silver mineralization found at the Vizcachitas Project.

Table 11.5: CRM Used During the 2015-2017 Drill Programme

	Cu (%)			Mo (ppm)			Ag (g/t)		
CRM Name	3 SD High	3 SD Low	Certified Value	3 SD High	3 SD Low	Certified Value	3 SD High	3 SD Low	Certified Value
Oreas 151b	0.20	0.17	0.18	62.00	48.00	55.00	0.80	0.30	0.60
Oreas 152b	0.40	0.35	0.38	91.00	71.00	81.00	1.10	0.60	0.90
Oreas 153b	0.72	0.63	0.68	195.00	132.00	163.00	1.70	1.20	1.50
Oreas 501b	0.29	0.23	0.26	122.00	76.00	99.00	1.20	0.40	0.80
Oreas 501c	0.31	0.24	0.28	107.00	89.00	98.00	0.60	0.20	0.40
Oreas 502b	0.83	0.71	0.77	272.00	203.00	238.00	2.60	1.60	2.10
Oreas 503b	0.60	0.46	0.53	368.00	270.00	319.00	2.10	1.00	1.50
Oreas 504b	1.23	0.98	1.11	567.00	430.00	499.00	3.70	2.40	3.10

A total of 788 CRMs were inserted into the sample sequence; 18 samples (2.3% of the CRMs) had assay results outside three standard deviations for either copper, molybdenum or silver. Twelve assays outside the three standard deviations were for the standards Oreas 501c and Oreas 151b for molybdenum. These are low grade molybdenum CRMs with narrow standard deviations (the three standard deviation limit for Oreas 151b is 62 ppm and the Certified Value is 55 ppm).

The results were reviewed in detail for each batch where there were assay results outside the three standard deviation limit. The ALS CRM results were downloaded from the ALS internet site, and the results of the batches were discussed with ALS. After reviewing the information it was decided that it was unnecessary to re-analyze these batches.

11.1.3.5 Second Laboratory

A total of 271 pulp samples and the related QA/QC samples were sent to a second laboratory for analysis, representing 5% of the routine samples. The laboratory used was Andes Analytical Assay (AAA) of Santiago, Chile, an ISO 9001:2008 IRAM and IQNET accredited laboratory.

The copper and molybdenum assays showed good correlation, but the silver results for the re-assayed samples showed a slight negative bias. Table 11.6, Table 11.7 and Table 11.8 summarize the results from AAA.

The QA/QC for the batch showed that of the 21 CRMs inserted into the batch, only one CRM had a value greater than the three standard deviation limit for molybdenum. For silver, out of the 21 CRMs inserted, Oreas 501b had three assays with low values outside the three Standard Deviation limit. The second laboratory used Atomic Absorption with a higher detection limit of

0.2 g/t Ag. The original samples were analyzed using the ALS geochemical procedure ME-MS61 using ultra-trace level ICP-MS analysis, with a detection limit for silver of 0.01 g/t Ag. The ALS ME-MS61 procedure was used because of the low detection limit for silver, which should mean that the precision at the level of 0.5 g/t to 2.0 g/t would be better than that for the Atomic Absorption analysis.

Table 11.6: Summary of Second Laboratory CRM Copper Values

RM	N	Cu (wt.%)		Observed Cu (wt.%)		Percent of
		Accepted	Std. Dev.	Average	Std. Dev.	Accepted
Oreas 503b	4	0.531	0.023	0.529	0.009	99.5%
Oreas 502b	4	0.773	0.020	0.767	0.007	99.2%
Oreas 501b	4	0.260	0.011	0.283	0.007	108.8%
Oreas 153b	3	0.678	0.015	0.690	0.012	101.8%
Oreas 152b	3	0.375	0.008	0.387	0.003	103.2%
Oreas 151b	3	0.182	0.005	0.186	0.001	102.4%
Total	21				Weighted Average	102.5%

Table 11.7: Summary of Second Laboratory CRM Molybdenum Values

RM	N	Mo (ppm)		Observed Mo (ppm)		Percent of
		Accepted	Std. Dev.	Average	Std. Dev.	Accepted
Oreas 503b	4	319.000	16.000	324.000	10.614	101.6%
Oreas 502b	4	238.000	11.000	244.750	2.872	102.8%
Oreas 501b	4	99.000	7.500	105.500	4.435	106.6%
Oreas 153b	3	163.000	10.000	164.667	6.658	101.0%
Oreas 152b	3	81.000	3.400	84.333	1.528	104.1%
Oreas 151b	3	55.000	2.200	62.667	5.508	113.9%
Total	21				Weighted Average	104.8%

Table 11.8: Summary of Second Laboratory CRM Silver Values

RM	N	Ag (g/t)		Observed Ag (g/t)		Percent of
		Accepted	Std. Dev.	Average	Std. Dev.	Accepted
Oreas 503b	4	1.540	0.190	1.400	0.216	90.9%
Oreas 502b	4	2.090	0.170	1.875	0.222	89.7%
Oreas 501b	4	0.778	0.128	0.300	0.141	38.6%
Oreas 153b	3	1.450	0.090	1.500	0.100	103.4%
Oreas 152b	3	0.861	0.096	0.733	0.058	85.2%
Oreas 151b	3	0.551	0.068	0.467	0.058	84.7%
Total	21				Weighted Average	80.8%

11.1.3.6 Specific Gravity Sampling and Determination

For drilling campaigns before 2015 Los Andes Copper took specific gravity measurements on the drill core using the water displacement method. Trained Los Andes Copper personnel conducted specific gravity determinations every 40 m downhole or more frequently if significant lithological changes occurred within the interval. The specific gravity samples ranged between 5 cm and 15 cm in length.

The determination procedure consisted of drying the sample, covering it with paraffin and weighing it in the air and under water. The method measures the volume of water displaced by the sample.

The specific gravity for the core samples from campaigns in 2015-2017 was calculated using the geometric density method using the following formula:

$$Density = m/v$$

Where,

m = weight of the core segment to be analyzed, and

v = volume of the core segment, calculated by $v = \pi r^2 \times length$

11.2 Los Andes Copper 2021-2022 QA/QC Results

11.2.1 Mechanical Preparation

The mechanical preparation of the first 1,900 drilling samples (47 batches) was carried out by Geoassay in Pudahuel (Santiago). ALS Patagonia carried out mechanical preparation of the remaining 2,079 samples (54 batches) in Colina (Santiago).

11.2.1.1 Geoassay Mechanical Preparation

The standard preparation procedure for diamond core consists of passing the half core pieces through a primary crusher, reducing them to <1/2", and then continuing by crushing the entire sample (6 kg to 8 kg) to 95% <10# ASTM (2 mm). Using a Jones Riffle Splitter, a fraction of approximately 1,000 g was separated after homogenization utilizing the same equipment. This sample was pulverized to 95% <150# Tyler (105 µm). From the resulting pulp, two envelopes of 80 g were separated and sent to the laboratory for analysis. The 840 g of excess pulp and the 7 kg of coarse reject (<2 mm) were packaged and returned to the Project for storage.

11.2.1.2 ALS Mechanical Preparation

The mechanical preparation of the diamond core was performed according to the PREP-31B ALS procedure. This procedure involves crushing the half core to <1/2" and then continuing by crushing of the entire sample (6 kg to 8 kg) to 70% <10# ASTM (2 mm). A Jones Riffle Splitter separated a fraction of approximately 1,000 g after homogenization. The sample was pulverized to 85% <200# Tyler (75 µm). The resulting pulp was separated into two envelopes of 125 g, one of which was sent to the laboratory for analysis and the other returned for storage. The 750 g of excess pulp and the 7 kg of coarse reject (<2 mm) were packaged and delivered to the warehouse for storage.

11.2.2 Chemical Analysis

Geoassay analyzed the first batches at its facilities in Pudahuel (Santiago). ALS re-analyzed all the samples for copper and molybdenum by atomic absorption (methods Cu-AA62 and Mo-AA62) in its laboratory in Lima (Peru). The samples were also analyzed using ICP-MS for 48 elements (MS-ME61 method). All samples with copper content above 7,000 ppm were further analyzed using the ore grade ICP method ME-OG62 (Cu).

All samples were dissolved using four acids (hydrochloric, nitric, perchloric and hydrofluoric), using the following masses 0.25 g (MS-ME61 and ME-OG62) and 0.4 g (Cu-AA62 and Mo-AA62).

11.2.3 Quality Assurance and Quality Control

The Quality Assurance and Quality Control (QA/QC) protocols implemented during the 2021-2022 drilling programme were designed to ensure that the sampling, mechanical preparation and analysis of the samples obtained during drilling and re-analysis of the historical pulps were carried out with an acceptable degree of quality.

Accuracy, precision and contamination were evaluated by incorporating control materials such as CRMs, Twin Samples, Coarse Duplicates, Pulp Duplicates and Blanks. These control samples were added to the sample stream in a ratio of 1 in 21 i.e. 5% CRMs, 5% Pulp Duplicates, 2.5% Twin Samples (1/2 control), 2.5% Coarse Duplicates and 5% blanks (2.5% Pulp and 2.5% Coarse).

The samples for mechanical preparation had Twin Samples and Coarse Blanks added to the sample stream. Sample bags were identified with the drill hole number and the from-to length. The Coarse Duplicates (identified in the same way) were added during the sample preparation.

Once the samples had been prepared, the sample envelopes containing the pulps were re-labelled with the final sample number. At this stage, the remaining control samples (CRMs, Pulp Blanks and Pulp Duplicates) were added to the sample sequence.

The CRMs were selected according to the estimated grade of the surrounding drill cores. A CRM with a grade as close as possible to the estimated grade of adjacent drilling samples was used.

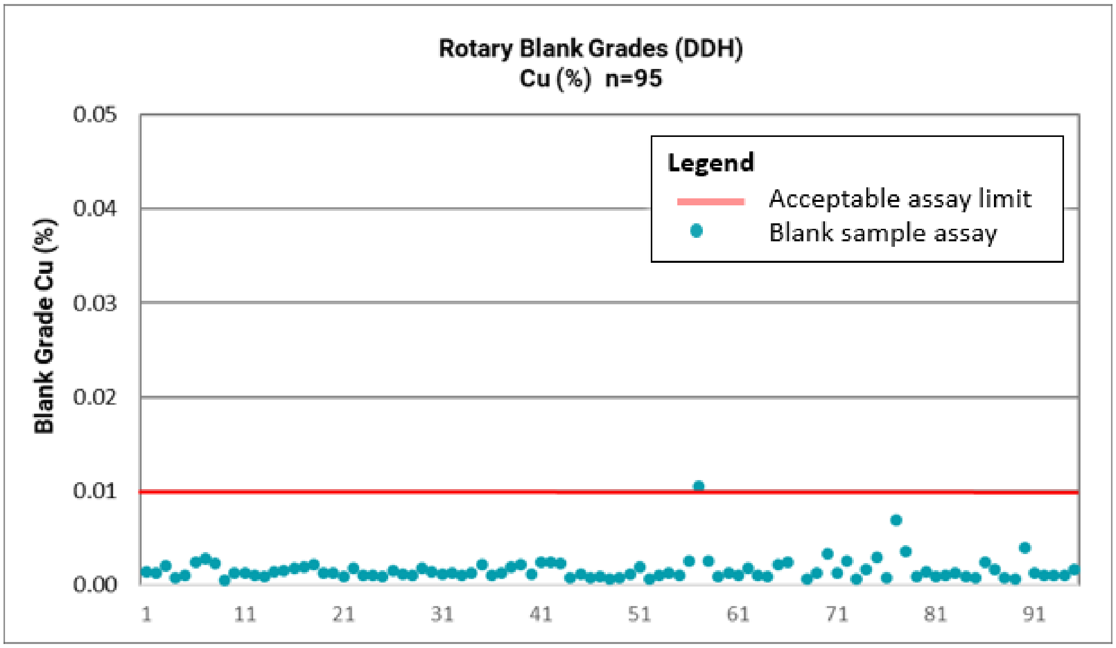
Once these procedures were completed, the batch was sent to the laboratory for analysis.

11.2.4 Blanks

11.2.4.1 Coarse Blanks

These were inserted at a rate of 1 in 42 (2.4%). The blank material was sterile quartz fragments (1/4” to 1/2”) purchased from a certified supplier. Figure 11.3 shows the results for the copper values for the 95 CRMs added as control samples and analyzed. None reported values higher than the accepted limit of 100 ppm Cu. Only one sample had a grade close to 100 ppm Cu, indicating no significant contamination events during the analytical process.

Figure 11.3: Coarse Blanks - Copper Values



Source: Los Andes Copper, 2022

11.2.4.2 Pulp Blanks

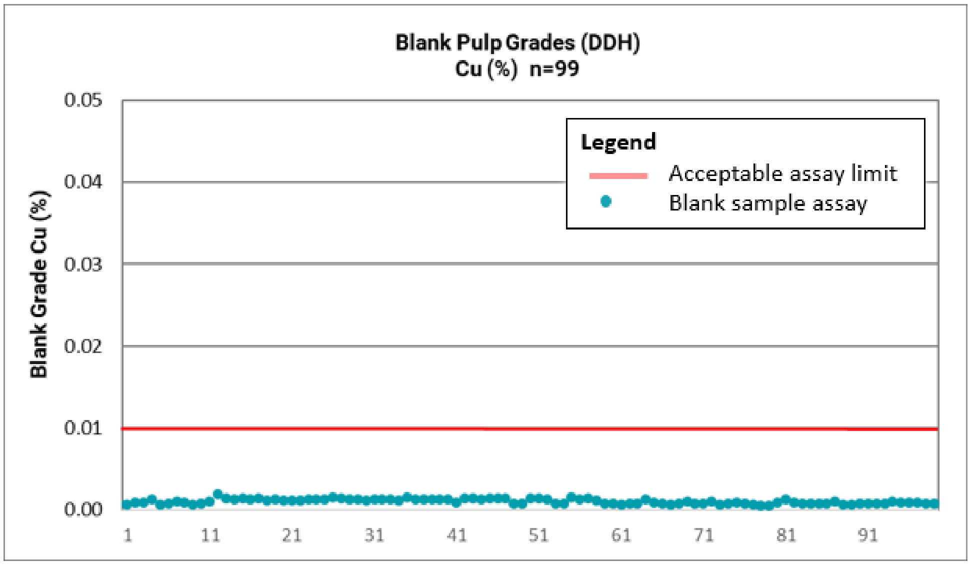
These were inserted at a rate of 1 in 40 (2.5%). The material used was silica pulverized to <150# (Tyler). Analysis of this material showed total copper levels less than or equal to 10 ppm (Figure 11.4).

The purpose of Pulp Blanks is to monitor the level of contamination produced during analytical processes. Eventually, this type of material may also indicate the existence of disorder or errors in the sample sequence.

Ninety-nine Blank Pulp samples were added as control materials in batches sent to the laboratory. None of the samples analyzed reported a value higher than the accepted limit (100 ppm Cu), indicating no significant contamination events during the analytical process.

The blank result for molybdenum and silver are contained in the QA/QC Technical Note (Martínez, 2022).

Figure 11.4: Pulp Blanks - Copper Values



Source: Los Andes Copper, 2022

11.2.5 Certified Reference Materials

The purpose of CRMs (standards) is to determine the degree of accuracy achieved by the analytical laboratory. During the execution of the 2021-2022 drilling campaign, five CRMs manufactured by ORE Research & Exploration Pty. Ltd. were used. The standards were OREAS 501d, OREAS 502c, OREAS 503d, OREAS 505, OREAS 507 and OREAS 504b, certified for around 57 elements, plus gold using fire assay.

These CRMs covered copper grades between 0.27% and 1.11% and were added with a frequency of 5%, i.e. one CRM for every 21 drill samples, following the blind insertion scheme described above.

The analysis values obtained for each CRM are presented in the following sections and control charts. These show the evolution of the reported grade as a function of time, standardized to the Z value to facilitate comparison between the CRMs.

11.2.5.1 Copper (CRM)

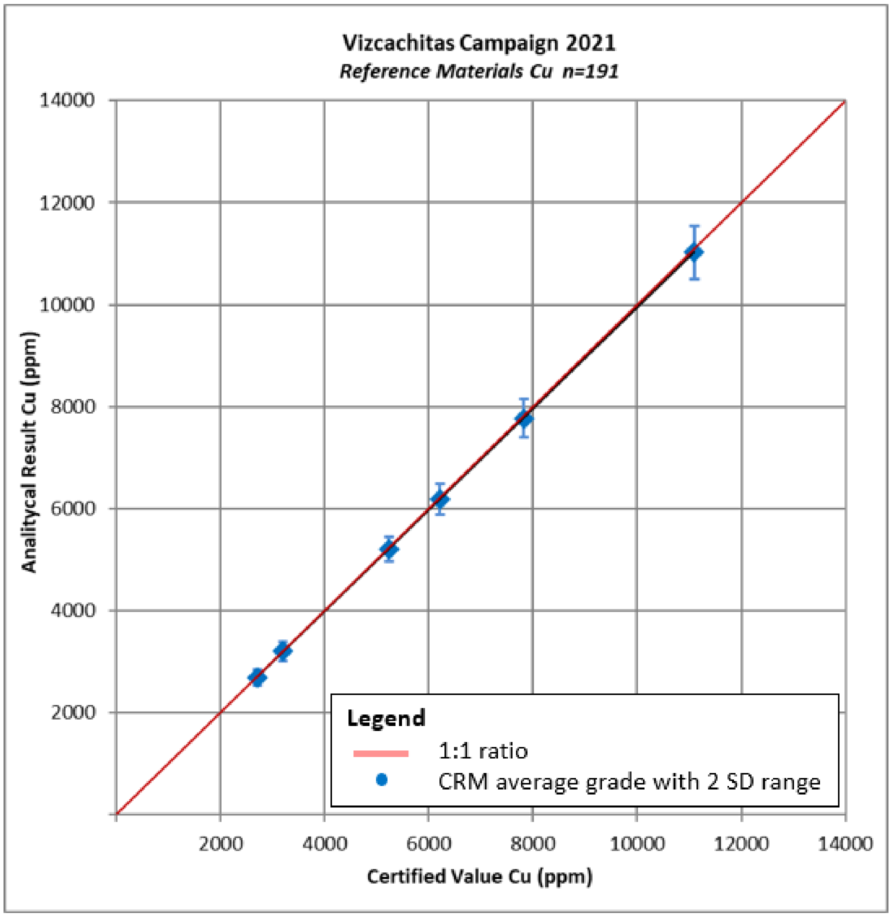
Figure 11.5 shows the result of plotting the expected value of each CRM used (X-axis) versus the average of the analytical results obtained for each reference material (Y-axis). The results are presented with a blue rhombus (mean) and two light blue bars (± one standard deviation). The six CRMs used were:

- OREAS 501d (2,720 ppm Cu)
- OREAS 502c (7,830 ppm Cu)
- OREAS 503d (5,240 ppm Cu)

- OREAS 505 (3,210 ppm Cu)
- OREAS 507 (6,220 ppm Cu)
- OREAS 504b (11,100 ppm Cu).

Figure 11.5 also shows that the average of the analytical results for each CRM falls very close to the central line; this means that the analytical accuracy achieved by the laboratory for copper was excellent.

Figure 11.5: Certified Reference Material - Copper Values

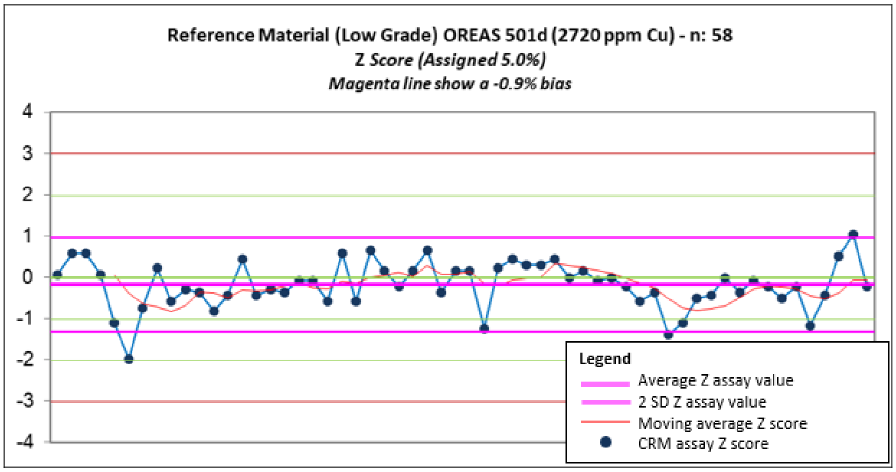


Source: Los Andes Copper, 2022

It is also possible to estimate the accuracy achieved from the length of the standard deviation of the analytical results for each CRM. The shorter the observed length, the better the accuracy. In this case, excellent analytical accuracy was achieved for copper.

CRM OREAS 501d has an expected grade of 2,720 ppm Cu, with a tolerance value of 5.0% for the analytical method. The results for this CRM are shown in Figure 11.6. Nearly all the 58 copper values reported for the OREAS 501d standard are within the range of Z -1 to 1, with an average of 2695 ppm Cu, practically equal to the expected value (2,720 ppm Cu), which implies a bias of -0.9% and a minimum under-estimation of the grades.

Figure 11.6: Certified Reference Material - OREAS 501d Z Score



Source: Los Andes Copper, 2022

The CRM results for the other copper CRMs and molybdenum, silver and arsenic are contained in QA/QC Technical Note (Martínez, 2022). The results reported for the CRMs show values consistent with the certified grades for these control samples. The biases observed are within the tolerance ranges and are low for all analyses. Only the arsenic shows a slight over-estimation that was accentuated towards the higher grades, although the bias remained within tolerance.

11.2.6 Pulp Duplicates

The purpose of Pulp Duplicates is to determine the degree of precision achieved during chemical analysis: the level of repeatability reached when re-analyzing the same sample under the same instrumental and operator conditions. Pulp Duplicates were added at 1 in 21 samples (5%).

The process of weighing and digesting the samples and the instrument reading incorporate errors that can be manifested in the fact that the results obtained for the same sample in different analyses are not always the same. The Relative Error is the quantitative measure used to evaluate the precision. For each pair of samples, precision is defined as the absolute value of the difference of the analysis result, divided by the half-sum of both the grades (original and duplicate), i.e.:

ER (Relative Error) = ABS(Value 1-Value 2)/((Value 1+Value 2)/2)

For Pulp Duplicates the acceptance criterion used by the industry is that 90% of the pairs have a Relative Error less than or equal to 10%.

Special treatment is required for samples with grades less than 15 times the detection limit. In these cases, minor variations in the grade generate high Relative Errors and it is necessary to define a different criterion for processing these results.

Among the alternatives found in the literature, it was determined that the most appropriate was Hyperbolic Method (Simon 2004), in which a Quadratic Hyperbolic equation defines the acceptance-rejection curve:

$$Y^2 = m^2 \cdot x^2 + b^2$$

The main advantage of this method is that the curve behaves linearly for the mid- to high-grade range; however, it is much more tolerant for the range between 1 to 15 times the detection limit.

Therefore, the Hyperbolic Method was used for normalizing the error for low-grade samples. The Y value calculated using the equation was considered the value equivalent to the tolerance, i.e. 10% for the Pulp Duplicates, to standardize the maximum value of the pair with respect to the Y value, thus calculated.

Pulp Duplicates are the sample immediately after the original sample. A second pulp sample was prepared by taking the material from the 75 g envelope. The original sample and its duplicate were analyzed in the same batch and so meet the requirement to maintain repeatable conditions.

11.2.6.1 Copper ICP (Pulp Duplicates)

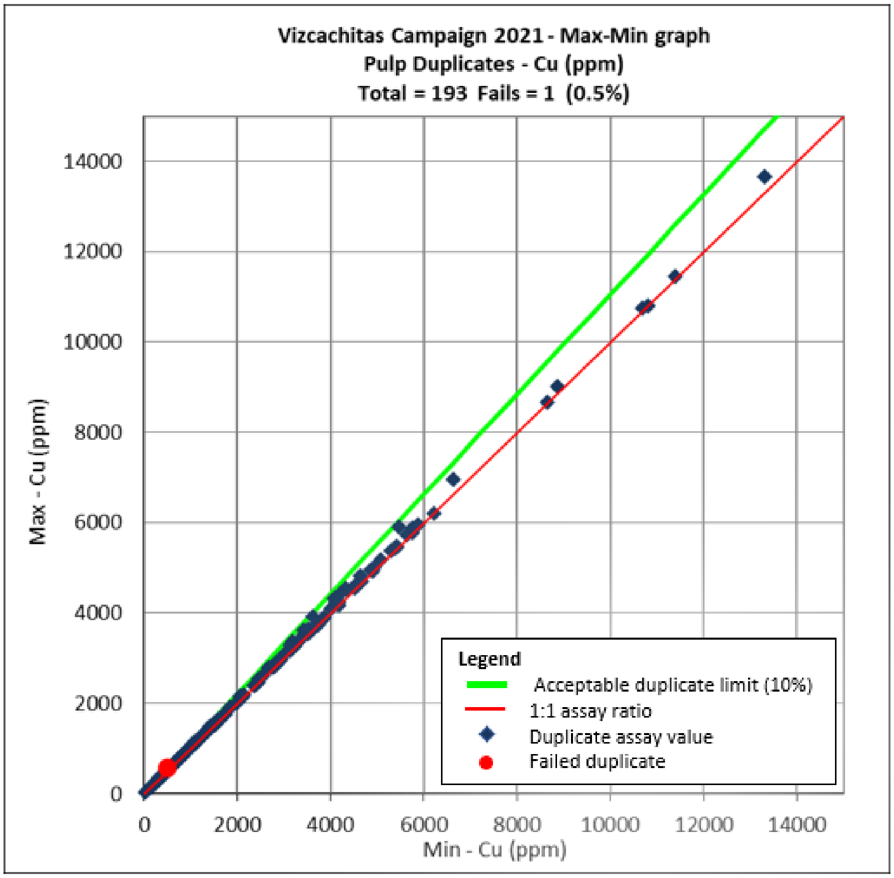
Examination of the copper results for 193 pairs of Pulp Duplicates indicates that the laboratory analytical accuracy was excellent (Figure 11.7).

The parameters used to define the Hyperbolic equation are:

- Practical Limit of Detection: 1 ppm Cu
- Tolerance Factor: 5
- Acceptance Limit: 10%

The rejection rate was 0.5%; 90% of the pairs had a Relative Error <4.3%, much lower than the rate accepted by Los Andes Copper (less than or equal to 10%). The normalized coefficient of variation was 2.1%.

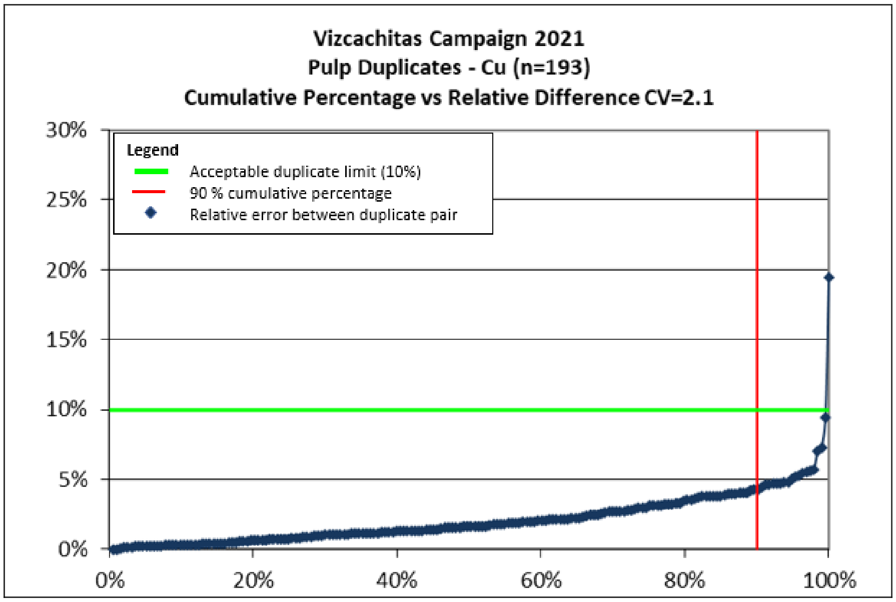
Figure 11.7: Pulp Duplicates – Maximum-Minimum Copper Values



Source: Los Andes Copper, 2022

Figure 11.8 shows the accumulated percentage versus relative difference; it was observed that 90% of the samples have a Relative Error <4.3%, much lower than the rate accepted by Los Andes Copper (less than or equal to 10%).

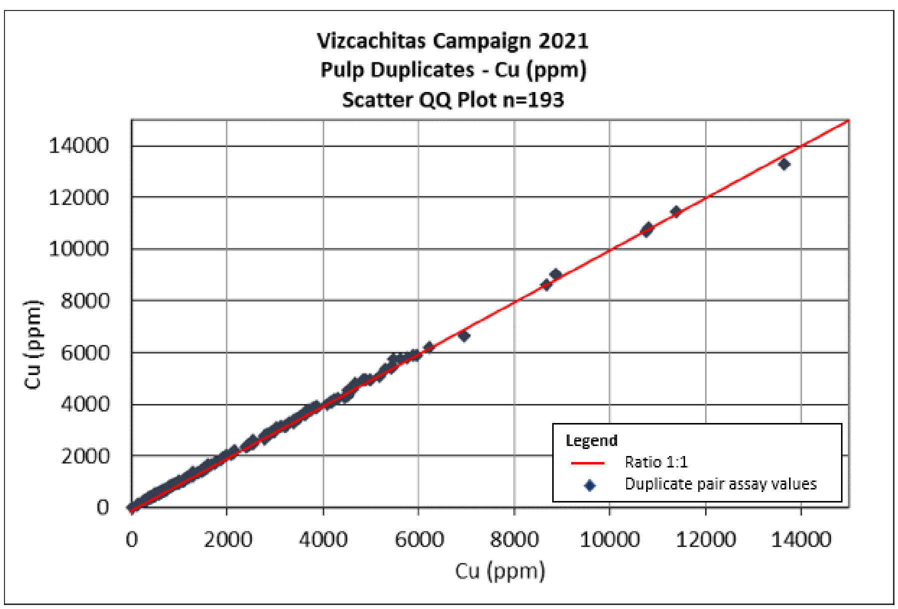
Figure 11.8: Pulp Duplicates - Percentage vs Relative Difference



Source: Los Andes Copper, 2022

Figure 11.9 shows the Pulp Duplicates scatter plot.

Figure 11.9: Pulp Duplicates - Scatter Q-Q Plot Copper Values



Source: Los Andes Copper, 2022

The scatter plot between the ordered values for each series of Pulp Duplicates shows no significant bias between the two datasets. Student's t-test results yielded a value of 1.05 (within the range of ±1.96) confirming that there was no global bias between the sets.

The CRM results for the other Pulp Duplicates and molybdenum, silver and arsenic are contained in the QA/QC Technical Note (Martínez, 2022). The Pulp Duplicate results show that the level of accuracy for these elements ranged from good to excellent, indicating that the laboratory performed well in terms of analytical accuracy.

11.2.7 Coarse Duplicates

Coarse Duplicates determine the degree of precision achieved during the mechanical preparation and chemical analysis. For this purpose, the duplicate was taken from the first division of the Jones Riffle Splitter, thus obtaining two sub-samples that are then prepared separately.

The mechanical and analytical preparation processes incorporate errors that can manifest in the results obtained for sub-samples from the same original sample.

Throughout the campaign Coarse Duplicates were included at a rate of 1 in 41 samples (2.5%). Coarse Duplicates identify the errors attributable to the mechanical preparation and the analytical process. Subtracting the analytical variance, which was calculated from the Pulp Duplicates, it is possible to isolate and estimate the magnitude of the variance of the error attributable to the mechanical preparation.

For Coarse Duplicates, the acceptance criterion used by Los Andes Copper was that 90% of the pairs have a Relative Error less than or equal to 20%.

11.2.7.1 Copper ICP (Coarse Duplicates)

Examination of the reported copper results for 96 pairs of Coarse Duplicates indicates that the level of accuracy achieved in the preparation and analysis stages was excellent.

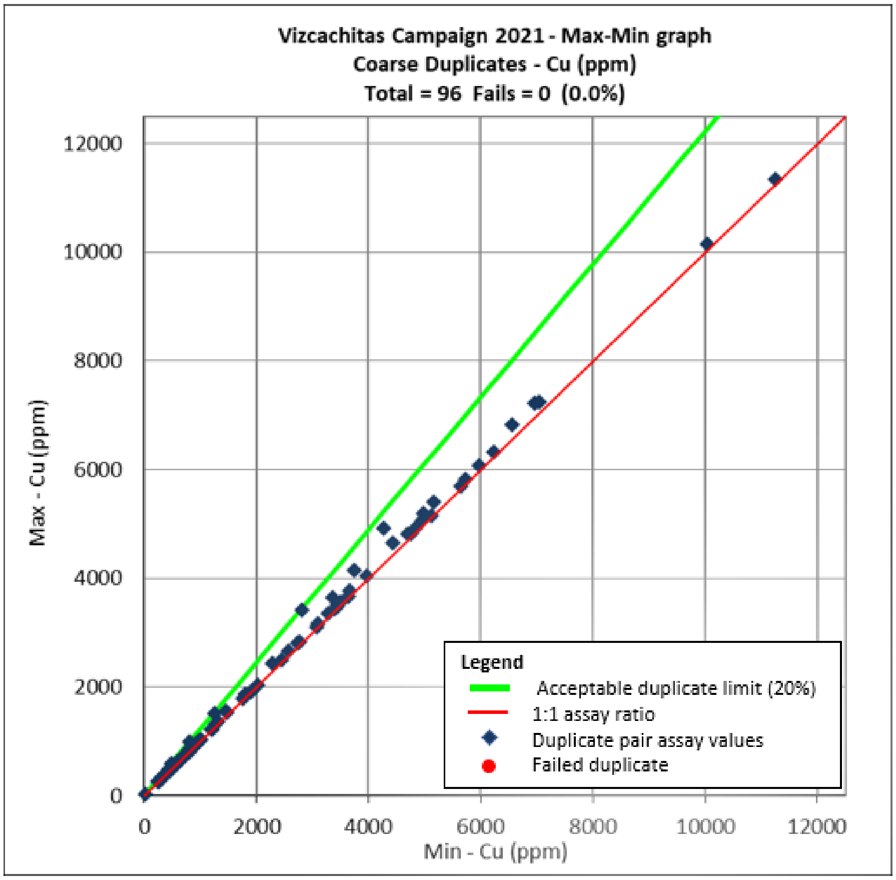
The parameters used to define the Hyperbolic equation are:

- Practical Limit of Detection : 1 ppm Cu
- Tolerance Factor : 7.5
- Acceptance Limit : 20%

The rejection rate was 0.0%; 90% of the pairs have a Relative Error <5.4%, much lower than the rate accepted by Los Andes Copper (less than or equal to 20%). The normalized coefficient of variation was 3.3%.

The maximum-minimum graph (Figure 11.10) shows the behaviour of the 26 pairs of Coarse Duplicates between 28 ppm to 15,000 ppm Cu. The overall rejection rate was 0.0%.

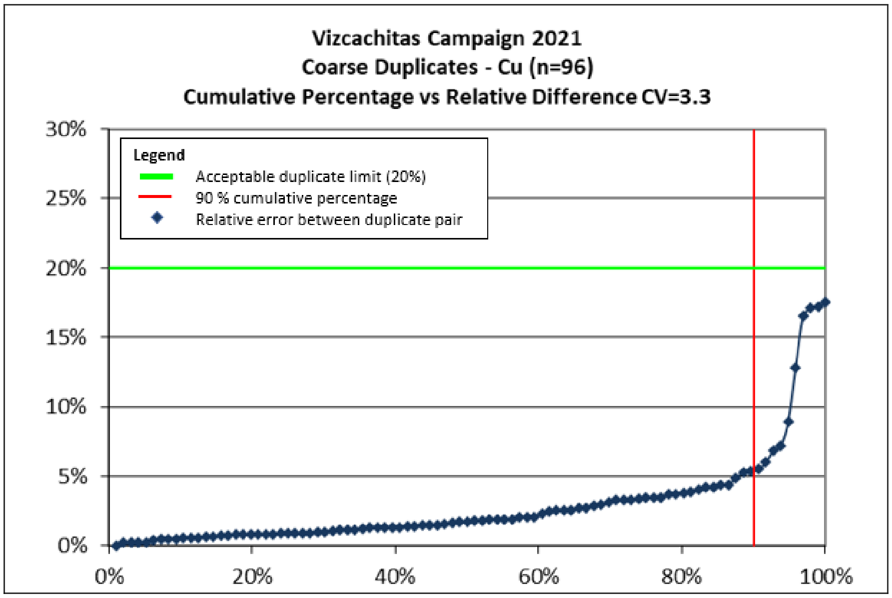
Figure 11.10: Coarse Duplicates – Maximum-Minimum Copper Values



Source: Los Andes Copper, 2022

The accumulated percentage versus relative difference graph (Figure 11.11) showed that 90% of the samples have a Relative Error <5.4%, much lower than the rate accepted by Los Andes Copper (less than or equal to 20%).

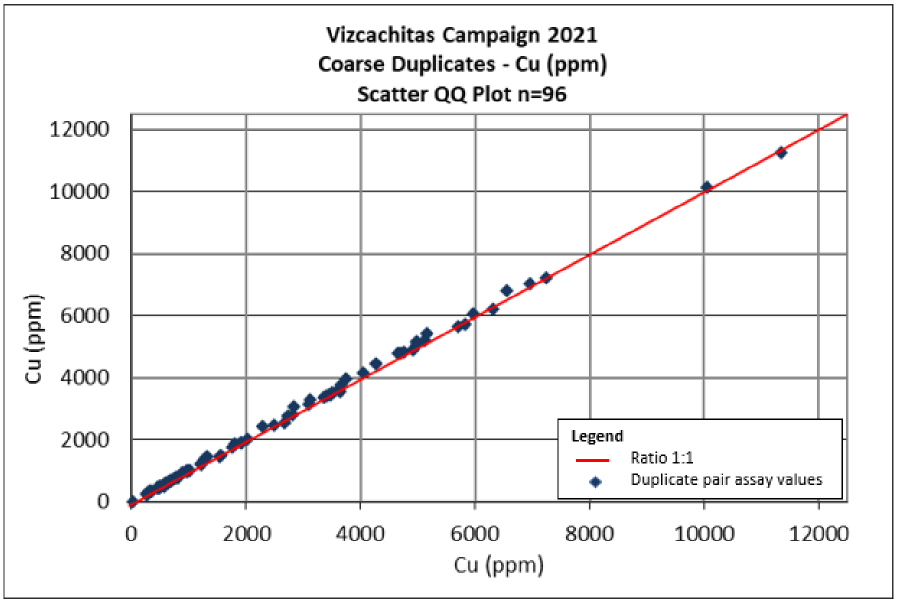
Figure 11.11: Coarse Duplicates – Percentage vs Relative Difference



Source: Los Andes Copper, 2022

The scatter plot between the ordered values for each series of Coarse Duplicates (Figure 11.12) shows no significant bias between the two datasets. The result of the Student's t-test yields a value of -1.80, which is within the range of +/-1.96, confirming that there was no global bias between the sets.

Figure 11.12: Coarse Duplicates – Scatter Q-Q Plot Copper Values



Source: Los Andes Copper, 2022

The Coarse Duplicates results for molybdenum, silver and arsenic are contained in the QA/QC Technical Note (Martínez, 2022) The Coarse Duplicate results indicate that the mechanical preparation had good to excellent levels of quality in the four main elements considered. Only the molybdenum has a lower accuracy (although within tolerance).

11.2.8 Twin Samples

Twin Samples are intended to determine the degree of precision achieved during cutting, mechanical preparation and chemical analysis of diamond cores, i.e. the level of repeatability achieved, by preparing and analyzing the other half of the same section of the core. Unlike Pulp Duplicates and Coarse Duplicates, which are duplicates because they are particulate materials, cutting a piece of core in half is not considered a sampling operation, hence the term Twin Samples is used instead of duplicates. Twin Samples are prepared by cutting the core lengthwise into two halves and generating two samples from the same core length.

The sampling and mechanical preparation processes, as well as the analytical processes, incorporate errors that can be manifested in the fact that the results obtained for a pair of sub-samples taken from the same original sample may not be the same.

Throughout the campaign, Twin Samples were added at a rate of 1 in 43 samples (2.5%). Twin Samples verify that the core cutting is carried out correctly. From the results, it was possible to conclude that the core cutting meets the standard set.

For Twin Samples, the acceptance criterion used by Los Andes Copper is that 90% of the duplicate pairs having a Relative Error less than or equal to 30%.

11.2.8.1 Copper ICP (Twin Samples)

Examination of the copper results reported for 93 pairs of Twin Samples indicates that the level of accuracy achieved in the cutting, preparation and analysis stages was acceptable.

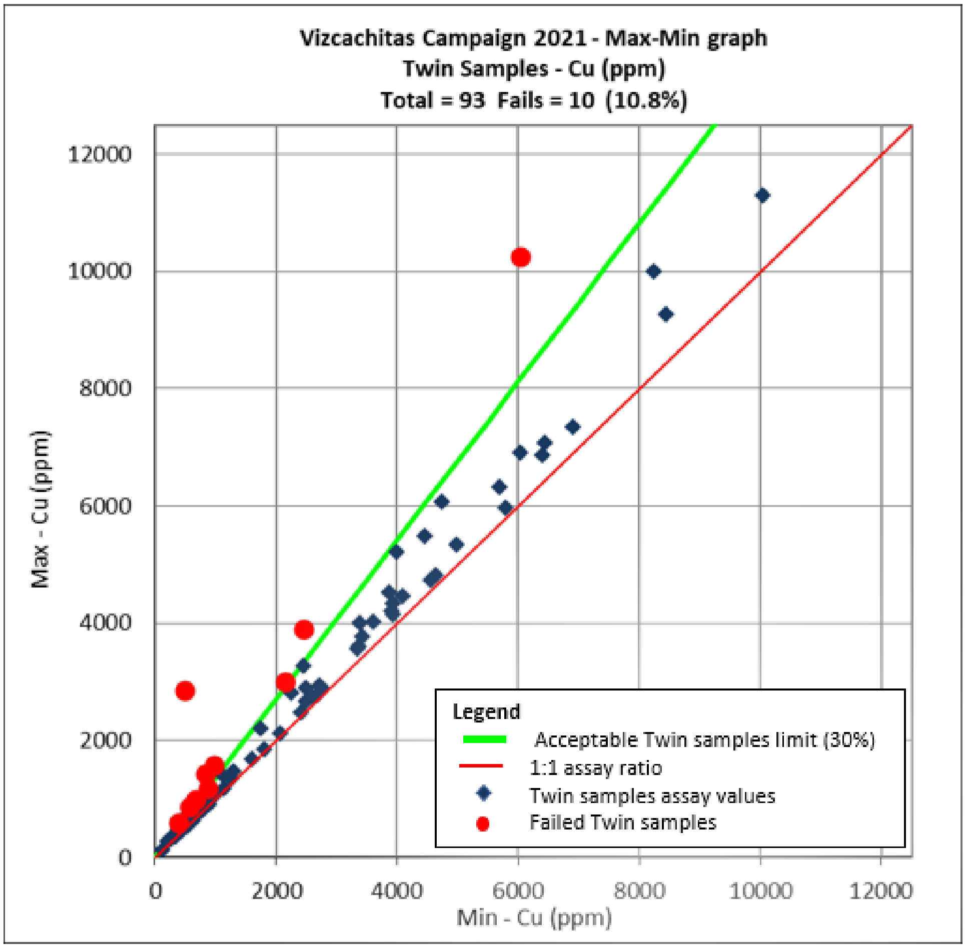
The parameters used to define the Hyperbolic equation are:

- Practical Limit of Detection: 1 ppm Cu
- Tolerance Factor: 8
- Acceptance Limit: 30%

The rejection rate was 10.8%; 90% of the pairs have a Relative Error <26.1%, lower than the rate accepted by Los Andes Copper (less than or equal to 30%). The normalized coefficient of variation was 14.9%.

The maximum-minimum graph (Figure 11.13) shows the behaviour of the 26 pairs of Twin Samples in the range between 127 ppm to 15,000 ppm Cu. The overall rejection rate was 10.8%, at the limit of tolerance.

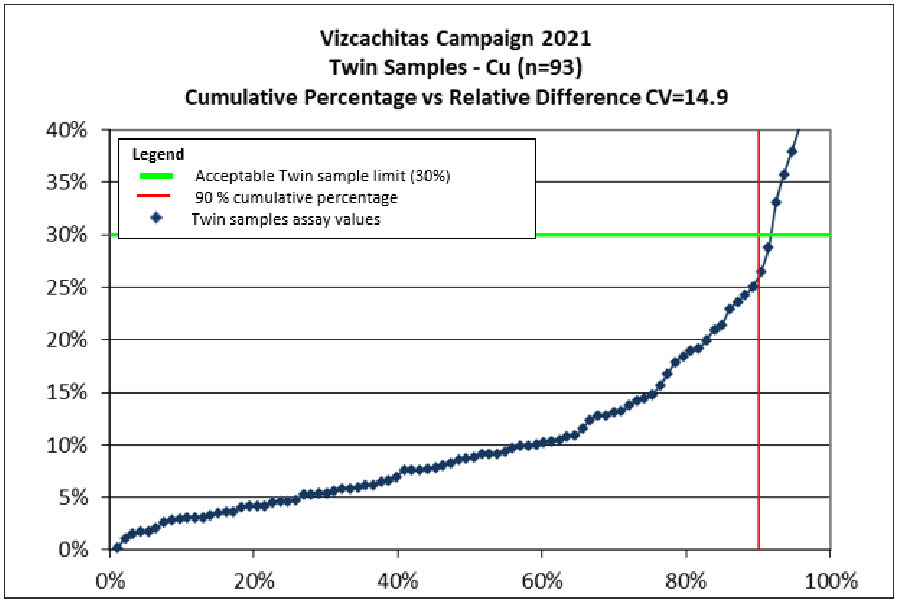
Figure 11.13: Twin Samples – Maximum-Minimum Copper Values



Source: Los Andes Copper, 2022

The accumulated percentage versus relative difference graph (Figure 11.14) showed that 90% of the samples have a Relative Error <26.1%, lower than the rate accepted by Los Andes Copper (less than or equal to 30%).

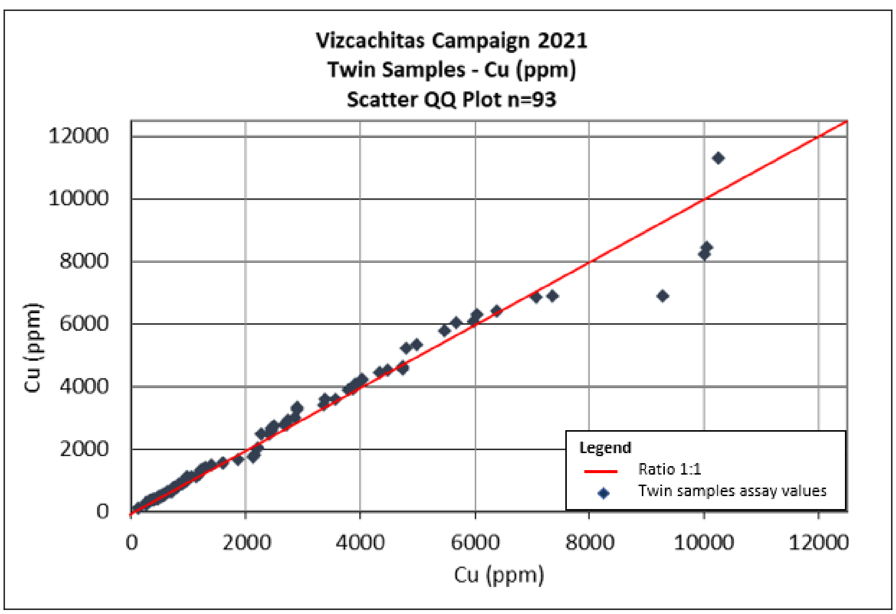
Figure 11.14: Twin Simples – Percentage vs Relative Difference



Source: Los Andes Copper, 2022

The scatter plot between the ordered values for each series of Twin Samples (Figure 11.15) shows no significant bias between the two data sets. The result of the Student's t-test yields a value of 0.10, within the range of ±1.96, confirming that there was no global bias between the sets.

Figure 11.15: Twin Samples – Scatter Q-Q Plot Copper Values



Source: Los Andes Copper, 2022

The Twin Sample results for silver are below the limit of 10% of the tolerance of failed samples. The results for molybdenum and arsenic are just above the tolerance limit, possibly due to the heterogeneity of the mineralization for molybdenum, which often shows a marked nugget effect. The variability for arsenic is attributable to the low grade, where the analytical precision was lower, although also not ruling out the presence of the nugget effect. Therefore, it was concluded that the longitudinal cut of the core was as even as possible.

11.2.9 Second Laboratory

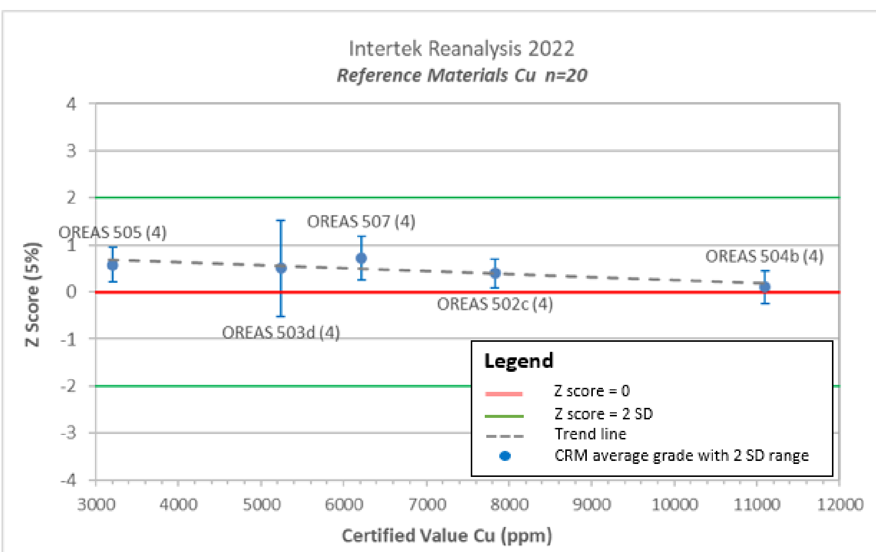
11.2.9.1 Introduction

To evaluate the overall performance of ALS, especially for the accuracy of the silver, arsenic, copper and molybdenum analysis, 5% of the drill samples were sent to a second laboratory. Intertek Genalysis, Maddington, Australia (Intertek), was selected because they have expertise in ICP-MS analysis. Intertek method 4A/MS48 analyzes the same 48 elements as the ALS ME-MS61 method.

Two hundred samples were randomly selected (5% of the total), covering the grade range for the deposit (0.1% Cu and 1.4% Cu). Additionally, 50 additional QA/QC samples (20 CRMs, 20 analytical duplicates and 10 Pulp Blanks) were inserted to check the performance of the second laboratory. The CRMs were the same as those used in the 2021-2022 campaign.

The copper results for the CRMs showed an average over-estimation of about 2.5% over the grade range between 3,210 ppm and 11,100 ppm, decreasing significantly towards high grades (Figure 11.16). This indicates that the analytical accuracy was good.

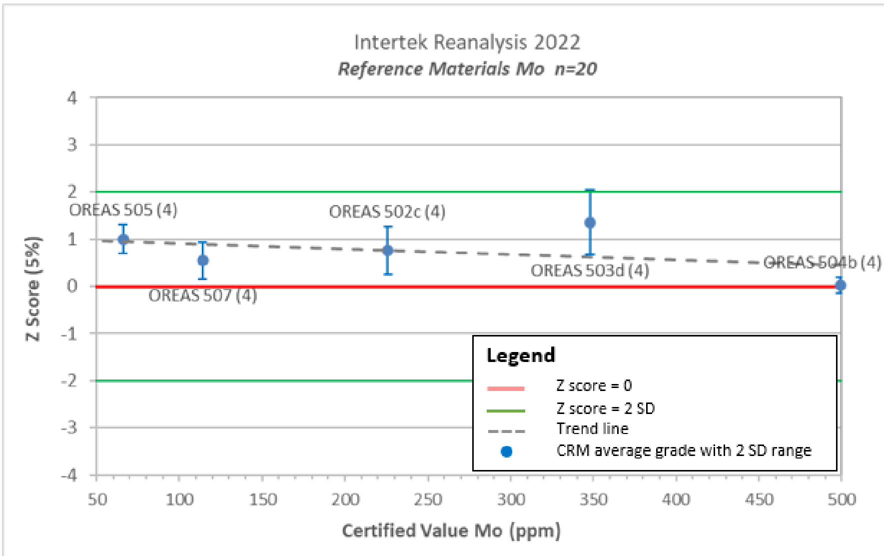
Figure 11.16: Intertek CRM Copper Values



Source: Los Andes Copper, 2022

Molybdenum CRM results showed an average over-estimation of about 4% over the grade range from 67 ppm to 499 ppm, again decreasing at high grades (Figure 11.17). Also indicating that the analytical accuracy was good.

Figure 11.17: Intertek CRM Molybdenum Values



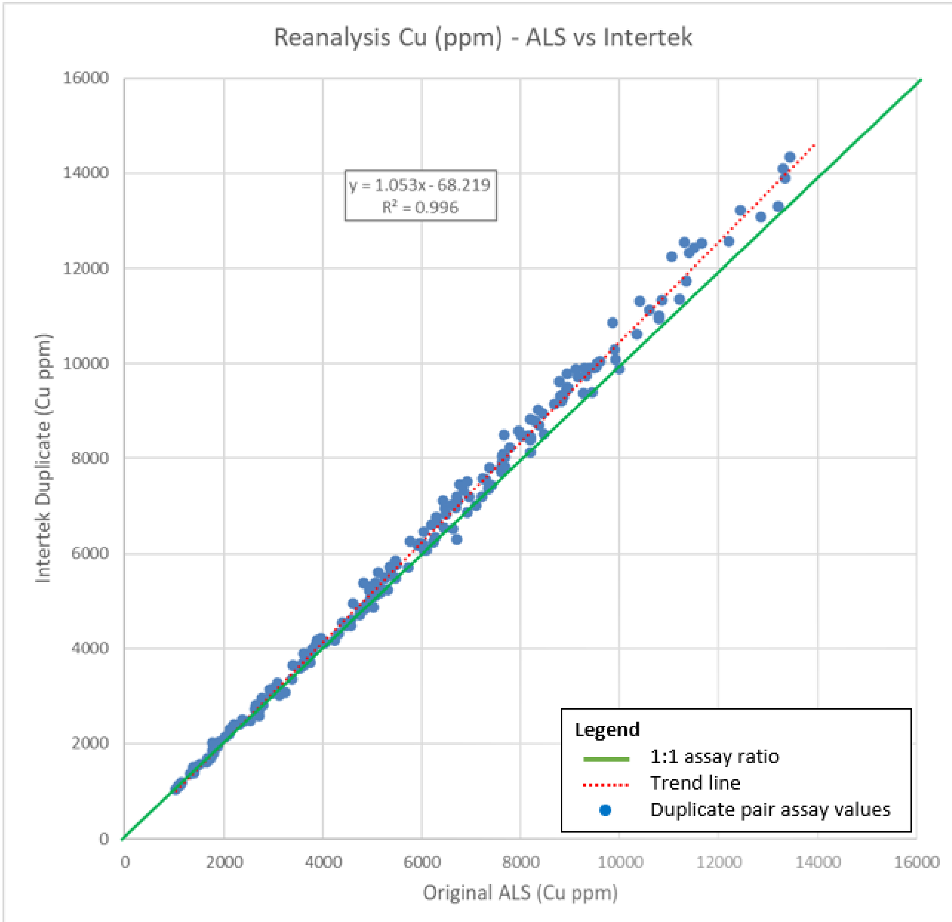
Source: Los Andes Copper, 2022

11.2.9.2 Re-analysis Results

Copper

The re-analysis results (original ALS values versus the Intertek re-analysis) show an excellent correlation (R^2 of 0.996) (Figure 11.18). However, there was a bias between the ALS laboratory and Intertek; the results reported by Intertek are higher by about 5% compared to ALS. This over-estimation was consistent with the results reported for the reference materials included in the re-analysis. Therefore, it can be concluded that Intertek over-estimated the values for copper (particularly in grades below 7,000 ppm) and that the original ALS results have better analytical accuracy when compared to the CRMs.

Figure 11.18: ALS vs Intertek re-analysis – Copper Values

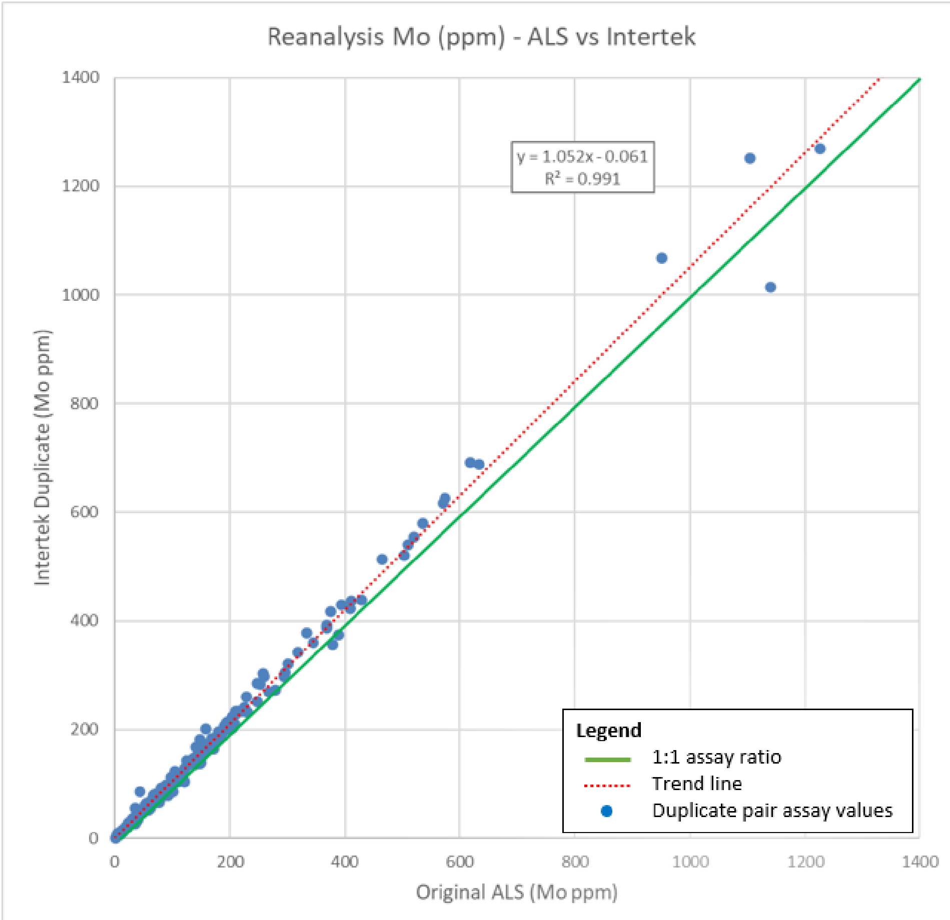


Source: Los Andes Copper, 2022

Molybdenum

The results of the Intertek re-analysis show an excellent correlation (R^2 of 0.991) (Figure 11.19). However, there was a bias between ALS and Intertek; the results reported by Intertek are higher by about 5% compared to ALS. This was consistent with the results reported for the reference materials included in the re-analysis. It can be concluded that Intertek over-estimated the molybdenum values (particularly at grades below 350 ppm) and that the original ALS results have better analytical accuracy when compared to the CRMs.

Figure 11.19: ALS vs Intertek re-analysis – Molybdenum Values



Source: Los Andes Copper, 2022

11.2.9.3 Conclusions

Examination of control samples indicates that although Intertek's performance was acceptable in terms of analytical precision and contamination levels, this laboratory over-estimated the copper and molybdenum grades by almost 5%. This conclusion was based on the regression curves of the original ALS results vs those reported by Intertek and was supported by the results for the CRMs.

The results reported by ALS have good precision and accuracy and can be used for resource estimation. The differences between the copper and molybdenum values between ALS and Intertek are due to an over-estimation by Intertek rather than problems with the original analysis.

11.2.10 Summary QA/QC 2021-2022

The QA/QC system implemented for the sampling, mechanical preparation, and analytical results of the 2021-2022 drilling campaign meet or exceed mining industry best practices.

- The analysis of the Blanks indicates no significant contamination events for the mechanical preparation and during the chemical analysis of drill samples.
- The results reported for the CRMs show values consistent with the certified grades for these control samples. The biases observed are within the tolerance ranges and are low for all analyses. Only the arsenic shows a slight over-estimation that was accentuated towards the higher grades, although the bias remained within tolerance.
- The Pulp Duplicate results show that the level of accuracy for the four elements (Cu, Ag, Mo, As) ranged from good to excellent, indicating that the laboratory performs well in terms of analytical accuracy.
- The Coarse Duplicate results indicate that the mechanical preparation had good to excellent levels of quality in the four main elements considered. Only the molybdenum has a lower accuracy (although within tolerance).
- The Twin Sample results show that the errors attributable to cutting the core, mechanical preparation and chemical analysis are at the limit or just above the accepted tolerance of 30%. For copper and silver these elements are at the tolerance limit (10% of failed samples). For molybdenum and arsenic these elements are above tolerance. For molybdenum this was possibly due to the heterogeneity of the mineralization, which often shows a marked nugget effect. For arsenic this could be attributable to the low grade of this element, where the analytical precision was lower, although also not ruling out the presence of the nugget effect. Therefore, it was concluded that the longitudinal cut of the core was as even as possible.

The 2021-2022 analytical results meet industry quality standards and therefore can be used to estimate Mineral Resources.

11.3 Opinion on the Adequacy of Sample Preparation and Assay Quality

In the QP's opinion, the sample preparation, security, analytical procedures, quality control procedures and quality assurance measures undertaken by Los Andes Copper provide confidence in the drill hole data collection and processing.

12. DATA VERIFICATION

12.1 Site Visits

On March 22 to 24, 2022, the Qualified Person (QP), Sergio Alvarado, visited the Project site, core cutting facility and warehouse.

12.1.1 Vizcachitas Project

The Vizcachitas Project site visit took place on March 23, 2022. The coordinates for eight drill holes were measured using a Garmin hand-held GPS to compare the coordinates to the database. There were minor differences, but these differences were within the precision of the handheld GPS.

Figure 12.1, Figure 12.2 and Figure 12.3 show drill collars and a drill in position on the Project site.

Figure 12.1: Drill Hole CMV-009, Looking North-East

Figure 12.2: Drill Hole CMV-012b, Drilling

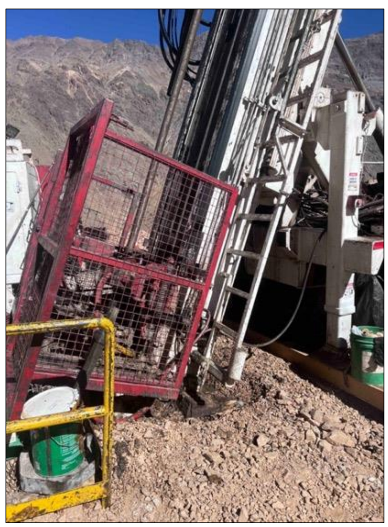


Figure 12.3: Drill Hole CMV-001b, Looking North

Source: Sergio Alvarado, 2022

12.1.2 Core Cutting Warehouse

The core cutting warehouse is in Rinconada, Valparaíso Region, 75 km from the Project site. Los Andes Copper has a strict protocol for chain of custody and transport. The core boxes are packed on pallets, sealed and registered at the Project site. The pallets are transported by truck directly from the Project site to the core-cutting warehouse. The Los Andes Copper staff check the core boxes on arrival against the shipping documents. Any variation from the register is immediately investigated.

The unpacked core is photographed, then cut with the diamond saw along the lines marked by the project geologist. Half of the cut core is placed in numbered sample bags and the other half is replaced in the core boxes. The core boxes and sample bags are then carefully repacked and registered before being sent by truck directly to the Quilicura logging warehouse (Figure 12.4 and Figure 12.5).

Figure 12.4: Pallets with Diamond Drilling Samples Received Directly from the Project Site

Figure 12.5: Diamond Drill Core Photography

Source: Sergio Alvarado, 2022

12.1.3 Quilicura Warehouse and Office

The drill hole logging and core storage warehouse is in Quilicura, Santiago, Metropolitan Region, 60 km from the core cutting warehouse.

On arrival the Los Andes Copper staff check the core boxes and samples against the shipping documents. Any variation from the register is immediately investigated.

The core boxes are placed on trestle tables where the geologist carries out the detailed logging for lithology, alteration, mineralization and veinlets. The core is logged using tablets running acQuire Technology Solutions Pty Ltd (acQuire) database software.

The database manager prepares the sample batch dispatch form and the geologists add Coarse Blank and Twin Samples to the sample sequence. The samples are then repacked and transported in a light truck to the ALS sample preparation laboratory in Quilicura. Figure 12.6 and Figure 12.7 show the Quilicura warehouse.

Figure 12.6: Core Boxes Arranged for Logging by Geologists

Figure 12.7: Storage of Core Boxes

Source: Sergio Alvarado, 2022

12.2 Data Validation

The historical exploration campaigns at the Vizcachitas Project have been reviewed and validated previously:

- 2008 AMEC International (Chile) S.A. reviewed and validated the database for GMC and Los Andes Copper for use in the resource estimation (AMEC, 2008)
- 2014 Coffey reviewed and validated the conversion of handwritten logs to a digital database and ensured that the database complied with the reporting standards so that it could be used in the resource estimation (Coffey, 2014)
- 2019 Tetra Tech reviewed and validated the information for the 2015-2017 drilling campaigns. The QP checked the: drill hole logging, drill collars, downhole survey, database, geological interpretation, alteration, mineralization and specific gravity, finding that the data quality complied with the NI 43-101 standards. This data was used for the 2019 PEA (Tetra Tech, 2019).

12.2.1 Logging Files

The QP reviewed the certified documents and paper records held in a document database and an Excel database related to:

- Type of drilling (DDH, diameter)
- Historical geological logs including: depth, lithology, structures, mineral zones, alteration, oxides mineralization, sulphide mineralization and gangue
- Drill core recovery for each drill interval
- Sample preparation and analysis forms.

12.2.2 Drill Hole Collars

The QP verified the collar coordinates for the exploration campaigns against the PhotoSat™ topography for the Project area. The QP determined that there is good correlation between the elevation of the drill hole collars and the topography.

12.2.3 Downhole Survey

In 2019 Tetra Tech reviewed the coordinates and trajectory data for 61 drill holes in the database. Tetra Tech confirmed that the database information was adequate and met the requirements of the Technical Report (Tetra Tech, 2019). The 2022 drill hole downhole survey showed consistent coordinates and trajectories, complying with the requirements for this Technical Report.

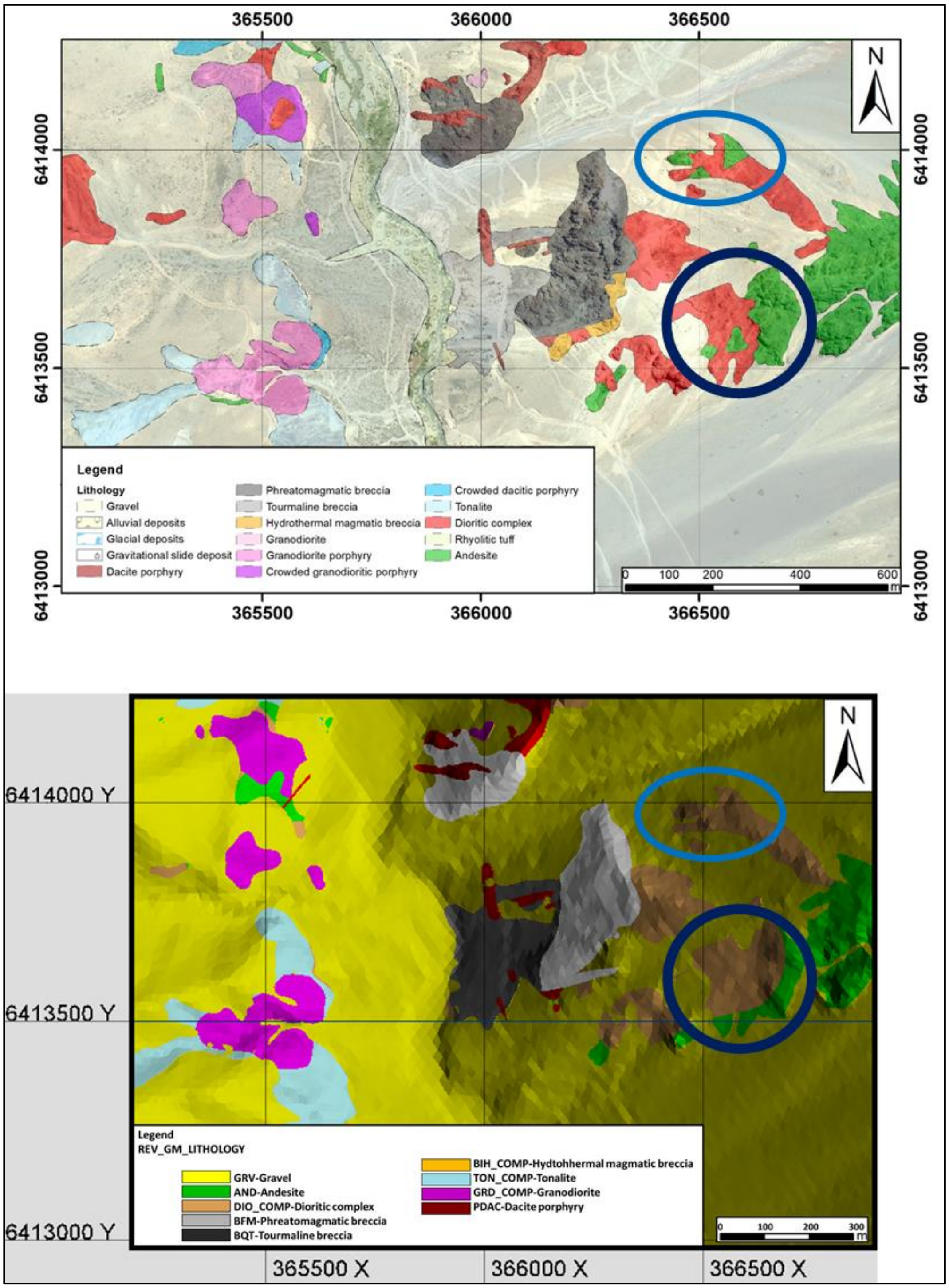
12.2.4 Database Review

The QP checked the acQuire database against pdf assay certificates provided by ALS for drill holes from the 2021-2022 campaign. The QP reviewed copper, molybdenum, silver and arsenic values from the multi-element ICP. No errors were identified.

12.3 Interpretation of Lithology, Alteration, Mineralization, Veinlets and Structures

The QP reviewed the lithological model of the Vizcachitas Project to verify the correlation between the drill hole logging, surface mapping, 2D sectional model and the 3D model generated in the Seequent Leapfrog Geo™ software. Figure 12.8 compares the surface geological mapping with the three-dimensional geological model. No significant discrepancies were identified, except that the surface mapping indicates diorite and andesites outcropping in the north-east but the 3D model only has the diorite outcropping.

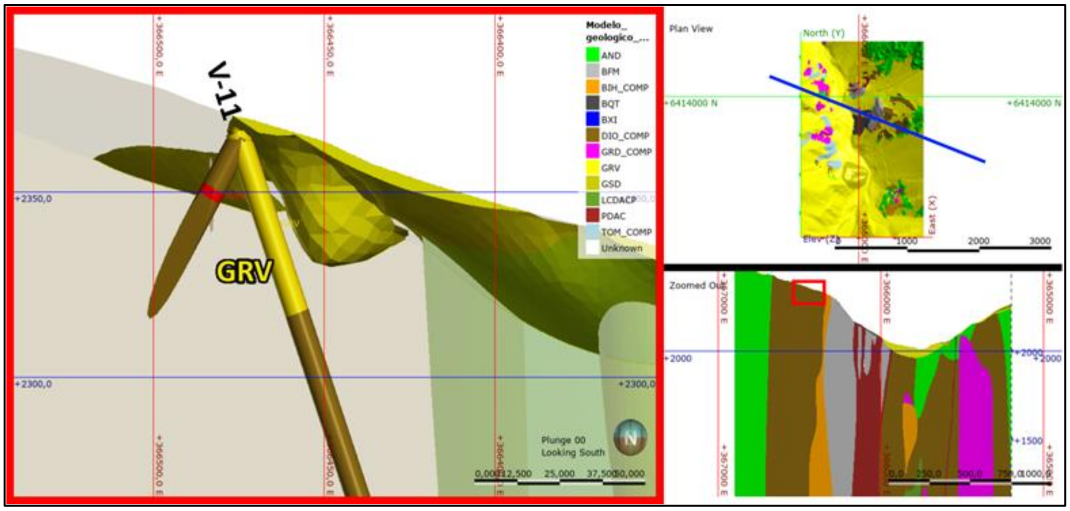
Figure 12.8: Top – Surface Geological Map;
Bottom – Surface View in a Three-Dimensional Model



Source: Tetra Tech, 2022

Figure 12.9 shows two minor inconsistencies between the 3D model and the drill holes on vertical section 2100. The left hand diagram shows the drill logged as gravel within the intrusive; the right hand diagram shows a drill hole mapped as breccia outside the 3D breccia model. Both inconsistencies are considered minor. The rest of the drilling correctly matched the 3D solids.

Figure 12.9: Inconsistencies Between Drill Holes and 3D Model, Vertical Section 2100



Note: Vertical section looking south
Source: Tetra Tech, 2022

The different scales for the models explain these differences. The logging was at a scale of 2 m, the sectional model was at a scale of 10 m, and the 3D model was at a scale of 30 m. The simplification of the model at each stage can create minor anomalies.

Similar reviews were carried out for the mineral zone and veinlets models. Some minor variations were identified between the drill hole database and the 3D model, but these could be explained by the scale of the 3D model.

A full review of the Qualified Person's data review and the site visit is contained in the QP's Technical Note (Alvarado, 2022).

12.4 Chapter 12 Opinion on Data Adequacy

In the opinion of the QP, the geological and geochemical data reviewed are an adequate and accurate reflection of the geology of the Vizcachitas Project. The data reviewed meets the standards of a NI 43-101 Technical Report and for use in the Mineral Resource estimation.

13. MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Previous Testwork

The Vizcachitas Project has been studied since 1998 over several metallurgical testing programmes. The objective of these programmes was to characterize the feed (sulphide or oxide) and define the process recovery method (froth flotation). All the testwork programmes were performed by SGS Minerals Chile and Lakefield Research.

13.1.1 Comminution Testwork

13.1.1.1 1998: Ball Bond Work Index

In October 1998 Lakefield Research Laboratories (Lakefield 1999a) determined standard Bond Ball Work indices (BWi) for five of the composite samples using standard laboratory procedures. Table 13.1 shows the results of the Bond tests. The work index values of the composites exhibited large variations ranging from a low of 10.1 kWh/st to a high of 13.5 kWh/st.

Table 13.1: Summary of Bond Ball Mill Work Index

Sample	W_i		P_{80}
	kW/t	kWh/st	µm
B	14.9	13.5	115
C	11.5	10.4	109
E	11.1	10.1	115
H	12.9	11.7	118
I	12.9	11.6	117

13.1.1.2 2008: SMC Test and Bond Abrasion Test

A total of six samples for comminution tests and six samples for abrasion tests were received at SGS Mineral Services in August 2008 (SGS Minerals Services, 2009).

The results of the comminution tests showed that the samples were hard to moderately hard. The results of the abrasion tests showed that the samples were moderately abrasive.

Table 13.2 and Table 13.3 show the results of the SMC tests (DWi, M_{ia}, Axb) and abrasion index (Ai), respectively.

Table 13.2: Summary of SMC Test

Sample	SG	SMC test		
		DW_i [kWh/m^3]	M_{ia} [kWh/t]	AxB
43506	2.58	7.50	22.30	34.28
43507	2.69	7.80	21.90	34.89
43508	2.57	6.60	20.20	39.40
43509	2.65	8.70	24.40	30.32
43510	2.55	5.70	18.20	44.49
43511	2.58	7.00	21.10	37.14

Table 13.3: Summary Abrasion Index Test

Sample	Initial Paddle weight (g)	Final Paddle weight (g)	Abrasion Index A_i
43512	94.00	93.68	0.32
43513	94.67	94.31	0.36
43514	94.31	96.91	0.41
43515	94.80	94.33	0.47
43516	94.33	93.97	0.36
43517	94.71	94.31	0.40

13.1.1.3 2017: Ball Bond Work Index

In 2017 Bond Work Index (BWi) tests were conducted on the 40 drill hole samples for sizing of the ball mills, the average result for the BWi was 13.1 kWh/t (11.9 kWh/st) (ranging between 7.4 kWh/t and 18.6 kWh/t (6.7 to 16.8 kWh/st), slightly higher than that determined in the tests in 1998 (BWi of 12.7 kWh/t).

13.1.1.4 2017: Starkey Test

In 2017, a Starkey test was conducted, measuring the time required to reach a given P80. This information was used to determine the specific energy consumption (SEC) of the SAG mill, once the grinding time (Starkey test/SEC) ratio was known.

The SEC is determined in a pilot plant with samples used in Starkey tests. Thus, an estimate of the expected SEC was made for the SAG mill when the empirical model of the SEC with grinding time (Starkey test) was known.

The average value of grinding time for the 40 samples was 66 minutes (ranging between 32 minutes and 103.9 minutes).

13.1.2 Leach Testwork

The database of leach tests completed included the following programmes:

- 1999 Testwork (Lakefield Research Chile S.A., 1999b)
    - Chemical analysis
    - Mineralogical analysis
    - Acid leach tests
    - Ferric leach tests
    - Bacterial-assisted leach tests
- 1999-2000 Testwork (Lakefield Research Chile S.A., 2000)
    - Chemical analysis
    - Mineralogical analysis
    - Column leach tests
- 2001 Testwork (Little Bear Laboratories, Golden, Colorado, 2001)
    - Chemical analysis
    - Column leach tests.

Testwork results concluded that the chalcopyrite samples from Vizcachitas are not amenable to conventional heap leaching using bacterial-assisted ferric leaching. The laboratory commented that secondary-enriched mineralization could be treated by this process, but further work was needed to improve copper dissolution kinetics and final copper extraction.

13.1.3 Flotation Testwork

The process route considered bulk and selective flotation of copper and molybdenum from the mineralized rock. A series of metallurgical tests aimed at understanding the behaviour of the Vizcachitas minerals to flotation processes was undertaken over several testwork campaigns. The database of flotation tests performed includes the following programmes:

- 1996 Testwork (Lakefield Research Chile S.A., 1996)
    - Chemical analysis
    - Mineralogical analysis
    - Primary flotation kinetics
    - Open cycle tests
- 1998 Testwork (Lakefield Research Chile S.A., 1999a)
    - Chemical analysis
    - Mineralogical analysis
    - Primary flotation kinetics
    - Open cycle tests
    - Locked cycle tests
- 2017-2017 Testwork (SGS Minerals Services, 2017-2018)
    - Chemical analysis
    - Primary flotation at different feed grain sizes
    - Locked-circuit tests (LCT).

13.1.3.1 1996 Testwork

Seven small samples from two drill holes were submitted to Lakefield Research, Canada in 1996 (Lakefield Research Chile S.A., 1996). These samples appeared to be coarse rejects from the drilling campaign at the time.

Table 13.4 presents the chemical analysis of the seven samples received. The head grades of the samples varied from 0.50% Cu to 1.31% Cu and 47 ppm Mo to 275 ppm Mo. Precious metal values were low but silver values were 5 g/t in one sample. Arsenic values were low at 7 ppm to 21 ppm.

Table 13.4: Chemical Analysis 1996 Samples

Sample	Cu (%)	Mo (ppm)	Fe (%)	S (%)	Au (ppm)	Ag (ppm)	As (ppm)
Saco 1	1.17	121	3.08	2.42	<0.01	4.40	16.00
Saco 2	0.74	47	3.58	1.80	0.01	2.20	8.00
Saco 3	0.50	73	6.35	0.92	0.02	1.30	12.00
Saco 4	1.31	57	5.05	3.52	0.02	5.10	8.00
Saco 5	0.95	117	3.62	1.40	0.02	1.50	8.00
Saco 6	0.64	115	2.84	2.80	0.03	1.00	7.00
Saco 7	0.67	275	3.21	3.70	0.01	1.30	21.00
Average	0.85	115	3.96	2.37	0.02	2.40	11.00
Maximum	1.31	275	6.35	3.70	0.03	5.10	21.00
Minimum	0.50	47	2.84	0.92	0.01	1.00	7.00
Standard dev.	0.30	77	1.27	1.05	0.01	1.66	5.29

Mineralogical analysis showed that the principal copper mineral was chalcopyrite but three of the samples had significant amounts of chalcocite and minor amounts of covellite. Pyrite values varied between 1% and 6% by weight.

Table 13.5 shows the mineralogical characterization and the species present in five of the samples, completed using automated quantitative mineralogy (QEMSCAN).

Table 13.5: Mineralogy 1996 Samples

Species	% Weight				
	Saco 1	Saco 3	Saco 5	Saco 6	Saco 7
Chalcopyrite	2.57	0.72	1.84	1.85	1.94
Chalcocite	0.24	0.26	0.35	-	-
Covellite	0.12	0.08	0.05	-	-
Tetrahedrite	0.02	-	-	-	-
Pyrite	2.66	1.06	1.22	4.02	5.65
Molybdenite	0.02	0.01	0.02	0.02	0.02
Limonite	0.09	0.22	0.04	-	-
Magnetite	-	0.62	0.11	-	0.04
Hematite	0.06	0.16	0.08	0.57	0.51
Rutile	0.10	0.04	0.15	-	0.03
Non-Sulphide Gangue	94.12	96.80	96.14	93.54	91.82

Flotation tests were carried out on five of the samples (rougher and open circuit cleaning tests). Rougher copper recoveries were very good and were in the range of 91% to >98% with good rougher grades. It should be noted that these tests were carried out at a relatively fine grind of 100 µm.

Table 13.6 shows the results for the maximum flotation time tested (12 minutes).

Table 13.6: Summary of Rougher Flotation 1996 Samples

Test	Sample	Conditions				Head				Rougher Concentrate			
						Head, Calc.		Head, Dir.		Grade		Recovery	
		MIBC	Reagent - Dosage	P_{80}	pH	Cu	Mo	Cu	Mo	Cu	Mo	Cu	Mo
		(g/t)	(g/t)	µm		(%)	(%)	(%)	(%)	(%)	(%)	(%)	(%)
S1-1	Saco 1	15.0	3477 - 40	100	8.0	1.19	0.010	1.17	0.012	10.40	0.043	97.50	49.30
S1-2		n/i	SIPX - 15	100	8.0	1.15	0.009	1.17	0.012	9.30	0.050	97.60	64.10
S3-1	Saco 3	20.0	3477 - 40	100	9.0	0.48	0.007	0.50	0.007	4.50	0.031	94.40	44.80
S3-2		12.5	SIPX - 15	100	8.0	0.46	0.007	0.50	0.007	4.70	0.035	91.00	45.00
S5-1	Saco 5	17.5	3477 - 45	100	8.5	0.96	0.008	0.95	0.012	9.80	0.045	97.60	57.50
S5-2		17.5	SIPX - 25	100	8.5	0.97	0.013	0.95	0.012	8.30	0.096	97.50	87.20
S6-1	Saco 6	15.0	3477 - 40	100	9.2	0.67	0.010	0.64	0.012	7.00	0.074	98.80	74.20
S6-2		17.5	SIPX - 25	100	9.4	0.63	0.013	0.64	0.012	6.00	0.108	98.10	88.50
S7-1	Saco 7	10.0	3477 - 40	100	9.1	0.68	0.029	0.67	0.028	6.50	0.169	98.70	61.50
S7-2		10.0	SIPX - 25	100	9.4	0.65	0.032	0.67	0.028	6.30	0.225	98.30	71.20

Ten open-cycle tests (OCT) were completed with three cleaning stages for different grind and regrind product sizes.

Results showed copper recoveries from the third cleaner stage of 98% with a regrind size of 20 µm to 30 µm and final concentrate grades between 30% Cu and 38% Cu. A finer regrind product produced a lower grade copper concentrate.

13.1.3.2 1998 Testwork

A total of 47 coarse reject samples were delivered to Lakefield Laboratories in October 1998 and blended into 11 composite samples (labelled A to K), largely based on lithology but also on type of mineralization (Lakefield Research Chile S.A., 1999a). The head grade of these composite samples varied from 0.43% Cu to 0.94% Cu and 0.005% Mo to 0.0140% Mo. Some composite samples contained silver grades up to 6 g/t. It was noted that some composite samples had significant values of acid soluble copper (up to 20%), which normally adversely affects the overall recovery of copper minerals. Table 13.7 illustrates the description and chemical analysis of the samples.

Table 13.7: Sample Description and Chemical Analysis

Composite	Rock type	Grade	Cu (%)	Cu (citric) (%)	Cu (sulfuric) (%)	Cu (NaCN) (%)	Mo (%)	Fe (%)	Ag (ppm)	Zn (ppm)
A	Andesite	High grade enrichment	0.94	0.09	0.10	0.52	0.01	3.38	1.60	36
B	Andesite	Mixed	0.54	0.08	0.09	0.24	0.01	5.07	1.24	40
C	Andesite	Primary mineralization	0.53	0.01	0.02	0.09	0.01	2.09	0.60	39
D	Granodiorite	High grade enrichment	0.85	0.07	0.06	0.49	0.01	2.37	8.00	36
E	Granodiorite	Mixed	0.49	0.08	0.10	0.15	0.01	2.51	0.08	62
F	Granodiorite	Primary mineralization	0.49	0.02	0.02	0.11	0.01	1.07	0.12	36
G	Diorite	High grade enrichment	0.78	0.09	0.08	0.39	0.01	4.12	5.85	42
H	Diorite	Mixed	0.54	0.03	0.03	0.16	0.01	3.28	2.53	40
I	Diorite	Primary mineralization	0.43	0.02	0.02	0.07	0.01	3.82	2.86	65
J	Breccia		0.53	0.05	0.04	0.17	0.01	3.60	2.17	107
K	Porphyritic Dacite		0.59	0.05	0.05	0.18	0.01	3.58	2.84	62

Mineralogical examination of the composite samples showed that the main copper mineral was chalcopyrite. A few composite samples contained large proportions of chalcocite and covellite. Three samples contained small amounts of tennantite/tetrahedrite and one sample contained 7% enargite.

Rougher flotation tests were carried out at two different grind sizes with P80 100 µm and 150 µm. The best results were obtained at the finer grind using Aeropromoter 3477 and diesel as a molybdenite promoter. Table 13.8 shows the results for Composite A. This was a relatively high-grade composite sample at over 0.9% Cu.

Table 13.8: Summary of Rougher Flotation Test Composite A

Test	Conditions						Calculated Head		Rougher Concentrate			
	Reagent (g/t)					P_{80} Grinding			Grade		Recovery	
	Collector			Frothers		µm	Cu	Mo	Cu	Mo	Cu	Mo
	3477	SIPX	Diesel	MIBC	DF-250 /MIBC		(%)	(%)	(%)	(%)	(%)	(%)
A1	40	-	-	10	-	100	0.94	0.014	9.60	0.130	94.40	81.200
A2	40	-	-	10	-	150	0.98	0.014	9.40	0.110	88.90	74.600
A3	40	-	15	10	-	100	0.97	0.014	9.60	0.140	94.20	93.700
A4	20	20	-	-	10	100	0.96	0.014	9.60	0.130	94.50	87.000
Average							0.96	0.014	9.50	0.128	93.00	84.100
Maximum							0.98	0.014	9.60	0.140	94.50	93.700
Minimum							0.94	0.014	9.40	0.110	88.90	74.600
Standard Deviation							0.02	0.00	0.13	0.010	2.74	8.150

The OCT consisted of rougher flotation, first cleaner, scavenger and second cleaner. The grind size P80 was 100 µm.

Table 13.9 shows the results of the tests for composites J and K (two tests for each composite). It was noted that the finer regrind product (P80 60 µm) generated a lower grade copper concentrate at the second cleaning stage. Tests with regrind size P80 28 µm resulted in concentrate copper grades of 30%. Table 13.9 also incorporates an estimate of recoveries for locked cycle testwork.

Table 13.9: Summary of Cleaner Flotation Result – Samples J and K

Test	Sample	Conditions	Head, Calc.		Rougher Concentrate				1st Cleaner Concentrate				2nd Cleaner Concentrate			
		P_{80}	Grade		Grade		Recovery		Grade		Recovery		Grade		Recovery	
		Regrind	Cu	Mo	Cu	Mo	Cu	Mo	Cu	Mo	Cu	Mo	Cu	Mo	Cu	Mo
		(µm)	(%)	(%)	(%)	(%)	(%)	(%)	(%)	(%)	(%)	(%)	(%)	(%)	(%)	(%)
J5	J	28	0.54	0.009	6.40	0.10	93.70	89.60	21.00	0.29	88.20	75.00	29.90	0.39	85.00	67.50
J6		60	0.48	0.010	6.20	0.12	92.10	90.40	15.60	0.31	89.20	87.30	20.90	0.41	86.60	84.50
K5	K	28	0.58	0.007	9.30	0.10	94.50	86.20	23.70	0.23	89.50	74.10	30.40	0.28	86.30	67.10
K6		60	0.62	0.007	9.10	0.10	92.80	86.60	21.40	0.22	89.10	80.10	28.50	0.28	86.50	75.80
Average			0.56	0.008	7.76	0.11	93.30	88.20	20.40	0.26	89.00	79.10	27.40	0.34	86.10	73.70
Maximum			0.62	0.010	9.31	0.12	94.50	90.40	23.70	0.31	89.50	87.30	30.40	0.41	86.60	84.50
Minimum			0.48	0.007	6.23	0.10	92.10	86.20	15.60	0.22	88.20	74.10	20.90	0.28	85.00	67.10
Standard Deviation			0.06	0.002	1.67	0.01	1.05	2.11	3.43	0.04	0.56	6.06	4.42	0.07	0.74	8.23
					Calculated Recovery (%) Cleaning Circuit (Locked Cycle)			Cu	97.30							
								Mo	89.40							

Five locked cycle tests using the same test circuit described for the OCT were completed. These tests were carried out using four samples obtained from mixing the 11 previous composites. The composites of the four samples tested are presented in Table 13.10.

Table 13.10: Composites Used in Locked Cycle Tests

Composite	Composite Represents
ADG	High Grade Enrichment Ore
BEH	Mixed Ore
CFI	Primary Mineralisation Ore
JK	Breccia and Dacitic Rock Type

The locked cycle test conditions were similar to those used for the OCTs. The main difference was that a small amount of sodium cyanide (NaCN) was added to the cleaning stage to suppress pyrite in two of the composite samples (JK and CFI). The regrind size P80 was 28 µm.

Table 13.11 shows the results of the locked cycle tests. Sample CFI (primary mineralization) was tested twice and did not generate a composite of commercial value.

Table 13.11: Summary of Locked Cycle Tests

Test	Sample	Head, Calc		Global			
				Grade		Recovery	
		Cu (%)	Mo (%)	Cu (%)	Mo (%)	Cu (%)	Mo (%)
Cycle 1	ADG	0.97	0.012	36.80	0.260	90.80	51.700
Cycle 1	BEH	0.57	0.009	27.50	0.370	88.00	74.300
Cycle 1	JK	0.59	0.007	31.40	0.290	92.10	72.500
Cycle 1	CFI	0.51	0.013	18.90	0.510	77.20	83.700
Cycle 2	CFI - R	0.49	0.010	19.10	0.560	79.20	80.500
Average		0.63	0.010	26.70	0.398	85.50	72.500
Maximum		0.97	0.013	36.80	0.560	92.10	83.700
Minimum		0.49	0.007	18.90	0.260	77.20	51.700
Standard Deviation		0.20	0.000	7.80	0.130	6.83	12.500

Apart from composite ADG, the head grades for the composites were in the range of 0.5% Cu to 0.6% Cu.

Although somewhat variable, the results can generally be regarded as satisfactory, apart from composite CFI, where both the final concentrate grade and recovery were poor.

Impurity analyses of the final concentrates showed that there were some impurities above the limit normally permitted without incurring penalties (0.024% Sb, 0.24% As). It was recommended that this should be reviewed in future testwork.

NaCN addition in rougher tests to depress pyrite may have produced the low copper recoveries noted in the locked cycle tests.

13.1.4 2008 Testwork

The metallurgical test programme conducted in 2008 was completed by SGS Minerals Services (Santiago) for Los Andes Copper.

A total of 6 samples were received in April 2008 and a further 5 samples were received in October 2008. These samples were coarse rejects from drilling campaigns.

Table 13.12 presents the chemical analysis of the eleven samples received. The samples had a copper grade between 0.34% and 0.76% and a molybdenum grade between 0.008% and 0.023%.

Table 13.12: Chemical Analysis of 11 Samples Received

Shipping	Sample	Cu (%)	Mo (ppm)	Fe (%)	S (%)	As (ppm)
1st Shipment	32825	0.57	0.010	3.20	2.50	0.008
	32826	0.37	0.021	3.00	1.00	0.005
	32827	0.66	0.013	2.10	1.40	0.021
	32828	0.53	0.015	3.70	1.50	0.007
	32829	0.43	0.008	2.00	1.20	0.005
	32830	0.76	0.011	4.80	1.30	0.006
3rd Shipment	43522	0.34	0.023	2.80	1.20	0.005
	43523	0.60	0.010	1.90	1.20	0.008
	43524	0.48	0.011	3.60	1.20	0.005
	43525	0.41	0.010	2.10	1.50	0.005
	43526	0.64	0.012	3.50	1.30	0.005
	Average	0.53	0.013	3.00	1.40	0.007
	Maximum	0.76	0.023	4.80	2.50	0.021
	Minimum	0.34	0.008	1.90	1.00	0.005
	Standard Dev.	0.13	0.005	0.90	0.40	0.005

Rougher flotation tests were carried out on the first set of samples delivered at three different grind sizes with P80 of 100 µm, 130 µm and 160 µm.

Table 13.13 and Table 13.14 summarize the results obtained in the rougher kinetics flotation tests with 12 minutes flotation time.

These tests carried out in 2008 showed that very good rougher recoveries in the range of 94% to >98% were achieved for all the composite samples. The grind had significantly less effect on the rougher recovery compared to previous testwork. The recoveries were less than 1% lower for the coarser grind than for the finer grind.

Rougher grades were lower than those from previous testwork in the range of 3% Cu to 6% Cu, but still sufficiently high to allow upgrading to commercial grade concentrates.

Molybdenum rougher recoveries were also very good at up to 96% Mo (range 88% to 96%), although the rougher grades were relatively low due to the lower head grades.

Table 13.13: Cumulative Cu Recoveries and Grades at Different P80

Sample	Copper					
	100 µm		130 µm		160 µm	
	Recovery (%)	Grade (%)	Recovery (%)	Grade (%)	Recovery (%)	Grade (%)
32825	98.60	4.00	98.30	5.30	98.30	5.40
32826	98.10	3.30	98.10	2.90	97.60	3.80
32827	97.80	6.00	97.40	5.70	97.40	5.90
32828	97.10	3.90	96.80	3.60	97.10	4.10
32829	98.50	4.00	98.60	4.20	97.70	4.60
32830	94.60	6.70	94.10	6.00	94.30	6.70
Average	97.50	4.70	97.20	4.60	97.10	5.10
Maximum	98.60	6.70	98.60	6.00	98.30	6.70
Minimum	94.60	3.30	94.10	2.90	94.30	3.80
Standard Dev.	1.50	1.40	1.70	1.20	1.40	1.10

Table 13.14: Cumulative Mo Recoveries and Grades at Different P80

Sample	Molybdenum					
	100 µm		130 µm		160 µm	
	Recovery (%)	Grade (%)	Recovery (%)	Grade (%)	Recovery (%)	Grade (%)
32825	87.60	0.101	73.00	0.053	93.00	0.127
32826	96.30	0.242	96.50	0.218	96.30	0.266
32827	94.10	0.134	94.20	0.129	89.50	0.139
32828	90.90	0.153	95.40	0.141	87.50	0.160
32829	91.40	0.108	91.50	0.106	90.50	0.111
32830	92.30	0.102	92.30	0.093	92.10	0.095
Average	92.10	0.14	90.50	0.123	91.50	0.150
Maximum	96.30	0.242	96.50	0.218	96.30	0.266
Minimum	87.60	0.101	73.00	0.053	87.50	0.095
Standard Dev.	3.00	0.054	8.80	0.056	3.10	0.061

Open circuit cleaning tests were carried out on the first six samples delivered to SGS. These were completed at two regrind sizes with P80 of 35 µm and 50 µm and included two stages of cleaning. An initial grind size P80 of 160 µm was used for all tests.

The results of these tests were disappointing and produced poor grades even after two stages of cleaning and were in the range of 14% Cu to 24% Cu for the finer regrind. The grades at the coarser regrind were even lower. The results of these tests are shown in Table 13.15 and Table 13.16 which also show estimates of recoveries for the locked cycle circuit.

The conditions for cleaner stages were changed (a finer regrind with P80 of 25 µm and three stages of cleaning) and tested on four samples. There was generally an improvement in recovery and concentrate grade as shown in Table 13.17.

Table 13.15: Copper and Molybdenum Recovery, Regrind at 35 µm – Two Cleaner Stages

Test	Sample	Regrind of 35 µm						Regrind of 35 µm					
		Cu recovery (%)			Cu grade (%)			Mo recovery (%)			Mo grade (%)		
		Rougher	1st Cleaner	2nd Cleaner	Rougher	1st Cleaner	2nd Cleaner	Rougher	1st Cleaner	2nd Cleaner	Rougher	1st Cleaner	2nd Cleaner
1	32,825	97.1	97.9	91.2	7.9	16.6	19.8	89.8	96.8	86.3	0.1	0.3	0.3
2	32,826	96.1	97.5	92.4	5.5	12.0	14.2	94.9	98.1	92.0	0.3	0.7	0.8
3	32,827	96.3	96.2	90.2	8.4	18.4	21.2	92.7	95.7	87.9	0.2	0.4	0.4
4	32,828	96.0	95.6	90.7	4.2	12.4	16.5	93.7	94.7	87.0	0.1	0.3	0.4
Average		96.1	96.2	89.4	6.0	15.2	19.0	88.8	95.2	86.2	0.1	0.4	0.4
Maximum		97.8	97.9	92.4	8.4	18.4	24.4	94.9	98.1	92.0	0.3	0.7	0.8
Minimum		93.4	92.4	80.2	4.2	12.0	14.2	73.1	89.7	73.6	0.1	0.2	0.3
Standard Dev.		1.5	2.0	4.6	1.8	2.8	3.6	8.0	2.9	6.5	0.1	0.2	0.2
Calculated Recovery (%) Cleaning Circuit (Locked Cycle)		99.1						93.0					

Table 13.16: Copper and Molybdenum Recovery, Regrind at 50 µm – Two Cleaner Stages

Test	Sample	Regrind of 50 µm						Regrind of 50 µm					
		Cu recovery (%)			Cu grade (%)			Mo recovery (%)			Mo grade (%)		
		Rougher	1st Cleaner	2nd Cleaner	Rougher	1st Cleaner	2nd Cleaner	Rougher	1st Cleaner	2nd Cleaner	Rougher	1st Cleaner	2nd Cleaner
7	32825	97.10	98.00	93.00	7.80	16.10	18.80	90.10	97.10	91.60	0.13	0.27	0.31
8	32826	97.20	97.60	93.70	5.40	13.50	17.00	94.90	98.00	93.90	0.29	0.72	0.91
9	32827	96.40	93.90	83.70	8.10	19.10	21.80	92.90	93.70	83.90	0.16	0.38	0.44
10	32828	96.40	95.00	87.70	4.10	13.20	16.90	93.60	93.70	81.20	0.11	0.34	0.40
11	32829	96.40	97.80	93.60	4.80	13.30	17.00	88.50	96.20	89.80	0.09	0.24	0.29
12	32830	93.00	95.40	84.70	3.90	13.50	20.60	84.60	95.60	83.80	0.06	0.19	0.29
Average		96.10	96.30	89.40	5.70	14.80	18.70	90.70	95.70	87.40	0.14	0.36	0.44
Maximum		97.20	98.00	93.70	8.10	19.10	21.80	94.90	98.00	93.90	0.29	0.72	0.91
Minimum		93.00	93.90	83.70	3.90	13.20	16.90	84.60	93.70	81.20	0.06	0.19	0.29
Standard Dev.		1.50	1.80	4.60	1.80	2.40	2.10	3.80	1.80	5.10	0.08	0.19	0.24
Calculated Recovery (%) Cleaning Circuit (Locked Cycle)		99.2						96.9					

Table 13.17: Copper and Molybdenum Recovery, Regrind at 25 µm – Three Cleaner Stages

Test	Sample	Regrind of 25 µm								Regrind of 25 µm							
		Cu recovery (%)				Cu grade (%)				Mo recovery (%)				Mo grade (%)			
		Rougher	1st Cleaner	2nd Cleaner	3rd Cleaner	Rougher	1st Cleaner	2nd Cleaner	3rd Cleaner	Rougher	1st Cleaner	2nd Cleaner	3rd Cleaner	Rougher	1st Cleaner	2nd Cleaner	3rd Cleaner
13	32825	97.40	93.20	81.10	69.20	4.70	16.60	27.10	31.60	90.80	89.90	78.20	67.40	0.09	0.30	0.47	0.54
14	32826	97.90	93.20	92.80	69.30	3.70	14.10	18.20	22.20	95.90	94.20	91.10	69.80	0.20	0.70	0.90	1.10
15	32827	95.70	91.30	89.80	82.60	9.90	27.00	32.80	34.50	92.50	88.70	85.10	81.20	0.20	0.50	0.60	0.60
16	32830	91.10	87.90	79.20	66.50	5.20	19.90	23.60	25.20	88.80	87.80	76.50	67.30	0.10	0.30	0.30	0.40
Average		95.50	91.40	85.70	71.90	5.90	19.40	25.40	28.40	92.00	90.20	82.70	71.40	0.14	0.45	0.58	0.66
Maximum		97.90	93.20	92.80	82.60	9.90	27.00	32.80	34.50	95.90	94.20	91.10	81.20	0.19	0.72	0.92	1.13
Minimum		91.10	87.90	79.20	66.50	3.70	14.10	18.20	22.20	88.80	87.80	76.50	67.30	0.07	0.28	0.32	0.35
Standard Dev.		3.10	2.50	6.60	7.30	2.70	5.60	6.20	5.70	3.00	2.80	6.70	6.60	0.06	0.21	0.25	0.33
Calculated Recovery (%) Cleaning Circuit (Locked Cycle)		98.0								94.6							

13.1.4.1 2017-2018 Flotation Testwork

During 2017 and 2018, 80 variability samples of approximately 50 kg were selected using geological criteria such as lithology (andesitic and breccia) and grade (high FeS2, high Mo and low grade). These samples were sent to SGS to obtain comminution and flotation parameters for geometallurgical characterization. Five composites were assembled using the variability samples (Andesitic 2017, High FeS_2, High Mo, First Years, Breccia, Andesitic 2018 and Hypogene Low Grade).

The 2017-2018 metallurgical programme included the following tests:

- Feed characterization: specific gravity, chemical (volumetric and FRX) and mineralogical (QEMSCAN BMA) analysis
- Rougher flotation testwork at different P80 grinding sizes for all samples
- Settling and environmental tests (ABA and NAG) for rougher tailings
- Open and locked cycle flotation tests for composites.

Table 13.18 to Table 13.20 summarize the flotation test results for the composites. This testwork was carried out using fresh water and Xanthate (C3330), dithiophosphate (AP3477), diesel and MIBC as reagents. Slurry was prepared at 38% solids and pH 10.5. This is referred to as the PEA reagent formula.

Testwork results from samples Andesitic 2017, High FeS_2 and High Mo were used in the preparation of the 2019 Preliminary Economic Assessment Report.

Table 13.18: Summary of Rougher Flotation Results 2017-2018 Testwork (PEA Reagent Formula)

Sample	Andesitic 2017	High FeS2	High Mo	Andesitic 2018	Breccia	First Years	Hypogene Low Grade
Testwork	2017	2017	2017	2018	2018	2018	2018
Cu in Head (%)	0.64	0.34	0.53	0.42	0.46	0.45	0.31
Mo in head (ppm)	150	230	330	210	230	140	80
Cu Rec. (210 µm)	94.7	96.2	97.4	94.3	94.5	91.2	95.7
Mo Rec. (210 µm)	87.7	91.9	94.3	91.1	83.9	79.2	88.8
Cu Rec. (240 µm)	-	-	-	92.7	93.9	91.5	95.2
Mo Rec. (240 µm)	-	-	-	90.6	86.2	86	90.4

Table 13.19: Summary of Open Cycle Flotation Results 2017-2018 Testwork

Sample	Andesitic 2017	High FeS2	High Mo	High Mo	Andesitic 2018	Breccia	First Years	Hypogene Low Grade
Testwork	2017	2017	2017	2017	2018	2018	2018	2018
Cu in head (%)	0.65	0.34	0.52	0.34	0.41	0.48	0.47	0.32
Mo in head (ppm)	140	220	320	360	200	220	130	110
Grinding P80 (µm)	210	210	210	240	280	280	280	280
Rougher Rec. Cu	94.8	95.8	97.5	96.6	91.8	92.8	88.2	94
Rougher Rec. Mo	86.7	87.4	88.5	94.8	91	84.6	84.6	90.8
Regrinding time	16'01"	12'07"	13'44"	13'44"	11'48"	13'07"	13'40"	11'45"
Regrinding P80 (µm)	45	45	45	45	45	45	45	45
Cleaner Rec. Cu	98.4	99.4	99.5	99.4	99.5	99.3	98.5	99.4
Cleaner Rec. Mo	56.7	94.3	85.4	96.1	97.3	96.4	94.7	96.8
Overall Rec. Cu	93.3	95.3	97	96	91.3	92.1	86.9	93.4
Overall Rec. Mo	49.1	82.5	75.6	91.1	88.6	81.5	80.1	87.9

Table 13.20: Summary of Locked Cycle Flotation Results 2017-2018 Testwork

Sample	Andesitic 2017	Andesitic 2017	Andesitic 2017	High FeS2	High FeS2	High Mo	Breccia
Testwork	2017	2017	2017	2017	2017	2017	2018
Cleaner Stages	1	1	1 without regrinding of scavenger concentrate	1	2	2	3
Cleaner pH	11.5	11.5	11.5	11.5	12.0	11.5	11.5
Cu in head, %	0.65	0.64	0.66	0.34	0.34	0.53	0.47
Mo in head, ppm	140	140	150	230	250	320	240
Grinding P80 (µm)	240	210	240	240	240	240	240
Rougher Rec. Cu	93.9	95.1	93.7	95.2	95.6	96.7	93.6
Rougher Rec. Mo	87.1	87.2	86.6	91.9	91.8	90.0	89.4
Overall Rec. Cu	92.8	94.5	92.8	94.8	94.9	96.3	91.4
Overall Rec. Mo	66.3	36.5	75.0	88.3	85.3	84.3	62.2
Overall Mass Pull	1.91	1.89	2.22	1.34	1.13	1.93	1.33
Cu in Final Conc.	31.70%	31.90%	27.50%	23.80%	28.40%	26.30%	32.50%
Mo in Final Conc.	0.29%	0.16%	0.47%	1.40%	1.33%	1.42%	0.67%

13.2 2020 Laboratory Testwork Programme

The main objective of the testwork was to obtain the metallurgical parameters that represent the first years of operation, defining the process design criteria and supporting the economic evaluation during the payback period. All tests were performed at SGS Minerals in Santiago, Chile, with exception of the pressure bed test (performed in Metso Laboratories in York, PA, USA) and the pressure and vacuum filtration tests for tailings (performed at Takraf, ex Tenova Delkor, in Santiago, Chile).

Table 13.21: Summary of the Metallurgical Testwork

Stage	Objective	Test
Ore Characterization	To determine ore bearing minerals, study liberation and define a proper reagent formula for the flotation stage.	Chemical Analysis and FRX
		Optical Microscopy
		QEMSCAN (BMA and PMA)
		Natural pH
		Specific Gravity
Comminution Parameters	To obtain metallurgical parameters for sizing and selection of crushers, HPGR and ball mills. Estimate energy and steel media consumption.	SMC Test
		Bond Abrasion Index
		Bond Ball Work Index
		PBT Test
		Grinding Kinetics
Flotation	To assess different types of water and to obtain recovery values for Cu and Mo.	Rougher Flotation
		Regrinding Kinetics
		Cleaner Flotation Kinetics
		Locked Cycle Test
		Selective Cu-Mo Flotation
		Final Concentrate Analysis
Water Recovery	Obtain parameters to calculate a water balance and estimate the water make-up.	Sample Preparation
		Particle Size Distribution Analysis
		Settling Test
		Pressure Filtration
		Vacuum Filtration
		Atterberg Limits of Filtered Tailings
Environmental	Measure acid generation from tailings	ABA and NAG
		TCLP

The methodologies used for chemical assays are described in Table 13.22.

Table 13.22: Summary of Chemical Analysis Methodologies Used in the Testwork Programme

Analyte	Methodology	Detection Limit
Cu < 5%	2g in 100mL / 15 HNO3,5 HCLO4 dry / 10 % HCL	0.00%
Cu > 5%	Short Volumetric Analysis	0.07%
Soluble Cu	1g in 100mL / 50 Citric Acid 1M / 1h 180 rpm / Centrifugate	0.00%
Fe	1g in 100mL / 10 HNO3, 5 HCLO4 dry / 10% HCL	0.01%
Mo	2 g in 100mL / 15 HNO3,5 HCLO4 dry / 10 % HCL	0.00%
Au	30g in 10ml / Fire Assay – Atomic Abs. 40% Aqua Regia	0.03 g/t
Ag	1g in 100 mL / 5 HF Repose / 5 HNO3 ,15 HCl,5 HCLO4 / 25% HCl	1 ppm
As	1g in 100mL / 10 HNO3,5 HCLO4,1 H2SO4 / Water 1% H2SO4	0.00%
Sb	1g in 100mL / 15 HCL, 5 HNO3 Syrup / 25% HCl	0.01%
Bi	1g in 100mL / 15 HCL, 5 HNO3 Syrup / 25% HCl	0.01%
S	Infrared Fusion (IR) LECO	0.01%
Hg	3mL HNO3 + 1mL HCl / Water Bath for 3h 60°C Agitate Each 30 Min	0.15 g/t
FRX	X-Ray Fluorescence For Optical Mineralogy	

13.2.1 Sample Selection

The metallurgical samples were selected from quartered drill core based on geological, spatial and timing defined by the mine plan. Composite samples were defined at this stage to study the response of the lithologies that are present in more than 5% in the mine plan and in approximately the first 10 years of the pit at a feed rate of 136 ktpd (N16Y and N712Y) to define conditions for further variability studies.

Table 13.23: Description of Composite Samples Used in the Metallurgical Testwork

Sample	Represents	Mass	Composition
AND	Andesite Lithology of Southern Area	90 kg	¼ LAV HQ drill cores
BXI	Breccia Lithology of Southern Area	90 kg	¼ LAV HQ drill cores
DIO	Diorite Lithology of Southern Area	90 kg	¼ LAV HQ drill cores
TON	Tonalite Lithology of Southern Area	90 kg	¼ LAV HQ drill cores
N16Y	Approx. First 5 Years of Northern Area	300 kg	¼ V2015-17 drill cores
N712Y	Approx. Years 6-10 of Northern Area	300 kg	¼ V2015-17 drill cores
NORTH	Blend of N16Y and N712Y	61 kg	-
SOUTH	Blend of AND, BXI, DIO, TON	20 kg	-

In further stages of the testwork programme, synthetic composites were prepared to represent the northern and southern areas. These composites were prepared by combining the composite samples above in the same proportion as found in the mine plan. Those composites were denominated NORTH and SOUTH, and they were used in the cleaner, locked cycle and molybdenum flotation tests.

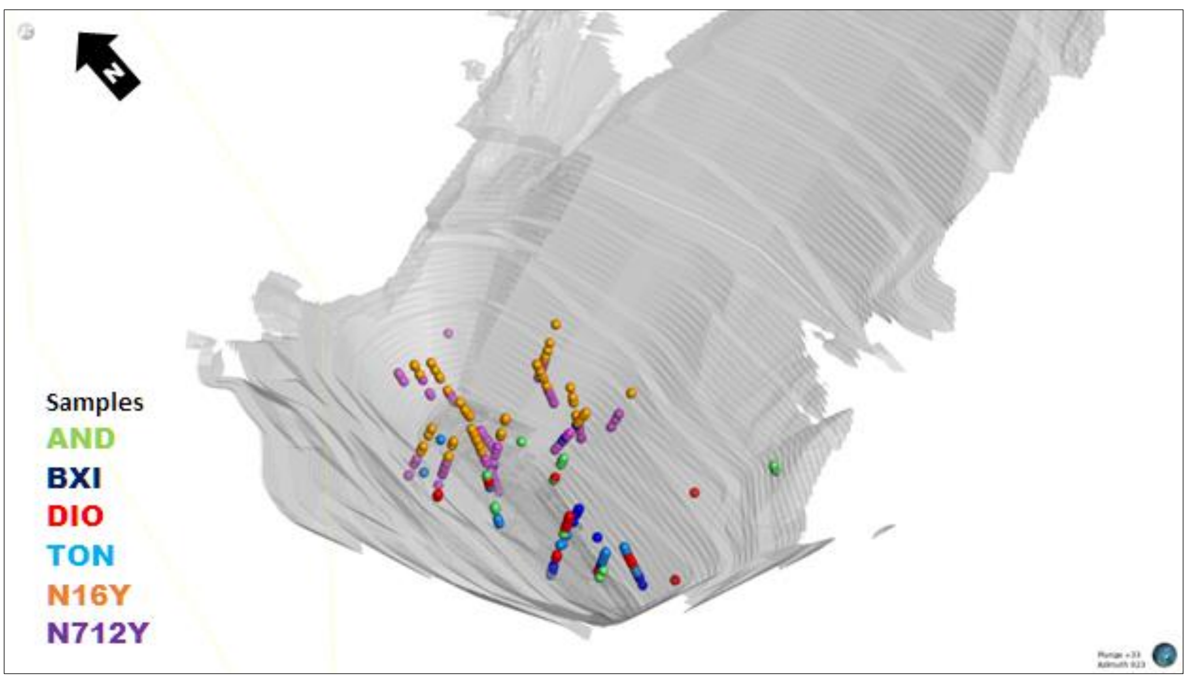
13.2.2 Sample Selection Methodology

Composites were prepared from individual drill core samples. To ensure the representativity of the metallurgical samples the following criteria were followed.

- Correspond to the first years of mine plan
- Drill core samples grades are above the cut-off grade (Cu >0.18%)
- Individual drill core samples matched the Cu grade distribution defined by the blocks contained in the first 10 years of the mine plan
- Individual core samples covered the entire volume defined in the first years of the mine plan
- N16Y and N712Y are defined by all drill cores belonging to the volume north of geological section N°1500.

Figure 13.1 shows the spatial location of the individual drill core samples contained in each Composite.

Figure 13.1: Individual Drill Core Samples in the 10 Year Pit



Source: Los Andes Copper, 2021

13.2.3 Sample Selection Validation

To validate the sample selection methodology Cu and Mo grades were compared between the values estimated from the mine plan (block model), the drill core database (individual drill core samples) and the chemical assays performed upon composites. Two statistical measurements were calculated:

- The variation coefficient that represents the "grade variability" of each sample
- The weighted average grade for each case.

If the weighted average grade is substantially the same as the sample assay grade, then the grade distribution of the mine plan and each individual core section also match.

Table 13.24 and Table 13.25 summarize the validation procedure for the samples considered in the PFS testwork. Both tables show that Cu and Mo results are as expected from the results of the chemical assays and that the composites represent the variability of the material (the coefficients of variation are similar between the individual core sections and those observed in the mine plan).

Table 13.24: Validation of Copper Grade (%) in Samples

Sample	Mine Plan			Drill Core Database			Sample
	Weighted Average	Std. Dev	Coeff. of Variation	Weighted Average	Std. Dev	Coeff. of Variation	Assay SGS
AND	0.37	0.14	0.38	0.37	0.12	0.31	0.37
BXI	0.46	0.14	0.30	0.45	0.12	0.28	0.43
DIO	0.46	0.15	0.32	0.43	0.14	0.32	0.45
TON	0.33	0.11	0.32	0.33	0.09	0.28	0.32
N16Y	0.50	0.27	0.53	0.51	0.32	0.62	0.48
N712Y	0.44	0.18	0.40	0.46	0.20	0.43	0.44

Table 13.25: Validation of Molybdenum Grade (ppm) in Samples

Sample	Mine Plan			Sample
	Weighted Average	Std. Dev	Coeff. of Variation	Assay SGS
AND	106	74	0.70	90
BXI	87	72	0.83	70
DIO	74	71	0.96	60
TON	255	278	1.09	190
N16Y	134	167	1.25	114
N712Y	123	130	1.05	102

13.2.4 Mechanical Preparation

Individual drill core sections were identified at the core shed. Trays containing previously half cut HQ diameter drill cores were shipped to a core cutting facility in San Felipe where the core was further quartered for the metallurgical samples. The cutting process was supervised by the Vizcachitas geological team.

After cutting, quarters of drill cores were sealed in plastic bags and labelled with the identification for each composite sample. As soon as this process was completed, the samples were sent to SGS Minerals in Santiago. Table 13.26 shows the date of reception and the mass of each sample in the inventory.

Table 13.26: Sample Inventory

Sample	Mass (Estimated)	Date of Reception	Mass (Inventory)
AND	90 kg	July 8, 2020	61 kg
BXI	90 kg	July 8, 2020	71 kg
DIO	90 kg	July 8, 2020	74 kg
TON	90 kg	July 8, 2020	79 kg
N16Y	300 kg	September 25, 2020	286 kg
N712	300 kg	October 27, 2020	306 kg

There are differences between the estimated and inventoried masses because a higher density was used to calculate each composite mass. A significant number of drill core sections at the shed contained broken rock, thus decreasing the apparent density of each section. This difference was corrected in the selection of samples N16Y and N712Y.

All samples were crushed at 100% -1 inch and particles were selected for SMC testing. After that, samples were crushed at 100% -3/4” to prepare sub-samples for Bond Abrasion and PBT tests. The remaining mass was crushed to 100% -6 mesh where a sub-sample for Bond Ball Work index testing was taken. Finally, the remaining sample was crushed to 100% -10 mesh and fed to a rotary cutter to form batches of 0.75 kg for feed characterization and flotation testwork.

Mass losses were logged in each crushing and classification stage. Particle size distribution and mass were measured in the 0.75 kg batch preparation stage to ensure that all batches had the same properties.

A further description of the sample preparation of these stages is provided in Section 11 of this report.

13.2.5 Feed Characterization

13.2.5.1 Specific Gravity

The specific gravity of each sample was estimated by volume displacement using a helium gas pycnometer. Results show that the specific gravity of the six samples are similar, and that the weighted average of all the samples is 2.74 (Table 13.27).

Table 13.27: Specific Gravity Results

Sample	Specific Gravity
AND	2.76
BXI	2.71
DIO	2.71
TON	2.69
N16Y	2.78
N712Y	2.75

13.2.5.2 Natural pH

Natural pH was measured after 10 minutes of grinding for each sample in a ceramic mill using ceramic balls and distilled water. The results are shown in Table 13.28.

Table 13.28. Natural pH Measurements

Sample	Natural pH
AND	7.36
BXI	7.43
DIO	7.53
TON	7.73
N16Y	7.22
N712Y	8.05

13.2.5.3 Chemical Analysis

Table 13.29 shows the solubility ratio (soluble copper over total copper) for each sample.

Table 13.29: Copper and Soluble Copper Chemical Analysis

Element	Cu (%)	Sol. Cu (%)	Cu Sol Ratio
AND	0.37	0.027	7.30
BXI	0.43	0.010	2.35
DIO	0.45	0.022	4.94
TON	0.32	0.009	2.82
N16Y	0.48	0.024	5.00
N712Y	0.44	0.011	2.49

Chemical analysis for Cu, Soluble Cu (citric), Fe, Au, Ag, As, Sb, Bi, S and a selection of oxides X-Ray diffraction (FRX) were performed on the six samples, the results are shown in Table 13.30.

No significant traces of deleterious elements such as As, Pb, Bi and Sb were detected in the samples.

Table 13.30: Chemical and FRX Analysis for Head Grades

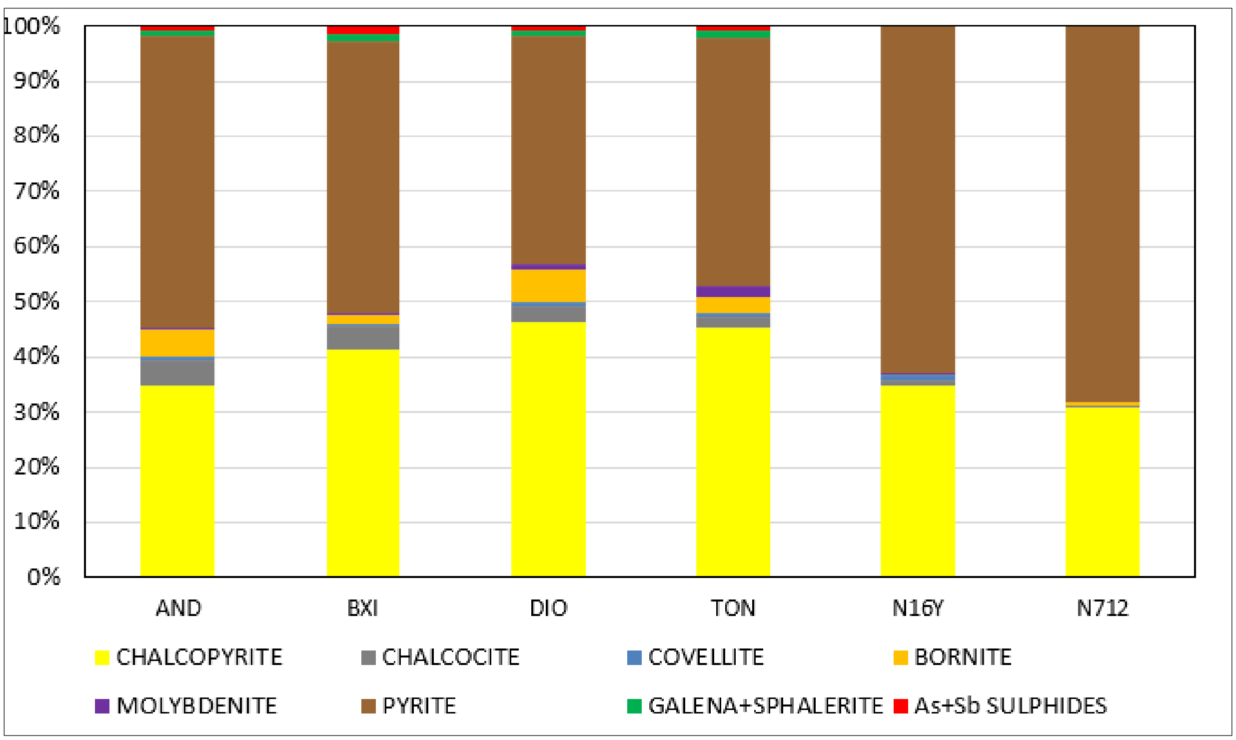
Element	Unit	Tolerance	AND	BXI	DIO	TON	N16Y	N712Y
Cu	%	0.0005	0.37	0.43	0.45	0.32	0.48	0.44
Mo	%	0.0005	0.009	0.007	0.006	0.019	0.0114	0.0102
Sol. Cu	%	0.001	0.027	0.01	0.022	0.009	0.024	0.011
Fe	%	0.02	3.64	2.3	2.08	1.71	3.49	3.3
Au	g/t	0.03	<0.03	<0.03	<0.03	<0.03	<0.03	<0.03
Ag	ppm	1	<1.0	1	2	1	<4.0	<4.0
As	%	0.003	0.007	0.006	0.01	0.009	<0.003	<0.003
Sb	%	0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005
Bi	%	0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005
S	%	0.01	1.05	1.33	0.86	0.87	1.79	2.06
Fe	%	0.01	4.53	2.64	2.43	1.9	4.32	3.74
Fe2O3	%	0.01	6.48	3.77	3.48	2.72	6.18	5.34
Si	%	0.04	27.6	29.7	30.5	31.4	28.1	28.1
SiO2	%	0.09	59.2	63.5	65.3	67.1	60	60.1
Ca	%	0.04	2.18	1.81	1.39	1.33	2.33	2.73
CaO	%	0.06	3.05	2.54	1.94	1.86	3.26	3.81
Mn	%	0.02	0.03	<0.02	<0.02	<0.02	<0.02	<0.02
MnO	%	0.02	0.04	0.02	<0.02	<0.02	0.02	0.03
Al	%	0.02	9.18	8.41	8.62	8.5	9.13	8.82
Al2O3	%	0.04	17.35	15.89	16.3	16.07	17.26	16.66
Ti	%	0.03	0.62	0.37	0.35	0.27	0.58	0.47
TiO2	%	0.05	1.04	0.62	0.58	0.45	0.97	0.79
Mg	%	0.03	1.47	1.1	0.96	0.78	1.42	1.26
MgO	%	0.05	2.44	1.82	1.6	1.3	2.35	2.08
P	%	0.01	0.081	0.042	0.045	0.039	0.1	0.06
P2O5	%	0.02	0.19	0.1	0.1	0.09	0.23	0.15
S	%	0.01	1.1	1.31	0.84	0.85	1.79	2.16
SO3	%	0.02	2.76	3.27	2.11	2.12	4.48	5.39
K	%	0.01	2.51	2.85	2.77	3.08	2.37	2.4
K2O	%	0.01	3.03	3.43	3.34	3.71	2.85	2.89
Zn	%	0.01	0.013	0.01	0.011	<0.010	<0.010	<0.010
ZnO	%	0.01	0.02	0.01	0.01	0.01	<0.01	<0.01
As	%	0.01	<0.01	<0.01	0.012	<0.01	<0.01	<0.01
As2O3	%	0.01	<0.01	<0.01	0.02	0.01	<0.01	<0.01
Pb	%	0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01
PbO	%	0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01
Na	%	0.06	2.57	2.88	2.88	2.82	2.73	2.78
Na2O	%	0.08	3.47	3.88	3.89	3.8	3.68	3.75
Mo	%	0.01	0.01	<0.01	<0.01	0.02	0.01	0.01
MoO3	%	0.01	0.021	<0.010	0.014	0.024	0.02	0.02
LoI	%	0.01	4.07	4.49	3.63	3.17	3.37	4.45

13.2.5.4 Mineralogical Analysis

Samples were milled to 70 mesh to prepare briquettes to identify opaque minerals via optical microscopy. The same briquettes for samples N16Y and N712Y were analyzed by QEMSCAN PMA to observe specific associations and liberation. Results from FRX analyses were used to account and balance grades with mineral species.

Mineralogical analyses are summarized on Figure 13.2 which shows that the Vizcachitas material is composed mainly of primary sulphides, chalcopyrite (30%-46%), pyrite (41%-68%), and less than 10% of secondary sulphides (mainly bornite and chalcocite). Secondary sulphides occur as thin coatings (*patinas*) over the primary copper sulphides.

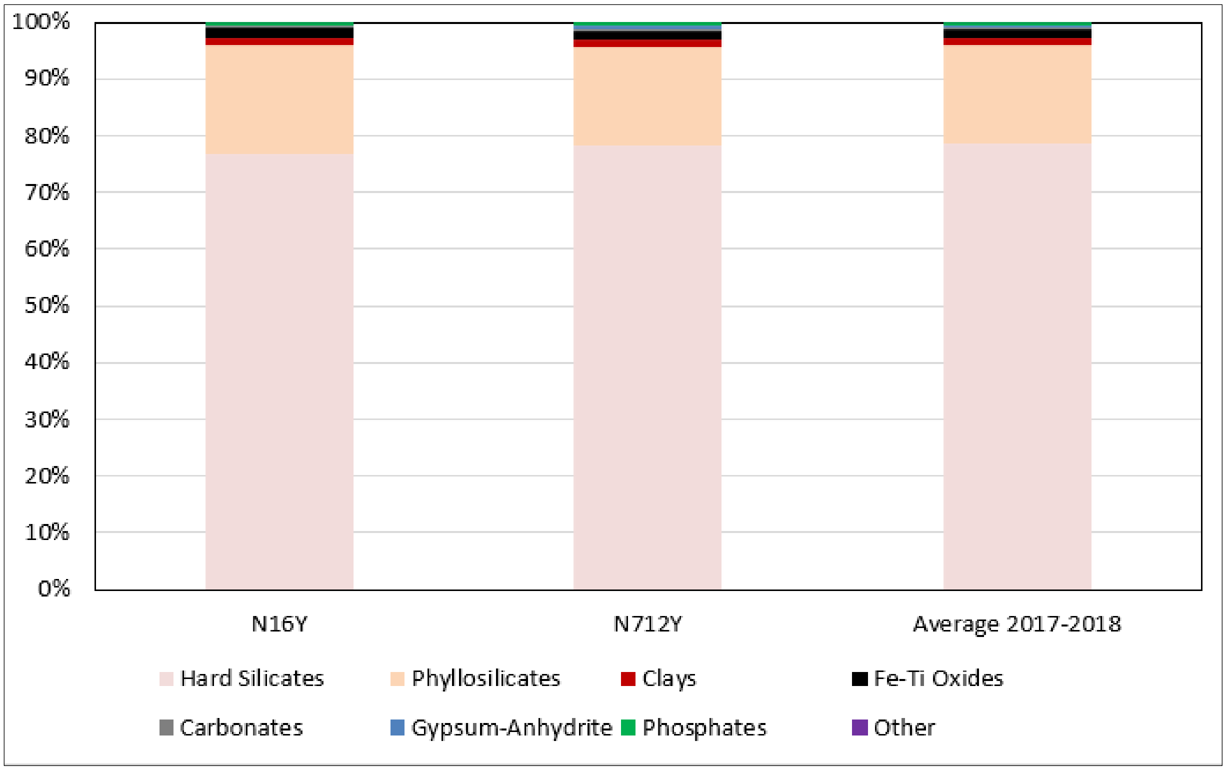
Figure 13.2: Sulphide Mineralogy of PFS Samples



Source: Los Andes Copper, 2021, based on QEMSCAN PMA results from 2020 testwork

The main gangue minerals are 79% hard silicates (quartz, k-feldspar, plagioclase/albite, epidote, tourmaline and amphiboles) and 17% phyllosilicates (sericite/muscovite, biotite and chlorite). Clays, such as montmorillonite and kaolinite, are present at less than 1.3%. Other gangue minerals such as carbonates, gypsum and iron oxides (magnetite, hematite and rutile) occur at less than 3%. Figure 13.3 shows the results for the gangue from samples N16Y and N712Y and the average of the 80 samples.

Figure 13.3: Gangue Composition and Average of 2017-18 Samples (QEMSCAN)



Source: Los Andes Copper, 2021, based on QEMSCAN PMA results from 2017-2018 testwork

According to the QEMSCAN results, copper sulphides and molybdenite are liberated to 83% and 74%, respectively. The secondary copper sulphides are mainly associated with primary copper sulphides. The outer surfaces of chalcopyrite particles have patinas of chalcocite and covellite which are commonly developed in zones above the ground water table.

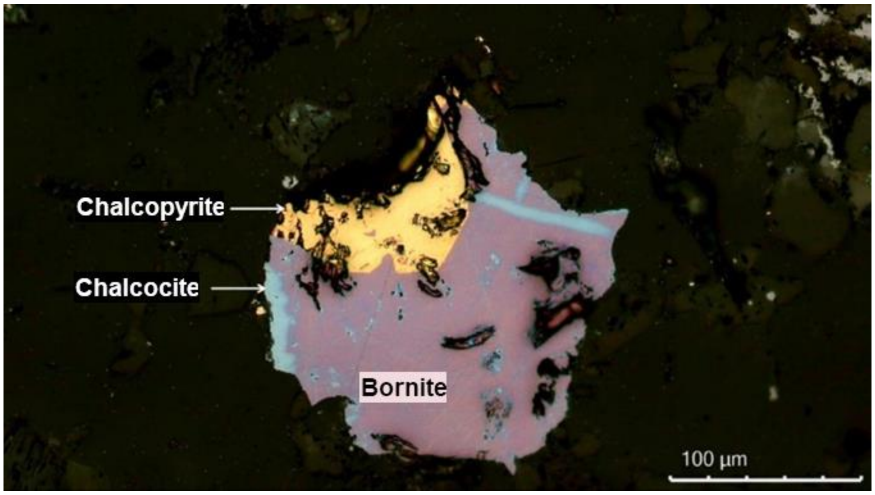
According to the QEMSCAN analysis, almost 80% of copper sulphides are associated with hard silicates and phyllosilicates. Pyrite is mainly found free, only 10% is associated with Cu sulphides.

Approximately, 60% of the non-free gangue (11% of total occurrence) is still recoverable by flotation methods as it has lateral associations on gangue. Hence, recovery losses should be associated to a lack of liberation of occluded and disseminated species which represent roughly 6% of the total of non-free particles.

Sulphide minerals that contain deleterious elements (such as Pb, As and Sb) are present at less than 1% and are mainly associated with pyrite.

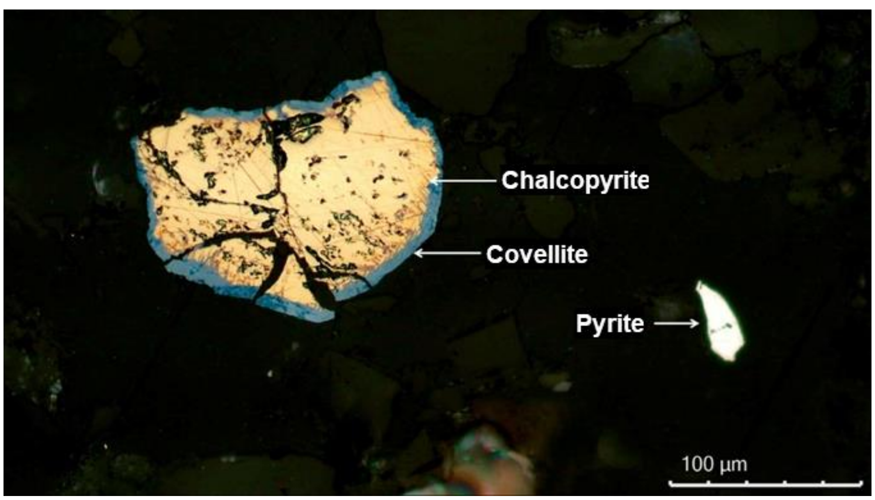
Figure 13.4 to Figure 13.11 show the mineralization in the composite samples.

Figure 13.4: Free Particle of Chalcopyrite-Bornite with Patinas of Chalcocite (AND Sample)



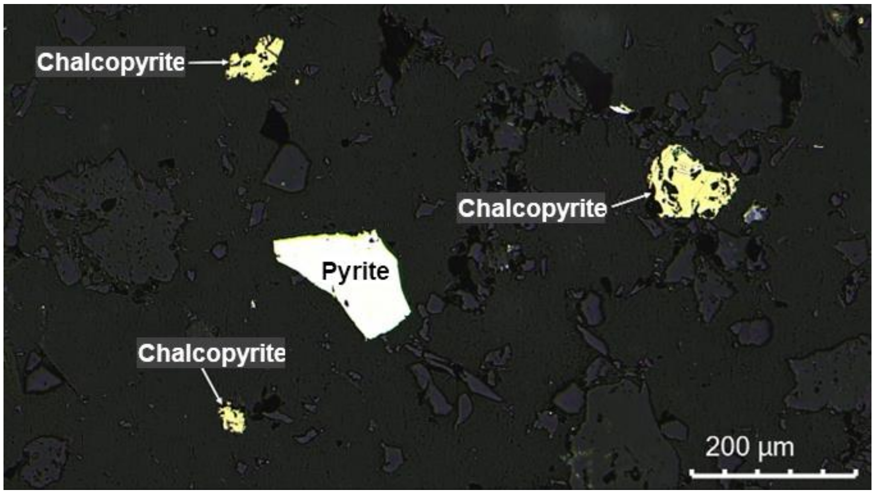
Source: SGS Optical Mineralogical Analysis from 2020 testwork

Figure 13.5: Free Particle of Chalcopyrite with Patinas of Covellite (BXI Sample)



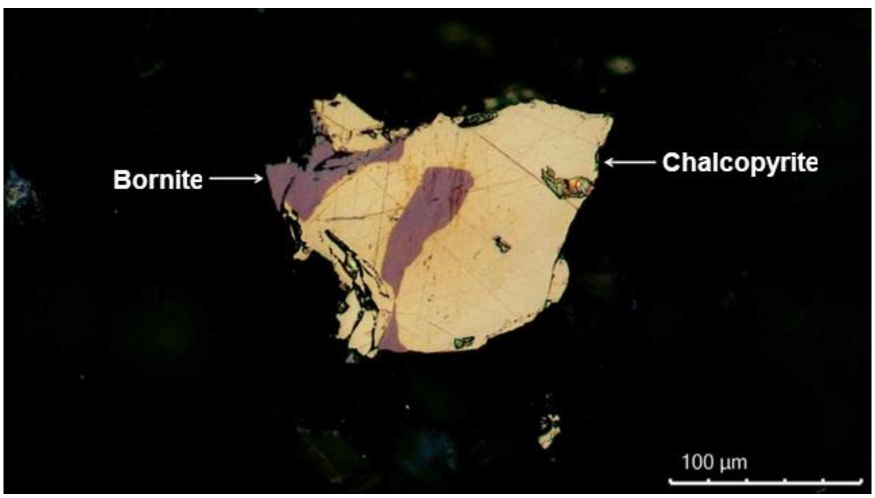
Source: SGS Optical Mineralogical Analysis from 2020 testwork

Figure 13.6: Free Chalcopyrite and a Pyrite Particle (N712Y Sample)



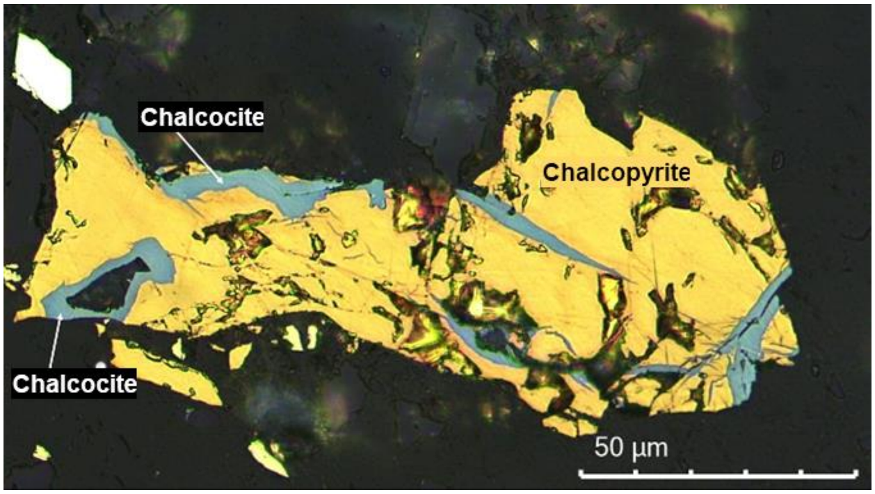
Source: SGS Optical Mineralogical Analysis from 2020 testwork

Figure 13.7: Free Chalcopyrite Particle with Interlocked Bornite (DIO Sample)



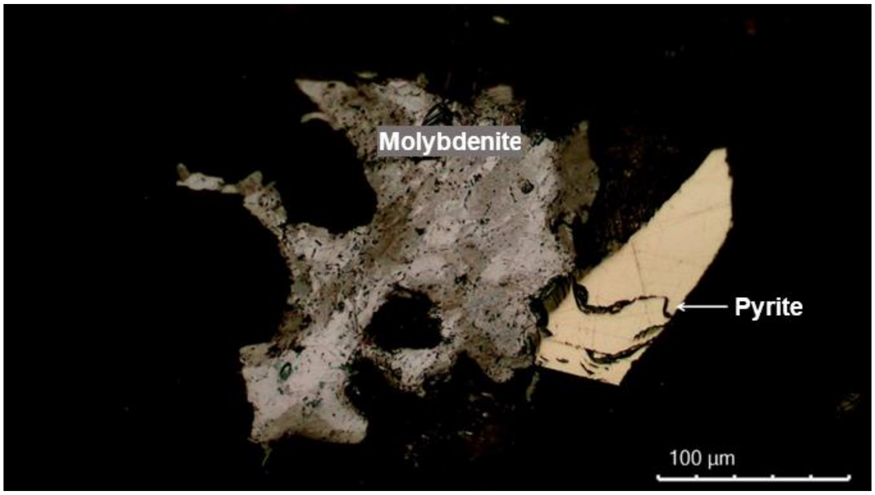
Source: SGS Optical Mineralogical Analysis from 2020 testwork

Figure 13.8: Free Chalcopyrite Particle with Interlocked Bornite (N16Y Sample)



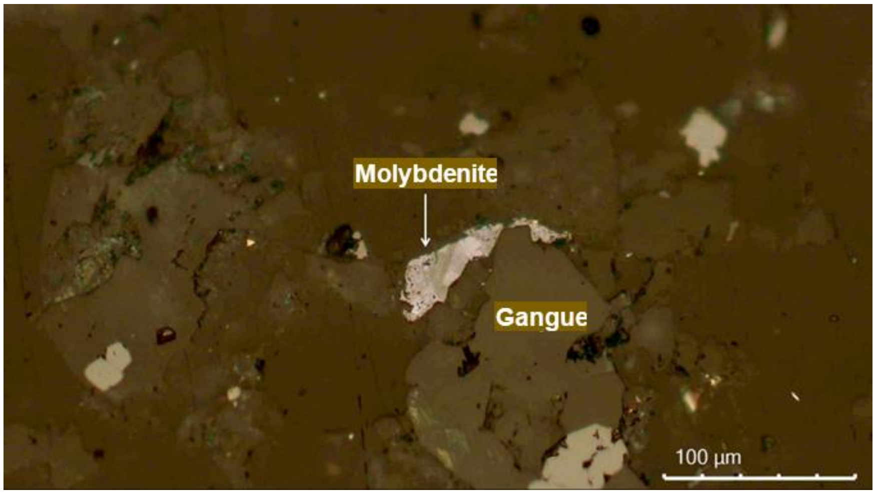
Source: SGS Optical Mineralogical Analysis from 2020 testwork

Figure 13.9: Molybdenite Associated with Pyrite (TON Sample)



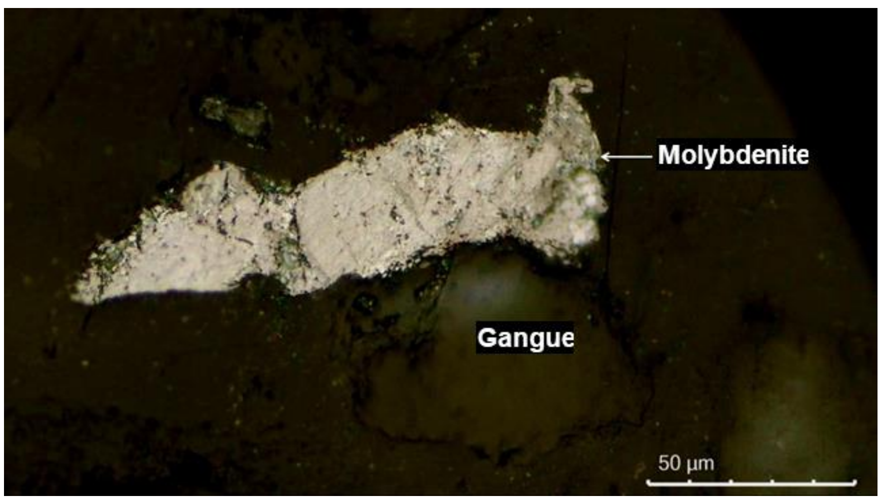
Source: SGS Optical Mineralogical Analysis from 2020 testwork

Figure 13.10: Large Molybdenite Particle Associated with Gangue (N16Y Sample)



Source: SGS Optical Mineralogical Analysis from 2020 testwork

Figure 13.11: Free Molybdenite (TON Sample)



Source: SGS Optical Mineralogical Analysis from 2020 testwork

13.2.6 Comminution Parameters

13.2.6.1 SMC Testing

The SMC (SAG Mill Comminution) test measures the resistance to impact breakage and provides the parameters to estimate the energy consumption of SAG, HPGR and conventional crushing circuits. The PFS samples were tested at SGS Minerals in Chile and the results shown in Table 13.31 and Table 13.32 were validated by JKTech in Australia.

In the historical JKTech database, 70% of the samples are softer than the AND sample and 50% are softer than the BXI, DIO and TON samples. This classifies the Vizcachitas material as medium to moderately hard.

Table 13.31: SMC Test Results

Sample	DWI, kWh/m^3	DWI (%)	Mia	Mih	Mic	SG	A	b	Axb	ta	SCSE
AND	7.3	59	21.6	16.3	8.4	2.61	65.4	0.54	35.3	0.35	10.3
BXI	6	41	18.6	13.4	6.9	2.59	69.2	0.62	42.9	0.43	9.41
DIO	5.5	35	17.7	12.6	6.5	2.54	70.5	0.65	45.8	0.47	9.11
TON	5.7	37	17.9	12.8	6.6	2.57	71.8	0.63	45.2	0.46	9.18
N16Y	7.9	67	23	17.6	9.1	2.62	66.1	0.5	33.1	0.33	10.64
N712Y	7.6	62	22	16.7	8.6	2.64	68.4	0.51	34.9	0.34	10.41

Table 13.32: Crusher Model Specific Energy Matrix

Ecs	14.5			20.6			28.9			41.1			57.8		
T10	10	20	30	10	20	30	10	20	30	10	20	30	10	20	30
AND	0.40	0.87	1.47	0.34	0.76	1.28	0.30	0.66	1.11	0.26	0.57	0.96	0.23	0.50	0.84
BXI	0.33	0.71	1.19	0.28	0.62	1.03	0.25	0.54	0.90	0.21	0.47	0.78	0.19	0.41	0.68
DIO	0.30	0.66	1.10	0.26	0.58	0.96	0.23	0.50	0.84	0.20	0.44	0.72	0.18	0.38	0.63
TON	0.31	0.68	1.12	0.27	0.59	0.97	0.24	0.51	0.85	0.20	0.44	0.73	0.18	0.39	0.64
N16Y	0.43	0.94	1.58	0.37	0.82	1.37	0.32	0.71	1.20	0.28	0.61	1.03	0.25	0.54	0.91
N712Y	0.40	0.88	1.48	0.35	0.77	1.28	0.31	0.67	1.12	0.26	0.58	0.97	0.23	0.51	0.85

13.2.6.2 Metso Pressure Bed Test

Sub-samples of AND, BXI, DIO and TON composites were shipped to Metso laboratories in York, PA, USA to perform pressure bed testing (PBT).

The HRC™ Packed Bed Compression Test was performed using 100 g to 500 g of -13.2 mm size material in a 76.2 mm cylinder. Maximum force and load rate are specified and controlled on a customized hydraulic press. Results are shown in Table 13.33.

The PBT results allow the parameter S1E or breakage rate to be obtained from the Herbst & Fuerstenau Population Balance Model for sizing and selection of HPGR and simulation of the particle size distributions from an HPGR circuit.

The PBT results showed that there were no significant variations of the breakage rates for the samples at a certain specific energy. Metso recommended a conservative value for the breakage rate parameter (S1E of 1.60 for AND) for sizing and selection of the HPGR circuit (Table 13.33).

Table 13.33: PBT Results

Sample	AND			BX1			DIO			TON		
SG	2.77	2.77	2.77	2.73	2.73	2.73	2.72	2.72	2.72	2.73	2.73	2.73
SE, kWh/t	0.99	1.30	1.60	1.00	1.30	1.59	1.00	1.29	1.59	0.99	1.30	1.59
S1E	2.07	1.74	1.60	2.37	1.91	1.69	2.10	1.88	1.66	2.25	1.80	1.62

13.2.6.3 Bond Tests

The Bond test results are shown in Table 13.34. The Bond tests included:

- Abrasion Index to calculate liners and steel media consumption
- Bond Ball Mill Work Index to size and calculate specific energy consumption of ball mills.

Table 13.34: Bond Abrasion Index and Ball Mill Work Index Tests Results

Sample	Abrasion Index, g	Work Index, kWh/t
AND	0.1896	11.66
BXI	0.1589	9.95
DIO	0.1931	9.93
TON	0.2009	10.11
N16Y	-	12.21
N712Y	-	12.28

13.2.6.4 Grinding Kinetics

Grinding kinetic tests were performed in a laboratory scale ball mill operating at 70% solids, 67 rpm and using 10 kg of 1 inch steel balls. Three tests were performed over different times to define a curve to interpolate the desired P80 for grinding within a range of ±10 µm. Grinding times for the PFS samples are shown in Table 13.35.

Table 13.35: Grinding Times to Obtain P80=240 µm

Sample	AND	BXI	DIO	TON	N16Y	N712Y
Time, min	6.19	4.57	4.62	4.52	5.23	6.06
P80, micron	243	240	235	233	239	248

The Project defined a grinding P80 of 240 µm. Based on results from previous test work (Table 13.18 and Table 13.19), flotation test work was performed with a P80 = 240 µm because it optimizes the recovery on a specific energy consumption, improves the process (flotation and water recovery) and results in good recoveries.

TETRA TECH

13.2.7 Flotation Testwork

13.2.7.1 Rougher Flotation Testwork

Rougher flotation tests were carried out using a revised reagent formula (called PFS-C) to improve copper and molybdenum recoveries. The parameters for this new formula are described in Table 13.36 and the results are shown in Table 13.37 and in Figure 13.12.

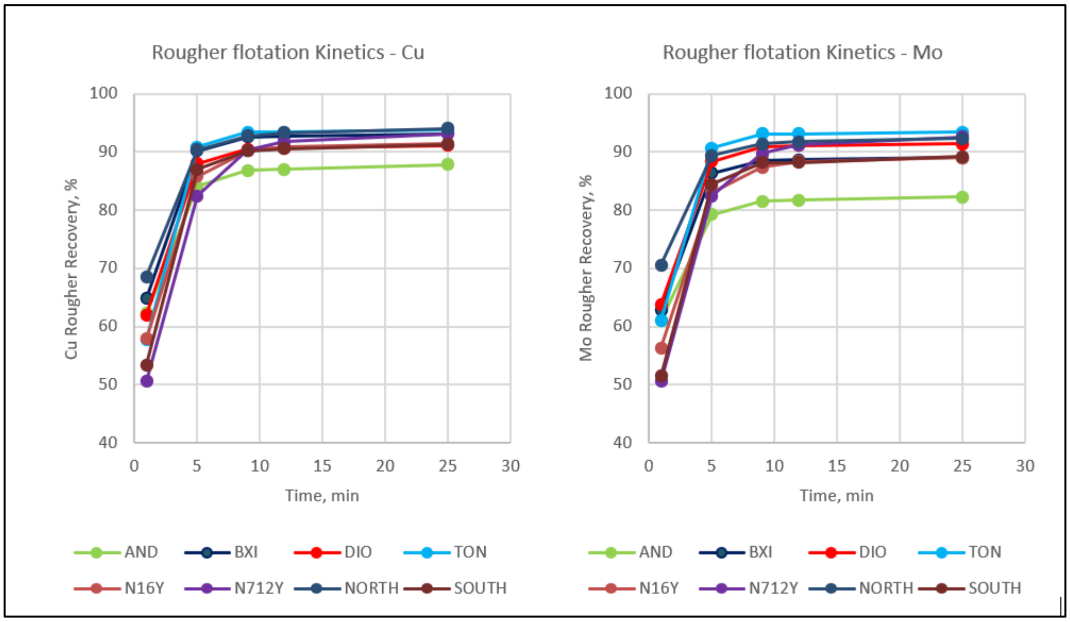
Table 13.36: Flotation Parameters for PFS-C Reagent Formula

Flotation	Value
Water	Fresh Water
Cell	Denver 2-Liter and Denver 4-Liter
Solids, %wt	30.5
pH	7.5
Speed, rpm	1400 (2-Liter Cell) and 1200 (4-Liter Cell)
Gas flow, L/min	10 (2-Liter) and 12 (4-Liter Cell)
Froth Scrap Frequency	1 in 10 Seconds
Flotation Time	1, 5, 9, 12, 25
Collectors	20 g/t C3330, 20 g/t MCC200 (14 g/t in Mill, 6 g/t in Upfront 5')
Frothers	5 g/t MIBC, 30 g/t DF400
Modifiers	20 g/t of Diesel, Lime to Adjust pH to 7.5

Table 13.37: Rougher Flotation Results Using the PFS-C Reagent Formula

Sample	Cu (%)	Mo (ppm)	Mass Pull	Cu Rec.	Mo Rec.	R. inf	K, 1/min
AND	0.39	105	7.19	87.8	82.4	89.52	3.15
BXI	0.42	86	6.25	93.2	89.0	95.56	2.97
DIO	0.44	77	5.49	91.1	91.4	93.63	2.80
TON	0.33	202	5.01	93.8	93.4	97.49	2.22
N16Y	0.48	110	7.56	91.4	89.1	93.81	2.37
N712Y	0.45	113	6.41	93.1	92.5	95.46	1.74
NORTH	0.47	108	8.76	94.1	92.4	95.61	3.45
SOUTH	0.37	106	4.66	91.3	89.2	94.77	2.01

Figure 13.12: Rougher Recovery vs. Time for PFS Samples



Source: Los Andes Copper, 2021, based on SGS Rougher Flotation results from 2020 testwork

13.2.7.2 Comparison Between PEA and PFS-C Formula

The differences between the reagent formula used in the 2017-18 testwork (PEA) and that used in the PFS study are shown in Table 13.38.

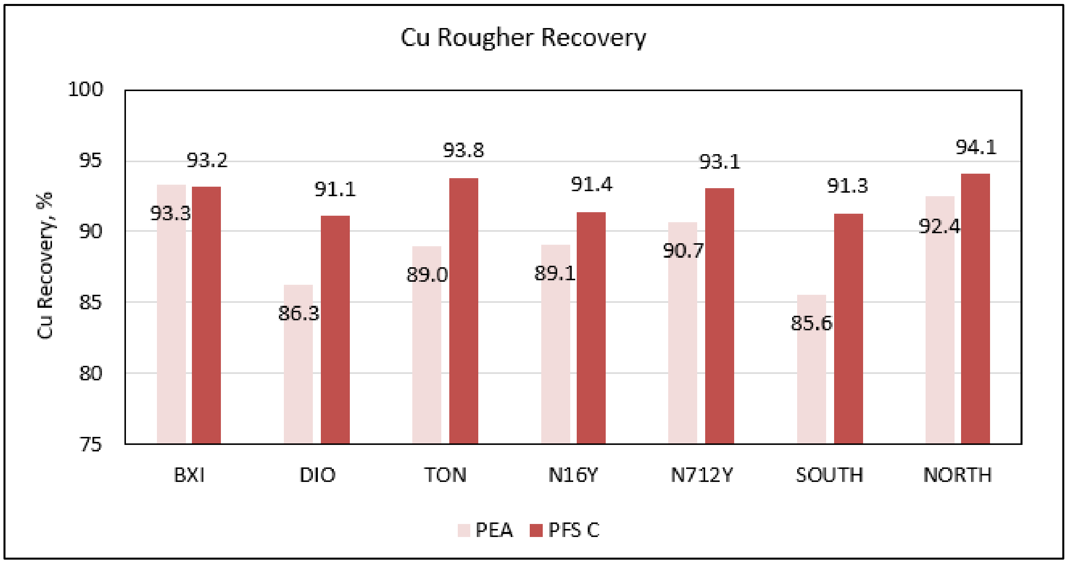
Table 13.38: Flotation Parameters for PEA and PFS-B Reagent Formula

Flotation	PEA	PFS-C
Agitation time	1 min	5 min
Solids, %wt	38.2	30.5
pH	10.5	7.5
Collectors	20 g/t C3330, 20 g/t AP3477	20 g/t C3330, 20 g/t MCC200
Frothers	15 g/t MIBC	5 g/t MIBC, 30 g/t DF400
Modifiers	20 g/t of Diesel, Lime	20 g/t of Diesel, Lime

Figure 13.13 and Figure 13.14 show the differences between the PEA and PFS-C reagent formulae in the rougher recoveries for the PFS samples. Sample AND was not tested using the PEA formula because there was not enough sample available. However, as shown in Table 13.37, AND samples were tested using the PFS-C formula.

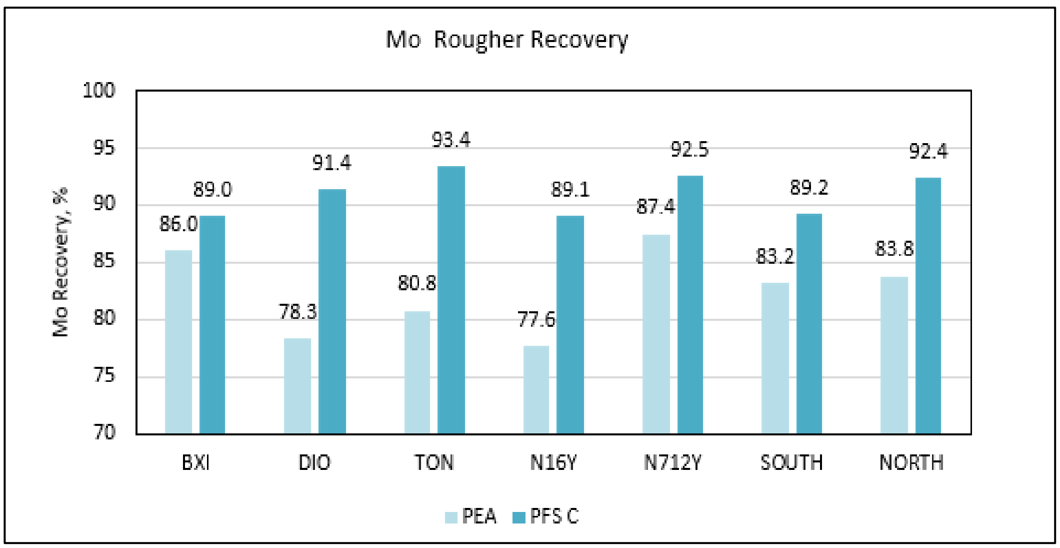
Results show that, on average, the PFS formula has an improved rougher recovery of 3.1% for copper and 8.6% for molybdenum over the PEA formula.

Figure 13.13: Cu Rougher Recovery – PEA vs. PFS-C Reagent Formula



Source: Los Andes Copper, 2021, based on SGS Rougher Flotation results from 2020 testwork

Figure 13.14: Molybdenum Rougher Recovery – PEA vs. PFS-C Reagent Formula



Source: Los Andes Copper, 2021, based on SGS Rougher Flotation results from 2020 testwork

Table 13.39: Comparison Between Results Obtained (%) with the PEA and PFS-C Formulae

Sample	Cu			Mo		
	PEA	PFS-C	Difference	PEA	PFS-C	Difference
BXI	93.3	93.2	-0.2	86.0	89.0	3.0
DIO	86.3	91.1	4.9	78.3	91.4	13.1
TON	89.0	93.8	4.8	80.8	93.4	12.7
N16Y	89.1	91.4	2.3	77.6	89.1	11.5
N712Y	90.7	93.1	2.4	87.4	92.5	5.1
SOUTH	85.6	91.3	5.7	83.2	89.2	6.0
NORTH	92.4	94.1	1.6	83.8	92.4	8.7
Average			3.1			8.6

13.2.8 Cleaner Flotation Testwork

13.2.8.1 Rougher Concentrate Characterization and Regrinding Kinetics

A mineralogical characterization of the concentrate (QEMSCAN PMA) showed that the main gangue minerals present in the concentrate are hard silicates (40%), pyrite (28%) and phyllosilicates (13.4%).

For the SOUTH and NORTH samples, the P80 was 175 and 203 µm, respectively. Regrinding kinetics were carried out for SOUTH and NORTH samples in a laboratory scale mill using 5 kg of 1 inch balls and operating at 70 rpm and 50% solids (%wt).

With the rougher flotation stage fed with a P80 of 240 µm, the regrinding stage must balance the liberation and final grades in the concentrate with a regrinding target lower than that defined in previous studies (45 µm). The effect of regrinding on the liberation of primary copper sulphides showed an increase from 83.8% to 88.1% (+5%) and the percentage of locked particles decreased from 3.0% to 1.7% (-43%). For secondary sulphides, the percentage of locked particles decreased from 8.1% to 3.9%. The proportion of locked molybdenite decreased from 7.7% to 1.6%.

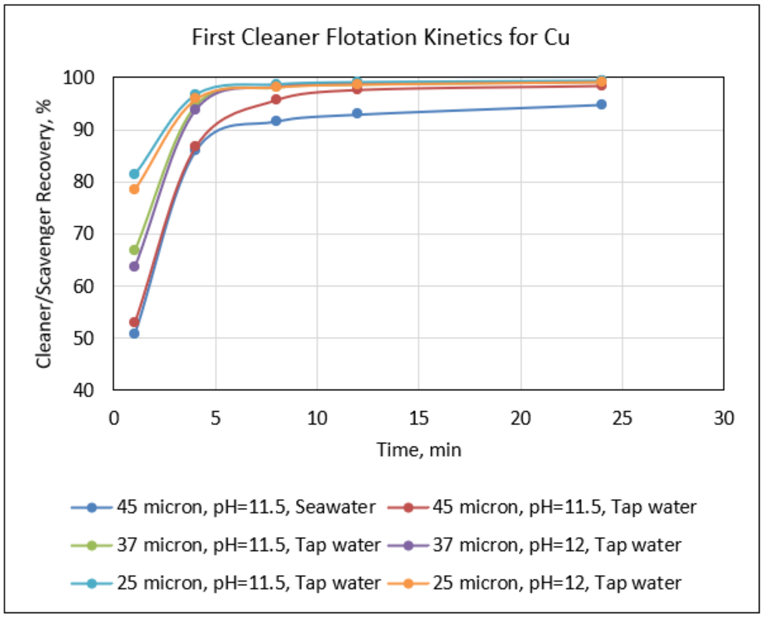
These results are reflected in the recoveries and concentrate grades in the cleaner stage, because higher recoveries and concentrate grades are normally expected for finer regrinding P80.

The results are shown in Table 13.40 and Figure 13.15 to Figure 13.19.

Table 13.40: Results of First Cleaner Flotation Kinetics

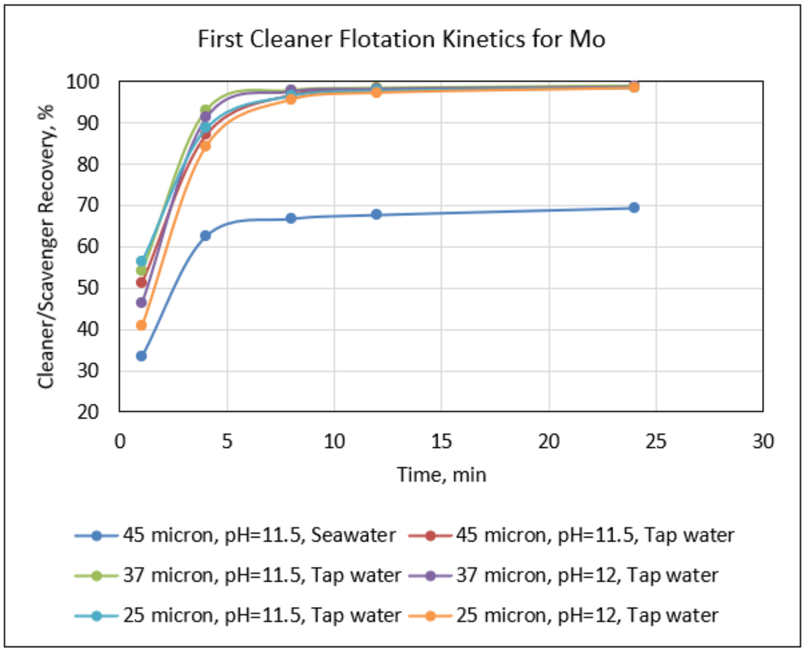
Test	1	2	3	4	5	6	7	8	9
Sample	NORTH	NORTH	NORTH	NORTH	NORTH	NORTH	N16Y	N712Y	SOUTH
Water	Sea	Fresh	Fresh	Fresh	Fresh	Fresh	Fresh	Fresh	Fresh
Regrinding P80, µm	45	45	37	37	25	25	25	25	25
pH	11.5	11.5	11.5	12	11.5	12	12	12	12
Cu, %	0.47	0.46	0.46	0.46	0.47	0.47	0.49	0.44	0.38
Mo, ppm	116	114	114	112	123	124	121	114	117
Rougher Flotation									
Mass Pull	9.61	8.73	8.65	8.64	8.49	8.57	8.53	7.78	6.36
Cu Recovery	93.1	92.1	93.7	92.9	92.9	93.1	91.6	93.5	91.9
Mo Recovery	89.8	92	91.2	91	90.3	91.9	90.9	95.2	92.8
Cleaner Flotation									
Mass Pull	23.4	24	23.9	20.5	20.1	18	17.4	24	22.8
Cu Recovery	86	86.7	94.8	93.9	96.6	95.9	95	96.4	94.8
Mo Recovery	62.7	87.1	93.1	91.4	88.8	84.4	77.9	85.2	84.7
Scavenger Flotation									
Mass Pull	8.8	8	8.9	7.3	6.7	7.7	6.7	4.9	6.5
Cu Recovery	8.8	11.7	4.5	5.4	2.7	3.3	3.6	2.6	4.0
Mo Recovery	6.8	11.7	5.9	7.5	9.9	14	20.2	12.7	12.8
Cleaner/Scavenger Flotation									
Mass Pull	32.1	32	32.8	27.8	26.9	25.7	24.1	28.9	29.3
Cu Recovery	94.9	98.4	99.3	99.3	99.3	99.2	98.6	99	98.8
Mo Recovery	69.5	98.8	99	98.8	98.7	98.4	98.1	97.9	97.4
Cu in Concentrate	16.8	17.7	19.7	22.4	24.6	27.3	28.7	21.2	22.9
Mo in Concentrate	0.29	0.43	0.47	0.52	0.58	0.62	0.58	0.5	0.64

Figure 13.15: First Cleaner Kinetics for Cu (NORTH sample)



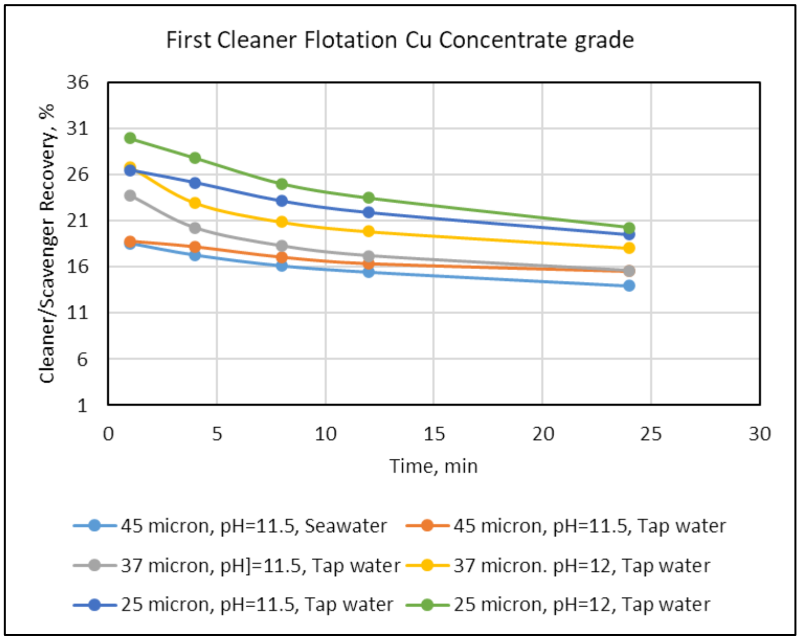
Source: Los Andes Copper, 2021, based on SGS Cleaner Flotation results from 2020 testwork

Figure 13.16: First Cleaner Kinetics for Mo (NORTH sample)



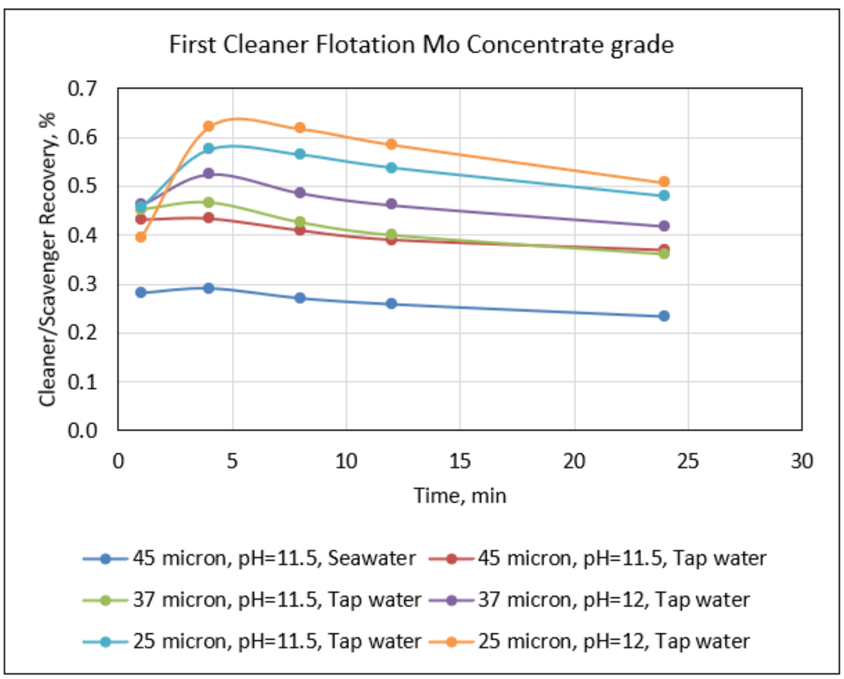
Source: Los Andes Copper, 2021, based on SGS Cleaner Flotation results from 2020 testwork

Figure 13.17: Cu Concentrate Grades in First Cleaner Flotation



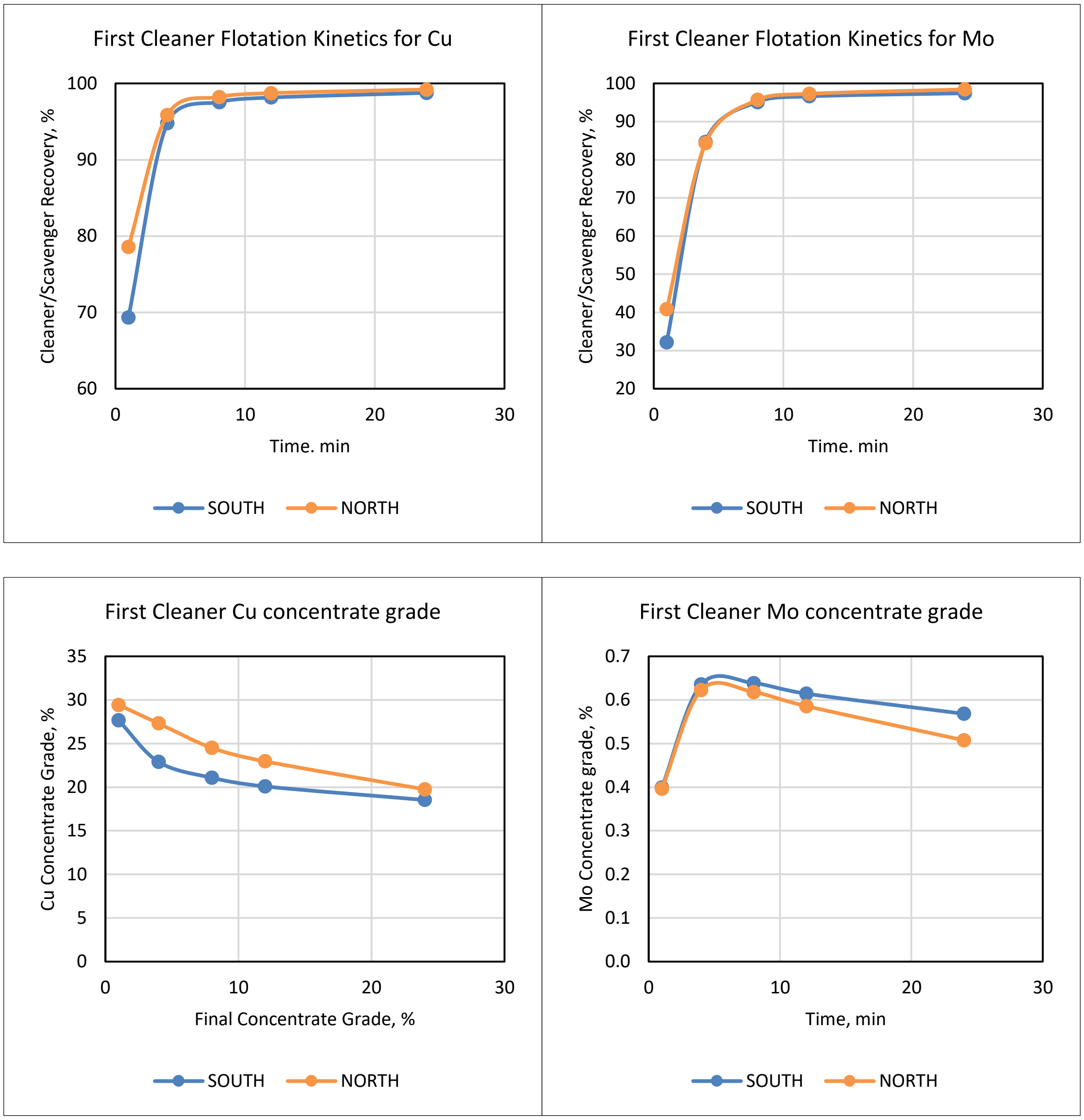
Source: Los Andes Copper, 2021, based on SGS
Cleaner Flotation results from 2020 testwork

Figure 13.18: Mo Concentrate Grades in First Cleaner Flotation



Source: Los Andes Copper, 2021, based on SGS
Cleaner Flotation results from 2020 testwork

Figure 13.19: First Cleaner Flotation Kinetics and Concentrate Grades for SOUTH and NORTH Samples



Source: Los Andes Copper, 2021, based on SGS Cleaner Flotation results from 2020 testwork

13.2.8.2 Open Cycle Test Results

First cleaner tests were performed using the NORTH sample at different conditions of pH and regrinding P80 by floating a rougher concentrate in a 2 L Denver cell operating at 1,100 rpm, 8 L/min and slurry solids between 15% and 20%. The best results were used to define the parameters to carry out tests using SOUTH, N16Y and N712Y samples. Cleaner flotation using sea water in the rougher stage was carried out to assess the effect of raw sea water on the concentrate grade.

Test 1 was performed with sea water only in the rougher stage and fresh water was used in the cleaner/scavenger flotation. Results showed that copper and molybdenum cleaner/scavenger recoveries decreased by 3.5% for copper and 29.3% for molybdenum. This confirmed that flotation using sea water is not recommended.

Tests 2 to 6 showed that the best cleaner/scavenger results were obtained by regrinding to P80 of 25 µm and operating the first cleaner with a pH of 12.

After the first 2 minutes of the NORTH sample cleaner flotation, a darker patch was observed in the concentrate froth as shown in Figure 13.20. This was related to the increase of molybdenum recovery and grades. Therefore, first cleaning residence times of 4 minutes were recommended.

Figure 13.20: Final Concentrate Containing Copper and Molybdenum

Source: Los Andes Copper, 2021

The scavenger Cu and Mo recoveries stabilized at a 21 minute flotation time.

Commercial grade concentrates were obtained with a single stage cleaner flotation. This is advantageous in terms of operating a simpler circuit, minimizing mass recycles, decreasing the footprint and decreasing the amount of flotation equipment.

13.2.8.3 Locked Cycle Testing

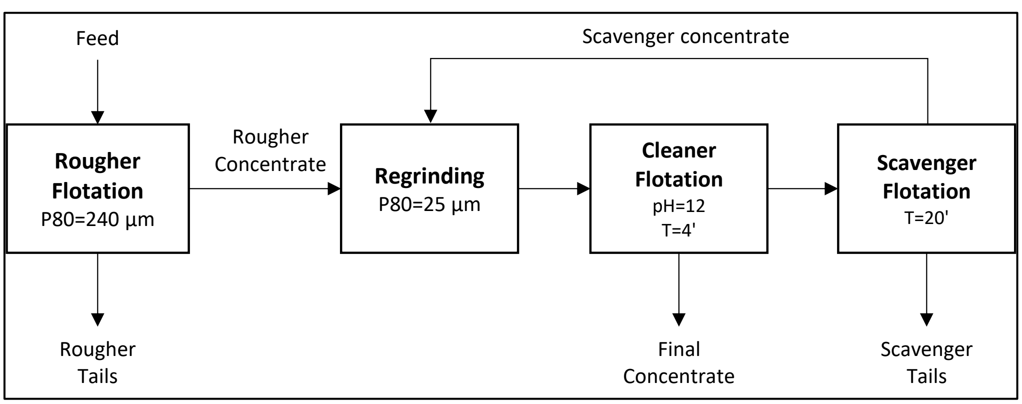
A locked cycle flotation test (LCT) was carried out using the NORTH sample and the best parameters obtained in the first cleaner kinetic tests. In the locked cycle, 18 rougher flotation tests were carried out to collect enough mass to perform six cleaner flotations at 15% to 20% solids in a 2 L cell. Only the NORTH sample was tested as there was not enough of the other samples available.

The flotation parameters and flowsheets are shown in Table 13.41 and Figure 13.21. The LCT results were balanced using the data from the last three cycles (steady state) using JKSimMet. The results are shown in Figure 13.22 and Table 13.42.

Table 13.41: Locked Cycle Test Parameters

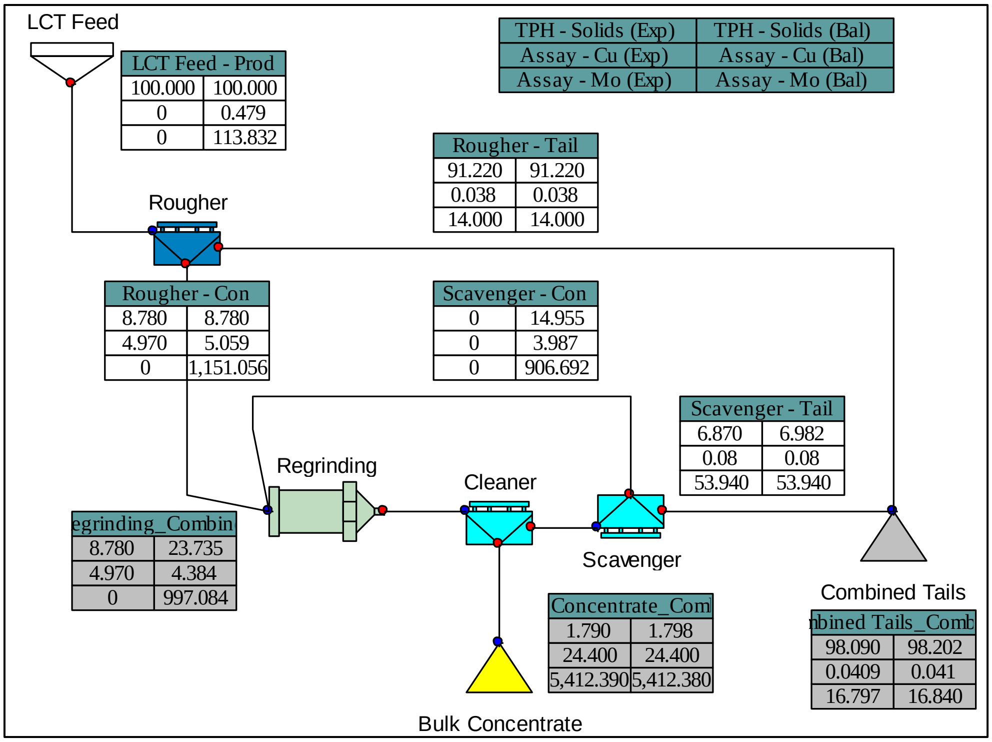
Sample	Rougher	Cleaner	Scavenger
Water	Fresh	Fresh	Fresh
Reagent Formula	PFS-C	-	-
P80, micron	240	25	25
% Solids, %wt	30.2	15-20%	15-20%
Cell	Denver 4-L	Denver 2-L	Denver 2-L
pH	7.5	12	12
Agitation Speed, rpm	1200	1100	1100
Gas Flow, L/min	12	8	8
Froth Scrap Frequency	1 in 10s	1 in 10s	1 in 10s
Flotation Time, min	25	4	20

Figure 13.21: Locked Cycle Test Circuit



Source: Los Andes Copper, 2021, based on laboratory testwork protocol

Figure 13.22: Mass Balance of Locked Circuit Flotation Test



Source: Los Andes Copper, 2021, based on Locked Cycle Test Mass Balance in JKSimMet

Table 13.42: Locked Cycle Flotation Test Results NORTH Sample

Results	Mass	Cu	Mo
Head	-	0.48%	114 ppm
Rougher Recovery, %	8.78	92.7	88.7
Cleaner/Scavenger Recovery, %	20.5	98.7	96.3
Overall Recovery, %	1.8	91.6	85.4
Final Concentrate Grade	-	24.40%	0.54%

Compared with the first cleaner flotation test, differences for Cu and Mo cleaner/scavenger recoveries were observed (-0.51% for Cu and -2.13 for Mo) (Table 13.43). This difference is associated with the recycle effect in the LCTs. Therefore, cleaner/scavenger recoveries and final concentrate grades were adjusted by these factors.

Table 13.43: Differences Between First Cleaner and Locked Cycle Test

Test	First Cleaner	Locked Cycle	Difference
Sample	NORTH	NORTH	
Cu, %	0.47	0.48	
Mo, ppm	124	114	
Rougher Mass Pull	8.57	8.78	0.21
Rougher Cu Recovery	93.1	92.7	-0.4
Rougher Mo Recovery	91.9	88.7	-3.2
Cleaner/Scavenger Mass Pull	25.7	20.5	-5.23
Cleaner/Scavenger Cu recovery	99.2	98.7	-0.51
Cleaner/Scavenger Mo recovery	98.4	96.3	-2.13
Overall Cu Recovery	92.4	91.5	-0.86
Overall Mo Recovery	90.4	85.4	-5.02
Cu in Concentrate	27.30%	24.40%	-2.91
Mo in Concentrate	0.62%	0.54%	-0.08

13.2.8.4 Final Concentrate Specification

The final copper and molybdenum concentrates generated from the LCTs were submitted for detailed analysis. The analysis confirmed that Vizcachitas can produce clean concentrates. The projected results show that final copper concentrate grades between 22.9% and 24.4% Cu are achievable. No elements were present at penalty levels in the copper concentrates (Sb and Bi are below 0.04% and 0.01%, respectively, the first year weighted average of As is 0.16% and Hg is approximately 5.2 ppm). Table 13.44 shows the final concentrate specification.

Table 13.44: Final Concentrate Specification

Sample Unit	Cu %	Mo %	Fe %	S %	Au g/t	Ag g/t	Sb %	Bi %	As %	Hg ppm	Ins %
NORTH	24.4	0.54	24.9	30.3	0.4	50	0.005	0.005	4.0	4	19.9
SOUTH	22.9	0.64	26.2	30.4	0.6	65	0.041	0.01	5.2	5.2	19.5

13.2.8.5 Selective Cu-Mo Flotation

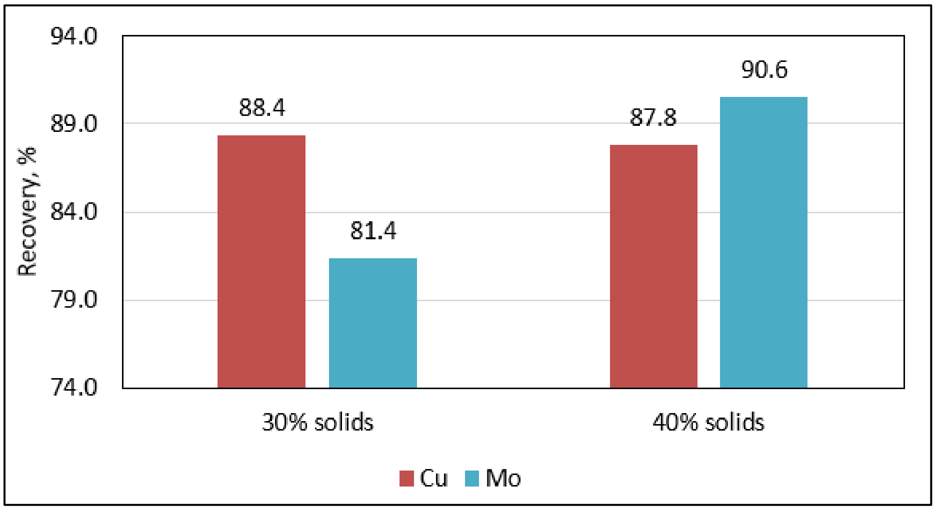
Two batches of 360 g and 570 g of bulk Cu-Mo concentrates were generated in LCTs for the NORTH sample for a fixed time Cu-Mo selective rougher flotation test. A 1 L Denver cell was used for testing; the main objective was to study the effect of the % solids on the Mo recovery.

Results showed that molybdenum recovery increased with the % solids of the slurry (Table 13.45 and Figure 13.23). Molybdenum rougher recoveries are similar to those observed in other operations located in the same metallogenic belt.

Table 13.45: Selective Cu-Mo Test Parameters

Sample	Test 1	Test 2
Water	Fresh	Fresh
% Solids, %wt	30%	40%
Cell	Denver 1-L	Denver 1-L
pH (Adjusted with H2SO4)	9	9
ORP (Adjusted with NaHS)	-450mV	-450mV
Agitation Speed, rpm	900	900
Nitrogen Gas Flow, L/min	2	2
Flotation Time, min	10	10

Figure 13.23: Selective Cu-Mo Flotation Results



Source: Los Andes Copper, 2021, based on Molybdenum selective flotation from SGS 2020 testwork

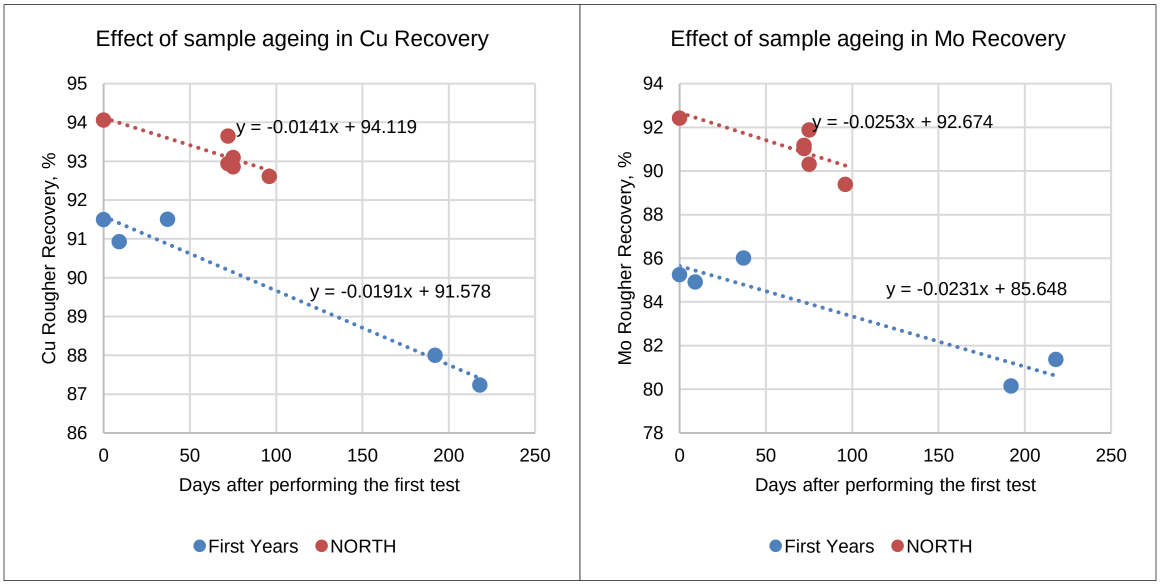
13.2.8.6 Effect of Sample Ageing on Flotation Performance

During the 2020 testwork at SGS the effect of sample ageing on the Cu and Mo rougher recoveries was observed. This effect was first observed at the end of the preliminary sea water assessment testwork, where the first years sample from the 2018 flotation testwork were tested two months apart and showed a decay in recovery.

To assess this effect the same sample was floated over time at the same conditions and parameters. Results showed that the rate of decrease for Cu and Mo rougher recoveries were 0.50% and 0.73% per month, respectively (Figure 13.24). It is recommended in future studies that a control sample is floated every month to assess this effect and improve the sample preparation process. It is possible that 10 mesh samples should be purged with nitrogen, sealed and stored in freezers to avoid ageing.

Ageing should also be considered in stockpile/bin management and in the mining plan in case material is stockpiled.

Figure 13.24: Effect of Sample Ageing on Cu and Mo Rougher Recovery



Source: Los Andes Copper, 2021, based on Rougher flotation test results from SGS 2020 testwork

13.2.8.7 Summary of Flotation Testwork

Values for Cu and Mo recoveries for each lithology used in mine planning are shown in Table 13.46 to Table 13.48.

Table 13.46: Summary of Cu Recoveries (%)

Sample	Grade	Rougher	Cleaner	Overall
AND	0.39	87.8	98.3	86.3
BXI	0.42	93.2	98.3	91.6
DIO	0.44	91.1	98.3	89.6
TON	0.33	93.8	98.3	92.2
NORTH	0.47	94.1	98.7	92.9

Table 13.47: Summary of Mo Recoveries (%)

Sample	Grade	Rougher	Cleaner	Bulk	Overall
AND	105	82.4	95.3	78.5	69.7
BXI	86	89	95.3	84.8	75.3
DIO	77	91.4	95.3	87.1	77.3
TON	202	93.4	95.3	89.0	79.0
NORTH	108	92.4	96.3	89.0	79.0

Table 13.48: Summary of Mass Pull (%)

Sample	Rougher	Cleaner	Overall
AND	7.20	24.10	1.73
BXI	6.30	24.10	1.51
DIO	5.50	24.10	1.32
TON	5.00	24.10	1.21
NORTH	8.80	20.50	1.80

The results show an overall bulk copper recovery between 86.3% and 92.9% and an average global mass of 1.51%.

For the selective Cu-Mo flotation stage, the overall Mo recovery was estimated by considering a 98% cleaner/scavenger recovery (benchmark) and a 90.6% Mo recovery obtained in the molybdenum rougher flotation stage.

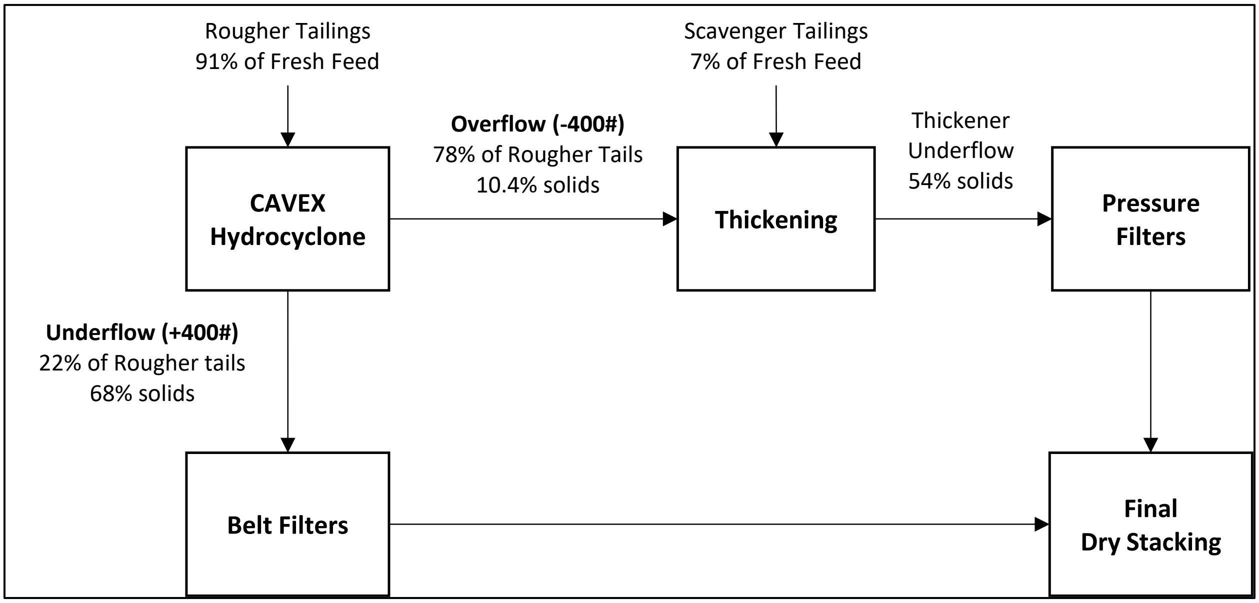
Based on the grades obtained in the final concentrates (Table 13.44) the silver recovery was estimated to be 75%.

13.2.9 Water Recovery Testwork

13.2.9.1 Fines Classification Study

To assess the feasibility of fine classification, Vizcachitas requested Weir Minerals to assess the fine classification performance using representative data of the particle size distribution of the Vizcachitas rougher tailings (Weir Minerals, 2020). The proposed classification circuit is shown in Figure 13.25.

Figure 13.25: Tailings Classification and Dewatering Circuit



Source: Los Andes Copper, 2021, based on Process Plant Design Criteria

Weir recommended that only the fine fraction (-400#) should be thickened, because the cyclone underflow (coarse fraction) can be fed to belt filters (68%).

The corrected classification function of the hydrocyclone was defined by the Rosin-Rammler model (King, 2012). Model parameters were adjusted to the Weir simulation data to define a function to estimate the coarse and fine fractions of samples based on the feed particle size distribution. The adjusted model is shown in the following equation:

$$c(d_p) = 0.16 + (1 - 0.16)\left(1 - \exp\left[0.693\left(\frac{d_p}{39.16}\right)^{1.91}\right]\right)$$

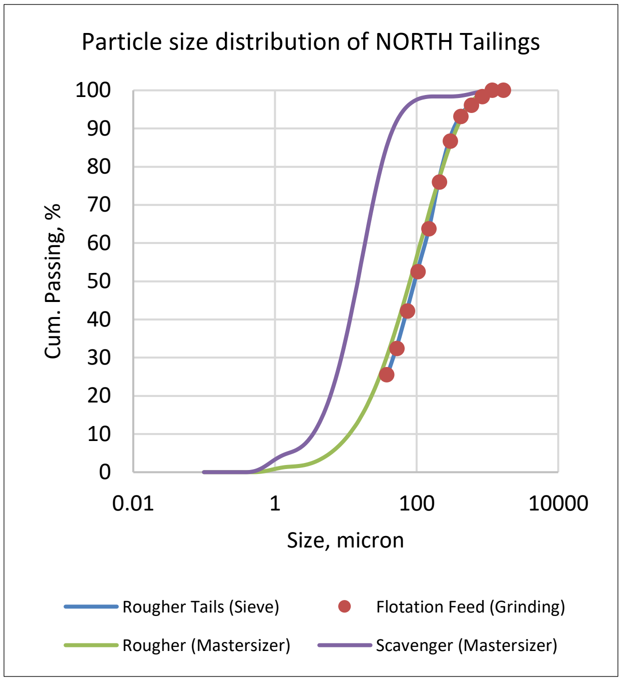
Where, $c(d_p)$ is the corrected classification curve for a size d_p in μm.

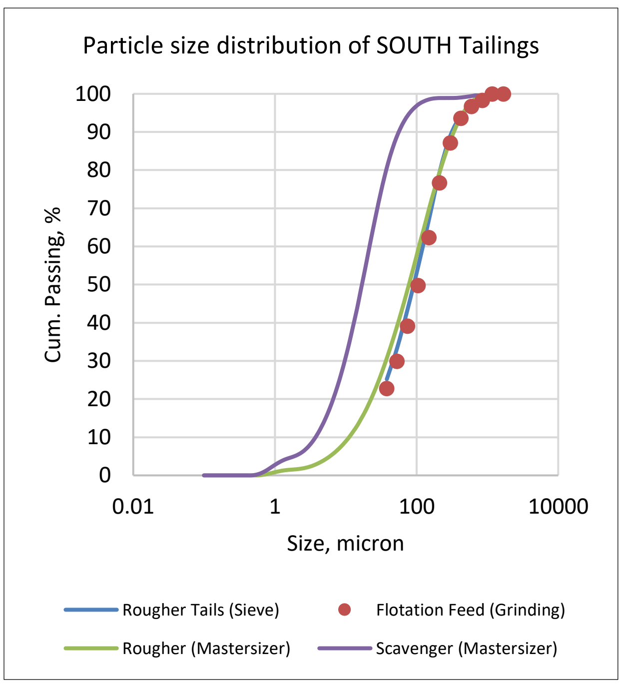
13.2.9.2 Tailings Characterization of Composites

NORTH and SOUTH rougher and scavenger tailings from LCT and cleaner tests, carried out at a grind of P80 240 μm and a regrind P80 of 25 μm, were analyzed by sieving and laser (MasterSizer). Figure 13.26 and Table 13.49 show the comparison between the results of these analyses compared with the particle size distribution obtained in the grinding stage.

The results show that corrected flotation feed curves can be used to estimate tailings particle size distribution; the sieve results from rougher tailings and flotation feed have similar shapes.

Figure 13.26: Particle Size Distribution of Tailings - NORTH and SOUTH Samples





Source: Particle Size Analysis from SGS 2020 testwork

Table 13.49: Particle Size Characterization - NORTH and SOUTH PFS Composites.

Sample	NORTH Composite Tailings				SOUTH Composite Tailings			
	Rougher (Sieve)	Flotation Feed	Rougher (Mastersizer)	Scavenger (Mastersizer)	Rougher (Sieve)	Flotation Feed	Rougher (Mastersizer)	Scavenger (Mastersizer)
P80	232	243	235	32	215	238	216	37
P50	201	206	81	15	187	209	78	17
P20	-	-	23	6	-	-	23	7
%-20 µm	-	-	18	65	-	-	21	64
%-10 µm	-	-	9	34	-	-	10	35

13.2.9.3 Sample Preparation for PFS Tailings Composites

Classification by hydrocyclones is not perfect, finer particles report to coarser products and vice versa. To represent this effect, rougher and scavenger tailings obtained from LCTs for AND, BXI, DIO, TON and NORTH samples were wet-classified using fresh water and laboratory scale sieves (150 and 400 mesh) to produce two sub-samples, M1 and M2, that represent the expected efficiency of the hydrocyclone as defined by Weir. The coarser fractions (underflow from the hydrocyclone) were M1. The finer fractions that include the synthetic overflow of the hydrocyclone and the remaining scavenger tailings were M2.

The amount of each sieve fraction was obtained by applying the classification model shown above to the particle size distributions obtained in the grinding kinetic stage at 240 µm.

The sample preparation protocol for each flotation tailing is shown in Figure 13.27.

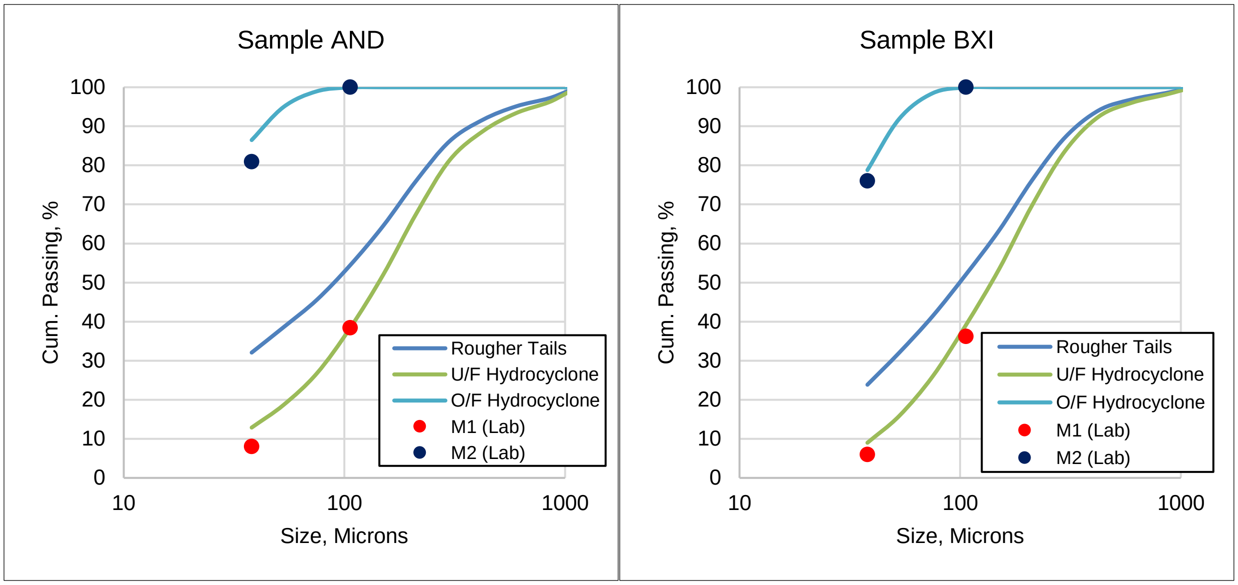
Figure 13.27: Sample Classification Protocol for LCT Rougher and Scavenger Tailings

SAMPLE	COMPOSITION				MASS	TEST
	ROUGHER TAILINGS			SCAVENGER TAILS		
	+150#	-150# +400#	-400#	Total		
M1	100% +150# 11.3 kg	4/5 fraction-150#+400# 5.1 kg	1/5 fraction-400# 1.2 kg	100% +150# 0 kg	**17.5 kg**	Vacuum Filtration
M2	100% +150# 0 kg	1/5 fraction-150#+400# 1.3 kg	4/5 fraction-400# 4.7 kg	100% Scav. Tails 1.6 kg	**7.6 kg**	Settling and Pressure Filtration

Source: Los Andes Copper, 2021, based on Laboratory Testwork Protocol

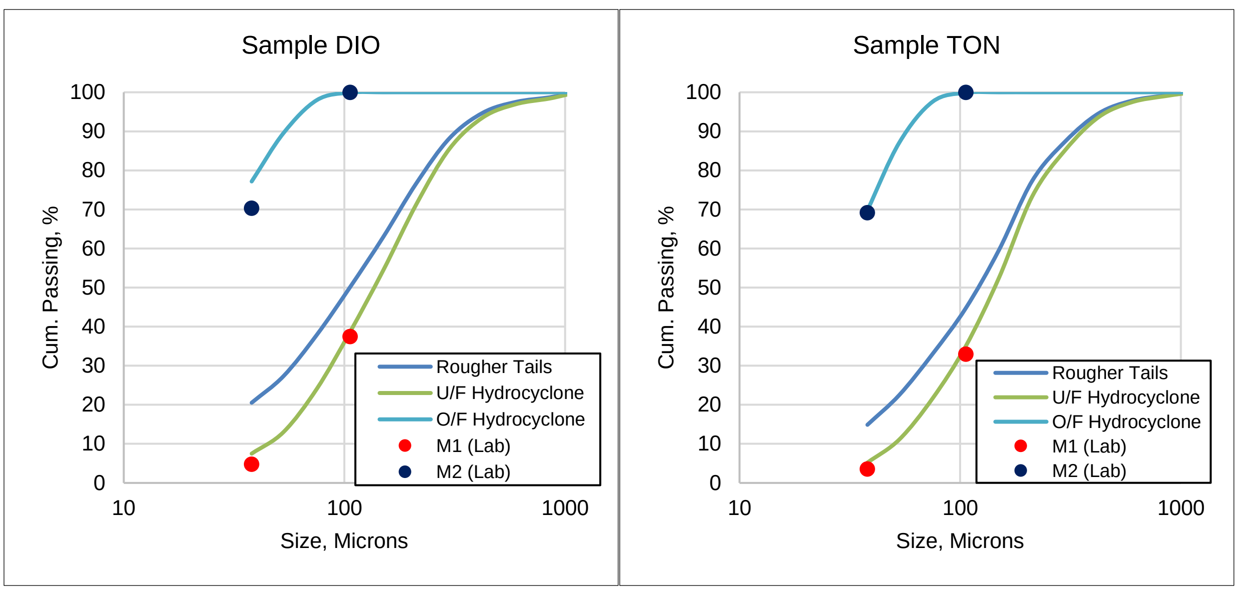
Figure 13.28 to Figure 13.30 show the mass obtained for each sub-sample obtained in the rougher tailings classification.

Figure 13.28: Sample Preparation for AND and BXI Rougher Tailings



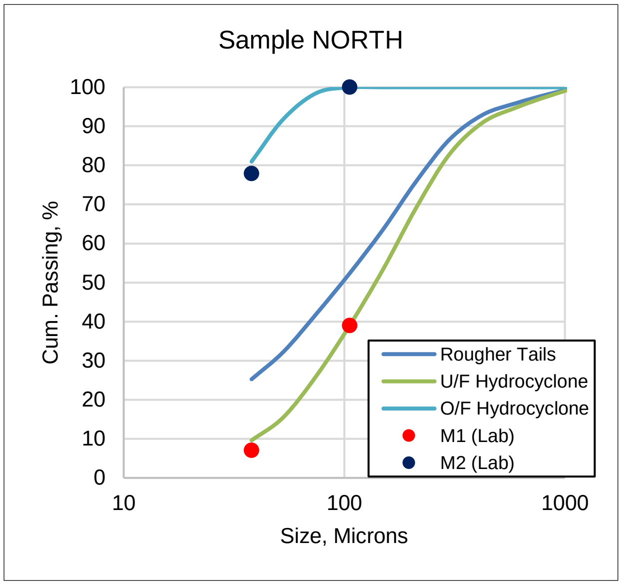
Source: Los Andes Copper, 2021, based on classification curves and laboratory measurements

Figure 13.29: Sample Preparation for DIO and TON Rougher Tailings



Source: Los Andes Copper, 2021, based on classification curves and laboratory measurements

Figure 13.30: Sample Preparation for NORTH Rougher Tailings



Source: Los Andes Copper, 2021, based on classification curves and laboratory measurements

After sieving, M2 samples were completed by adding the scavenger tailings for each sample. Finally, the slurry density was adjusted to obtain a solids percent representative of the hydrocyclone underflow (68 %wt) for M1 and the thickener discharge (54 %wt) for M2. The M1 and M2 samples weights by lithology are shown in Table 13.50.

Table 13.50: Summary of Sub-Samples M1 and M2 for PFS Filtration Testwork

Sub Sample	AND	BXI	DIO	TON	NORTH
M1 (Coarse or U/F)	12.8 kg	14.6 kg	16.4 kg	16.9 kg	17.2 kg
M2 (Fine or O/F)	6.7 kg	5.9 kg	5.5 kg	5.2 kg	7.8 kg

13.2.9.4 Settling Testwork

Static settling tests were carried out using the M2 (fine) fractions (-400 mesh rougher + scavenger tailings) of the AND, BXI, DIO, TON and NORTH PFS composites.

The results in Table 13.51 show that the finer fractions of Vizcachitas tailings can achieve suitable final slurry solids (55.5% in average) for pressure filters. Settling rates are improved with the addition of a flocculant. For the first 10 years of operation, a settling velocity of 2.65 m/h at 55.7 solids %wt can be expected.

Table 13.51: Settling Testwork Results for PFS Tailings Composites

Samples	Without Flocculant		With Flocculant (TEC 2050)	
	Settling Velocity, m/h	Final Solids (24 h), %	Settling Velocity, m/h	Final Solids (24 h), %
AND	0.57	55.2	1.98	55.2
BXI	0.77	56.8	2.52	56.1
DIO	0.58	56.1	2.75	54.8
TON	0.56	55	2.42	55.6
NORTH	0.67	57	3	56

13.2.9.5 Filtration Testwork

Filtration testwork was carried out by SGS using vacuum filtration on the coarse fraction (M1) and pressure filtration on the fine fraction (M2) of the Vizcachitas tailings. The particle size distribution was calculated from the sieving process during the sample preparation (Table 13.52).

Table 13.52: Particle Size Distribution of M1 and M2 Samples

Sample	Fraction	AND	BXI	DIO	TON	NORTH
M1	- 150#	38.4	36.2	37.5	33	39
	- 400#	8	6	4.8	3.6	7
M2	- 150#	100	100	100	100	100
	- 400#	80.9	76	70.3	69.2	77.9

The results in Table 13.53 show that Vizcachitas tailings have good filtration rates, and the implementation of a dry stacked filtered tailings deposit is feasible. The presence of fine particles influences filtration rates; the AND sample contains a higher proportion of -400 mesh particles and it can be seen that the results are not as good for this lithology.

For the first years of operation, the expected filtration rate for vacuum filters is 1.92 t/h/m^2 with a cake moisture of 14.7%; for pressure filters the expected filtration rate is 0.55 t/h/m^2 with a cake moisture of 15.5%.

Table 13.53: Summary of Preliminary Filtration Testwork (2018 composites)

Sample	Fraction	Filter Type	Pressure bar	% Solids In feed	Average Cake Moisture, %	Average Filtration Rate, t/h/m^2
AND	M1	Vacuum	0.6	65	16.5	1.80
	M2	Pressure	12.0	54	16.3	0.51
BXI	M1	Vacuum	0.6	65	14.6	1.92
	M2	Pressure	12.0	54	14.9	0.62
DIO	M1	Vacuum	0.6	65	14.5	2.00
	M2	Pressure	12.0	54	15.4	0.62
TON	M1	Vacuum	0.6	65	14.1	2.00
	M2	Pressure	12.0	54	15.4	0.64
NORTH	M1	Vacuum	0.6	65	14.2	1.91
	M2	Pressure	12.0	54	15.3	0.51

13.2.9.6 Testwork Performed at Takraf

M1 and M2 sub-samples of AND, BXI, DIO and TON tailings were mixed to create a SOUTH composite using the proportions given in the mine plan. This sample, along with M1 and M2 sub-samples of the NORTH sample tailings were sent to Takraf laboratories in Chile (ex Tenova Delkor) to perform settling, vacuum, pilot pressure filtration and Atterberg limits testwork (Takraf, 2021). Tailings particle size distributions were estimated based on laboratory measurements and the Weir classification curves (Table 13.54 and Table 13.55). Takraf received fine and coarse fractions of M1 and M2 samples for the NORTH and SOUTH composites.

The samples were considered representative of the process because, they were collected from LCTs. The samples were conditioned with fresh water and lime was added to adjust the pH to 10.5-11.0.

Table 13.54: Particle Size Distribution of NORTH Tailings Composites

Stream	Rougher Tails	Scavenger Tails	O/F Cyclone	U/F Hydrocyclone Belt Filter Feed	Pressure Filter Feed
P80, µm	235	32	36	291	35
P50, µm	81	15	18	126	18
P20, µm	23	6	7	54	7
%-20 µm	18	65	56	5	58
%-10 µm	9	34	28	2	29

Table 13.55: Particle Size Distribution of SOUTH Tailings Composites

Stream	Rougher Tails	Scavenger Tails	O/F Cyclone	U/F Hydrocyclone	Pressure Filter
				Belt Filter Feed	Feed
P80, µm	216	37	36	267	36
P50, µm	78	17	18	120	18
P20, µm	23	7	7	53	7
%-20 µm	19	59	56	6	57
%-10 µm	9	31	28	2	28

13.2.9.7 Vacuum Filtration Test

The vacuum filtration tests were performed in a proprietary vacuum filter, testing different cloth types and cake thickness for a slurry feed at 58% solids. The design filtration rates for both samples included a safety factor of 25%. Test results are shown in Table 13.56 (SGS test are included for comparison).

Table 13.56: Vacuum Filtration Test Results (50 mm thickness)

Composite	Sample	SGS	Takraf
Filtration Rate, t/h/m2	SOUTH	1.91	2.9
	NORTH	1.92	2.8
Cake Moisture	SOUTH	15.10%	15.00%
	NORTH	14.70%	14.00%

13.2.9.8 Settling Test for Fine Fraction

Preliminary flocculant screening tests showed that the MF155 flocculant is the most appropriate for thickening (Table 13.57). The recommended pulp dilution to maximize flocculation is 10% solids using a flocculant dosage of 6 g/t to 11 g/t. The maximum compaction was 68.1% and 62% solids in the discharge was selected for design.

Table 13.57: Settling Test Results

Composite	Sample	Value
pH	SOUTH	10.5-11.0
	NORTH	10.5-11.0
Flocculant dosage (MF 155)	SOUTH	6-8 g/t
	NORTH	9-11 g/t
Yield Stress (unsheared) @62% solids	SOUTH	< 30 Pa
	NORTH	< 50 Pa
Rise rate	SOUTH	3.53 m/h
	NORTH	3.53 m/h

13.2.9.9 Pressure Filtration Tests

The pressure filtration tests were performed in a pilot vacuum filter using a 50 mm chamber. The filtration pressure was 10 bar and the squeezing pressure was 14 bar. Test results are shown in Table 13.58 (SGS test results are included for comparison).

Table 13.58: Pressure Filtration Test Results (50 mm thickness)

Composite	Sample	SGS	Takraf
% Solids in feed	SOUTH	54.00%	62.00%
	NORTH	54.00%	62.00%
Filtration Rate, t/h/m²	SOUTH	0.59	0.79
	NORTH	0.51	0.81
Cake Moisture	SOUTH	15.70%	15.00%
	NORTH	15.50%	14.00%

13.2.9.10 Atterberg Limits

Atterberg limits were tested at IDIEM laboratories (ASTM D4318-10) in Santiago using the SOUTH sample (IDIEM, 2021). Results show that the plastic limit for the fine fraction is 21% moisture and the liquid limit is 27% moisture. Cake moistures obtained in this testwork were below this value, confirming that the fine fraction can be conveyed.

13.2.10 Environmental Testwork

Tailings composites including scavenger tailings from the SOUTH and NORTH PFS composites were analyzed for ABA (acid-base accounting), NAG (net acid generation test) and TCLP (toxicity characteristic leaching procedure). The results are summarized in Table 13.59 to Table 13.61.

Neutralization (NP) and acid generation (AP) potential values were estimated from chemical analysis (carbonates and sulphur). Based on the net neutralization potential rate (Net NP) and NP/AP ratios the Vizcachitas material has acid generation potential. Both samples have potential for acid generation. The NORTH composite generates 19 kg H_2SO_4/t.

The TCLP results from eight elements show that no hazardous elements are present in the leached solution.

Table 13.59: Summary of ABA Test Results

Sample	S (%)	S^{-2} (%)	CO_3^{-2} (%)	SO_4^{-2} (%)	Paste pH	NP	AP	Net NP	NP/AP
SOUTH	0.5	0.42	0.29	0.04	6.44	12.43	13.13	-0.67	0.95
NORTH	1.51	1.2	0.28	0.66	8.04	12.46	37.63	-20.01	0.47

Table 13.60: Summary of NAG Test Results

Sample	NAG pH	NAG kg H_2SO_4/t pH 4.5	Type
SOUTH	3.33	3.9	Acid generator
NORTH	2.51	19	Acid generator

Table 13.61: Summary of TCLP Test Results (mg/L).

Sample	As	Ba	Cd	Cr	Hg	Ag	Pb	Se
SOUTH	0.01	<0.1	<0.01	<0.05	<0.0005	<0.01	<0.05	<0.001
NORTH	0.01	<0.1	<0.1	<0.05	<0.0005	<0.01	<0.05	<0.001

13.2.11 Quality Assurance and Quality Control

QA/QC in the metallurgical testwork programme was focused on:

- Ensuring the representativity of the composite samples
- Tracking mass losses in the sample preparation stages
- Ensuring proper preparation of 100% -10 meshes batches
- Controlling chemical analysis deviations in the flotation testwork.

13.2.11.1 General QA/QC Procedures

The activities of sample selection and cutting from drill cores were supervised by the Los Andes Copper geological and metallurgical team.

All the samples were handled and transferred between laboratories only by members of the Los Andes Copper team.

The Los Andes Copper team performed periodical visits to the metallurgical laboratories (this was limited during 2020 due to COVID-19 restrictions). During the lockdown the supervision process was performed by video calls or phone calls with the SGS and Takraf teams.

The rougher and cleaner flotation tests were carried out by the same operator at SGS Minerals Santiago. The LCTs were performed by senior SGS operators.

For flotation results, all tests were required to comply with the mass balance criteria: calculated head grade and assayed head grade must not exceed 6% for copper and 30% for molybdenum. All results for Cu and Mo were within the set ranges and the relative error showed no specific trend through the testwork.

13.2.11.2 Chemical Assays

Precision results show a correlation coefficient over 0.9950 between the original and duplicates for Cu, Fe and Mo using the AAS022D methodology; Cu and Mo using the AAS023D methodology; Cu using the CON013V_2 methodology, As using the AAS030G methodology; S using LEC010B; Ag using AAS042D; and Au using the AAS030G.

The accuracy of reference materials is within the 95% confidence interval (normal distribution).

In terms of repeatability, only 0.09% of the total of analytical determinations carried out did not confirm the results.

14. MINERAL RESOURCE ESTIMATES

14.1 Available Data

All the drill hole data is stored in an acQuire Technology Solutions Pty Ltd database (Section 11). A total of 182 drill holes and 60,924 m of drilling have been drilled on the project. Some of these drill holes are not used for the resource estimate. Drill holes that did not reach bedrock or drill holes that failed to reach their objective and were re-drilled are not included in the resource estimate. The collar locations for the Place Dome drill holes cannot be located, so those were not used for the resource estimate.

The drilling database for the Vizcachitas resource estimate was closed on July 6, 2022, and uses 168 diamond drill holes from the 1996 to 2022 drilling campaigns. The number of drill holes and metres drilled by General Minerals Corporation and Los Andes Copper is summarised for each campaign in Table 14.1. Of these 168 drill holes, only drill hole V2015-06B did not have any assay information as it was drilled over its whole length within the barren diatreme and therefore was not sampled.

Table 14.1: Drill Hole Campaigns

Campaign	GM		LAC		Total No. Drill Holes	Total Length (m)
	No. Drill Holes	Length (m)	No. Drill Holes	Length (m)		
1996-1998	60	15,794			60	15,794
2007-2008			79	22,616	79	22,616
2015-2017			17	11,819	17	11,819
2021-2022			12	8,398	12	8,398
Grand Total	60	15,794	108	42,833	168	58,628

Table 14.2 to Table 14.5 detail the metres drilled for each lithology, mineralized zone, C-veinlets and B-veinlets.

Table 14.2: Metres Drilled by Lithological Code

Lithology Code	Description		Drilled Metres
101	AND	Andesite	18,039
102	DIO_COMP	Fine Grained Diorite	14,963
103	GRD_COMP	Granodiorite	3,179
104	TON_COMP	Tonalite	6,564
105	LCDACP	Late Crowded Dacitic Porphyry	596
106	BIH_COMP	Hydrothermal Breccia	5,830
107	BXI	Igneous Contact Breccia	619
108	BFM	Diatreme / Phreatomagmatic Breccia	1,306
109	BQT	Quartz-Tourmaline Breccia	14
110	PDAC	Dacitic Porphyry	1,531
111	GRV	Gravel	5,783
112	GSD	Gravitational Slide Deposit	205
		Total (m)	58,628

Table 14.3: Metres Drilled by Mineral Zone

Mineralized Zone Code	Description		Drilled Metres
301	OVR	Overburden	5,947
302	LEA	Leached	2,964
303	SUP	Supergene	7,881
304	HYP	Hypogene	41,835
		Total (m)	58,628

Table 14.4: Metres Drilled by C-Veinlets

Veins Code	Description Frequency Veins	Drilled Metres
401	C(high)	17,271
402	C(mid)	10,598
403	C(low)	15,836
404	C(grv)	5,520
405	C(novet)	9,071
	Total (m)	58,297

Table 14.5: Metres Drilled by B-Veinlets

Veins Code	Decription Frecuency Veins	Drilled Metres
411	B(high)	11,932
412	B(mid)	15,051
413	B(low)	16,781
414	B(grv)	5,410
415	B(novet)	9,069
	Total (m)	58,243

Note: The total length of the logged veinlets does not exactly match the total length of the logged lithology. The veinlet logging was carried out in 2020, and some core sections could not be properly logged due to loss of core or sections with poor recovery. This explains the minor difference in the total length.

14.2 Geological 3-D Model and Domains

Los Andes Copper geologists prepared a sectional geological model using Micromine software, and the consulting geologists at GeoEstima then prepared the 3D geological model using Leapfrog software. From this model, Los Andes Copper prepared 3D .dxf files for lithology, structure, mineral zone, C-veinlets, B-veinlets and alteration and provided this data to Tetra Tech. The codes for each unit are shown in Table 14.6 to Table 14.9.

Table 14.6: Lithology Model Codes

Lithology	Solids.dxf	Code
Andesite	AND	101
Fine Grained Diorite	DIO_COMP	102
Granodiorite	GRD_COMP	103
Tonalite	TON_COMP	104
Late Crowded Dacitic Porphyry	LCDACP	105
Hydrothermal Breccia	BIH_COMP	106
Igneous Contact Breccia	BXI	107
Diatreme / Phreatomagmatic Breccia	BFM	108
Quartz-Tourmaline Breccia	BQT	109
Dacitic Porphyry	PDAC	110
Gravel	GRV	111
Gravitational Slide Deposit	GSD	112

Table 14.7: Mineral Zone Model Codes

Mineral Zone	Solids.dxf	Code
Overburden	GRV	301
Leached	LIX	302
Supergene	SUP	303
Hypogene	HYP	304

Table 14.8: Alteration Model Codes

Alteration	Solids.dxf	Code
Gravel	GRV	201
Guartz-Sericite-(Chlorite)	QTZ-SER	202
Argillic-Sericite-Chlorite-Carbonate	ARC-BE-CLO-CAR	203
Quartz-Chlorite	QTZ-CL	204
Quartz- K-Feldspar-Anhydrite	QTZ-KFELD-ANH	205
Biotite-Chlorite-Sericite	BIO-CL-SER	206
Biotite-Chlorite-Anhydrite	BIO-CL-ANH	207
Biotite-Magnetite-Anhydrite	BIO-MGT-ANH	208
Biotite-Chlorite-Epidote	BIO-CL-EP	209
Chlorite-Quartz-Magnetite	CL-QTZ-MGT	210
Biotite-Silica	BIO-SIL	211

Table 14.9: Veinlet Model Codes

Veinlets	Solids.dxf	Code
C Veinlets (high)	GM_VET_C - High	401
C Veinlets (mid)	GM_VET_C - Mid	402
C Veinlets (low)	GM_VET_C - Low	403
C Veinlets (grv)	GM_VET_C - GRV	404
C Veinlets (novet)	GM_VET_C - Non_Vet	405
B Veinlets (high)	GM_VET_B - High	411
B Veinlets (mid)	GM_VET_B - Mid	412
B Veinlets (low)	GM_VET_B - Low	413
B Veinlets (grv)	GM_VET_B - GRV	414
B Veinlets (novet)	GM_VET_B - Non_Vet	415
A+B+C Veinlets (high)	GM_VET_EDM_A_B_C - High	421
A+B+C Veinlets (mid)	GM_VET_EDM_A_B_C - Mid	422
A+B+C Veinlets (low)	GM_VET_EDM_A_B_C - Low	423
A+B+C Veinlets (grv)	GM_VET_EDM_A_B_C - GRV	424
A+B+C Veinlets (nvet)	GM_VET_EDM_A_B_C - Non_Vet	425
EDM Veinlets (high)	GM_VET_EDM - High	431
EDM Veinlets (mid)	GM_VET_EDM - Mid	432
EDM Veinlets (low)	GM_VET_EDM - Low	433
EDM Veinlets (grv)	GM_VET_EDM - GRV	434
EDM Veinlets (nvet)	GM_VET_EDM - Non_Vet	435

14.3 Method for Estimate and Tools

The Mineral Resources for the Vizcachitas Project were estimated using Ordinary Kriging. The Minesight™ version 15.60-2 software was used, supported by the MSDA tool for statistical analysis of the database for copper, molybdenum, silver and arsenic model statistics.

Supervisor version 8 software was used for statistical and variographic analysis.

14.4 Specific Gravity

The Vizcachitas density data set includes 3,093 density determinations using the water displacement method. The density variable in the block model was assigned with values calculated as an average in each geological estimation unit (UGE) in both the Hypogene and Supergene zones. Before this step, a statistical analysis was performed for each lithological unit to identify outliers to avoid influencing the values calculated. The densities used are shown in Table 14.10 and Table 14.11.

Table 14.10: Density by Lithology, Hypogene Zone

Lithology Numberical Code	Code	Description	Density gr/cm³
101	AND	Andesite	2.63
102	DIO_COMP	Fine Grained Diorite	2.69
103	GRD_COMP	Granodiorite	2.57
104	TON_COMP	Tonalite	2.57
105	LCDACP	Late Crowded Dacitic Porphyry	2.66
106	BIH_COMP	Hydrothermal Breccia	2.65
107	BXI	Igneous Contact Breccia	2.71
108	BFM	Diatreme / Phreatomagmatic Breccia	2.62
110	PDAC	Dacitic Porphyry	2.59

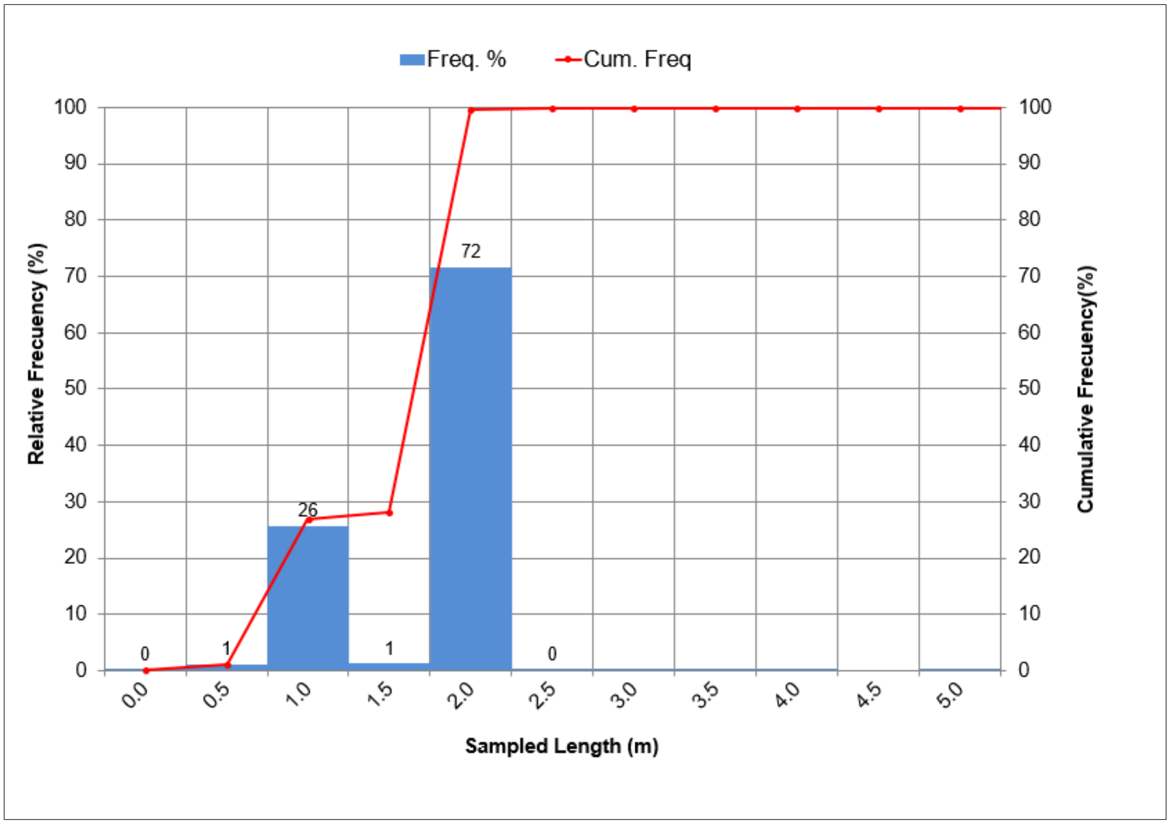
Table 14.11: Density by Lithology, Supergene Zone

Lithology Numberical Code	Code	Description	Density gr/cm³
101	AND	Andesite	2.59
102	DIO_COMP	Fine Grained Diorite	2.55
103	GRD_COMP	Granodiorite	2.48
104	TON_COMP	Tonalite	2.51
106	BIH_COMP	Hydrothermal Breccia	2.52
108	BFM	Diatreme / Phreatomagmatic Breccia	2.55
109	BQT	Quartz-Tourmaline Breccia	2.55
110	PDAC	Dacitic Porphyry	2.52

14.5 Composites

The drill hole assays from the Vizcachitas resource database were composited at 2 m intervals resulting in. 29,367 composites. A total of 72% of the original samples are 2 m in length, and 26% are 1 m in length (Figure 14.1). All composites with a total length of less than 1 m were excluded. All intervals with no sampling or no recovery were assigned a grade of 0.0% Cu.

Figure 14.1: Sample Length



Source: Los Andes Copper, 2022

14.6 Assay Statistics

Table 14.12 to Table 14.19 show the statistical analysis for each lithological, mineral and veinlet intensity zone unit for copper, molybdenum and silver (the veinlet intensity for silver was not reviewed).

14.6.1 Copper

Table 14.12: Copper Grade Statistics by Lithology

Lithology Code	Description	N° Samples	Length (m)	Minimum (%)	Maximum (%)	Mean (%)	Std. Devn.	Variance	Co. of Variation
101	AND	8,932	17,835	0.001	2.490	0.324	0.220	0.05	0.679
102	DIO_COMP	7,494	14,968	0.004	2.279	0.281	0.235	0.06	0.834
103	GRD_COMP	1,499	2,987	0.007	1.220	0.183	0.145	0.02	0.792
104	TON_COMP	3,236	6,454	0.001	1.635	0.296	0.165	0.03	0.557
105	LCDACP	273	545	0.063	1.055	0.413	0.191	0.04	0.461
106	BIH_COMP	2,984	5,954	0.003	2.530	0.406	0.287	0.08	0.706
107	BXI	224	448	0.012	0.837	0.221	0.184	0.03	0.831
108	BFM	488	968	0.002	0.851	0.101	0.114	0.01	1.125
110	PDAC	766	1,532	0.000	0.810	0.122	0.158	0.02	1.296
111	GRV	138	276	0.002	1.070	0.133	0.194	0.04	1.454
Total		26,034	51,966	0.000	2.530	0.298	0.230	0.05	0.769

Table 14.13: Copper Grade Statistics by Mineralized Zone

Mineralized Zone Code	Description	N° Samples	Length (m)	Minimum (%)	Maximum (%)	Mean (%)	Std. Devn.	Variance	Co. of Variation
301	Overburden	131	262	0.002	0.609	0.078	0.098	0.010	1.250
302	Leached	1,353	2,706	0.001	1.604	0.122	0.121	0.015	0.985
303	Supergene	3,891	7,781	0.007	2.490	0.474	0.277	0.077	0.584
304	Hypogene	20,659	41,217	0.000	2.530	0.278	0.206	0.042	0.740
Total		26,034	51,966	0.000	2.530	0.298	0.230	0.053	0.769

Table 14.14: Copper Grade Statistics by C-Veinlet Type

Veins C	Description	N° Samples	Length (m)	Minimum (%)	Maximum (%)	Mean (%)	Std. Devn.	Variance	Co. of Variation
401	C Veinlets (high)	8,510	16,991	0.013	2.490	0.426	0.192	0.037	0.451
402	C Veinlets (mid)	5,252	10,484	0.001	1.625	0.259	0.163	0.026	0.629
403	C Veinlets (low)	7,896	15,760	0.001	2.150	0.179	0.196	0.038	1.094
404	C Veinlets (grv)	138	276	0.002	1.070	0.133	0.194	0.038	1.454
405	No C Veinlets	4,238	8,454	0.000	2.530	0.320	0.286	0.082	0.894
Total		26,034	51,966	0.000	2.530	0.298	0.230	0.053	0.769

14.6.2 Molybdenum

Table 14.15: Molybdenum Grade Statistics by Lithology

Lithology Code	Description	N° Samples	Length (m)	Minimum (ppm)	Maximum (ppm)	Mean (ppm)	Std. Devn.	Variance	Co. of Variation
101	AND	8,932	17,835	0.438	4018.990	95.669	155.387	24,145	1.624
102	DIO_COMP	7,494	14,968	0.900	2850.000	94.923	158.133	25,006	1.666
103	GRD_COMP	1,499	2,987	1.430	2130.000	60.524	134.632	18,126	2.224
104	TON_COMP	3,236	6,454	1.800	5000.000	124.134	185.403	34,374	1.494
105	LCDACP	273	545	6.700	4560.000	357.940	417.875	174,619	1.167
106	BIH_COMP	2,984	5,954	0.355	5550.000	166.743	228.311	52,126	1.369
107	BXI	224	448	1.000	1225.000	49.524	99.551	9,910	2.010
108	BFM	488	968	1.000	1695.000	28.799	95.224	9,068	3.307
110	PDAC	766	1,532	0.438	2580.000	55.395	151.894	23,072	2.742
111	GRV	138	276	3.300	459.972	61.616	76.562	5,862	1.243
Total		26,034	51,966	0.355	5550.000	104.854	176.390	31,113	1.682

Table 14.16: Molybdenum Grade Statistics by Mineral Zone

Mineralized Zone Code	Description	N° Samples	Length (m)	Minimum (ppm)	Maximum (ppm)	Mean (ppm)	Std. Devn.	Variance	Co. of Variation
301	Overburden	131	262	3.300	459.972	64.427	83.696	7,005	1.299
302	Leached	1,353	2,706	0.438	781.000	57.778	79.250	6,281	1.372
303	Supergene	3,891	7,781	0.600	1,972.030	101.108	122.825	15,086	1.215
304	Hypogene	20,659	41,217	0.355	5,550.000	108.908	189.071	35,748	1.736
Total		26,034	51,966	0.355	5,550.000	104.854	176.390	31,113	1.682

Table 14.17: Molybdenum Grade Statistics by B-Veinlet Type

Veins B	Description	N° Samples	Length (m)	Minimum (ppm)	Maximum (ppm)	Mean (ppm)	Std. Devn.	Variance	Co. of Variation
411	B Veinlets (high)	8,510	16,991	0.013	2.490	0.426	0.192	0.037	0.451
412	B Veinlets (mid)	5,252	10,484	0.001	1.625	0.259	0.163	0.026	0.629
413	B Veinlets (low)	7,896	15,760	0.001	2.150	0.179	0.196	0.038	1.094
414	B Veinlets (grv)	138	276	0.002	1.070	0.133	0.194	0.038	1.454
415	No B Veinlets	4,238	8,454	0.000	2.530	0.320	0.286	0.082	0.894
Total		26,034	51,966	0.000	2.530	0.298	0.230	0.053	0.769

14.6.3 Silver

Table 14.18: Silver Grade Statistics by Lithology

Lithology Code	Description	N° Samples	Length (m)	Minimum (ppm)	Maximum (ppm)	Mean (ppm)	Std. Devn.	Variance	Co. of Variation
101	AND	6,425	12,826	0.010	53.900	1.025	1.026	1.05	1.001
102	DIO_COMP	6,370	12,723	0.020	9.490	0.780	0.609	0.37	0.781
103	GRD_COMP	1,115	2,222	0.030	8.618	0.755	0.606	0.37	0.803
104	TON_COMP	2,159	4,302	0.030	100.000	1.190	2.366	5.60	1.989
105	LCDACP	273	545	0.060	2.780	0.985	0.504	0.25	0.512
106	BIH_COMP	2,421	4,828	0.030	100.000	1.163	2.199	4.84	1.891
107	BXI	218	436	0.040	5.010	0.808	0.726	0.53	0.899
108	BFM	373	742	0.014	47.884	0.452	2.891	8.36	6.390
110	PDAC	490	980	0.010	60.300	0.640	2.758	7.61	4.309
111	GRV	106	212	0.040	3.340	0.881	0.728	0.53	0.826
Total		19,950	39,817	0.010	100.000	0.942	1.434	2.06	1.522

Table 14.19: Silver Grade Statistics by Mineral Zone

Mineralized Zone Code	Description	N° Samples	Length (m)	Minimum (ppm)	Maximum (ppm)	Mean (ppm)	Std. Devn.	Variance	Co. of Variation
301	Overburden	97	194	0.040	3.340	0.909	0.755	0.569	0.831
302	Leached	1,080	2,160	0.023	100.000	0.933	3.603	12.978	3.860
303	Supergene	2,799	5,597	0.010	17.450	1.330	0.985	0.971	0.741
304	Hypogene	15,974	31,866	0.010	100.000	0.875	1.218	1.484	1.393
Total		19,950	39,817	0.010	100.000	0.942	1.434	2.056	1.522

Note: GMC and Los Andes Copper in 2007-2008 only assessed for copper and molybdenum using atomic absorption. Since 2017 Los Andes Copper has assayed all core using multielement ICP-MS and has re-assayed some of the historical pulp samples using the same method. A total of 19,950 samples have now been assayed for silver.

14.7 Exploratory Data Analysis

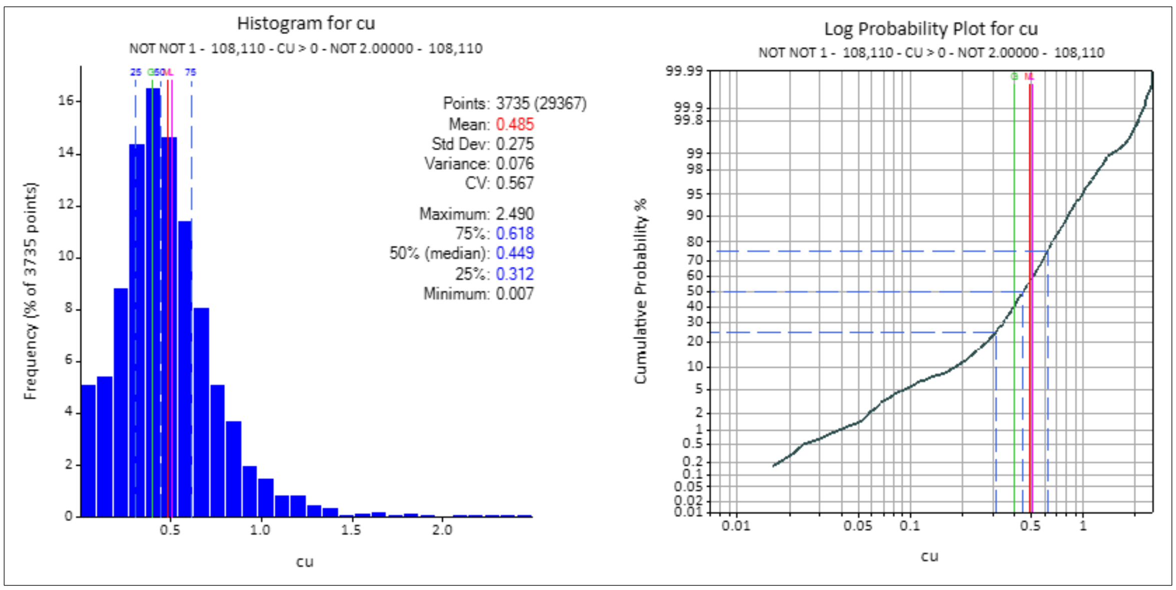
The exploratory data analysis (EDA) complemented the statistical analysis carried out in the previous stage, with histograms and log probability graphs to define the geological estimation units based on a combination of lithological units, veinlet intensity and mineral zones.

14.7.1 Copper

14.7.1.1 Histogram and Log Probability Plots

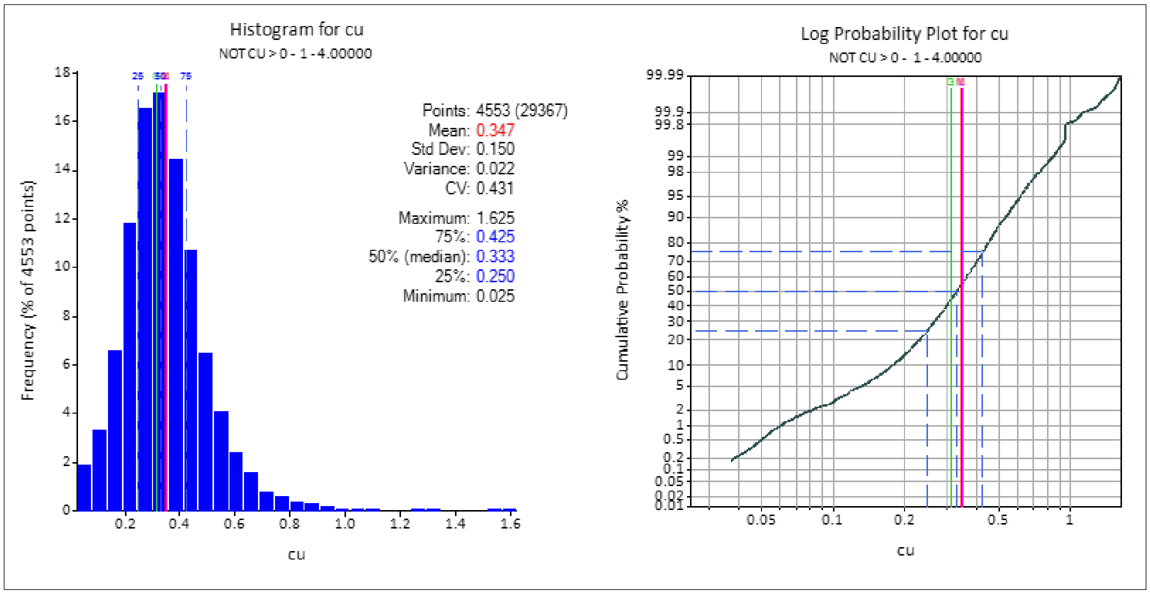
The copper EDA defined 12 geological estimation units for copper (UGECU). The histograms and log probability graphs (Figure 14.2 to Figure 14.7) show the most significant UGEs. These six units represent approximately 70% of the total mineral resources.

Figure 14.2: Histogram and Log Probability Graph – UGECU2



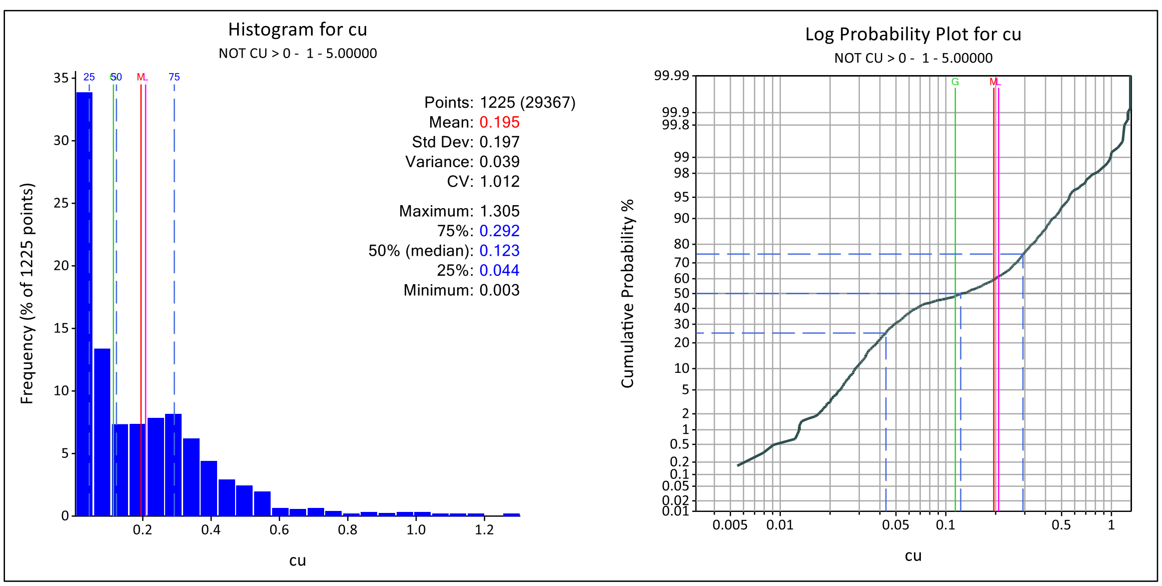
Source: Los Andes Copper, 2022

Figure 14.3: Histogram and Log Probability Graph – UGECU4



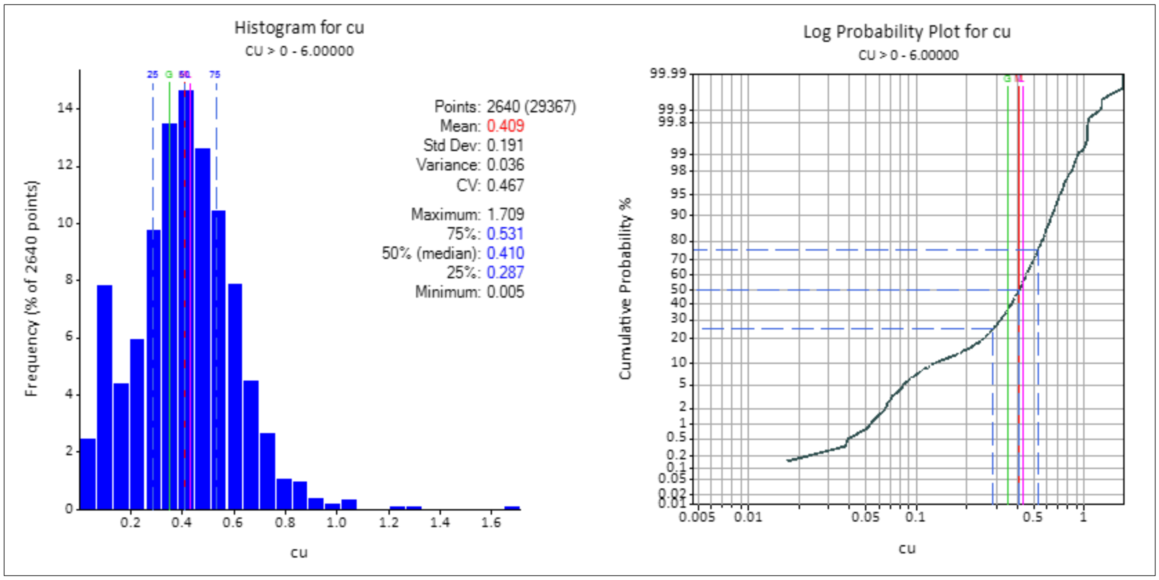
Source: Los Andes Copper, 2022

Figure 14.4: Histogram and Log Probability Graph – UGECU 5



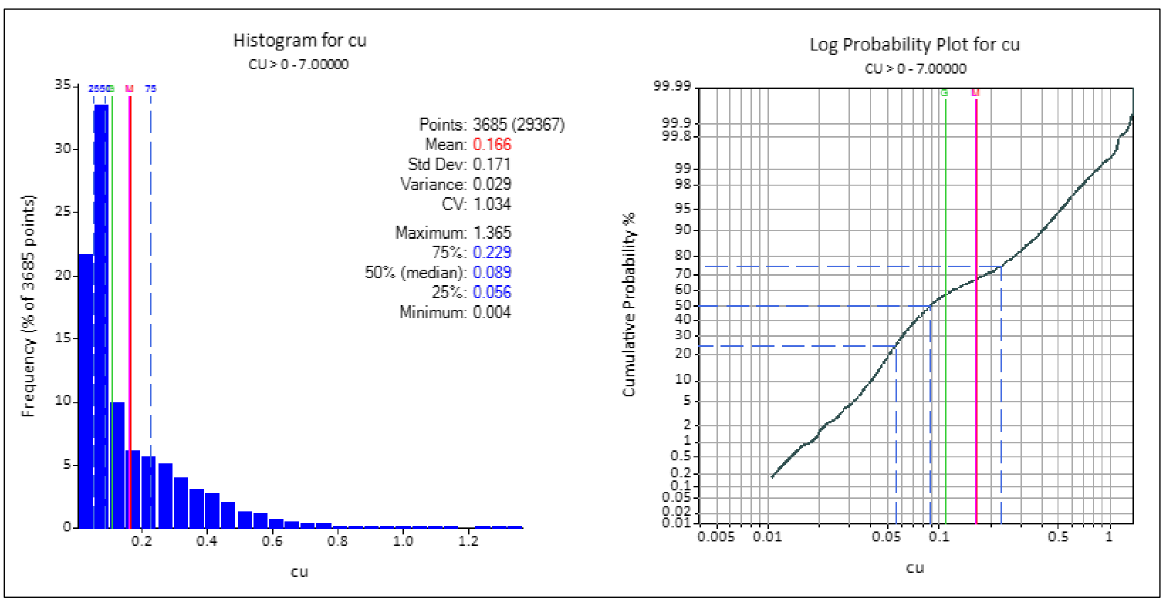
Source: Los Andes Copper, 2022

Figure 14.5: Histogram and Log Probability Graph – UGECU 6



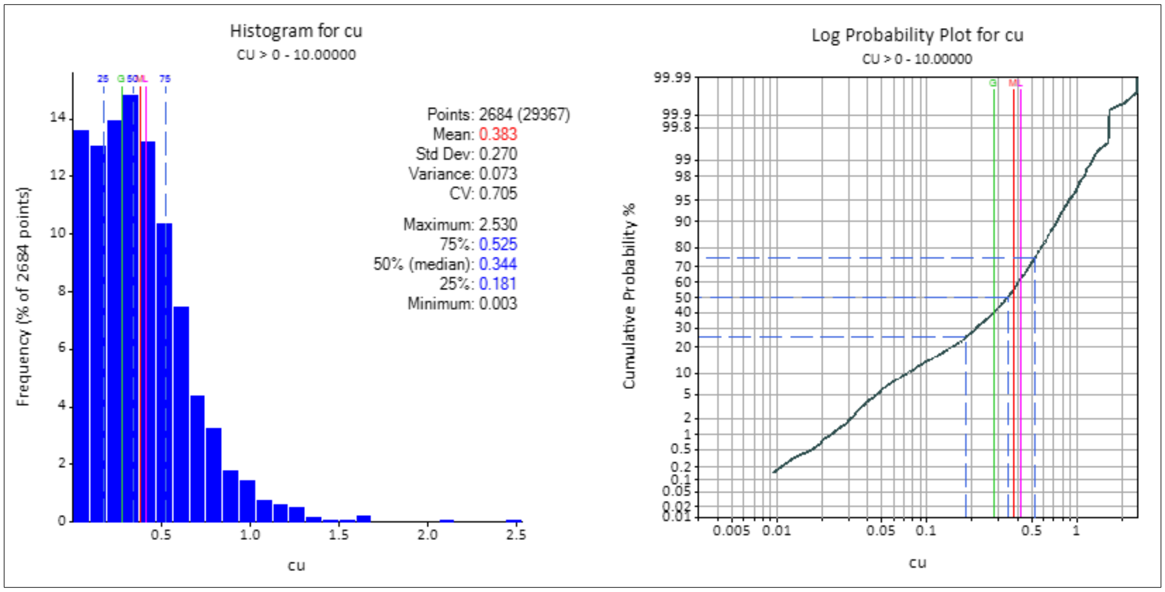
Source: Los Andes Copper, 2022

Figure 14.6: Histogram and Log Probability Graph – UGECU 7



Source: Los Andes Copper, 2022

Figure 14.7: Histogram and Log Probability Graph – UGECU 10



Source: Los Andes Copper, 2022

The copper estimation units (UGECU) were built from the lithology and C-veinlet domains, except for the Leached and Supergene units. Andesite and Diorite units were sub-divided into two classes using the C-veinlet intensity. The Granodiorite and Tonalite units did not need to be sub-divided as there is only one statistical population (Table 14.20 and Table 14.21).

Table 14.20: Copper Estimation Units

UGECU	Lithology		C-type veinlets		Mineralized Zone
	Code	Description	Code	Description	
1	-	-	-	-	Leached
2	-	-	-	-	Supergene
3	108 / 110	Diatreme & Dacitic Porphyry	-	-	Supergene
4	101	Andesite	401 / 402	C veinlets (high-med)	Hypogene
5	101	Andesite	403	C veinlets (low)	Hypogene
6	102	Diorites	401 / 402	C veinlets (high-med)	Hypogene
7	102	Diorites	403	C veinlets (low)	Hypogene
8	103	Granodiorites	-	-	Hypogene
9	104	Tonalites	-	-	Hypogene
10	106 /107	Hydrothermal & Igneous Breccia	-	-	Hypogene
11	105	Late Crowded Dacitic Porphyry	-	-	Hypogene
12	South of Campamento Fault				

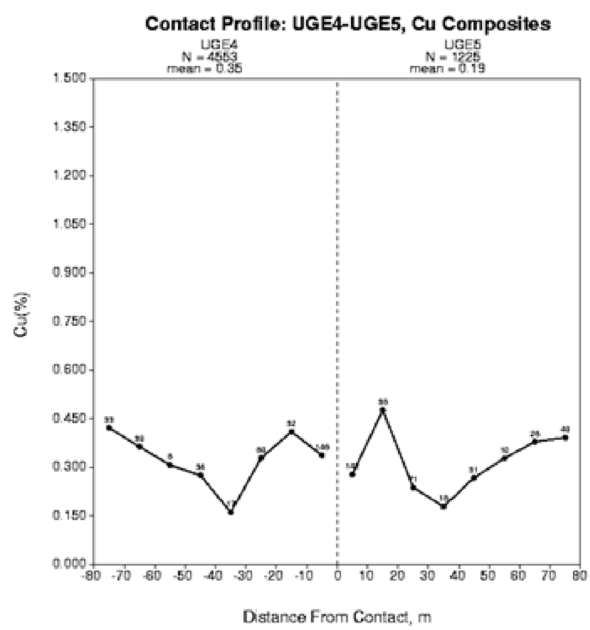
Table 14.21: Copper Statistics by UGECU

UGECU	N° Samples	Length (m)	Minimun (%)	Maximun (%)	Mean (%)	Standard Deviation
0	131	262	0.002	0.735	0.084	0.114
1	1,290	2,580	0.001	1.604	0.127	0.121
2	3,735	7,469	0.007	2.490	0.485	0.275
3	1,060	2,112	0.000	0.851	0.128	0.150
4	4,553	9,087	0.025	1.625	0.347	0.150
5	1,225	2,443	0.003	1.305	0.195	0.197
6	2,640	5,269	0.005	1.709	0.409	0.191
7	3,685	7,361	0.004	1.365	0.166	0.171
8	1,305	2,599	0.007	1.220	0.173	0.135
9	2,495	4,973	0.004	1.031	0.271	0.126
10	2,684	5,354	0.003	2.530	0.383	0.270
11	273	545	0.063	1.055	0.413	0.191
12	958	1,912	0.001	0.938	0.095	0.111
Total	26,034	51,966	0.000	2.530	0.298	0.229

14.7.1.2 Contact Analysis

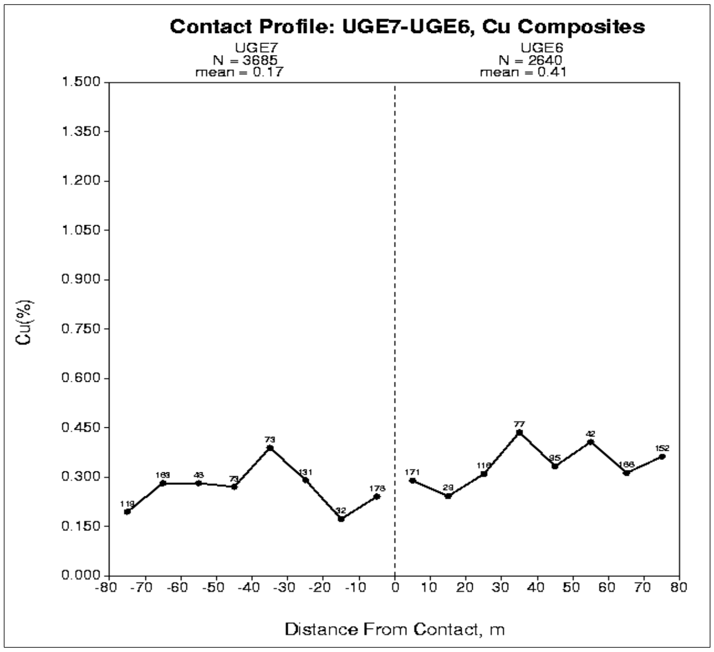
Contact analysis was performed on the UGEs, particularly those with a combination of lithology and veins, where a gradual transition from one UGE to another neighbour is expected. Figure 14.8 to Figure 14.10 show the units that present a gradual transition (soft contacts) for copper grades between units. These soft contacts mean that when estimating the block-grade composites near the contact, the neighbouring units were also included in the kriging estimation.

Figure 14.8: UGECU4 v/s UGECU5 Contact Analysis



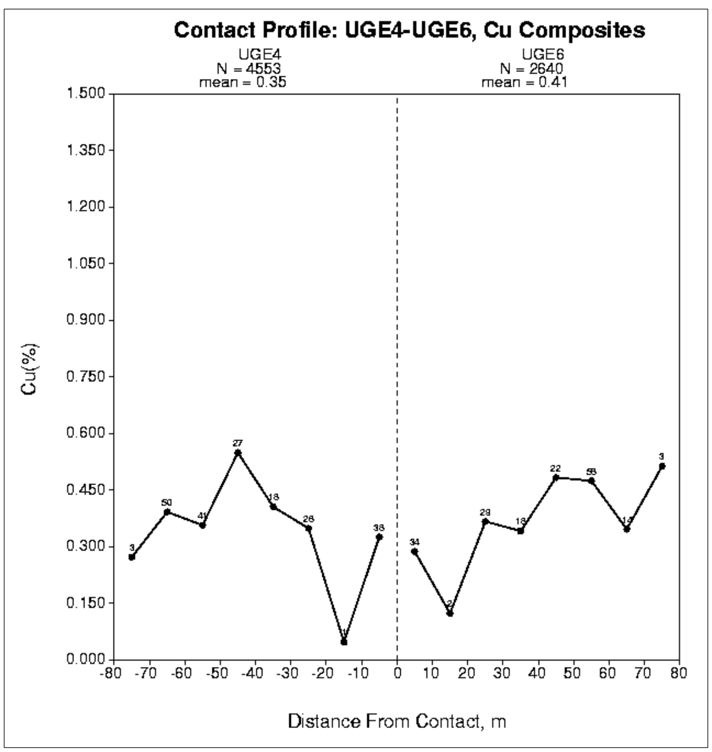
Source: Los Andes Copper, 2022

Figure 14.9: UGECU7 v/s UGECU6 Contact Analysis



Source: Los Andes Copper, 2022

Figure 14.10: UGECU4 v/s UGECU6 Contact Analysis



Source: Los Andes Copper, 2022

14.7.2 Molybdenum

14.7.2.1 Histogram and Log Probability Plots

Molybdenum estimation units (UGEMO) were defined from the lithology and B-veinlets units (Table 14.22 and Table 14.23). The B-veinlet intensity has a strong correlation with the molybdenum grade. Histogram and log probability plots of the units are presented in Figure 14.11 to Figure 14.14.

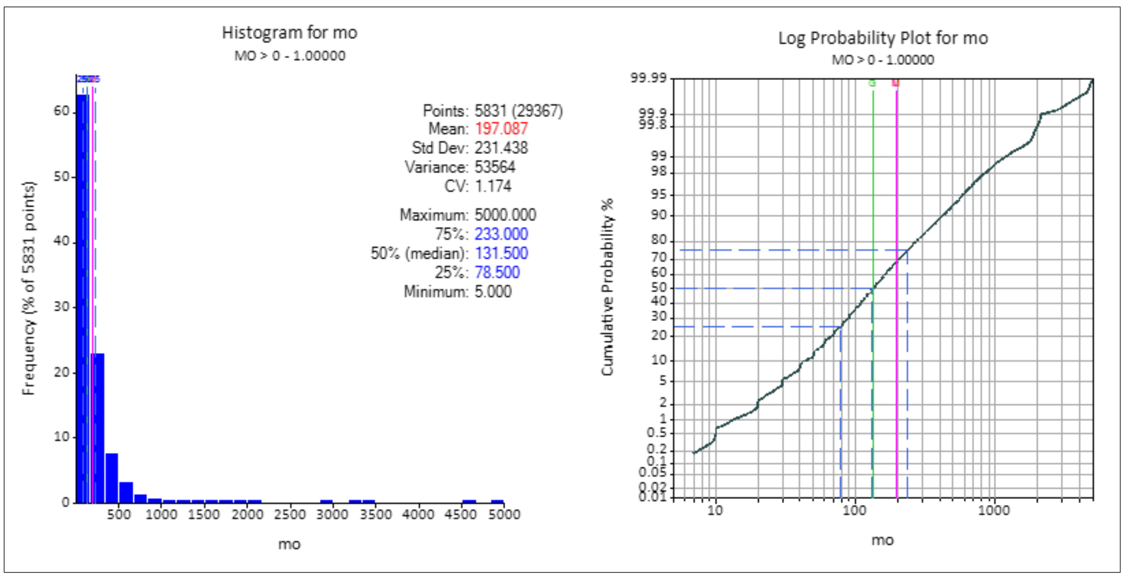
Table 14.22: Molybdenum Estimate Units

UGEMO	Lithology	B-type veinlets		Mineralized
	Description	Code	Description	Zone
1	All Lithologies Less Breccias, Diatreme and Dacitic Porphyry	411	B Veinlets (high)	All
2		412	B Veinlets (mid)	All
3		413	B Veinlets (low)	All
4	Hydrothermal & Igneous Breccia	-	-	All
5	No B Veinlets Less Breccias	-	-	All
6	Diatreme & Dacitic Porphyry	-	-	All
7	South of Campamento Fault	-	-	All

Table 14.23: Composites Statistics for Molybdenum by UGEMO

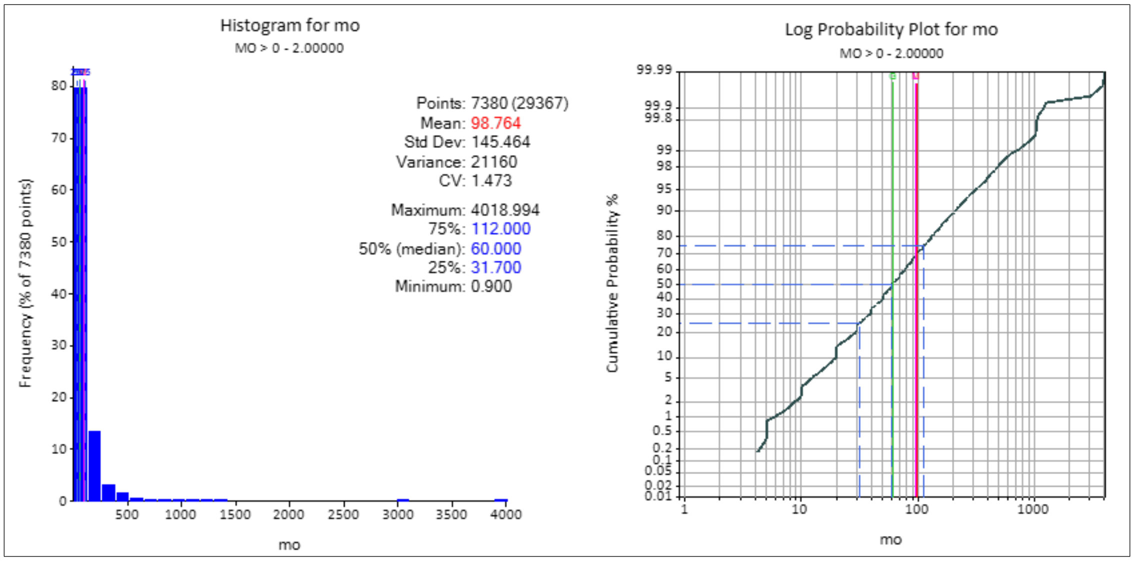
UGEMO	N° Samples	Length (m)	Minimum (ppm)	Maximum (ppm)	Mean (ppm)	Std. Devn.	Variance	Co. of Variation
1	5,831	11,646	5.000	5,000	196.901	231.258	53,480	1.174
2	7,380	14,733	0.900	4,019	98.783	145.507	21,172	1.473
3	7,361	14,689	0.438	2,409	36.508	84.092	7,071	2.303
4	3,208	6,402	0.355	5,550	158.543	223.755	50,066	1.411
5	21	42	5.000	160	49.992	48.306	2,333	0.966
6	1,163	2,318	0.438	2,580	47.825	138.141	19,083	2.888
7	958	1,912	0.940	370	11.881	22.784	519	1.918
Total	25,922	51,742	0.355	5,550	105.049	176.669	31,212	1.682

Figure 14.11: Molybdenum Histogram and Log Probability Graph - UGEMO1



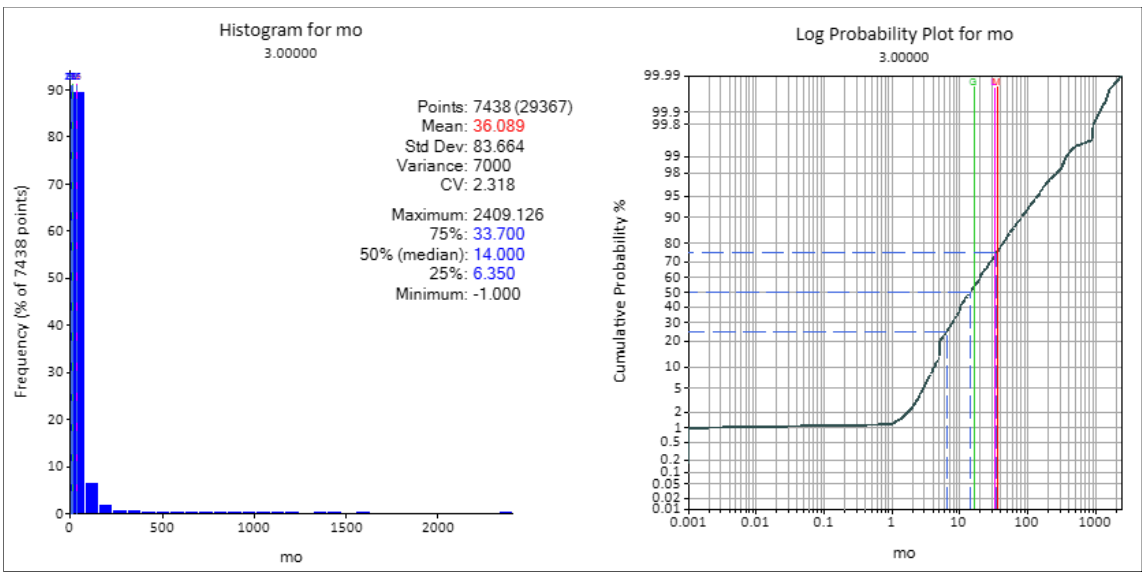
Source: Los Andes Copper, 2022

Figure 14.12: Molybdenum Histogram and Log Probability Graph - UGEMO2



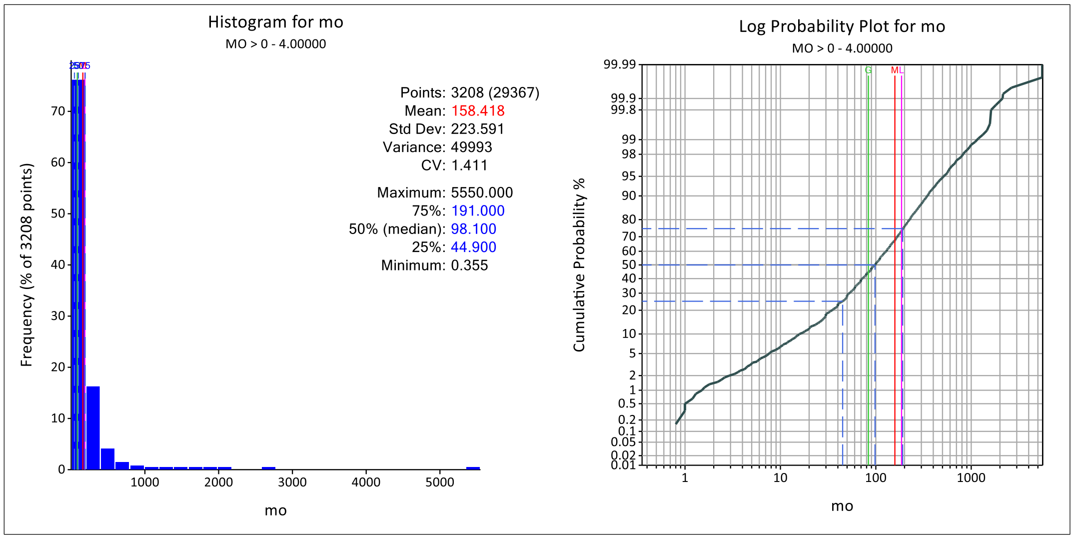
Source: Los Andes Copper, 2022

Figure 14.13: Molybdenum Histogram and Log Probability Graph - UGEMO3



Source: Los Andes Copper, 2022

Figure 14.14: Molybdenum Histogram and Log Probability Graph - UGEMO4

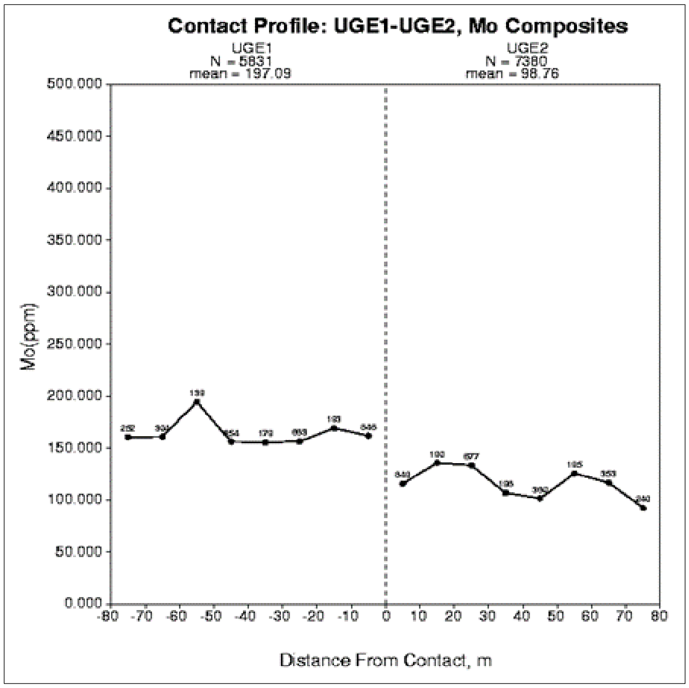


Source: Los Andes Copper, 2022

14.7.2.2 Contact Analysis

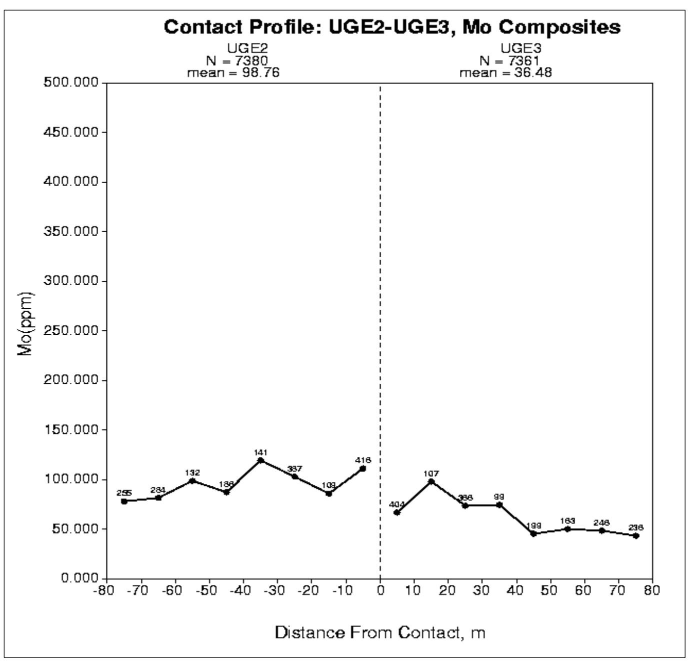
Contact analysis between the molybdenum domains was carried out to determine the type of contact between each molybdenum estimation unit. Figure 14.15 to Figure 14.19 show a gradual transition of the grade between units (soft contact). For the soft contacts, composites from both UGEs were used for the estimation.

Figure 14.15: UGEMO1 v/s UGEMO2 Contact Analysis



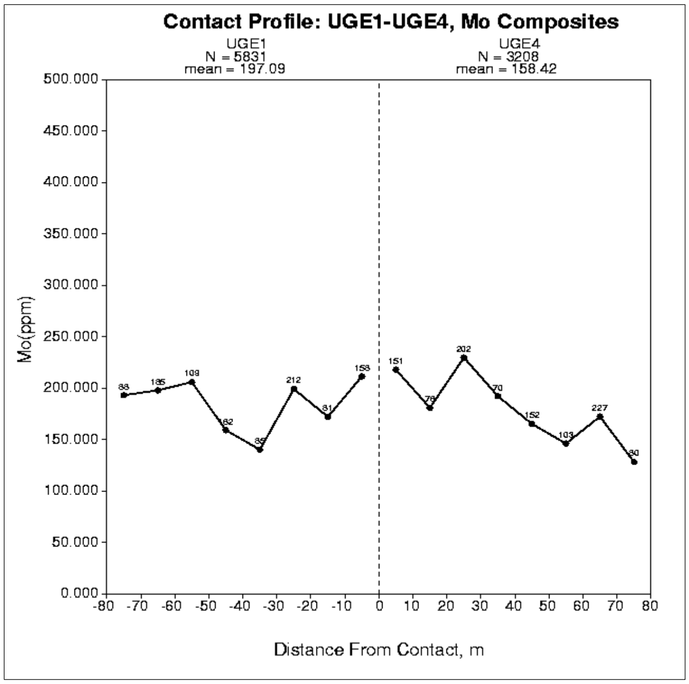
Source: Los Andes Copper, 2022

Figure 14.16: UGEMO2 v/s UGEMO3 Contact Analysis



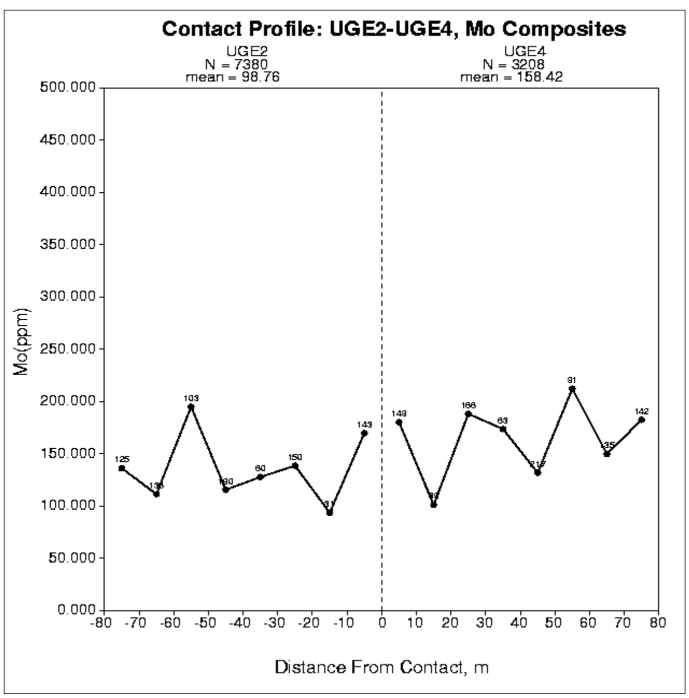
Source: Los Andes Copper, 2022

Figure 14.17: UGEMO1 v/s UGEMO4 Contact Analysis



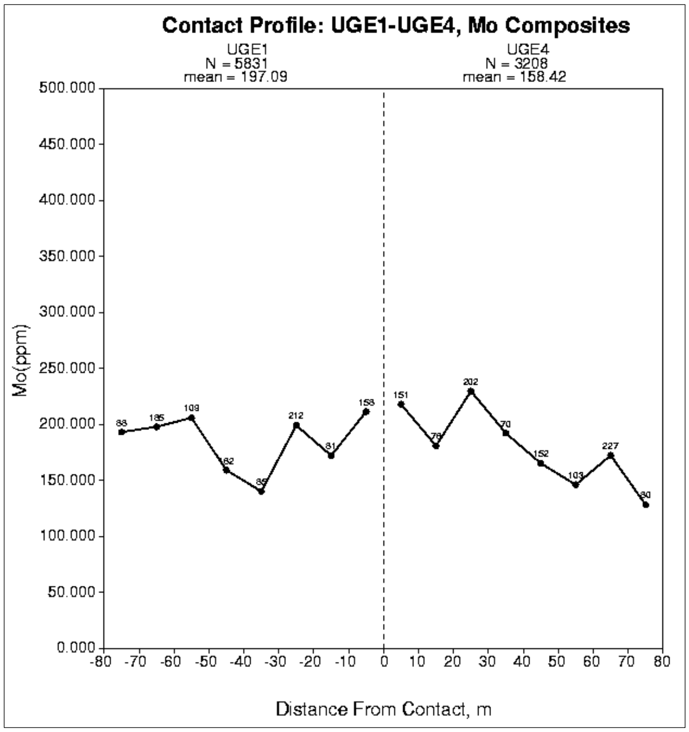
Source: Los Andes Copper, 2022

Figure 14.18: UGEMO2 v/s UGEMO4 Contact Analysis



Source: Los Andes Copper, 2022

Figure 14.19: UGEMO1 v/s UGEMO4 Contact Analysis



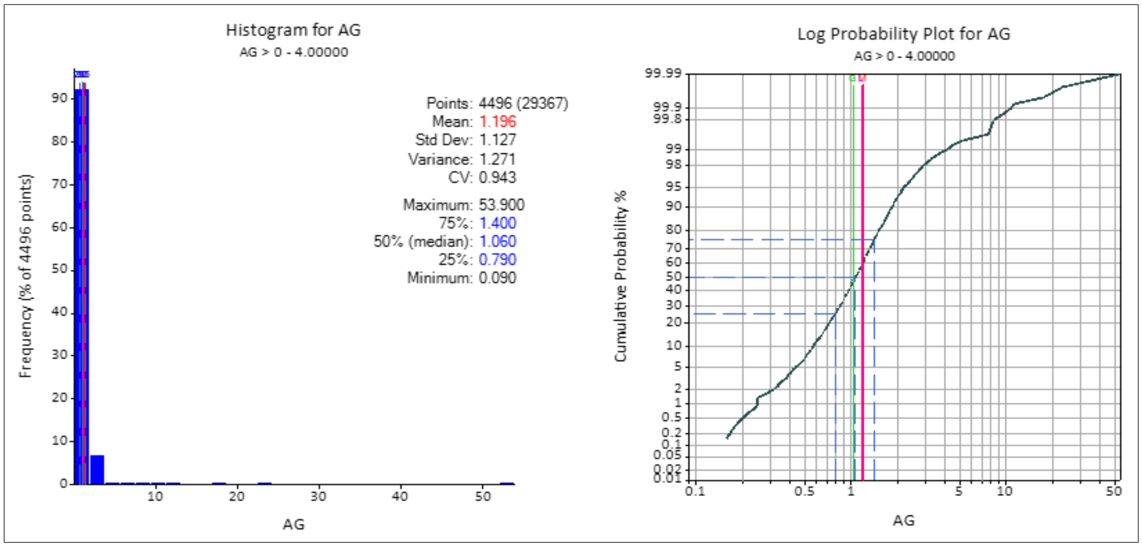
Source: Los Andes Copper, 2022

14.7.3 Silver

14.7.3.1 Histogram and Log Probability Plots

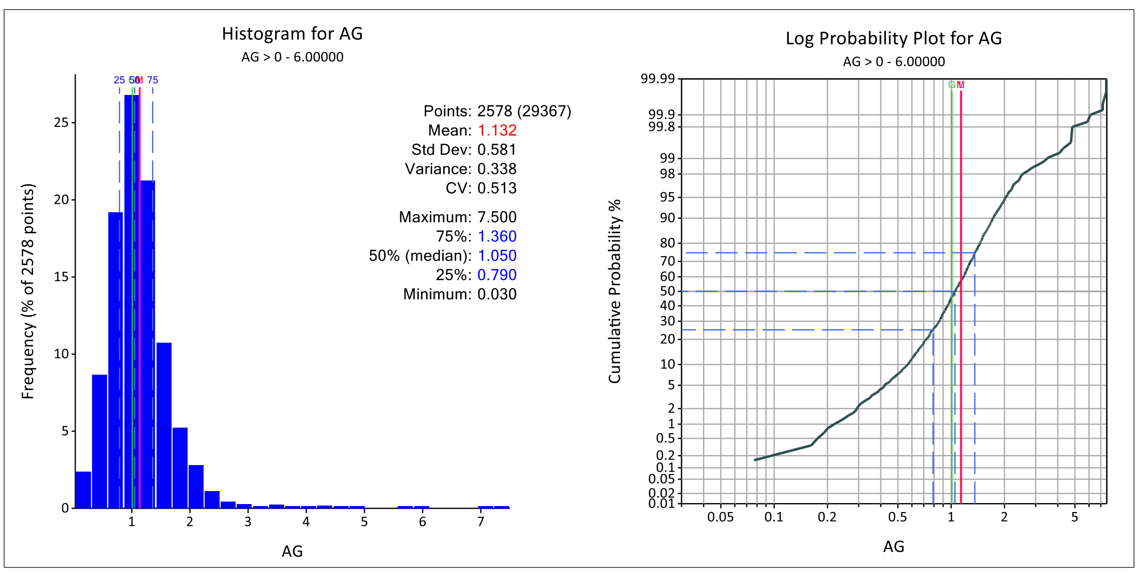
The silver estimation units (UGEAG) were defined based on the copper units. The histograms and log probability graphs (Figure 14.20 to Figure 14.22) show geological units for silver estimation representative of the deposit.

Figure 14.20: Silver Histogram and Log Probability Graph - UGEAG4



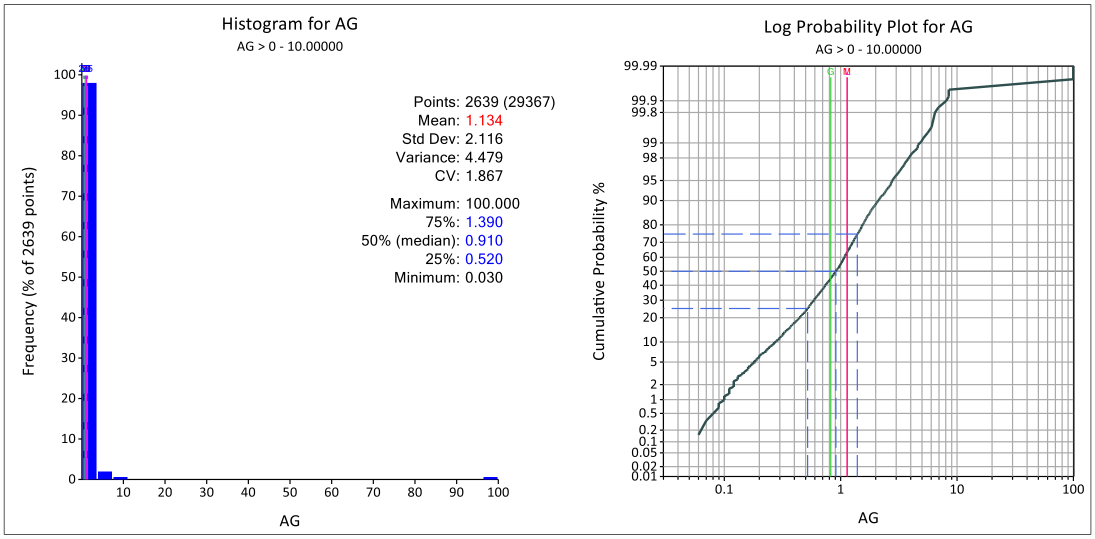
Source: Los Andes Copper, 2022

Figure 14.21: Silver Histogram and Log Probability Graph - UGEAG6



Source: Los Andes Copper, 2022

Figure 14.22: Silver Histogram and Log Probability Graph - UGEAG10



Source: Los Andes Copper, 2022

Table 14.24 compares the copper and silver UGEs, and Table 14.25 shows the composite statistics for silver UGEs.

Table 14.24: Copper Estimation Units Compared to Silver Estimation Units

UGECU	Lithology		C-type veinlets		UGEAG
	Code	Description	Code	Description	
1	-	-	-	-	-
2	-	-	-	-	-
3	108 / 110	Diatreme & Dacitic porphyry	-	-	3
4	101	Andesite	401 / 402	C veinlets (high-med)	4
5	101	Andesite	403	C veinlets (low)	5
6	102	Diorites	401 / 402	C veinlets (high-med)	6
7	102	Diorites	403	C veinlets (low)	7
8	103	Granodiorites	-	-	8
9	104	Tonalites	-	-	9
10	106 /107	Hydrothermal & Igneous breccia	-	-	10
11	105	Late crowded dacitic porphyry	-	-	11
12	South of Campamento Fault				12

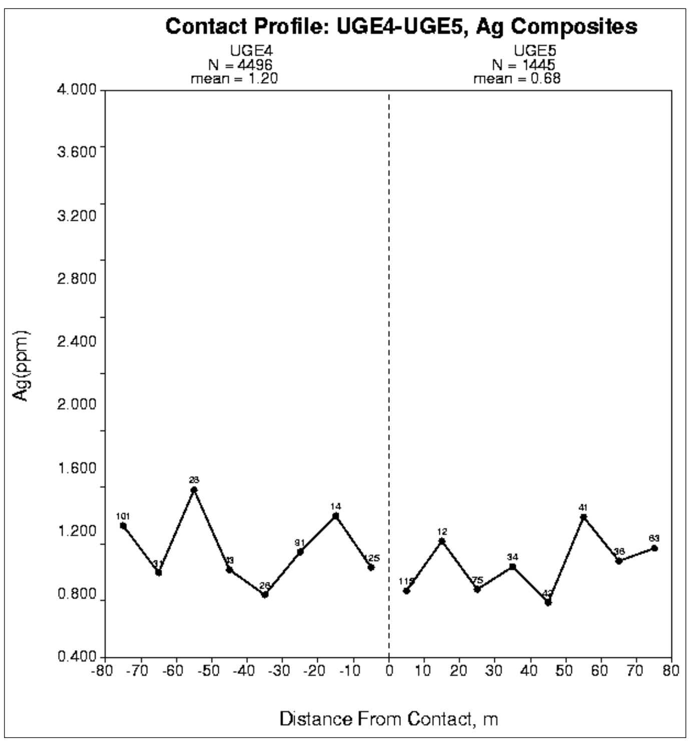
Table 14.25: Composites Statistics for Silver by UEAG

UGEAG	N° Samples	Length (m)	Minimum (ppm)	Maximum (ppm)	Mean (ppm)	Std. Devn.	Variance	Co. of Variation
3	373	742.09	0.014	47.884	0.4524	2.8907	8.3562	6.3902
4	4,496	8,975.18	0.090	53.900	1.1964	1.1279	1.2722	0.9427
5	1,445	2,885.87	0.010	5.910	0.6779	0.569	0.3238	0.8393
6	2,578	5,147.73	0.030	7.500	1.1325	0.5814	0.3381	0.5134
7	3,792	7,575.49	0.020	9.490	0.5398	0.5004	0.2504	0.927
8	1,115	2,221.63	0.030	8.618	0.7547	0.6057	0.3668	0.8026
9	2,159	4,302.45	0.030	100.000	1.1896	2.3656	5.5961	1.9886
10	2,639	5,264.31	0.030	100.000	1.1337	2.1186	4.4883	1.8687
11	273	545.15	0.060	2.780	0.9853	0.5039	0.254	0.5115
12	490	977.25	0.040	5.720	0.4625	0.4811	0.2314	1.0403
Total	19,360	38,637.15	0.010	100.000	0.9499	1.3859	1.9206	1.4589

14.7.3.2 Contact Analysis

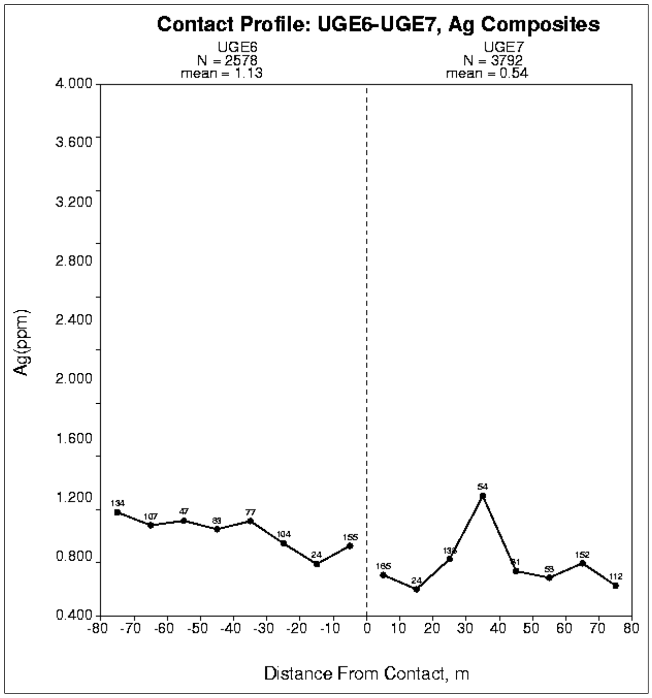
Contact analysis was performed at the contact between the silver UGEs. Figure 14.23 to Figure 14.25 show a gradual transition of silver grade between the estimation units (soft contacts).

Figure 14.23: UGEAG4 v/s UGEAG5 Contact Analysis



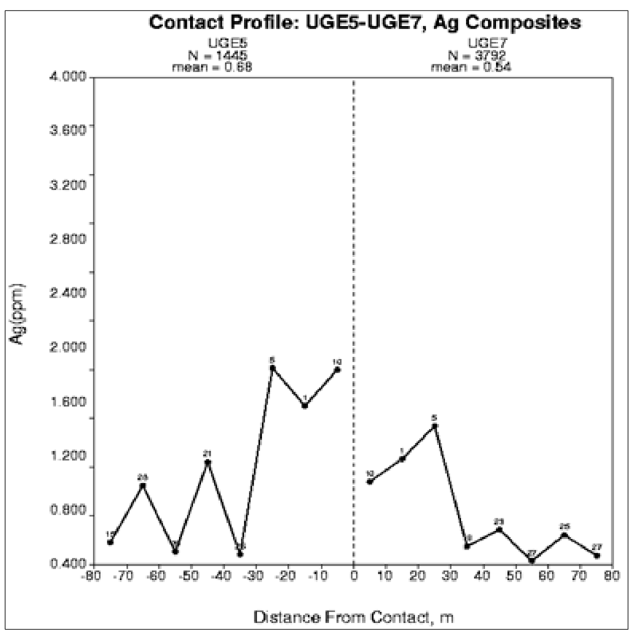
Source: Los Andes Copper, 2022

Figure 14.24: UGEAG6 v/s UGEAG7 Contact Analysis



Source: Los Andes Copper, 2022

Figure 14.25: UGEAG5 v/s UGEAG7 Contact Analysis



Source: Los Andes Copper, 2022

14.8 High-Grade Capping

High-grade capping was implemented after reviewing the log probability graphs (Section 14.7) in combination with the EDA, using a step function of 15% (breaks or jumps in the ordered grades). The Cu, Mo and Ag outliers are shown in Table 14.26 to Table 14.28, respectively.

Table 14.26: Copper Grade Outliers by UGECU

UGECU	Topcut Value Cu (%)	N° of Samples
1	0.7	5
2	No/Capp	0
3	0.2	14
4	1.1	5
5	No/Capp	0
6	1.03	9
7	No/Capp	0
8	0.75	7
9	No/Capp	0
10	1.55	8
11	No/Capp	0
12	0.2	107

Table 14.27: Molybdenum Grade Outliers by UGEMO

UGEMO	Topcut Value Mo (ppm)	N° of Samples
1	2,127	7
2	1,297	4
3	1,016	10
4	1,600	7
5	400	7
6	10	546
7	10	263

Table 14.28: Silver Grade Outliers by UGEAG

UGEAG	Topcut Value Ag (g/t)	N° of Samples
3	1.7	2
4	9.0	7
5	3.0	7
6	4.8	6
7	3.2	17
8	3.1	8
9	5.8	13
10	6.3	6
11	No Capp	0
12	2.0	6

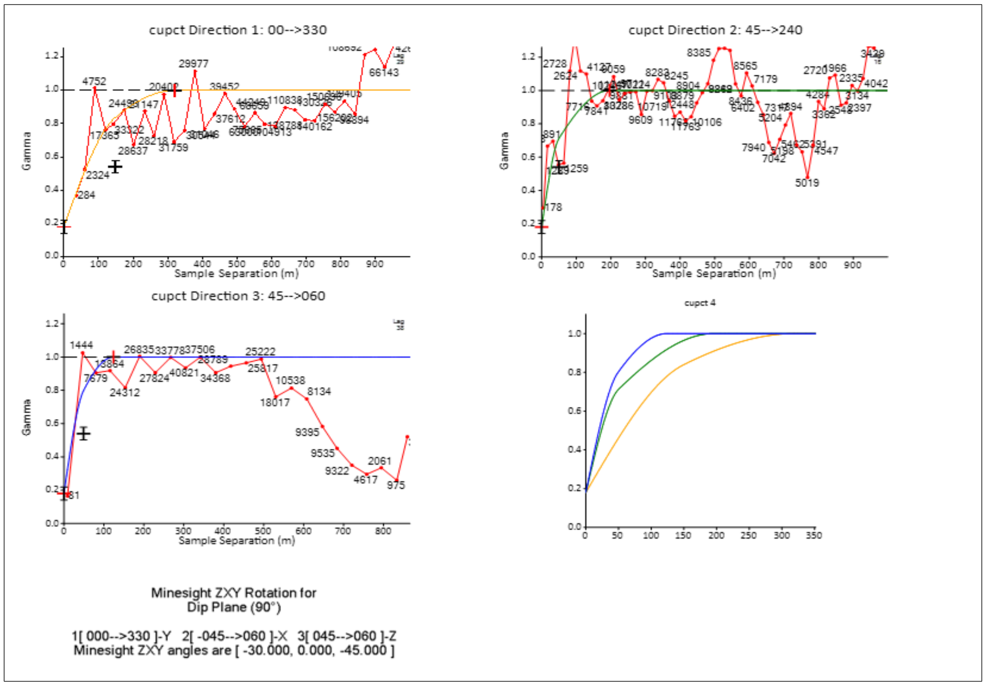
14.9 Spatial Analysis - Variography

Variographic analysis was carried out to determine the direction, anisotropies and extent of the mineralization for each geological estimation unit. This analysis was performed for the copper, molybdenum and silver as shown below.

14.9.1 Copper

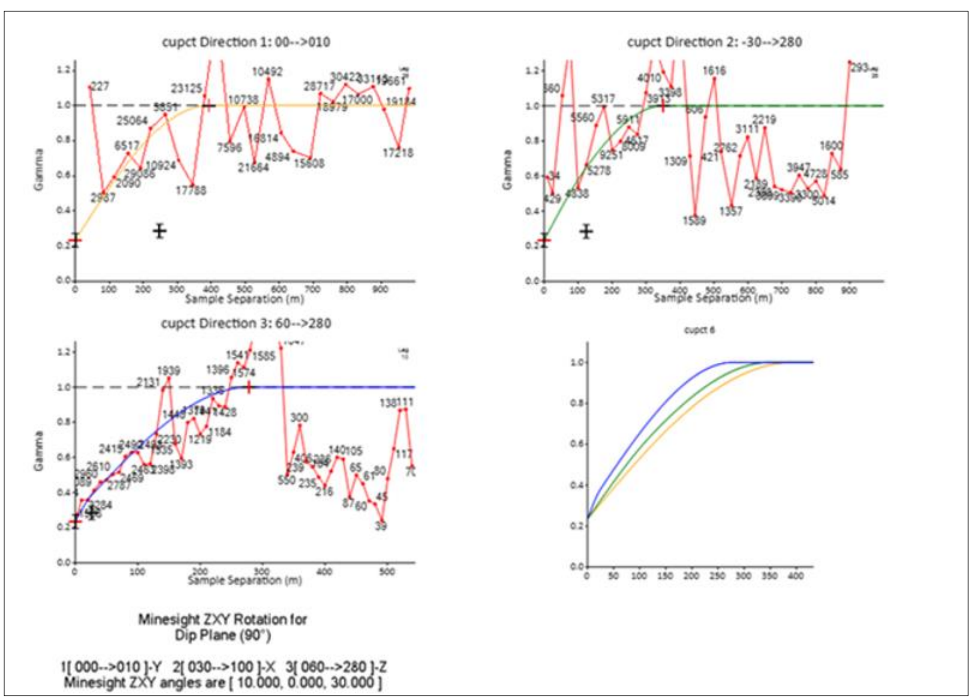
The variography for Cu is shown in Figure 14.26 to Figure 14.28 and the variographic structures are shown in Table 14.29 (Structures 1 and 2) and Table 14.30 (Structures 3 and 4).

Figure 14.26: UGECU4 Variography



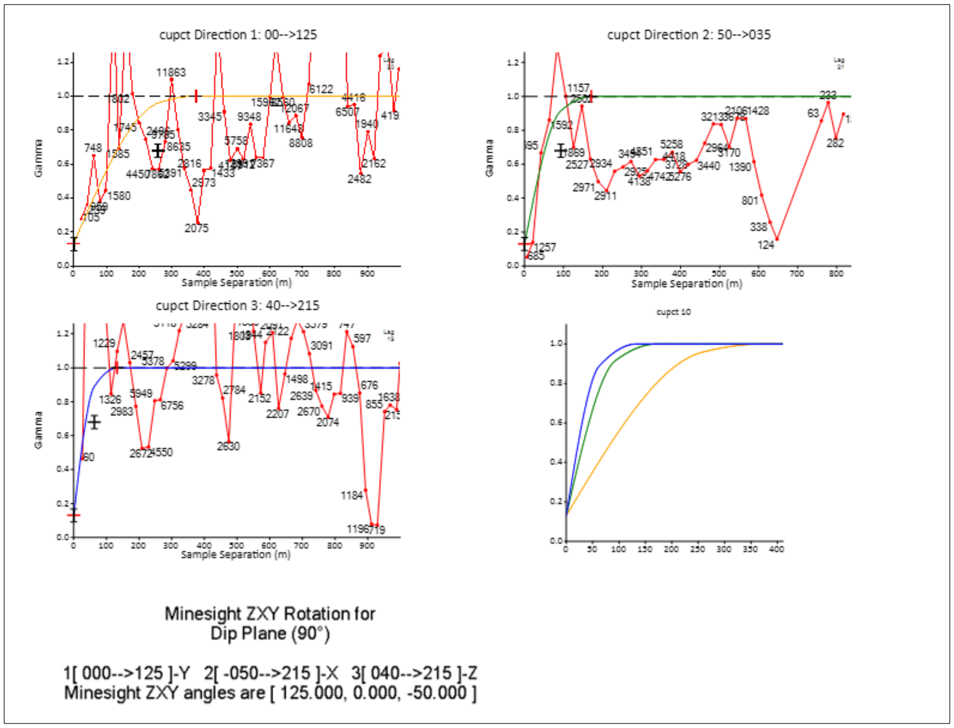
Source: Los Andes Copper, 2022

Figure 14.27: UGECU6 Variography



Source: Los Andes Copper, 2022

Figure 14.28: UGECU10 Variography



Source: Los Andes Copper, 2022

Table 14.29: Copper Variographic Models for Structures 1 and 2

UGECU	Angles			Variance Contribution			Direction First Structure			Direction Second Structure		
	Z	X	Y	C o	C1	C2	D 1	D2	D3	D 1	D2	D3
1	114.090	-11.436	-9.772	0.170	0.260	0.570	163	41	8	253	145	21
2	105.000	0.000	0.000	0.130	0.230	0.640	80	89	24	257	206	49
4	-30.000	0.000	-45.000	0.180	0.360	0.460	149	50	49	320	196	124
5	110.000	0.000	-25.000	0.020	0.010	0.970	564	127	156	683	296	219
6	10.000	0.000	30.000	0.230	0.050	0.720	248	124	26	393	350	278
7	160.000	0.000	0.000	0.010	0.060	0.840	236	272	13	468	324	195
8	115.000	0.000	-40.000	0.060	0.370	0.570	304	105	41	305	228	164
9	145.000	0.000	-15.000	0.180	0.200	0.620	98	149	46	450	247	228
10	125.000	0.000	-50.000	0.130	0.550	0.320	258	93	63	374	171	133
11	135.000	0.000	0.000	0.290	0.250	0.460	172	269	237	310	364	346
12	110.000	0.000	-25.000	0.020	0.010	0.970	564	127	156	683	296	219

Table 14.30: Copper Variographic Models for Structures 3 and 4

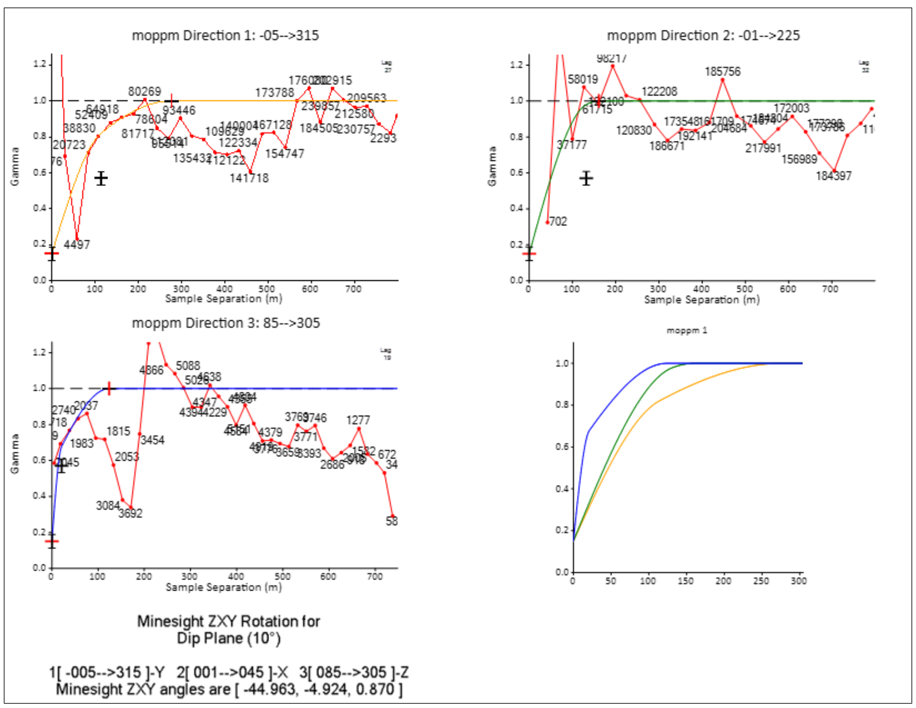
UGECU	Direction Third Structure			N° Samples		Max. N° Drill Holes	Direction Fourth Structure			N° Samples		Max. N° Drill Holes
	D 1	D2	D3	Min	Max		D 1	D2	D3	Min	Max	
1	375	220	200	4	24	2						
2	390	315	90	3	24	2						
4	480	285	180	3	24	2	675	595	495	2	24	2
5	595	445	335	3	24	2	600	600	600	2	24	2
6	585	525	420	3	24	2	645	645	545	2	24	2
7	690	480	300	3	24	2	740	695	640	2	24	2
8	455	342	245	3	24	2	600	600	490	2	24	2
9	670	370	340	3	24	2	700	700	500	2	24	2
10	560	255	200	3	24	2	690	690	500	2	24	2
11	465	545	515	3	24	2	595	595	545	2	24	2

The angles and ranges obtained from the models were incorporated into the MineSight modelling software that Tetra Tech used for the resource estimation. The adjusted theoretical variogram corresponds to the spherical model.

14.9.2 Molybdenum

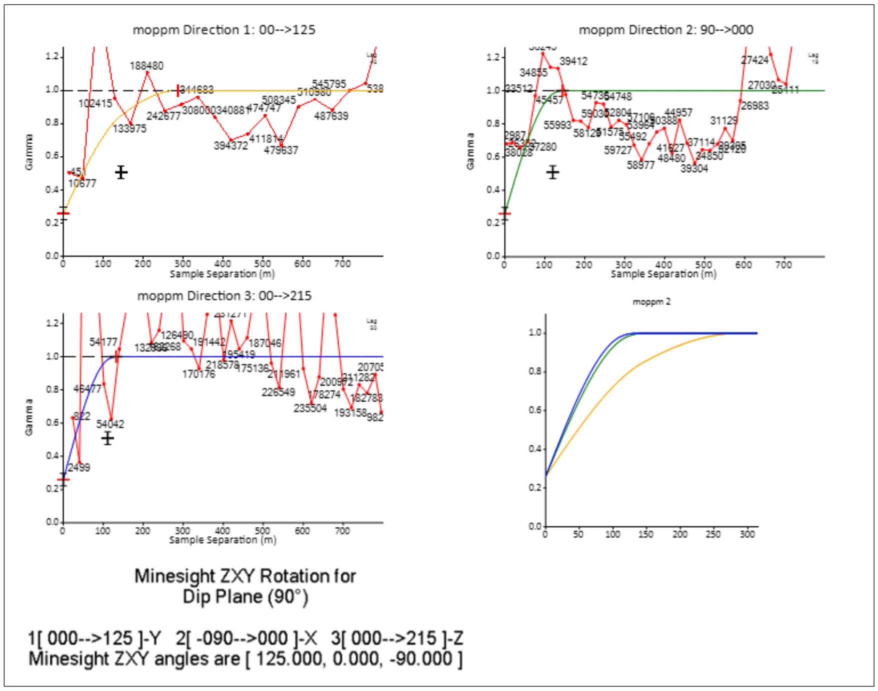
The variography for Mo is shown in Figure 14.29 to Figure 14.31 and the variographic structures are shown in Table 13.31 and Table 13.32.

Figure 14.29: Variography UGEMO 1



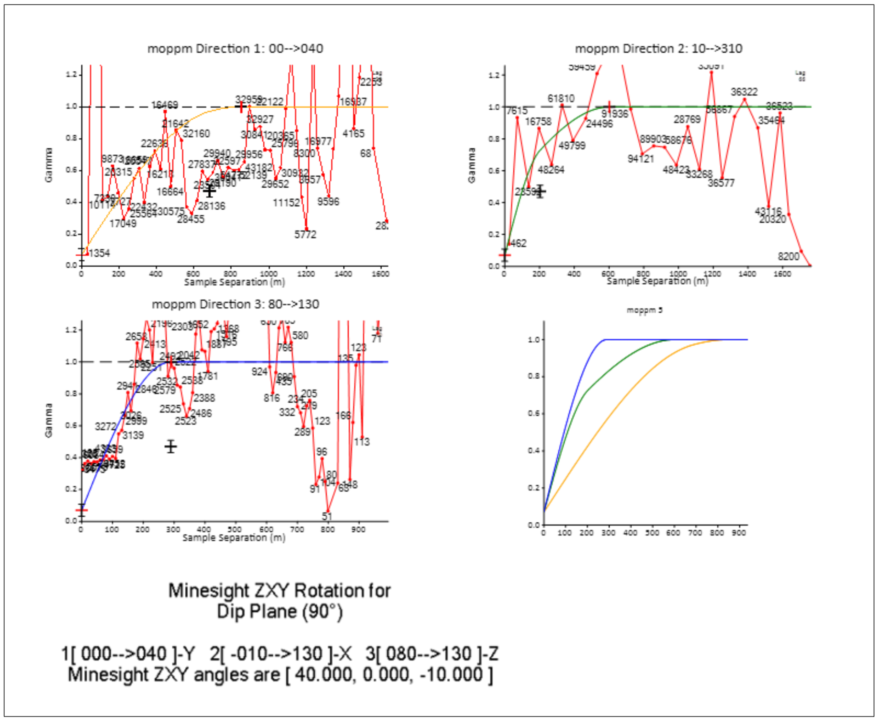
Source: Los Andes Copper, 2022

Figure 14.30: Variography UGEMO 2



Source: Los Andes Copper, 2022

Figure 14.31: Variography UGEMO 4



Source: Los Andes Copper, 2022

Table 14.31: Molybdenum Variographic Model for Structures 1 and 2

UGEMO	Angles			Variance Contribution			Direction First Structure			Direction Second Structure		
	Z	X	Y	C o	C1	C2	D 1	D2	D3	D 1	D2	D3
1	-44.963	-4.924	0.870	0.150	0.420	0.430	114	132	21	277	161	124
2	125.000	0.000	-90.000	0.260	0.250	0.490	145	120	111	287	144	132
3	15.000	0.000	0.000	0.040	0.240	0.720	599	211	77	675	530	369
4	40.000	0.000	-10.000	0.070	0.400	0.530	683	203	289	852	602	290
5	55.000	0.000	0.000	0.150	0.070	0.780	547	263	222	548	357	223
6	40.000	0.000	-10.000	0.070	0.400	0.530	683	203	289	852	602	290
7	15.000	0.000	0.000	0.040	0.240	0.720	599	211	77	675	530	369

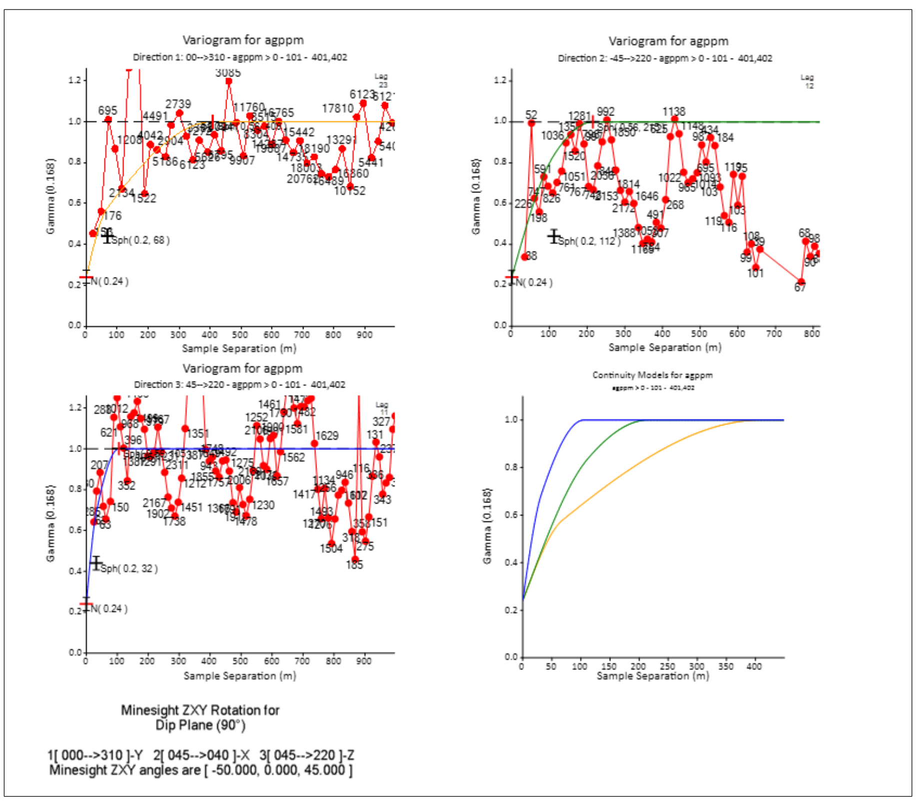
Table 14.32: Molybdenum Variographic Model for Structures 3 and 4

UGEMO	Direction Third Structure			N° Samples		Max. N° Drill Holes	Direction Fourth Structure			N° Samples		Max. N° Drill Holes
	D 1	D2	D3	Min	Max		D 1	D2	D3	Min	Max	
1	350	250	250	4	24	2	400	400	400	4	24	2
2	400	300	200	4	24	2	400	400	400	4	24	2
3	680	550	400	4	24	2	850	650	550	3	24	2
4	850	600	300	4	24	2	900	700	400	3	24	2
5	650	450	350	4	24	2	700	600	500	4	24	2

14.9.3 Silver

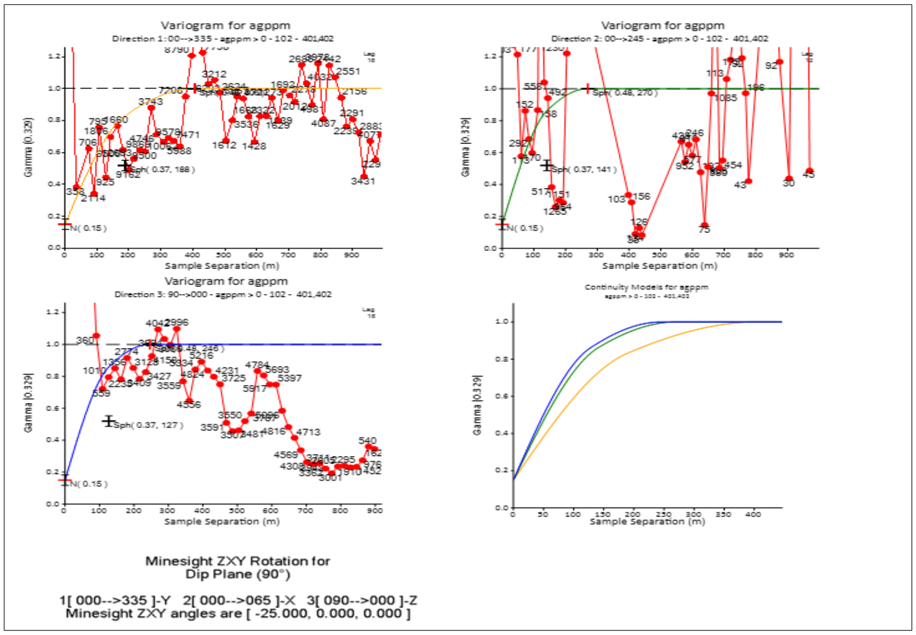
The variography for Ag is shown in Figure 14.32 to Figure 14.34 and the variographic structures are shown in Table 14.33.

Figure 14.32: Variography UGEAG 4



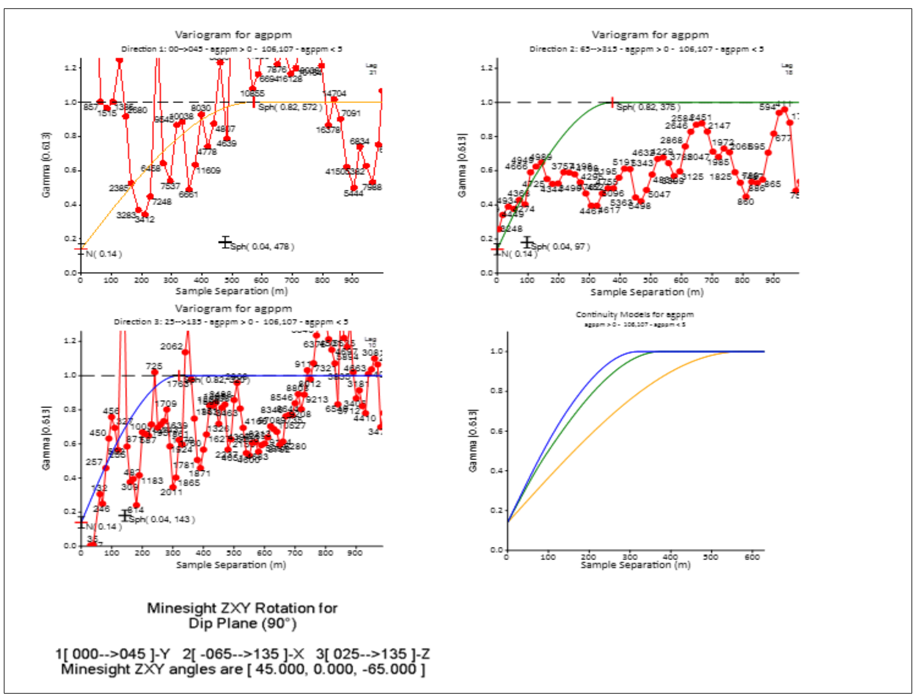
Source: Los Andes Copper, 2022

Figure 14.33: Variography UGEAG 6



Source: Los Andes Copper, 2022

Figure 14.34: Variography UGEAG 10



Source: Los Andes Copper, 2022

Table 14.33: Silver Variographic Model

UGEAG	Angles			Variance Contribution			Direction First Structure			Direction Second Structure		
	Z	X	Y	C o	C1	C2	D 1	D2	D3	D 1	D2	D3
3	120.000	0.000	-50.000	0.250	0.270	0.480	186.000	95.000	111.000	327.000	221.000	168.000
4	-50.000	0.000	45.000	0.240	0.230	0.530	68.000	112.000	32.000	409.000	215.000	105.000
5	90.000	0.000	-105.000	0.270	0.240	0.490	129.000	141.000	175.000	350.000	268.000	176.000
6	-25.000	0.000	0.000	0.150	0.370	0.480	188.000	141.000	127.000	406.000	270.000	246.000
7	25.000	0.000	80.000	0.290	0.110	0.600	353.000	186.000	189.000	519.000	478.000	278.000
8	10.000	0.000	0.000	0.220	0.360	0.420	337.000	118.000	34.000	338.000	275.000	217.000
9	-75.000	0.000	0.000	0.260	0.460	0.280	167.000	141.000	135.000	343.000	277.000	244.000
10	45.000	0.000	-65.000	0.140	0.040	0.820	478.000	97.000	143.000	572.000	375.000	320.000
11	0.000	0.000	90.000	0.180	0.030	0.790	95.000	95.000	95.000	252.000	252.000	252.000
12	90.000	0.000	-105.000	0.270	0.240	0.490	129.000	141.000	175.000	350.000	268.000	176.000

14.10 Resource Block Model

The Vizcachitas block model originates at UTM WGS84 East 364,900, North 6,412,800 and an elevation of 900 masl. The block dimensions are 20 m x 20 m x 15 m (Table 14.34).

Table 14.34: Block Model Dimensions

	East	North	Elev
Minimun	364,900	6,412,000	400
Maximun	367,060	6,414,700	3,040
Blocks (m)	20	20	15
Blocks N°	108	135	176

14.10.1 Interpolation Plan

14.10.1.1 Copper

Table 14.35 shows the Kriging interpolation plan for the first and second searches, R1 and R2. Table 14.36 shows the Kriging plan for the third and fourth searches, R3 and R4.

Table 14.35: Ordinary Kriging Estimate Plan, Copper Grade (R1 and R2)

UGECU	First Search			Second Search			Topcut value	N° Samples		Max. Comp.DH
	Radius 1	Radius 2	Radius 3	Radius 1	Radius 2	Radius 3		Min	Max	
1	160	40	40	250	145	80	0.70	4	24	3
2	80	90	30	260	210	60	No/Capp	4	24	3
4	150	50	50	320	190	120	1.10	4	24	3
5	250	130	150	400	300	220	No/Capp	4	24	3
6	250	120	30	390	350	280	1.03	4	24	3
7	240	270	30	460	320	200	No/Capp	4	24	3
8	300	105	41	305	228	164	0.75	4	24	3
9	98	149	46	450	247	228	No/Capp	4	24	3
10	258	93	63	374	171	133	1.55	4	24	3
11	172	269	237	310	364	346	No/Capp	4	24	3
12	600	600	600				0.20			3

Table 14.36: Ordinary Kriging Estimate Plan, Copper Grade (R3 and R4)

UGECU	Third Search			N° Samples		Max. Comp.DH	Fourth Search			N° Samples		Max. Comp.DH
	Radius 1	Radius 2	Radius 3	Min	Max		Radius 1	Radius 2	Radius 3	Min	Max	
1	375	220	200	4	24	2						
2	390	315	90	3	24	2						
4	480	285	180	3	24	2	675	595	495	2	24	2
5	595	445	335	3	24	2	600	600	600	2	24	2
6	585	525	420	3	24	2	645	645	545	2	24	2
7	690	480	300	3	24	2	740	695	640	2	24	2
8	455	342	245	3	24	2	600	600	490	2	24	2
9	670	370	340	3	24	2	700	700	500	2	24	2
10	560	255	200	3	24	2	690	690	500	2	24	2
11	465	545	515	3	24		595	595	545			

14.10.1.2 Molybdenum

Table 14.37 shows the Kriging interpolation plan for the first and second searches, R1 and R2. Table 14.38 shows the Kriging plan for the third and fourth searches, R3 and R4.

Table 14.37: Ordinary Kriging Estimate Plan, Molybdenum Grade (R1 and R2)

UGEMO	First Search			Second Search			Topcut value	N° Samples		Max. Comp.DH
	Radius 1	Radius 2	Radius 3	Radius 1	Radius 2	Radius 3		Min	Max	
1	114	132	21	277	161	124	2,127	4	24	3
2	145	120	111	287	144	132	1,297	4	24	3
3	400	211	77	675	530	369	1,016	4	24	3
4	400	203	289	852	602	290	1,600	4	24	3
5	400	263	222	548	357	223	400	4	24	3
6	800	600	300				10	4	24	3
7	850	650	550				10	4	24	3

Table 14.38: Ordinary Kriging Estimate Plan, Molybdenum Grade (R3 and R4)

UGEMO	Third Search			N° Samples		Max. Comp.DH	Fourth Search			N° Samples		Max. Comp.DH
	Radius 1	Radius 2	Radius 3	Min	Max		Radius 1	Radius 2	Radius 3	Min	Max	
1	350	250	250	4	24	2	400	400	400	4	24	2
2	400	300	200	4	24	2	400	400	400	4	24	2
3	680	550	400	4	24	2	850	650	550	3	24	2
4	850	600	300	4	24	2	900	700	400	3	24	2
5	650	450	350	4	24	2	700	600	500	4	24	2
6												
7												

14.10.1.3 Silver

Table 14.39 shows the Kriging interpolation plan for the first and second searches, R1 and R2. Table 14.40 shows the kriging plan for the third and fourth searches, R3 and R4.

Table 14.39: Ordinary Kriging Estimate Plan, Silver Grade (R1 and R2)

UGEAG	First Search			Second Search			Topcut Value	N° Samples		Max. Comp.DH
	R1	R2	R3	R1	R2	R3		Min	Max	
3	180	100	100	300	200	160	1.68	4	24	3
4	70	110	30	400	215	100	9.00	4	24	3
5	120	140	170	350	270	250	3.00	4	24	3
6	180	140	120	400	270	240	4.80	4	24	3
7	350	180	180	500	450	270	3.20	4	24	3
8	300	120	30	330	270	250	3.10	4	24	3
9	160	140	130	300	270	240	5.80	4	24	3
10	350	100	140	500	375	320	6.30	4	24	3
11	95	95	95	250	250	250	nocapp	4	24	3
12	600	600	600				2.0	4	24	3

Table 14.40: Ordinary Kriging Estimate Plan, Silver Grade (R3 and R4)

UGEAG	Third Search			N° Samples		Max. Comp.DH	Fourth Search			N° Samples		Max. Comp.DH
	R1	R2	R3	Min	Max		R1	R2	R3	Min	Max	
3	450	390	375	3	24	2	645	645	590	3	24	2
4	480	285	180	3	24	2	675	595	495	2	24	2
5	595	445	335	3	24	2	600	600	600	2	24	2
6	585	525	420	3	24	2	645	645	545	2	24	2
7	690	480	300	3	24	2	740	695	640	2	24	2
8	455	342	245	3	24	2	600	600	490	2	24	2
9	670	370	340	3	24	2	700	700	500	2	24	2
10	560	255	200	3	24	2	690	690	500	2	24	2
11	465	545	515	3	24	2	595	595	545	2	24	2
12												

14.11 Block Model Validation

The Ordinary Kriging block model estimate for both copper and molybdenum grades were validated using the Nearest Neighbour (NN) estimation for radii 1 and 2 of the Kriging estimate.

A visual inspection was also carried out section by section, comparing the grades for copper, molybdenum and silver estimated with Ordinary Kriging and the composite grades.

14.11.1 Copper

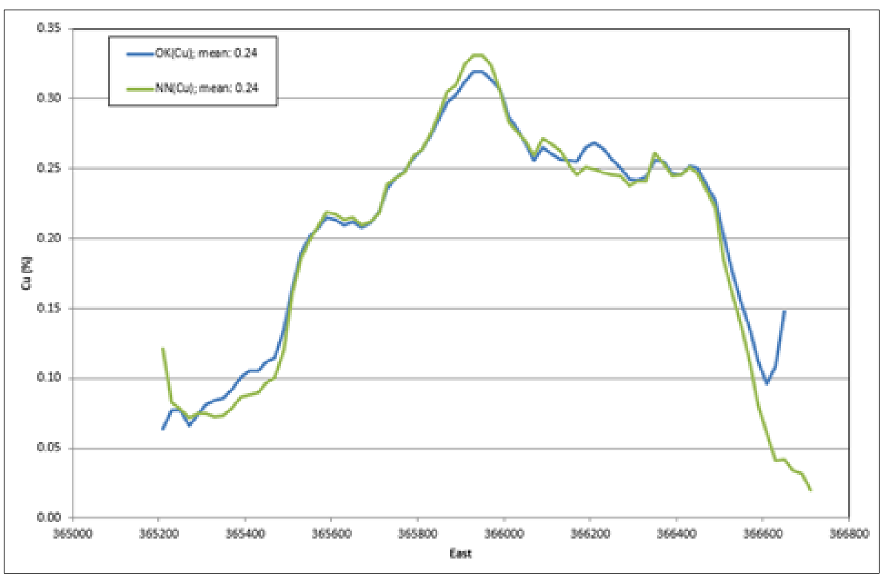
The Mineral Resource estimate was completed using Ordinary Kriging and validated by comparing the results to those generated using the Nearest Neighbour method. The maximum deviation was approximately 5%, considered acceptable by general industry standards (Table 14.41).

Table 14.41: Copper Grade Estimate Validation

UGECU	Composites (1) Mean (%)	Decluster (2) Mean(%)	OK (3) Mean (%)	Diff.(a) (1)-(3)/(1)	Diff.(b) (2)-(3)/(2)	NN (4) Mean CUT	(OK-NN)/OK (3)-(4)/(3)	N° of Blocks	Distribution % Blocks
1	0.110	0.103	0.094	17%	9.2%	0.088	6.0%	8,118	2%
2	0.469	0.393	0.421	11%	-7.1%	0.423	-0.5%	13,566	3%
4	0.336		0.323	4%		0.311	3.7%	31,003	8%
5	0.158	0.161	0.175	-10%	-8.9%	0.170	3.2%	95,812	23%
6	0.408		0.411	-1%		0.414	-0.6%	30,792	7%
7	0.165	0.165	0.152	9%	8.1%	0.159	-4.8%	135,800	33%
8	0.173		0.174	-1%		0.170	2.2%	25,695	6%
9	0.271	0.260	0.248	9%	4.4%	0.236	4.7%	47,102	11%
10	0.383		0.382	0%		0.379	0.6%	20,807	5%
11	0.413	0.418	0.457	-10%	-9.4%	0.436	4.6%	2,181	1%

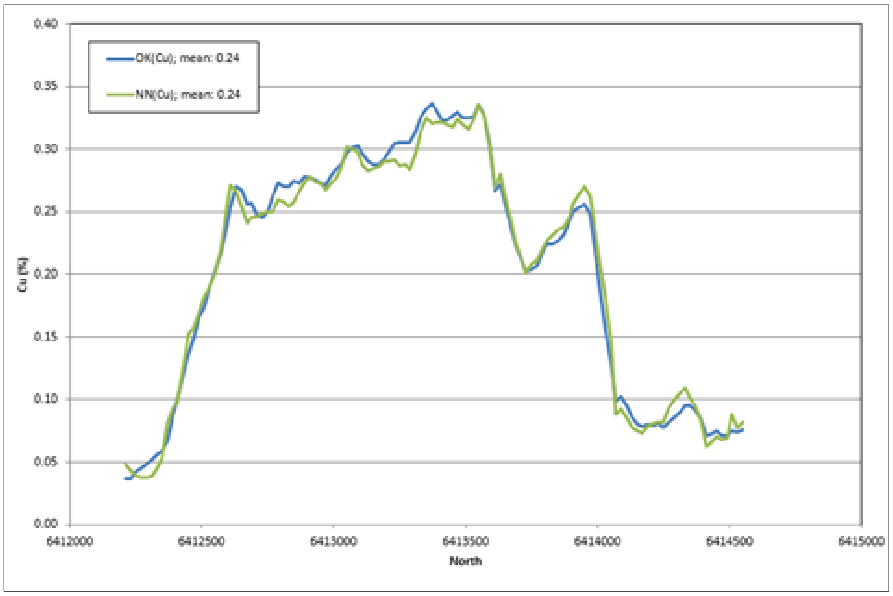
The Mineral Resource estimate was also validated using drift analysis, comparing the results obtained from the Ordinary Kriging estimation with a reference estimate made by the Nearest Neighbour method. A comparison was made graphically by observing the estimated block grade versus the composites grade on various east-west, north-south and elevation sections. Figure 14.35 to Figure 14.37 show representative sections for these three orientations.

Figure 14.35: East-West – Ordinary Kriging vs Nearest Neighbour for Copper



Source: Los Andes Copper, 2022

Figure 14.36: North-South – Ordinary Kriging vs Nearest Neighbour for Copper



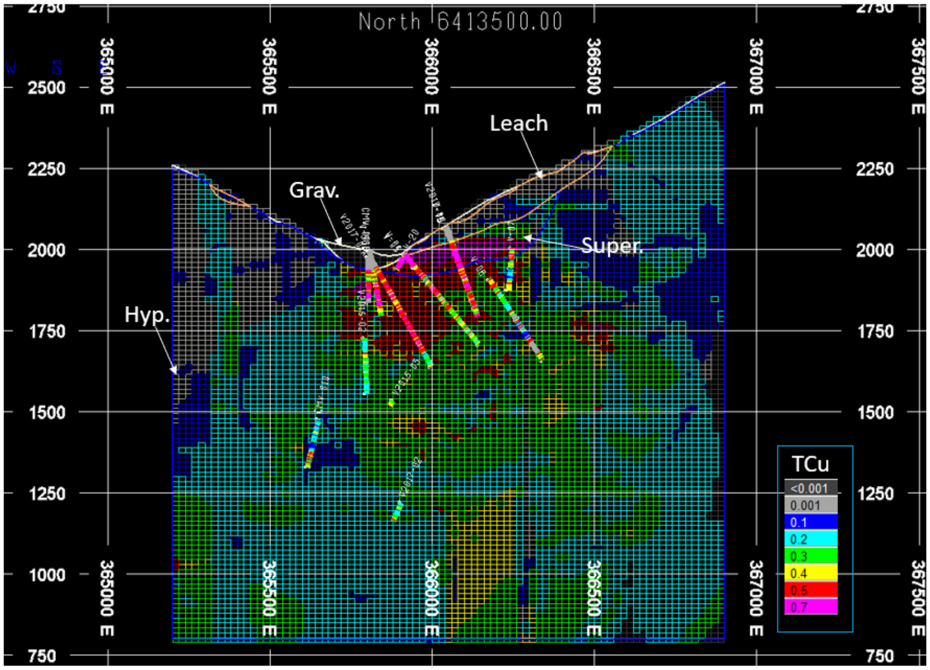
Source: Los Andes Copper, 2022

Figure 14.37: Elevation – Ordinary Kriging vs Nearest Neighbour for Copper

Source: Los Andes Copper, 2022

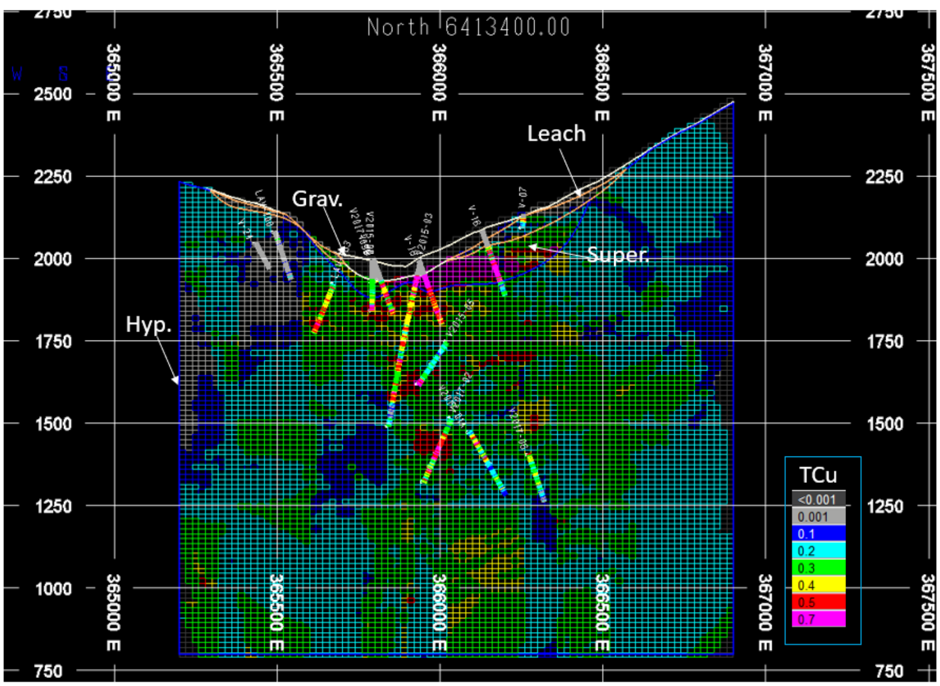
Figure 14.38 and Figure 14.39 show two representative east-west sections through the block model and Figure 14.40 shows a representative horizontal section through the block model to show the visual match between the sample composite grades and the estimated block grade for copper.

Figure 14.38: Vertical Section 6,413,500 North – Copper Grade (%)



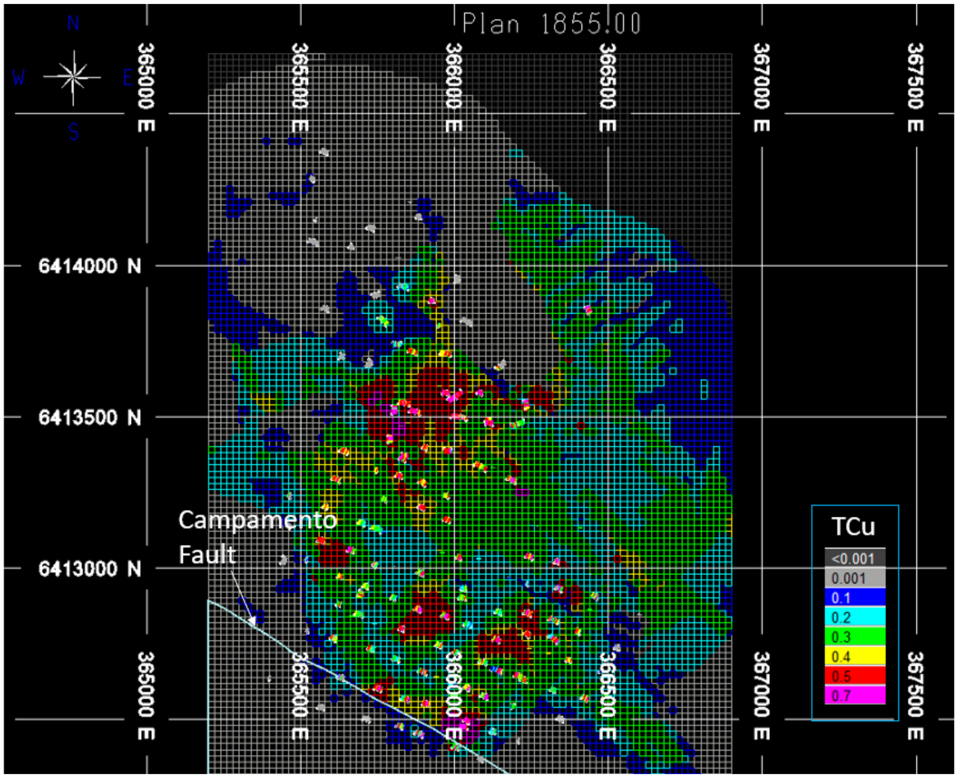
Source: Los Andes Copper, 2022

Figure 14.39: Vertical Section 6,413,400 North – Copper Grade (%)



Source: Los Andes Copper, 2022

Figure 14.40: Horizontal Section 1,855 masl – Copper Grade (%)

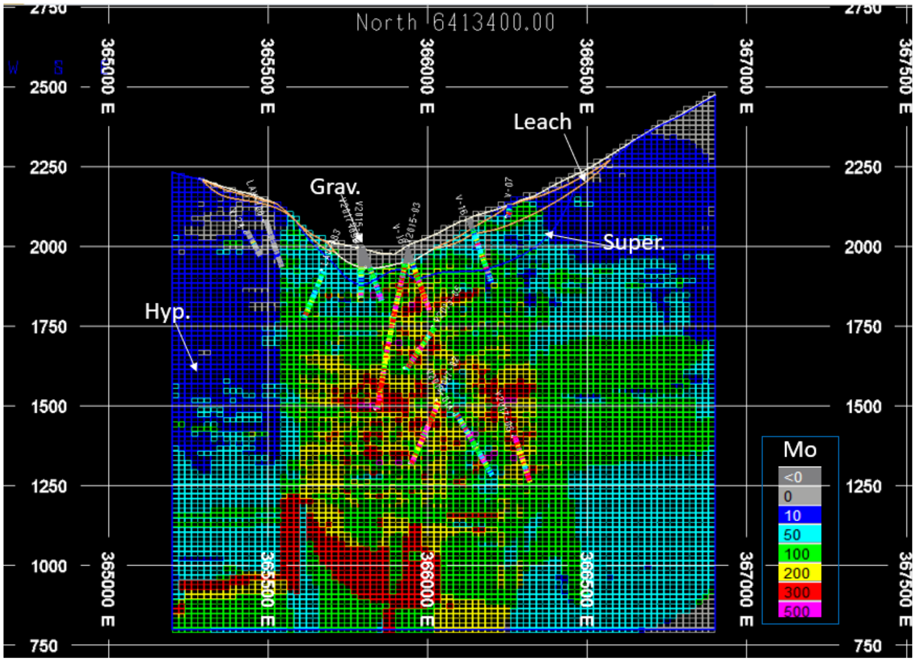


Source: Los Andes Copper, 2022

14.11.2 Molybdenum

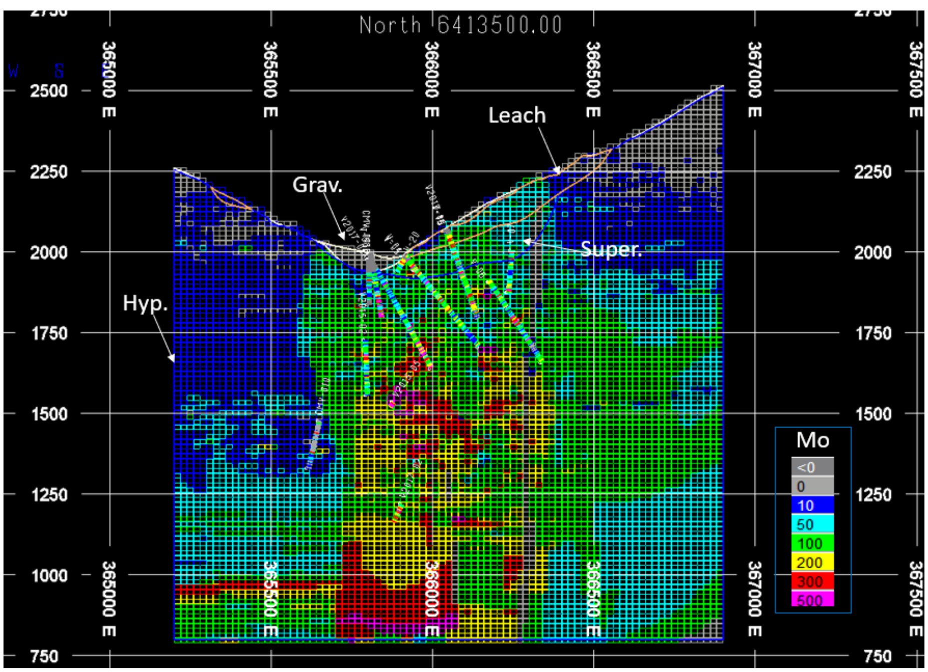
Figure 14.41 and Figure 14.42 show two representative east-west sections through the block model and Figure 14.43 shows a representative horizontal section through the block model, to show the match between the sample composite grades and the estimated block grade for molybdenum.

Figure 14.41: Vertical Section 6,413,400 North – Molybdenum Grade (ppm)



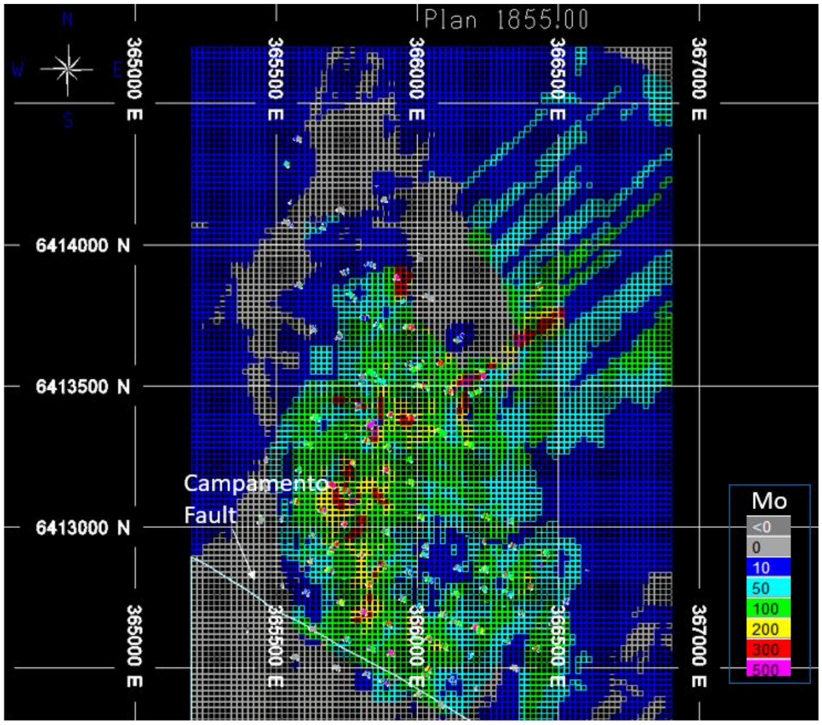
Source: Los Andes Copper, 2022

Figure 14.42: Vertical Section 6,413,500 North – Molybdenum Grade (ppm)



Source: Los Andes Copper, 2022

Figure 14.43: Horizontal Section 1,855 masl – Molybdenum Grade (ppm)



Source: Los Andes Copper, 2022

14.12 Classification of Mineral Resources

The categorization criteria used in this Technical Report has been based on the criteria used in the previous 2014 PEA by Coffey and the 2019 PEA by Tetra Tech. This classification links the geological information with the geostatistics, in which the tonnage associated with a specific production period is considered.

The geostatistical methodology provides tools to establish confidence levels for resource estimates. These methods involve the evaluation of estimate variances for large blocks (quarterly or annual production equivalent). This method provides an estimate of global confidence or confidence over large areas and is not dependent on local data. For this method, 90% confidence limits are calculated using the Ordinary Kriging coefficient of variation of the composites (Davis, 1997). The following criteria illustrate the calculation for Measured and Indicated Resources:

Tonnage and grade reliability can be defined as follows:

- Measured Resources for a quarterly production with ±15% accuracy at a 90% confidence limit.
- Indicated Resources for an annual production with ±15% accuracy at a 90% confidence limit.

A monthly production block was used with the dimensions:

- $323 \text{ m} \times 323 \text{ m} \times 15 \text{ m} = 1{,}564{,}935 \text{ m}^3$
- $1{,}564{,}935 \text{ m}^3 \times 2.6 \text{ t/m}^3 = 4{,}080{,}000$ t. (136,000 t/d)

For this evaluation Indicator Kriging was used with a cut-off grade of 0.25% Cu for four copper estimation units using the following parameters and as shown in Table 14.42.

- CL (90% confidence limits) = $\pm 1.645 \times \sigma$ relative, where σ relative = $\sqrt{\sigma^2}$ relative (annually or quarterly variance)
- σ^2 relative (annually) = σ^2 relative (monthly)/12 (for Indicated Mineral Resources)
- σ^2 relative (monthly) = σ^2 OK (monthly) x CV^2 (Ordinary Kriging Variance and Coefficient Variation)

Table 14.42: Estimation Units for Indicator Kriging used for Categorization

UGECU	Lithology		Veins		Mineralized Zone	
	Code	Description	Code	Description	Code	Description
2	-	-	-	-	303	Supergene
4	101	Andesite	401/402	C:High/Mid	304	Hypogene
6	102	Diorite	401/402	C:High/Mid	304	Hypogene
10	106/107	Breccias	-	-	304	Hypogene

The results of this analysis are shown in Table 14.43.

Table 14.43: Drill Hole Spacing Data Used to Establish Confidence Levels for Resource Categorization

UEIND2

Mesh	200	180	160	140	120	100	80
OK	0.374	0.360	0.116	0.081	0.071	0.031	0.020
CV	0.581	0.581	0.581	0.581	0.581	0.581	0.581
Monthly Relative Variance	0.126	0.122	0.039	0.027	0.024	0.010	0.007
Annual Relative Variance	0.011	0.010	0.003	0.002	0.002	0.001	0.001
V.A. Root	0.103	0.101	0.057	0.048	0.045	0.029	0.023
A.P. Confidence Limit 90% (± 1.645)	16.87%	16.56%	9.38%	7.87%	7.34%	4.83%	3.86%
T.P. Confidence Limit 90% (± 1.645)	33.74%	33.13%	18.75%	15.74%	14.68%	9.67%	7.73%

UEIND4

Mesh	200	180	160	140	120	100	80
OK	0.518	0.502	0.156	0.113	0.103	0.044	0.029
CV	0.461	0.461	0.461	0.461	0.461	0.461	0.461
Monthly Relative Variance	0.110	0.107	0.033	0.024	0.022	0.009	0.006
Annual Relative Variance	0.009	0.009	0.003	0.002	0.002	0.001	0.001
V.A. Root	0.096	0.094	0.053	0.045	0.043	0.028	0.023
A.P. Confidence Limit 90% (± 1.645)	15.75%	15.52%	8.64%	7.36%	7.03%	4.61%	3.71%
T.P. Confidence Limit 90% (± 1.645)	31.51%	31.03%	17.28%	14.71%	14.05%	9.22%	7.42%

UEIND6

Mesh	200	180	160	140	120	100	80
OK	0.394	0.375	0.142	0.102	0.086	0.040	0.025
CV	0.467	0.467	0.467	0.467	0.467	0.467	0.467
Monthly Relative Variance	0.086	0.082	0.031	0.022	0.019	0.009	0.006
Annual Relative Variance	0.007	0.007	0.003	0.002	0.002	0.001	0.000
V.A. Root	0.085	0.082	0.051	0.043	0.039	0.027	0.021
A.P. Confidence Limit 90% (± 1.645)	13.92%	13.57%	8.35%	7.09%	6.49%	4.42%	3.53%
T.P. Confidence Limit 90% (± 1.645)	27.85%	27.14%	16.70%	14.17%	12.98%	8.84%	7.05%

UEIND10

Mesh	200	180	160	140	120	100	80
OK	0.584	0.567	0.175	0.136	0.125	0.055	0.035
CV	0.705	0.705	0.705	0.705	0.705	0.705	0.705
Monthly Relative Variance	0.290	0.282	0.087	0.068	0.062	0.027	0.017
Annual Relative Variance	0.024	0.023	0.007	0.006	0.005	0.002	0.001
V.A. Root	0.155	0.153	0.085	0.075	0.072	0.048	0.038
A.P. Confidence Limit 90% (± 1.645)	25.57%	25.22%	14.00%	12.36%	11.85%	7.82%	6.22%
T.P. Confidence Limit 90% (± 1.645)	51.15%	50.43%	27.99%	24.72%	23.69%	15.63%	12.44%

The following criteria were considered for resource classification:

- Measured Resources: Three drill holes as a minimum, and an average distance <80 m.
- Indicated Resources: Two drill holes as a minimum, and an average distance <180 m.
- Inferred Resources: One drill hole as a minimum, and a distance <400 m.

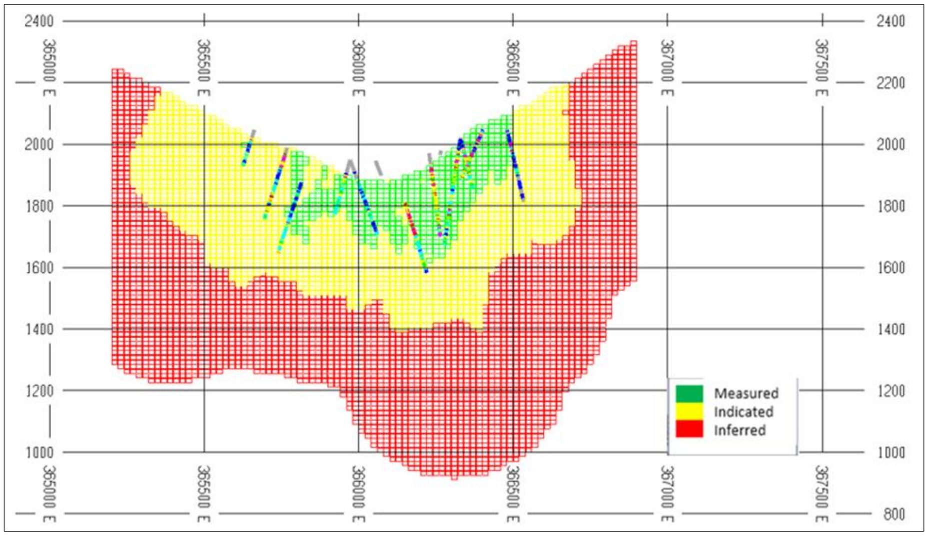
To avoid having Inferred blocks included with Measured blocks (salt and pepper effect), a smoothing algorithm using an Inverse Distance Weighting (IDW) estimation was used. This smoothing was not applied to the already categorized Measured and Indicated blocks estimated by Ordinary Kriging. Based on this, the following criteria were used for the smoothing:

- Measured Resources: Three drill holes as a minimum and an average distance <80 m
- Indicated Resources: Two drill holes as a minimum and an average distance <140 m

- Inferred Resources: One drill hole as a minimum and a distance <400 m

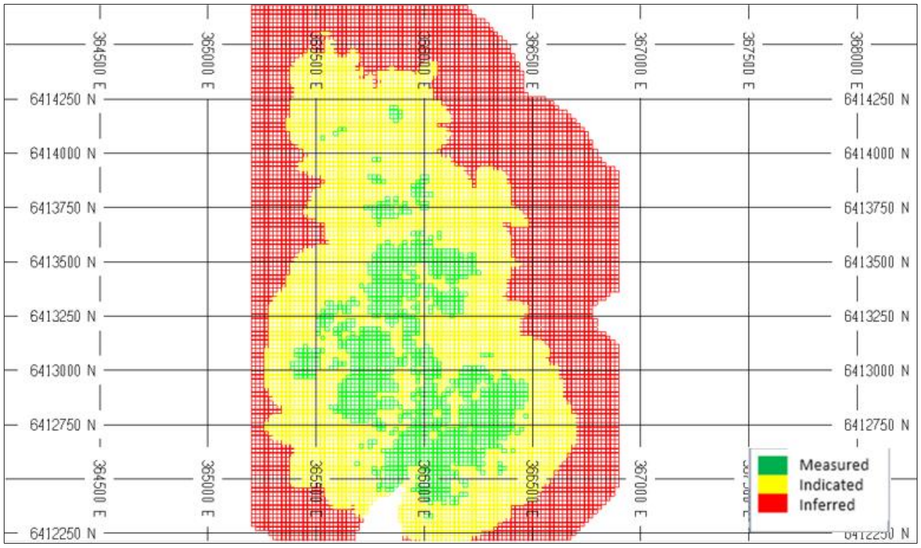
The copper resource categorization is shown on a representative vertical section in Figure 14.44 and on a representative horizontal section in Figure 14.45.

Figure 14.44: Copper Resource Categorization, Vertical Section 6,412,720 North



Source: Los Andes Copper, 2022

Figure 14.45: Copper Resource Categorization, Horizontal Section 1,870 masl



Source: Los Andes Copper, 2022

14.13 Reasonable Prospects for Eventual Economic Extraction

To assess the reasonable prospects for eventual economic extraction, a Whittle pit shell was prepared to constrain the estimated resource blocks using general technical and financial assumptions listed below.

- Plant cost : US$5.4/t
- Energy cost : US$65/MWh
- Mine cost : US$1.58/t
- Cu selling cost : US$0.48/lb
- Mo selling cost : US$1.68/lb
- Ag selling cost : US$2.5/oz
- Cu recovery : Variable by lithology and section, averaging 91.1%
- Mo recovery : Variable by lithology, averaging 74.8%
- Silver recovery : 75.0%
- Material to concentrate : Supergene + Hypogene
- Cu price : US$3.68/lb
- Mo price : US$12.90/lb
- Ag price : US$22.0/oz
- Pit sloped angles : 44° to 52°

The Mineral Resources are contained within an open pit shell to demonstrate the prospects of eventual economic extraction. Only blocks within the Whittle pit shell are included in the Mineral Resources.

14.14 Mineral Resource Statement

The in-pit Vizcachitas Mineral Resources have an Effective Date of February 7, 2023 and are reported using a 0.25% copper cut-off as follows:

- **Measured Mineral Resources** are 273 million tonnes grading 0.433% copper, 139 ppm molybdenum and 1.3 g/t silver for a 0.482% copper equivalent.
- **Indicated Mineral Resources** are 1,268 million tonnes grading 0.373% copper, 158 ppm molybdenum and 1.0 g/t silver for a 0.426% copper equivalent.
- **Measured and Indicated Mineral Resources** are 1,541 million tonnes grading 0.383% copper, 155 ppm molybdenum and 1.1 g/t silver for a 0.436% copper equivalent.
- **Inferred Mineral Resources** are 1,823 million tonnes grading 0.342% copper, 123 ppm molybdenum and 0.9 g/t silver for a 0.384% copper equivalent.

Table 14.44 to Table 14.47 present the Measured, Indicated, Measured plus Indicated, and Inferred Mineral Resources, respectively for a range of copper cut-off grades. The base case for the estimation of resources is 0.25% Cu.

Table 14.44: Measured Resources

Measured Resources									
Cut-Off (Cu %)	Tonnage (Mt)	Cu (%)	Mo (ppm)	Ag (g/t)	CuEq (%)	Cu (Mlb)	Mo (Mlb)	Ag (Moz)	CuEq (Mlb)
0.18	317	0.403	135	1.2	0.451	2,821	95	12.3	3,154
0.20	308	0.409	136	1.2	0.457	2,784	93	12.1	3,110
0.25	273	0.433	139	1.3	0.482	2,605	84	11.0	2,900
0.30	226	0.466	138	1.3	0.515	2,320	69	9.3	2,564
0.35	180	0.502	137	1.3	0.551	1,991	54	7.6	2,186

Table 14.45: Indicated Resources

Indicated Resource									
Cut-Off (Cu %)	Tonnage (Mt)	Cu (%)	Mo (ppm)	Ag (g/t)	CuEq (%)	Cu (Mlb)	Mo (Mlb)	Ag (Moz)	CuEq (Mlb)
0.18	1,606	0.340	150	1.0	0.390	12,036	530	51.6	13,815
0.20	1,525	0.348	151	1.0	0.399	11,697	509	49.7	13,408
0.25	1,268	0.373	158	1.0	0.426	10,416	442	42.8	11,901
0.30	951	0.405	164	1.1	0.460	8,492	343	33.3	9,643
0.35	644	0.444	171	1.1	0.501	6,298	243	23.6	7,113

Table 14.46: Measured and Indicated Resources

Measured and Indicated Resources									
Cut-Off (Cu %)	Tonnage (Mt)	Cu (%)	Mo (ppm)	Ag (g/t)	CuEq (%)	Cu (Mlb)	Mo (Mlb)	Ag (Moz)	CuEq (Mlb)
0.18	1,923	0.351	147	1.0	0.400	14,859	624	63.9	16,972
0.20	1,833	0.358	149	1.0	0.409	14,480	602	61.8	16,517
0.25	1,541	0.383	155	1.1	0.436	13,021	526	53.8	14,801
0.30	1,176	0.417	159	1.1	0.471	10,812	412	42.6	12,208
0.35	824	0.457	164	1.2	0.512	8,288	297	31.2	9,298

Table 14.47: Inferred Resources

Inferred Resource									
Cut-Off (Cu %)	Tonnage (Mt)	Cu (%)	Mo (ppm)	Ag (g/t)	CuEq (%)	Cu (Mlb)	Mo (Mlb)	Ag (Moz)	CuEq (Mlb)
0.18	2,956	0.294	112	0.8	0.333	19,187	729	80.2	21,683
0.20	2,764	0.302	114	0.9	0.340	18,376	693	76.3	20,748
0.25	1,823	0.342	123	0.9	0.384	13,747	495	55.3	15,444
0.30	1,180	0.379	129	1.0	0.423	9,853	336	38.4	11,009
0.35	655	0.423	142	1.1	0.472	6,117	205	23.3	6,824

Notes

1. Mineral Resources were classified using CIM Definition Standards (2014), and CIM Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines (2019).
2. The Mineral Resources effective date is February 7, 2023.
3. Mineral Resources are inclusive of Mineral Reserves.
4. Copper equivalent grade has been calculated using the following formula: CuEq (%) = Cu (%) 0.000288 x Mo (ppm) + 0.00711 x Ag (g/t), using the metal prices: US$3.68/lb Cu, US$12.9/lb Mo and US$21.79/oz Ag, with

metallurgical recoveries of 91.1% for copper, 74.8% for molybdenum and 75% for silver based on the PFS metallurgical testwork.
5. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability
6. The quantities and grades of reported Inferred Mineral Resources are uncertain in nature, and further exploration may not result in their upgrading to Indicated or Measured status.
7. Mineral Resources were prepared by Maria Loreto Romo and Severino Modena both full-time employees of Tetra Tech Sudamérica, and Ricardo Muñoz, a consultant part of the Tetra Tech Sudamérica team, all are Qualified Persons as defined by National Instrument 43-101.
8. Due to rounding, numbers may not add precisely to the totals.
9. All Mineral Resources are assessed for reasonable prospects for eventual economic extraction (RPEEE).

14.15 Factors That May Affect the Mineral Resource Estimate

Tetra Tech is not aware of any existing environmental, permitting, legal, socio-economic, marketing, political or other factors that could materially affect the Mineral Resource estimate, other than the analysis and considerations presented in this TR. The Mineral Resource estimate could be materially affected by future changes in the following factors:

1. Modifications to the metallurgical cut-off grade as a result of changes in process technology
2. Future drilling campaigns may result in changes to the lithological/reserves model
3. Future drilling campaigns may result in changes to metallurgical recovery per species
4. Changes to the input parameters to the constraining pit shell, the operating mine design and mine plan based on that pit shell
5. For the Whittle pit shell the metal prices used as reference are the long-term market consensus prices and may be re-evaluated by the market
6. Geotechnical issues that may arise as a result of updated information from future drilling campaigns.

15. MINING RESERVES ESTIMATES

This TR was developed using industry standards of mine planning techniques for the final pit definition and optimal economic mine schedule based on the Lerchs-Grossmann algorithm. The detailed operating mine phase design using MinePlan from Hexagon AB and the strategic mine plan scheduling was developed using COMET software (Commercial Optimal Mine Exploitation Technology).

15.1 Key Assumptions, Parameters, Methods, and Considerations

15.1.1 Block Model

The effective date for the Mineral Reserve estimate is December 2, 2022.

The Mineral Reserve estimate for the Project is based on the block model outlined in Chapter 14, and the design reserve pit described in Chapter 16. Although the Mineral Resources slightly increased on February 7, 2023 with an updated cost and metal prices scenario after the Mineral Reserves were determined, the result of this update is that the Mineral Reserves may be marginally under-estimated.

15.1.2 Dilution

The Vizcachitas mineralization lacks complexity in the mineral/waste contacts and has a smooth grade distribution around a barren central diatreme. Based on experience with similar deposits and using engineering judgment, Tetra Tech assumed for this TR that the potential for losses and dilution is minimal due to the homogeneity of the grades in the deposit and the cut-off grade selected. Therefore, no dilution was added to the block model during the mine planning design process. It is planned to operate Vizcachitas as an open pit mining operation with large electric rope shovels equipped with high-precision GPS devices, this will also reduce any relevant dilution. A bench-by-bench review of dilution values is recommended in future studies.

15.1.3 Economic Pit Definition

The design reserve pit was based on a Lerchs–Grossmann (LG) optimization process using Whittle4X/GEOVIA software from Dassault Systèmes to determine the final economic pit limit (Table 15.1) and a series of nested pits for increments of referential commodity prices contained in the resource model. This software allows the optimal size and shape of the pit to be determined and the generation of production plans to maximize the tonnage of economically mineable feed for the process plant.

Table 15.1: Ultimate Pit Values

Final Pit	Unit	Values
Total Rock	kt	5,812,978
Total Ore	kt	1,722,908
Total Waste	kt	4,090,070
Strip ratio	-	2.37
Cu grade	%	0.3509
Fine copper	t	5,504,577
Mo grade	ppm	142.31
Ag grade	ppm	1.06

Table 15.2 shows the economic parameters used as input information for the nested pits. As of the Effective Date of the Mineral Reserves (December 2, 2022), the long-term consensus metal prices were lower than the ones used in the financial model in Chapters 21 and 22. Other parameters used as input information are also different from those developed later and presented in Section 22, as they do not reflect subsequent studies and information obtained through the TR preparation. The use of a lower copper price in this Chapter results in a statement of reserves which is more conservative than if the final metal prices of US$3.68/lb had been used.

After the nested pits determination, twelve detailed operating phases were designed to prioritize the higher economic benefit zones within the mineral extraction plan, while maintaining suitable working widths that would enable high productivity mining sequences using large scale mining equipment.

Table 15.2: Summary of Economic Parameters for Nested Pits Estimate

Economic Parameters		
Resources Parameters	**Unit**	**Value**
Block Model Estimation		*mdrec.csv*
Block Size	m	20 x 20 x 15
Economic Parameters	**Unit**	**Value**
Cu Price	US$/lb	3.5
Mo Price	US$/kg	24
Rec Cu	%	by Lithology
Rec Mo	%	by Lithology
TC/RC	US$/lb	
Metallurgical Deductions	US$/lb	
Credit Ag By-Product	US$/lb	0.35
Freight & Insurance	US$/lb	
Selling Costs	US$/lb	
Sample, Analysis, Financial Costs	US$/lb	
Total by Sold Cu	US$/lb	0.350
Process Costs	**Unit**	**Value**
Crushing	US$/$t_{proc}$	0.65
Flotation	US$/$t_{proc}$	2.32
Mo Processing Plant	US$/$t_{proc}$	
Desalination & Water Pumping	US$/$t_{proc}$	0.58
Tailings	US$/$t_{proc}$	1.14
On-Site Infrastructure	US$/$t_{proc}$	0.29
Off-Site Infrastructure	US$/$t_{proc}$	0.42
General Administration	US$/$t_{proc}$	
TOTAL Process	US$/$t_{proc}$	5.40
Overhead	US$/$t_{proc}$	0.30
TOTAL Process + Overhead	US$/$t_{proc}$	5.70
Mine Costs	**Unit**	**Value**
Base Bench	m	1995
Mine Cost (Ex Haulage)	US$/t	0.80
Increment Up Loaded Truck (In Pit)	US$/t per bench	0.032
Increment Down Loaded Truck (In Pit)	US$/t per bench	0.014
Haulage Cost to Plant (Ex Pit)	US$/t	0.33
Haulage Cost to Waste Dump (Ex Pit)	US$/t	0.79
Mine Cost ORE (L&H) Base Bench	US$/t	1.13
Mine Cost WASTE (L&H) Base Bench	US$/t	1.59
Differential WASTE - ORE	US$/t	0.46

15.1.4 Optimization Considerations

The mine plan was optimized by analyzing numerous scenarios that utilized high-, medium- and low-grade stockpiling strategies, operational constraints and a variable cut-off grade profile. The total rock movement and the production daily rate were defined considering a mill feed of 136,000 tpd.

The strategic LOM plan was developed using Measured and Indicated Mineral Resources as plant feed. Inferred Mineral Resources were considered to be waste. Only hypogene and supergene mineralization was considered as plant feed. For the mill feed, the cut-off grade was set at 0.18% Cu considering metallurgical processing constraints.

The mining operation was designed as an open pit using electric rope shovels and mining trucks as loading and hauling equipment working as an autonomous haulage system (AHS) operation.

15.1.5 Topography

The Project is located in the Rocín River valley. Vizcachitas is a greenfield project, therefore there are no existing stockpiles and no previous mining operations.

15.1.6 Geotechnical Parameters

A preliminary pit slope geotechnical assessment was performed by Tetra Tech in 2022. Based on 174 Rock Quality Designation (RQD) samples available from several drilling campaigns carried out up and including 2022, overall pit slopes varied from 44º to 52º, inclusive of geotechnical berms and ramp allowances. The pit wall design criteria for this study are shown in Table 15.3. Section 16.3 provides a more detailed discussion.

Table 15.3: Summary of Geotechnical Parameters

Zone	Number	Bench Height (m)	Overall Slope Angle (OSA) (°)	Inter-Ramp Slope Angle (IRA) (°)	Face Angle (°)	Height Between Ramps (m)	Berm Witdh (m)	Berm (m)	Backbreak (m)	Toe-Toe (m)
Z_I (Broken)	1	15	44	50.3	80	120	25	9.8	2.64	12.5
Z_II (Rock)	2	15	52	56.0	85	225	25	8.8	1.31	10.1

15.1.7 Infrastructure Considerations

The Project considers mine infrastructure, equipment and systems required for crushing, grinding, flotation, tailings filtration and dry stacking, and concentrate filtration, storage and handling. In addition to the mine and process facilities the required infrastructure to support the planned operation have been considered.

15.2 Mineral Reserve Statement

The Mineral Reserve estimate considered technical, economic, environmental and legal modifying factors. Economic parameters considered production rate, long-term commodity prices and costs, geotechnical and geometrical design criteria for the operating pit design, and metallurgical recoveries by species.

Table 15.4 summarizes the Mineral Reserves as of December 2, 2022.

Table 15.4: Mineral Reserve Statement

Category	Tonnage (Mt)	Grade				Contained Metal			
		Cu (%)	Mo (ppm)	Ag (g/t)	CuEq (%)	Cu (Mlb)	Mo (Mlb)	Ag (Moz)	CuEq (Mlb)
Proven	302	0.41	135	1.2	0.45	2,714	89.8	11.9	3,031
Probable	918	0.34	136	1.1	0.39	6,908	275.3	31.8	7,858
Proven & Probable	1,220	0.36	136	1.1	0.40	9,623	365.0	43.6	10,889

Notes:
1. Mineral Reserves were classified using CIM Definition Standards (2014) and CIM Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines (2019)
2. Mineral Reserves have an effective date of December 2, 2022
3. Mineral Reserves are included within the Mineral Resources
4. The Qualified Person for the estimate is Mr. Severino Modena, BSc, Mining Engineer, MAusIMM, Member of the Chilean Mining Commission, and Tetra Tech Sudamérica General Manager
5. The Mineral Reserve has a metallurgical cut-off based on the process plant design of 0.18% Cu for direct mill feed
6. Due to rounding, numbers may not add precisely to the totals
7. The Mineral Reserve estimate uses a marginal phase analysis through a cut-off grade optimization software (COMET)
8. The Mineral Reserves are contained within operational phases defined using a COMET optimized mining schedule, which includes a stockpiling strategy. Key inputs for this process are:
   i. Metal prices of US$3.5/lb copper and US$12/lb molybdenum
   ii. Mining Cost of US$1.59/t at a reference elevation of 1,990 masl, plus costs adjustments of US$0.014/t per bench above reference and US$0.032/t per bench below reference
   iii. Process cost of US$5.7/t milled (inclusive of general and administrative costs of US$0.30/t milled)
   iv. Overall pit slopes angles varying from 44° to 52°
9. Process recoveries are based on lithology for both copper and molybdenum, except for one sector with a fixed copper recovery
10. Cu grades are reported as percentages, Mo and Ag grades are reported as parts per million (ppm)
11. The strip ratio (waste:ore) is 2.33. There are 2,855 Mt of waste in the ultimate pit
12. The Mineral Reserve statement considers the mill feed at the primary crusher as the reference point.

15.3 Factors that May Affect the Mineral Reserves

Tetra Tech is not aware of any existing environmental, permitting, legal, socio-economic, marketing, political or other factors which could materially affect the Mineral Reserve estimate other than the analysis and considerations presented in this TR. The Mineral Reserve estimate could be materially affected by future changes in the following factors:

1. Modifications to the metallurgical cut-off grade as a result of changes in process technology
2. Future drilling campaigns may result in changes to the lithological/reserves model
3. Future drilling campaigns may result in changes to metallurgical recovery per species
4. Changes to the input parameters to the constraining pit shell, the operating mine design and mine plan based on that pit shell
5. For the Mineral Reserve estimate, the metal prices used as reference are the long-term market consensus prices and may be re-evaluated by the market
6. Geotechnical considerations that may arise as a result of updated information from future drilling campaigns

7. Ability to maintain mining permits and/or surface rights
8. Changes to environmental laws that may have an impact on mine design studies
9. Environmental permits affecting Project infrastructure
10. Social licence to operate with neighbouring communities.

15.4 Comments on Mineral Reserve Estimate

Mineral Reserves are reported under the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), Definition Standards (2014) and Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines (2019).

16. MINING METHODS

16.1 Mine Planning Considerations

An open pit mining method has been selected for the Vizcachitas deposit mainly based on the copper and molybdenum grades and the continuity of the mineralization occurring near the surface, which results in a low strip ratio for open pit mining. The mine has been scheduled to operate 365 days per year, considering negligible snow related downtime and minor heavy rainfall events. The plan is based on two 12 hour shifts per day. Mining operations include in-house drilling, blasting, loading, hauling and earthworks, as well as outsourced services. This TR presents the results obtained for the mine plan developed for a mill throughput of 136,000 tpd.

The pit optimization process was preceded by an economic valuation of the block model within the resource pit, where all blocks have an associated economic value that is negative for waste and Inferred Resources and positive or negative for the Measured and Indicated Resources. This was accomplished using the block model described in Chapter 14. Although the Mineral Resources slightly increased on February 7, 2023 with an updated cost and metal prices scenario after the Mineral Reserves were determined, the result of this update is that the Mineral Reserves may be marginally under-estimated. The pit optimization process was performed using Whittle4X and the Lerchs–Grossman optimization algorithm.

Based on the pit optimization results, a set of operating mining phases were designed which considered average phase widths between 90 m and 150 m, sufficient to support mining for the 136,000 tpd scenario. It is planned to carry out part of the pre-stripping using earthmoving contractors.

The marginal phases analysis and the mine plan schedule were prepared using the COMET strategic planning software to maximize the total Project value, obtain the best cut-off grade strategy, and comply with technical-operating restrictions (such as maximum development of phases by period, vertical distance and inter-phase interferences).

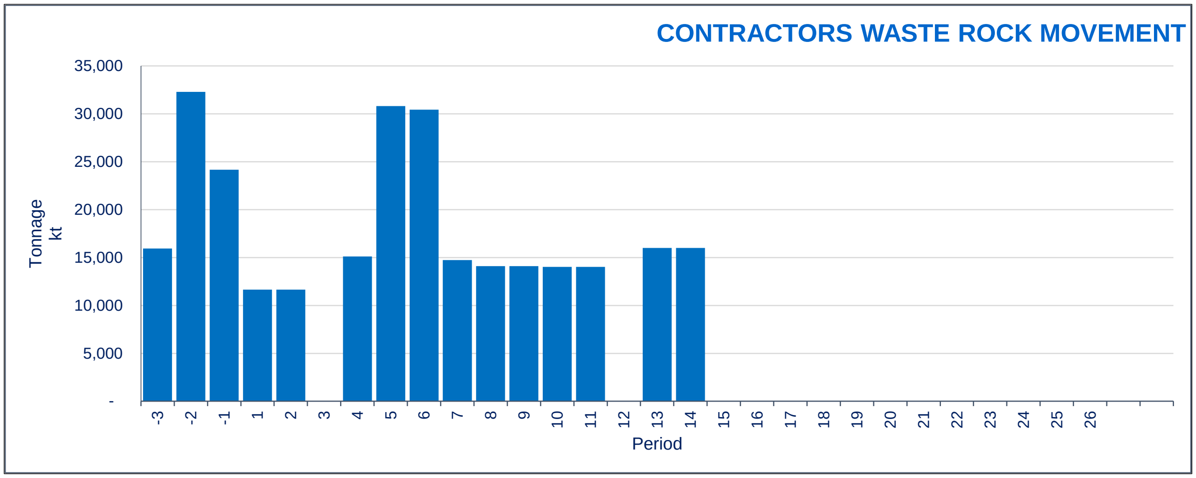
The strategic mine plans considered the following conditions:

- Bench height: 15 m (the same block height as the PFS resource model)
- Maximum vertical head by phase: 10 benches per year, equivalent to a maximum vertical development of 150 m per year
- Minimum number of active phases by period: two, except for the last years of the mine plan when only one phase is operated
- Number of stockpiles: three; high-grade (HG), medium-grade (MG) and low-grade (LG)
- The mill begins production in the first quarter of the fourth year after the start of construction
- Metallurgical cut-off grade: 0.18% Cu
- Maximum number of active phases by period: five

- Some benches at the top of each phase will be developed by contractors. This is not included in the strategic mine plan. The main reason for using contractors is to remove waste rock on narrow mining widths at the start of each phase with smaller equipment.

The total waste rock movement by contractors per period is shown in Figure 16.1.

Figure 16.1: Contractor Total Waste Rock Movement



Source: Tetra Tech, 2023

The mine plan options were assessed at a detail level, determining the hauling distances for mineralized material and waste, and then estimating the equipment fleet and the purchasing requirements over time.

16.2 Bench Height

The bench height defined was 15 m, this is the same as the block height used in the geological resource model. A bench height of 15 m (instead of 10 m used in previous studies) has the following major advantages:

- Drilling and blasting cost savings (reduced metres and operating time)
- Earthworks cost savings (removing an intermediate level)
- Introduction of larger, heavier duty equipment
- Reduction in numbers of equipment and operators.

16.3 Geotechnical Parameters

The methodology used in this PFS for the evaluation of the geotechnical stability of slopes, considered numerical modeling techniques using finite element analysis. The purpose was to identify the shape of potentially unstable zones and to calculate the safety factors according to rock mass strength properties.

A new geotechnical interpretation, different from that in the PEA, was prepared incorporating additional information from new drilling and rock mechanics tests. This resulted in the identification of two zones with very different geotechnical characteristics which enabled a better understanding of the elements that will control the failure modes with greater impact on the overall stability.

After the geotechnical zoning was updated, the final slope stability geometry was assessed using the strength reduction factor method. In this method, a safety factor is calculated which corresponds to the number by which shear resistance must be divided to cause a functional failure in the slope (Duncan, 1996). A simple way to calculate this factor using numerical methods is to intentionally reduce shear resistance until the slope collapses. The resulting safety factor is then the quotient between the initial strength and the strength value in the failure.

The following acceptability criteria were established to define the conditions when the slope will be considered to have a functional geotechnical failure condition:

(1) Functional failure occurs for displacements over 10 cm (experience of geotechnical monitoring results and definition of alert thresholds)
(2) Functional failure occurs for maximum shear unit deformations over 0.005 (corresponding to the same 10 cm, but over a length of 20 m which could correspond to the height of a bench).

For the analysis of results a review of the unit deformations due to shear stress in E and W sections was carried out, assessing qualitatively the rock mass extents that did not achieve the acceptability criteria for overall wall slope stability.

16.3.1 Geotechnical Background Information

Geotechnical information has been obtained from the drilling campaigns executed between 1993 and 2022. In 2018 a geotechnical interpretation using RMR indices (Bieniawski, 1989) and GSI (Hoek, 2013) was conducted as part of the PEA (FF GeoMechanics, 2018). This report was based on seven mapping cells and a review of metres drilled; it resulted in the definition of the eight geotechnical units shown in Table 16.1.

Table 16.1: Definition of Geotechnical Units, June 2018

UGT	Geological Unit	Lithology	GSI	RMR
1	Overburden	Gravel	20 - 25	30
2	Leached	Leached rock	42 - 47	53
3	Fractured rock	Varied	30 - 35	41
4	Fractured host rock	Upper Andesite	50 - 55	60
5	Intrusive complex	Hypogene intrusives and igneous breccias	67 - 72	71
6	Hydrothermal breccia	Hypogene hydrothermal breccia	66 - 71	70
7	Discordant bodies	Hypogene diatreme and dacitic porphyry	63 - 68	74
8	Host rock	Hypogene Lower andesite	72 - 77	74

Given that no high contrast differentiation was identified among the eight units, a new geotechnical characterization was completed for this TR, supported by a larger amount of data and better representativeness of the area of interest. Based on these criteria, RQD records available from the drill holes were considered as an alternative to represent the geotechnical context in the pit.

The Geotechnical Report prepared by Tetra Tech (Tetra Tech, Pre-Feasibility Geomechanical Study Vizcachitas Project, 2023) developed a new geotechnical model supported by RQD records and the geotechnical properties for the units based on laboratory assays results. Table 16.2 shows the information used in this TR compared to the previous analysis.

Table 16.2: Data Used for Geotechnical Zoning

2018 Geotechnical Zoning	2022 Geotechnical Zoning
Supported by 7 geotechnical cells at the surface and a review of 314 m drill holes (from 4 drill holes).	Supported by records from 154 drill holes, with 26,419 RQD data.
Rock mechanics assays: 16 triaxial assays, 16 UCS assays, and 8 indirect traction assays.	Additional Rock mechanics assays: 72 triaxial assays, and 19 UCS assays.

It is recommended that more analysis is conducted on background information from structures (discontinuities) during the feasibility phase, using oriented drill holes and mapping cells on the surface.

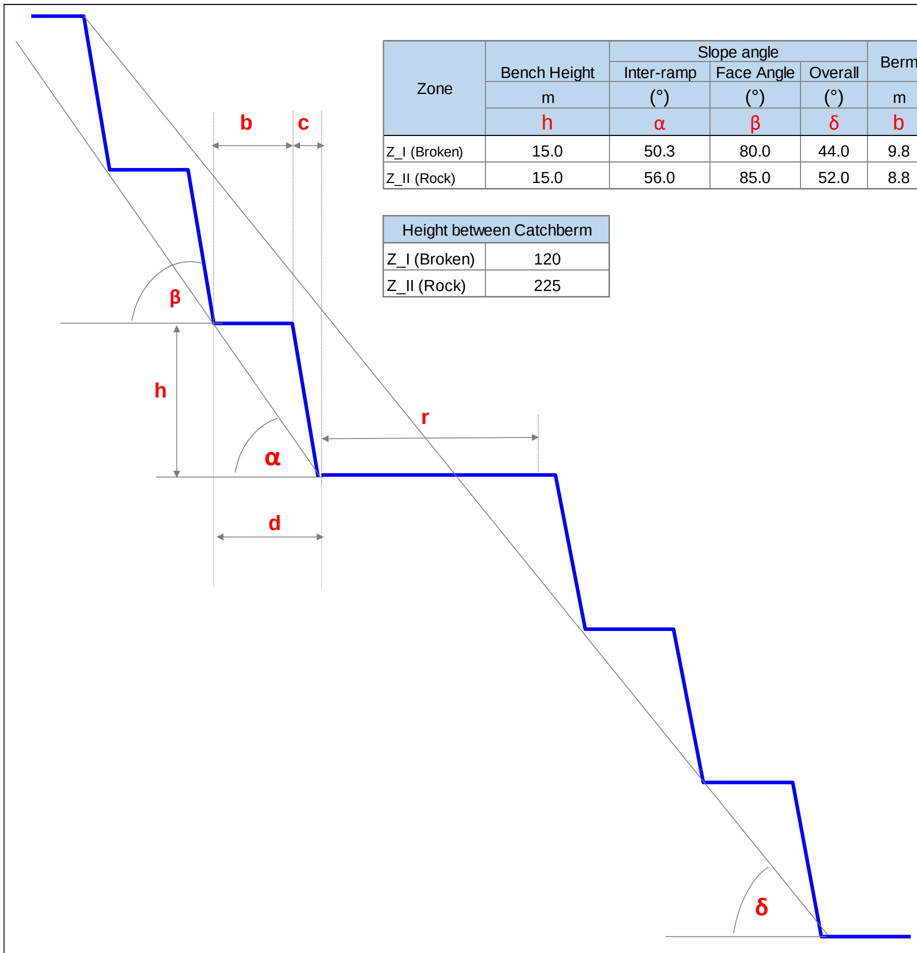
16.3.2 Evolution of Mine Design

The evaluation process for slope stability is iterative. It starts with conservative angles, evaluating the design stability, and concluding whether it is feasible or not to increase the angles if the acceptability criteria are met. If not, the geometry (angles and height) should be modified until the initial criteria are met.

The stability of east and west slopes geometry was assessed at the end of mining (final angles), assuming that other geometry (prior to the final one) could not have a greater height and should not have higher angles. It is recommended that during the feasibility study this be complemented with the analysis of intermediate stages that could have a higher angle, but with lower overall slope height. At this stage, the purpose was to identify any potential fatal flaws which may affect subsequent excavation and determine a complete final slope rather than design intermediate stages which are usually operationally controlled using geotechnical monitoring systems.

With the new geotechnical characterization, a final wall geometry was evaluated and defined by overall slope angles and inter-ramp slope angles as shown in Figure 16.2

Figure 16.2: Mine Design Geometry



Zone	Bench Height	Slope angle			Berm	Backbreak	Toe-Toe	Ramp Width
		Inter-ramp	Face Angle	Overall				
	m	(°)	(°)	(°)	m	m	m	m
	h	α	β	δ	b	c	d	r
Z_I (Broken)	15.0	50.3	80.0	44.0	9.8	2.64	12.5	38.0
Z_II (Rock)	15.0	56.0	85.0	52.0	8.8	1.31	10.1	38.0

Height between Catchberm	
Z_I (Broken)	120
Z_II (Rock)	225

Source: Tetra Tech, 2023

16.4 Waste Dump Design Parameters

Table 16.3 presents the parameters used for waste dump design.

Table 16.3: Geometrical Parameters for Waste Dumps

Parameters Dump Design		
Parameters	**Value**	**Unit**
Module Height	60.0	m
Face Slope	37.0	°
Ramp Width	38.0	m
Ramp Slope	10.0	%
Berm Width	40.0	m

16.5 Hydrology and Hydrogeology

ITASCA Chile developed a hydrogeological model for the Project (ITASCA, 2022; ITASCA, 2023). ITASCA proposed the mine dewatering operation by building a series of pumping wells around the Rocín River upstream (to the north) of the open pit, to extract underground water flows. This series of pumping wells would be built before starting mining and will operate until the end of the LOM. Other pumping wells within the pit would be built on an as-needed basis according to the pit development.

The ITASCA estimate considers dewatering systems in two zones: for the north sector there is an estimated water flow of 14 L/s with 5 to 8 wells, and within the pit there is an estimated water flow of 10 L/s with 4 to 8 wells located in the pit. ITASCA considered that the north sector wells should reach the water table level.

A provision under Other in the Operating Costs includes water management (dewatering) in the pit and waste dumps (among other activities).

16.6 Treatment Capacity

The Vizcachitas process plant is designed for a throughput rate of 136,000 tpd and includes the infrastructure, equipment and systems required for crushing, grinding, flotation, tailings filtration and dry stacking, concentrate filtration, and storage and handling of concentrates.

The mill is only fed with Measured and Indicated Resources reported in the Mineral Resources of this TR. The feed is direct from the pit or from stockpiles generated from the mining operation.

16.7 Optimum Pit Shells

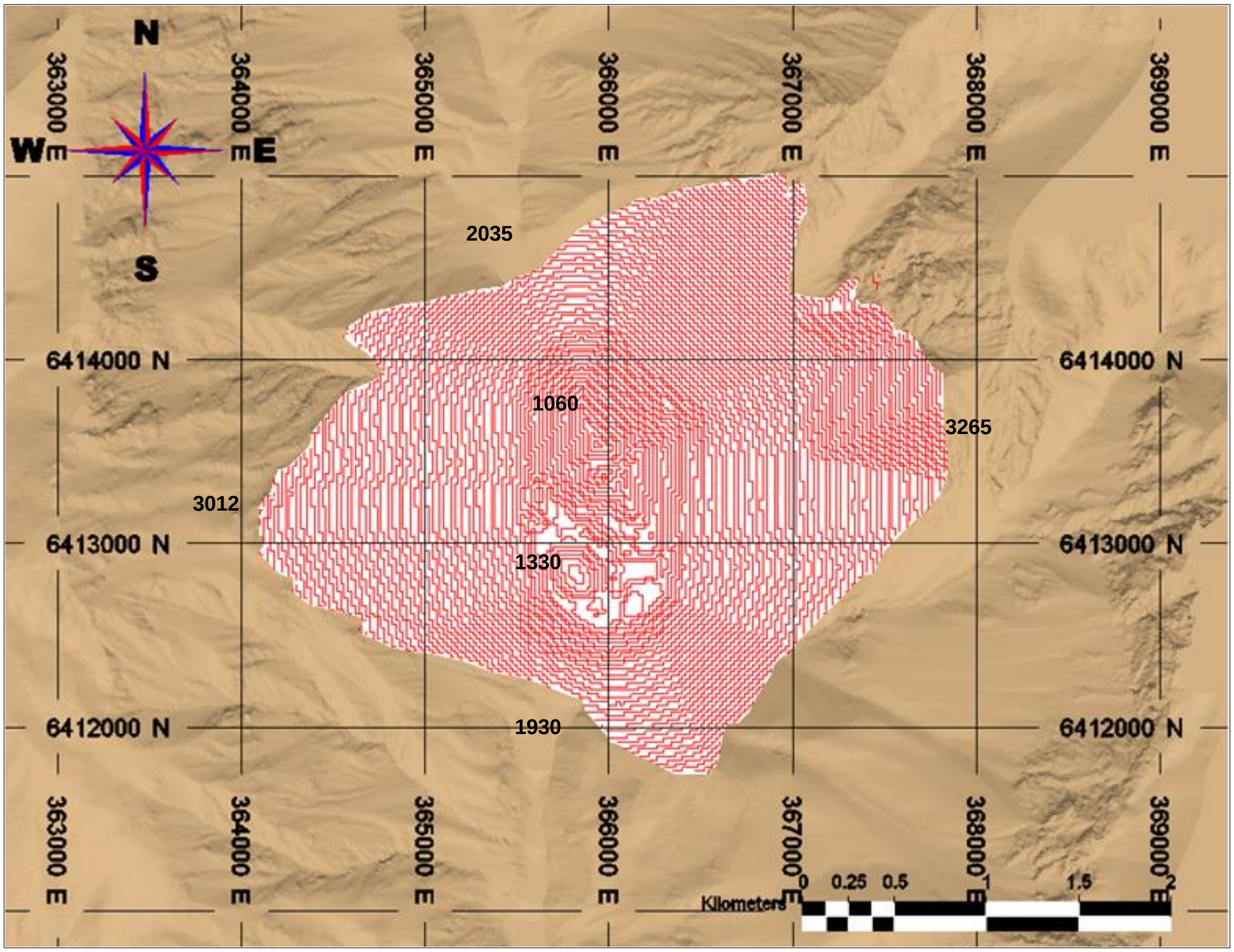
To define the final boundary and the extraction sequence for the design of the operating phases in each case under analysis, the final pit was optimized using the existing resource model following the economic and geotechnical parameters defined for the Project.

This exercise was performed using the Lerchs-Grossman algorithm introduced into the Whittle4X software.

16.7.1 Final Pit Optimization Results

Figure 16.3 shows the final economic pit limit obtained for the 136,000 tpd case. With this limit, the total mill feed is 1,723 Mt with a 0.35% Cu grade and the total rock movement is 5,813 Mt.

Figure 16.3: Final Whittle Pit



Source: Tetra Tech, 2023

Table 16.4 and Table 16.5 summarize the final pit optimization results based on incremental copper prices, with variations from US$0.76/lb to US$3.50/lb (Revenue Factor (RF) 0.22 to 1.00).

Table 16.4 highlights the pits used in the design phase (light gray). These pits are shown in Figure 16.4 along with the design phases.

Table 16.4: Summary Final Pit Optimization, Pits 1-48

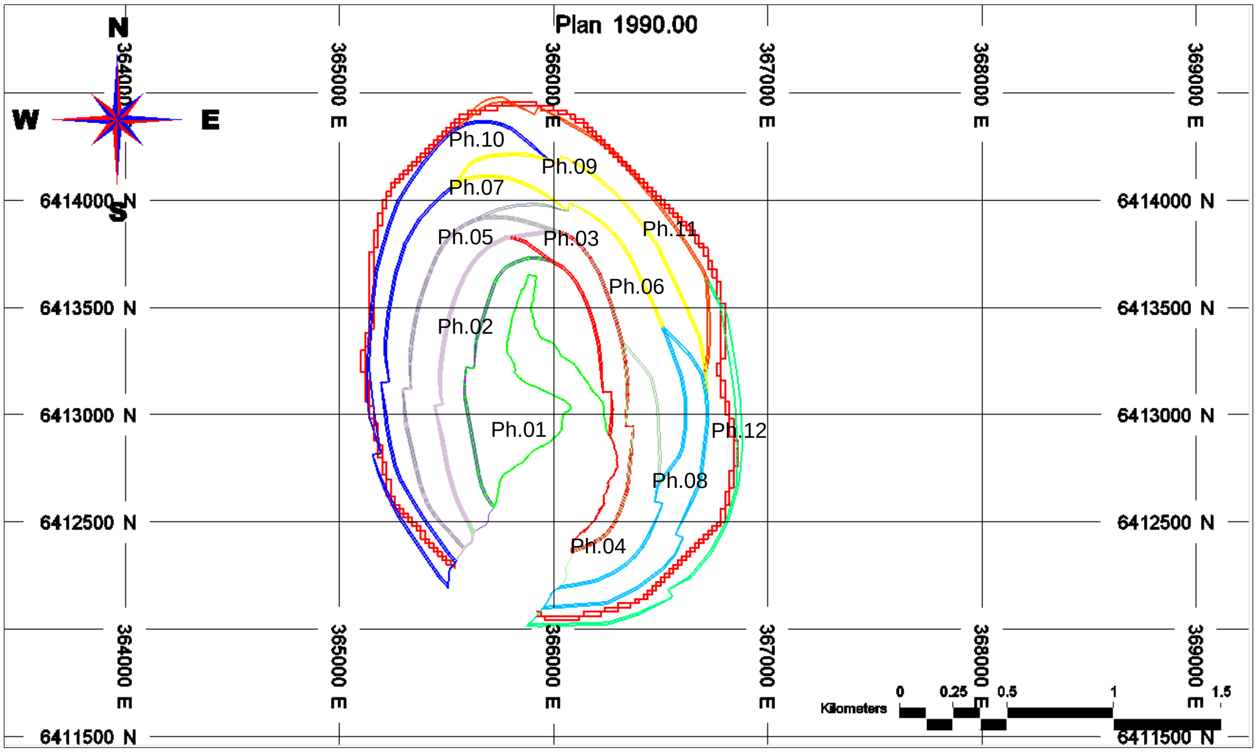
Pit	Revenue Factor	Copper Price (USD/lb)	Total Rock (kt)	Ore (kt)	Strip Ratio	Cu (%)	Mo (ppm)	Ag (g/t)
1	0.22	0.76	15	15	0.00	0.75	93.97	1.7
2	0.22	0.78	163	136	0.20	0.76	58.60	1.5
3	0.23	0.81	215	188	0.14	0.75	53.18	1.5
4	0.24	0.84	387	341	0.13	0.71	58.50	1.5
5	0.25	0.87	9,422	3,196	1.95	0.89	112.85	1.9
6	0.26	0.90	10,760	3,848	1.80	0.86	115.23	1.8
7	0.26	0.92	15,177	6,132	1.48	0.79	116.97	1.7
8	0.27	0.95	17,199	7,219	1.38	0.76	116.24	1.6
9	0.28	0.98	21,009	9,245	1.27	0.72	115.77	1.6
10	0.29	1.01	65,298	29,889	1.18	0.63	140.67	1.6
11	0.30	1.04	92,046	44,076	1.09	0.60	137.82	1.5
12	0.30	1.06	107,391	52,116	1.06	0.59	138.80	1.5
13	0.31	1.09	135,860	67,715	1.01	0.57	139.04	1.5
14	0.32	1.12	152,300	76,089	1.00	0.56	137.83	1.5
15	0.33	1.15	201,071	107,671	0.87	0.52	143.38	1.4
16	0.34	1.18	233,172	126,893	0.84	0.50	144.95	1.4
17	0.34	1.20	277,702	150,176	0.85	0.50	142.06	1.3
18	0.35	1.23	332,269	178,810	0.86	0.48	141.05	1.3
19	0.36	1.26	389,380	210,192	0.85	0.47	140.00	1.3
20	0.37	1.29	431,162	238,698	0.81	0.46	140.65	1.3
21	0.38	1.32	492,253	276,877	0.78	0.44	141.21	1.3
22	0.38	1.34	547,726	309,517	0.77	0.44	139.54	1.3
23	0.39	1.37	635,445	362,066	0.76	0.42	136.58	1.2
24	0.40	1.40	759,605	428,422	0.77	0.41	132.99	1.2
25	0.41	1.43	805,360	457,286	0.76	0.41	132.31	1.2
26	0.42	1.46	874,542	495,823	0.76	0.40	132.22	1.2
27	0.42	1.48	966,313	543,203	0.78	0.40	132.07	1.2
28	0.43	1.51	1,009,097	567,145	0.78	0.39	132.17	1.2
29	0.44	1.54	1,143,541	631,844	0.81	0.39	132.43	1.2
30	0.45	1.57	1,222,474	666,219	0.83	0.38	132.82	1.2
31	0.46	1.60	1,295,571	700,553	0.85	0.38	133.33	1.2
32	0.46	1.62	1,372,249	732,750	0.87	0.38	134.14	1.2
33	0.47	1.65	1,425,784	757,359	0.88	0.38	134.07	1.2
34	0.48	1.68	1,535,141	800,323	0.92	0.37	134.29	1.2
35	0.49	1.71	1,645,278	840,588	0.96	0.37	134.41	1.2
36	0.50	1.74	1,717,112	863,994	0.99	0.37	134.72	1.2
37	0.50	1.76	1,842,517	905,751	1.03	0.37	134.95	1.2
38	0.51	1.79	1,947,766	941,016	1.07	0.37	135.19	1.2
39	0.52	1.82	2,075,274	983,999	1.11	0.37	135.06	1.2
40	0.53	1.85	2,218,856	1,028,685	1.16	0.36	135.39	1.2
41	0.54	1.88	2,278,634	1,045,500	1.18	0.36	135.69	1.2
42	0.54	1.90	2,397,454	1,084,618	1.21	0.36	136.33	1.1
43	0.55	1.93	2,481,294	1,109,654	1.24	0.36	136.72	1.1
44	0.56	1.96	2,563,746	1,133,588	1.26	0.36	137.07	1.1
45	0.57	1.99	2,673,049	1,162,952	1.30	0.36	137.10	1.1
46	0.58	2.02	2,756,345	1,185,668	1.32	0.36	137.43	1.1
47	0.58	2.04	2,832,887	1,206,067	1.35	0.36	137.87	1.1
48	0.59	2.07	2,932,355	1,232,362	1.38	0.36	137.80	1.1

Table 16.5 : Summary Final Pit Optimization, Pits 49-98

Pit	Revenue Factor	Copper Price	Total Rock (kt)	Ore (kt)	Strip Ratio	Cu (%)	Mo (ppm)	Ag (g/t)
49	0.60	2.10	2,990,411	1,245,381	1.40	0.36	138.30	1.1
50	0.61	2.13	3,107,906	1,277,621	1.43	0.36	138.14	1.1
51	0.62	2.16	3,171,765	1,292,500	1.45	0.36	138.41	1.1
52	0.62	2.18	3,198,369	1,299,050	1.46	0.36	138.36	1.1
53	0.63	2.21	3,330,818	1,330,034	1.50	0.36	138.66	1.1
54	0.64	2.24	3,450,403	1,358,156	1.54	0.36	138.53	1.1
55	0.65	2.27	3,480,057	1,364,800	1.55	0.36	138.50	1.1
56	0.66	2.30	3,551,012	1,379,927	1.57	0.35	138.66	1.1
57	0.66	2.32	3,565,675	1,384,149	1.58	0.35	138.68	1.1
58	0.67	2.35	3,616,099	1,394,863	1.59	0.35	138.66	1.1
59	0.68	2.38	3,671,455	1,407,721	1.61	0.35	138.55	1.1
60	0.69	2.41	3,763,541	1,423,850	1.64	0.35	138.81	1.1
61	0.70	2.44	3,823,508	1,437,146	1.66	0.35	138.74	1.1
62	0.70	2.46	3,939,792	1,460,622	1.70	0.35	138.95	1.1
63	0.71	2.49	4,027,956	1,479,087	1.72	0.35	139.03	1.1
64	0.72	2.52	4,030,430	1,479,576	1.72	0.35	139.01	1.1
65	0.73	2.55	4,109,098	1,493,359	1.75	0.35	139.24	1.1
66	0.74	2.58	4,116,545	1,494,852	1.75	0.35	139.20	1.1
67	0.74	2.60	4,183,812	1,507,296	1.78	0.35	139.24	1.1
68	0.75	2.63	4,223,484	1,513,781	1.79	0.35	139.11	1.1
69	0.76	2.66	4,229,663	1,514,750	1.79	0.35	139.09	1.1
70	0.77	2.69	4,269,223	1,522,973	1.80	0.35	139.04	1.1
71	0.78	2.72	4,360,289	1,537,389	1.84	0.35	139.44	1.1
72	0.78	2.74	4,364,532	1,538,282	1.84	0.35	139.41	1.1
73	0.79	2.77	4,390,347	1,543,457	1.84	0.35	139.24	1.1
74	0.80	2.80	4,510,751	1,561,462	1.89	0.35	139.48	1.1
75	0.81	2.83	4,544,360	1,566,417	1.90	0.35	139.52	1.1
76	0.82	2.86	4,717,376	1,591,154	1.96	0.35	140.01	1.1
77	0.82	2.88	4,776,676	1,599,915	1.99	0.35	140.16	1.1
78	0.83	2.91	4,847,650	1,609,082	2.01	0.35	140.40	1.1
79	0.84	2.94	4,905,072	1,617,909	2.03	0.35	140.36	1.1
80	0.85	2.97	4,914,567	1,619,593	2.03	0.35	140.28	1.1
81	0.86	3.00	4,919,192	1,620,022	2.04	0.35	140.32	1.1
82	0.86	3.02	4,921,004	1,620,180	2.04	0.35	140.32	1.1
83	0.87	3.05	5,034,598	1,634,071	2.08	0.35	140.69	1.1
84	0.88	3.08	5,040,565	1,635,065	2.08	0.35	140.67	1.1
85	0.89	3.11	5,169,760	1,651,550	2.13	0.35	140.82	1.1
86	0.90	3.14	5,170,472	1,651,644	2.13	0.35	140.82	1.1
87	0.90	3.16	5,172,750	1,651,959	2.13	0.35	140.80	1.1
88	0.91	3.19	5,293,818	1,665,028	2.18	0.35	141.26	1.1
89	0.92	3.22	5,313,000	1,668,098	2.19	0.35	141.18	1.1
90	0.93	3.25	5,359,445	1,673,930	2.20	0.35	141.23	1.1
91	0.94	3.28	5,370,247	1,675,216	2.21	0.35	141.18	1.1
92	0.94	3.30	5,443,847	1,682,662	2.24	0.35	141.49	1.1
93	0.95	3.33	5,445,435	1,683,003	2.24	0.35	141.50	1.1
94	0.96	3.36	5,508,011	1,690,718	2.26	0.35	141.44	1.1
95	0.97	3.39	5,630,731	1,703,777	2.30	0.35	141.81	1.1
96	0.98	3.42	5,631,803	1,704,041	2.30	0.35	141.81	1.1
97	0.99	3.47	5,806,181	1,722,054	2.37	0.35	142.33	1.1
98	1.00	3.50	5,812,978	1,722,908	2.37	0.35	142.31	1.1

Phases 10, 11 and 12 define the final operational extraction limit which is based on the economic final pit shell (red line in Figure 16.4)

Figure 16.4: Nested Pits and Phase Design



Source: Tetra Tech, 2023

16.8 Mine Design

The study considered the best set of mine phase designs from several design iterations. Each phase was designed with its own access roads to enable rock haulage from the loading area to the destination at the primary crusher, stockpiles or waste dumps.

The access roads for the top benches were designed with a width of 25 m to operate smaller trucks in narrow areas. Those areas will be mined by earthmoving contractors. Other roads were designed with a width of 38 m to operate high capacity autonomous haulage operation (AHS) mining trucks. For both cases internal ramps inside each phase will connect to exit ramps and roads.

To guarantee slope stability, the mine design includes catch berms of 25 m along the wall slope every eight benches in poor quality rock zones and every 15 benches in good quality rock zones.

16.8.1 Operating Criteria

The minimum operating widths for loading equipment during mining on the benches are:

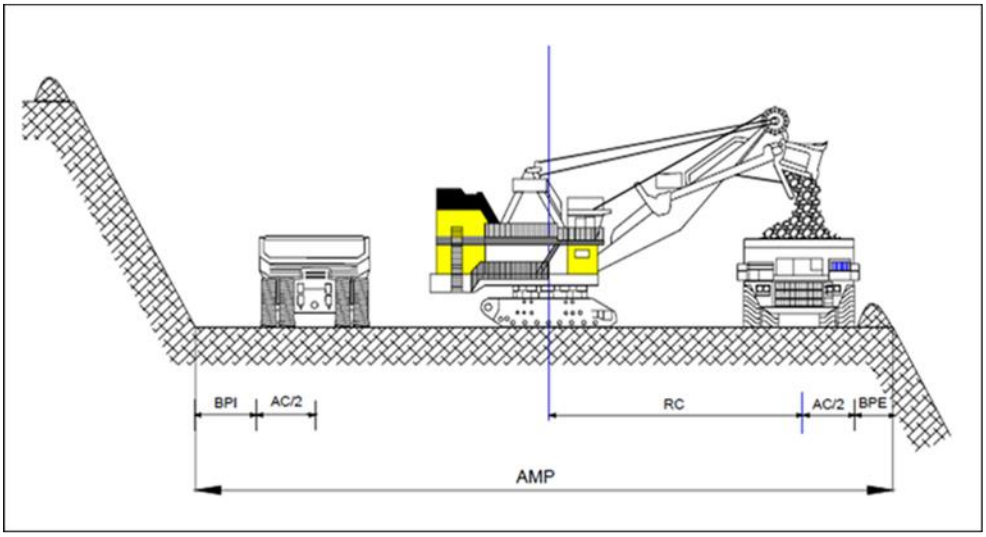
- In production blasting with control strip:
  - 73 yd³ rope shovel with 300 t trucks loading from both sides : 70 m
  - 73 yd³ rope shovel with 300 t trucks loading from one side : 55 m
- In bench closing blasting:
  - PC hydraulic shovel with 300 t trucks loading from one side: 50 m

Figure 16.5 to Figure 16.7 show the loading scheme for the mining operating widths identified. These parameters allow operation of loading equipment at optimum performance. Table 16.6 and Table 16.7 show the parameters used for the mining width calculation for the rope shovel and hydraulic shovel, respectively.

Table 16.6: Mining Width Parameters for 73 yd^3 Shovel

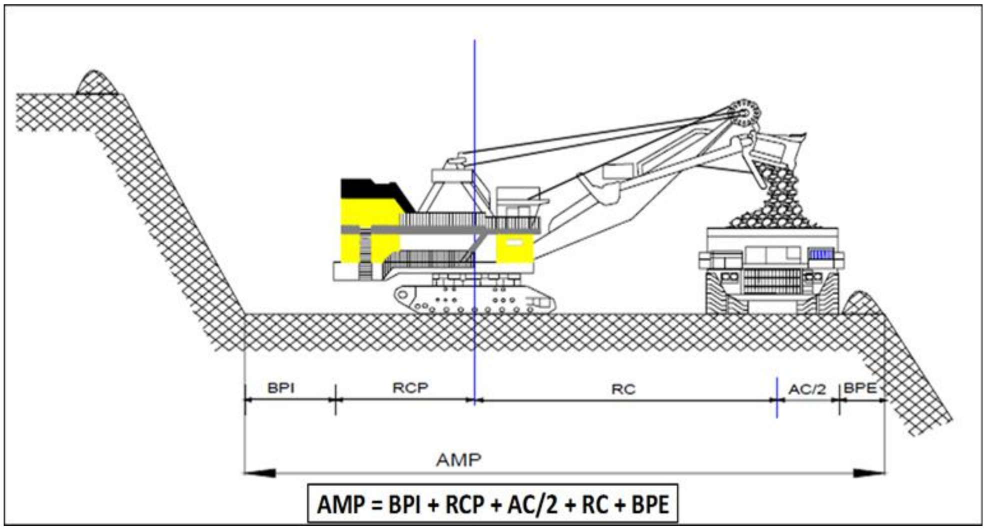
Field (m)	Description	Rope Shovel	
		Both Sides	One Side
BPE	External Safety Berm	10	10
BPI	Internal Safety Berm	6	8
RCP	Counter Weight Radius	-	9
RC	Loading Radius	22	22
AC/2	Truck Width	10	10
AMP	Minimum Loading Width	70	55

Figure 16.5: 73 yd^3 Shovel Loading, Both Sides



Source: Tetra Tech, 2023

Figure 16.6: 73 yd^3 Shovel Loading, One Side

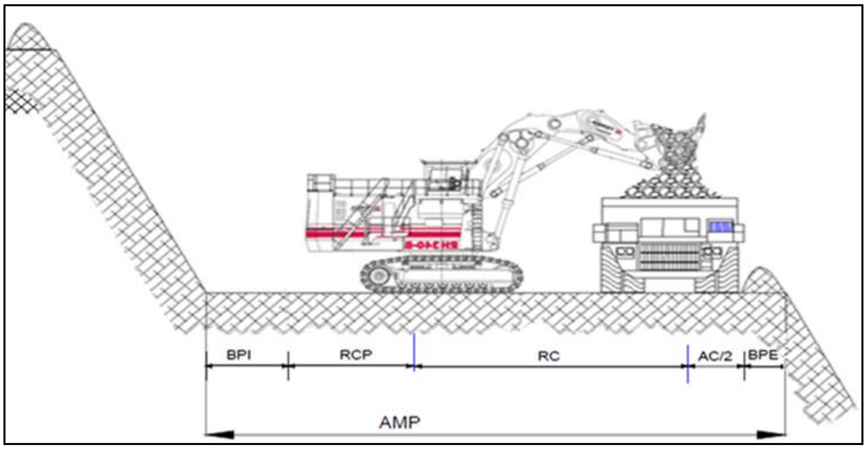


Source: Tetra Tech, 2023

Table 16.7: Mining Width Parameters for Hydraulic Shovel Loading

Field (m)	Description	Hydraulic Shovel
BPE	External Safety Berm	10
BPI	Internal Safety Berm	5
RCP	Counter Weight Radius	10
RC	Loading Radius	20
AC/2	Truck Width	5
AMP	Minimum Loading Width	50

Figure 16.7: Hydraulic Shovel Loading, One Side



Source: Tetra Tech, 2023

The recommended loading area size is shown in Table 16.8.

Table 16.8: Loading Area Size Recommendation

Loading Area Size Depth (m) x Width (m)	Shovel and Backhoe
Recommended Area Size	105 x 85
Minimum Area Size	70 x 50
Recommended Area Size with Reverse Entry	70 x 40
Minimum Area Size with Reverse Entry	50 x 30

16.8.2 General Mine Design Criteria

Operating phase designs use the geotechnical parameters defined in Figure 16.2.

Minimum operating widths for production are 70 m and 55 m (Figure 16.5 and Figure 16.6), and the minimum operating width for bench closures is 50 m (Figure 16.7).

According to the widths defined for the phases and considering the depth where each phase is developed from the surface to the lower part of the pit, the amount of mineralization contained (with a cut-off grade ≥0.18% Cu) in each phase ranges from 2 years to 3 years of feed to the mill.

Benches located in high points may be narrower, requiring the use of smaller capacity equipment (bulldozers, front-end loaders and motor graders) to facilitate development (pre-stripping) by contractors.

The mine plans were developed using the criteria above to assign the feed directly to the mill feed or to the stockpiles.

16.8.3 Phase Design

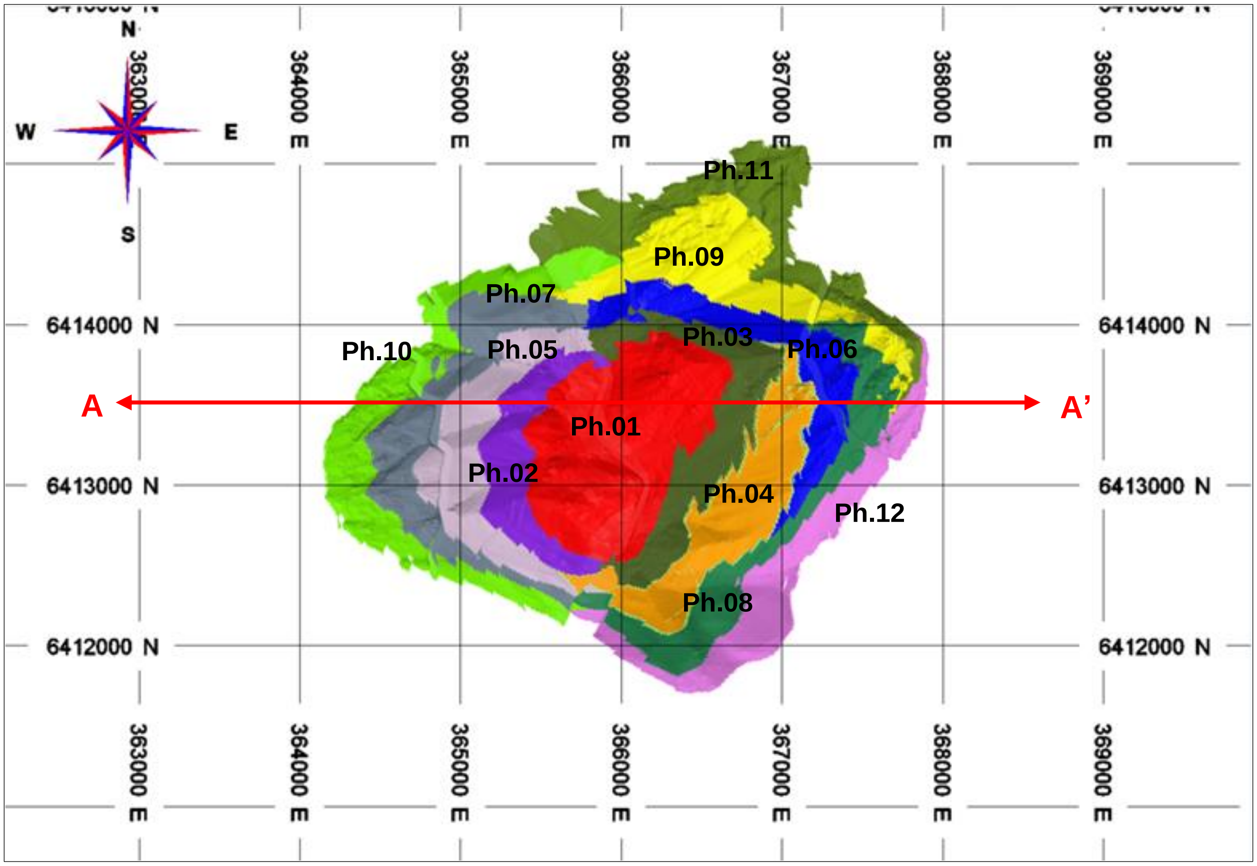
The phase design consists of the development of operational geometries where ramps, accesses and catch berms are detailed, following the extraction sequence defined in Figure 16.4 and the inter-ramp angles in Figure 16.2.

The operating mine design includes 12 phases totaling 1,421 Mt of mineral with an average Cu grade of 0.35% and 4,325 Mt of waste (including waste movement by contractors), using a cut-off grade greater than or equal to 0.18% (Table 16.9). The final condition of the mine is shown in Figure 16.8 for the final operating phases (phase 12 in the south-east, phase 11 in the north-east and phase 10 in the west). Figure 16.9 shows a view of the design phases in a section view AA' (N-6,413,500) showing only Measured and Indicated Resources. It can be seen that the last three phases on the east side reach an elevation of approximately 3,000 masl. On the west side only two phases reach an elevation above 2,500 masl (maximum 2,750 masl). Detailed designs for the phases are shown in Figure 16.10 to Figure 16.21.

Table 16.9: Summary by Operating Phase

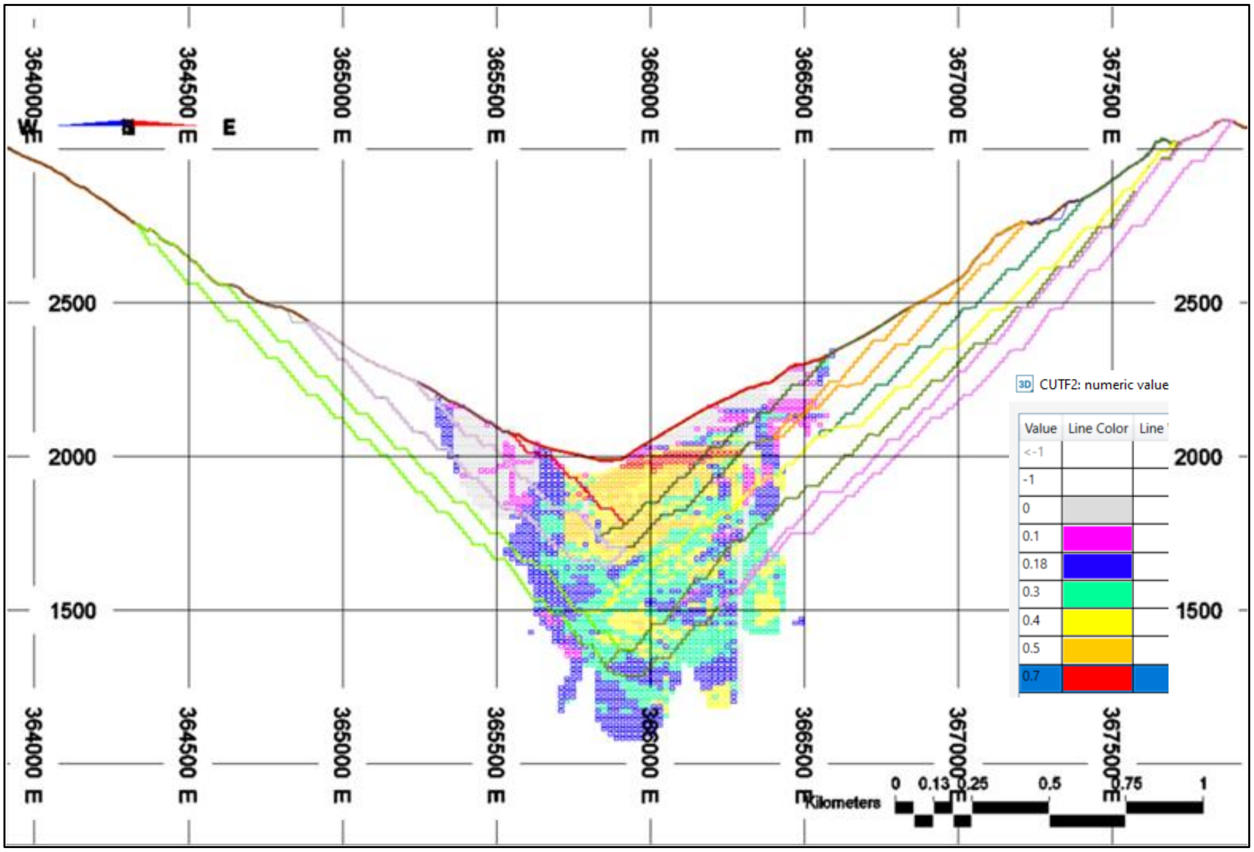
Phase	Mineral Feed Measured & Indicated Resource Cu CoG ≥ 0.18%				Waste and Inferred Resource	Total	Strip Ratio	Global Index	Mineral Feed to Plant (Years)
	Tonnage (kt)	Cu Grade (%)	Mo Grade (ppm)	Ag Grade (g/t)	Tonnage (kt)	Tonnage (kt)			
1	139,260	0.47	133.8	1.3	136,401	275,662	0.98	0.24	2.8
2	129,441	0.33	143.3	1.2	89,296	218,737	0.69	0.20	2.6
3	117,187	0.37	124.7	1.1	151,293	268,480	1.29	0.16	2.4
4	131,484	0.35	95.3	1.2	158,607	290,091	1.21	0.16	2.6
5	131,533	0.33	139.3	1.2	253,747	385,280	1.93	0.11	2.6
6	137,915	0.35	131.7	1.0	320,298	458,212	2.32	0.11	2.8
7	101,585	0.31	141.9	1.0	352,298	453,883	3.47	0.07	2.0
8	100,837	0.34	97.9	1.2	383,938	484,775	3.81	0.07	2.0
9	133,012	0.35	186.2	0.8	470,197	603,208	3.54	0.08	2.7
10	95,430	0.32	138.4	1.1	527,159	622,588	5.52	0.05	1.9
11	83,310	0.36	211.8	0.8	590,524	673,834	7.09	0.04	1.7
12	119,565	0.32	92.1	0.9	616,144	735,709	5.15	0.05	2.4
All Phases	1,420,557	0.353	134.98	1.08	4,049,902	5,470,459	2.85	0.092	28.6

Figure 16.8: Plan View Operating Phase Design



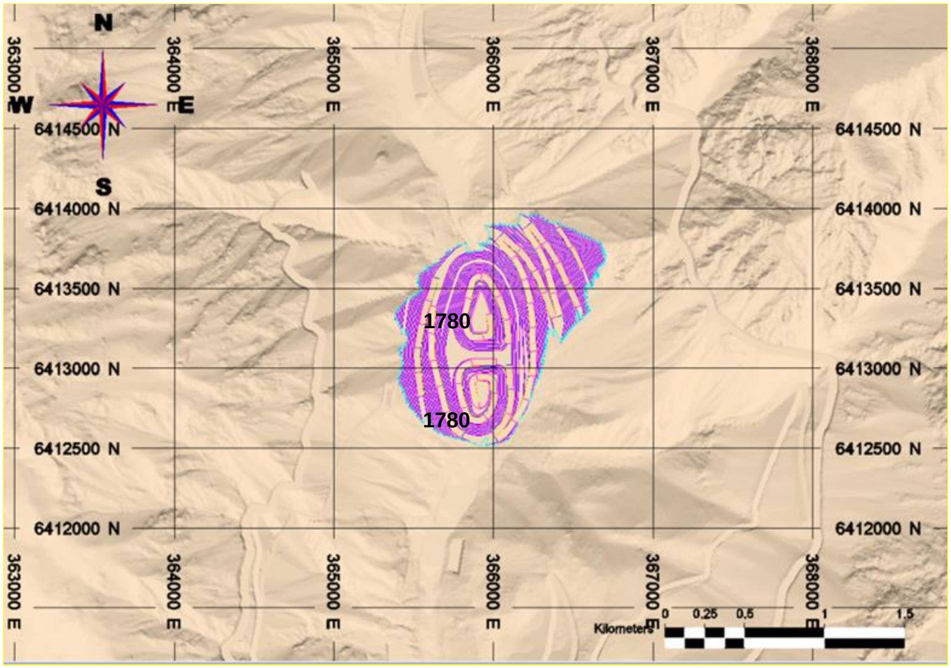
Source: Tetra Tech, 2023

Figure 16.9: Section View AA', N-6.413.500



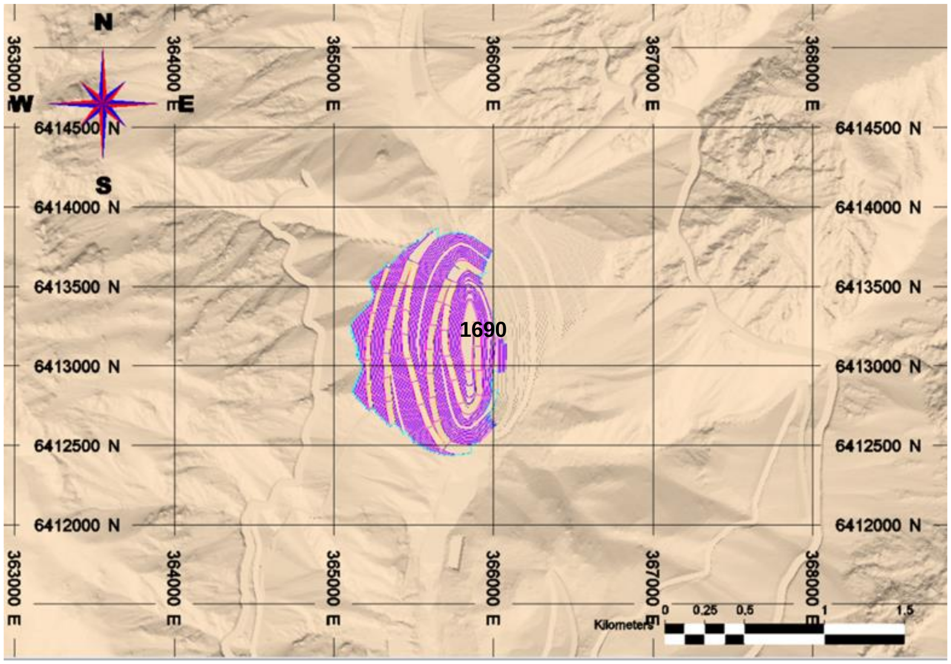
Source: Tetra Tech, 2023

Figure 16.10: Detail Design Phase 1



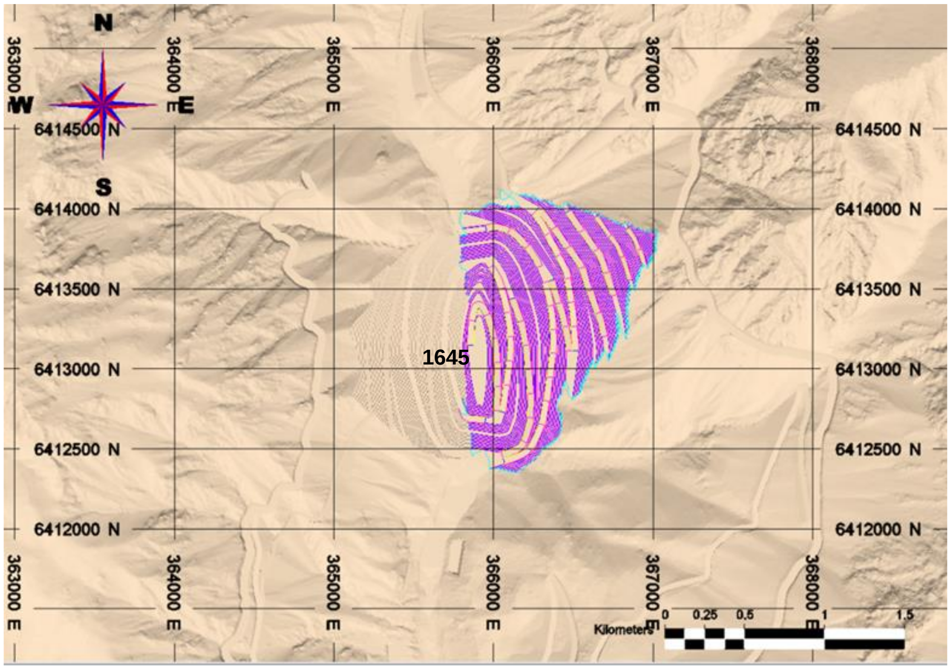
Source: Tetra Tech, 2023

Figure 16.11: Detail Design Phase 2



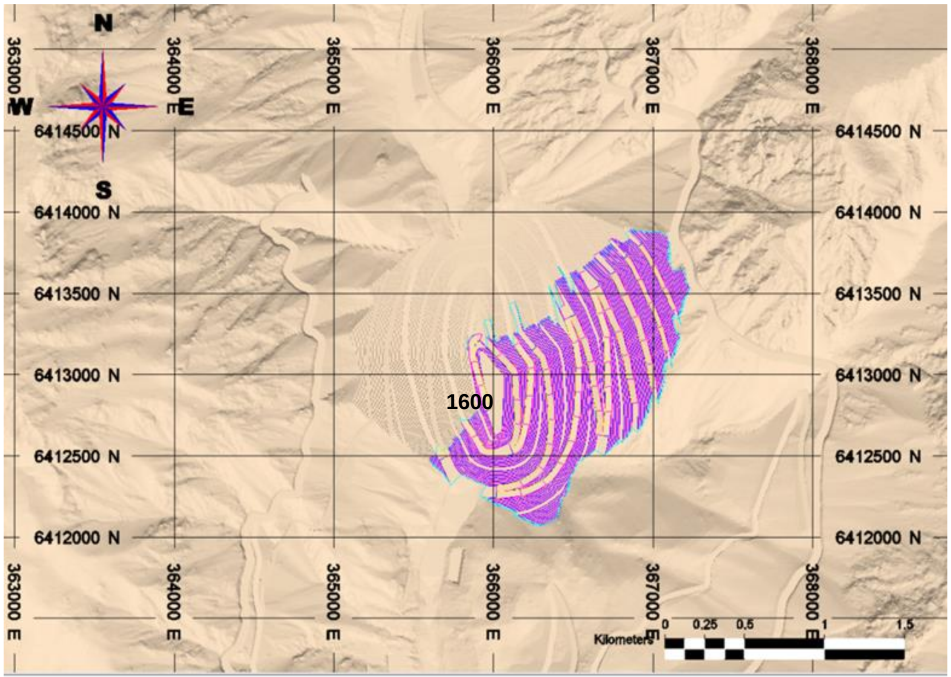
Source: Tetra Tech, 2023

Figure 16.12: Detail Design Phase 3



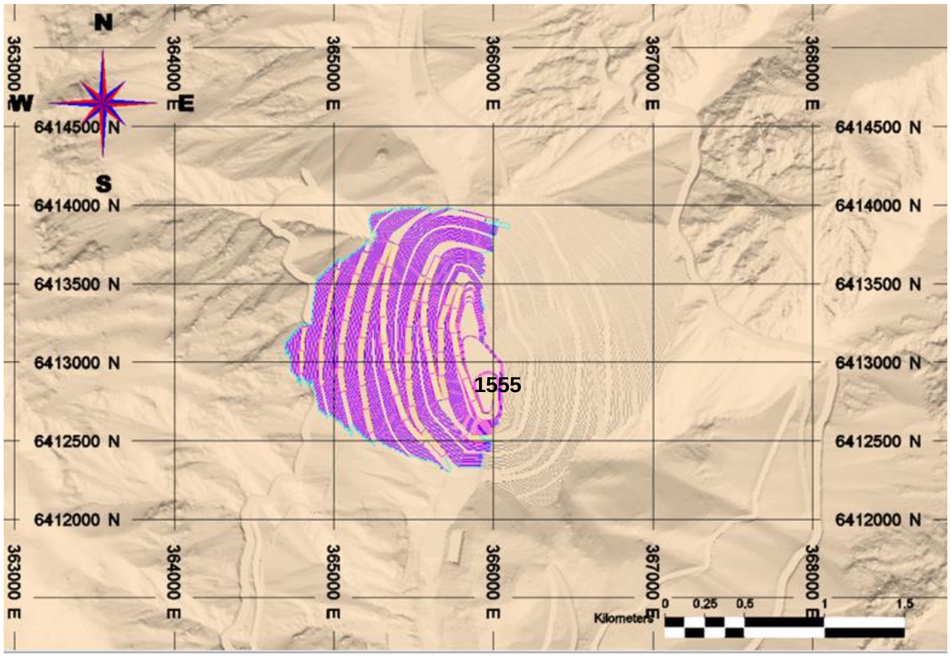
Source: Tetra Tech, 2023

Figure 16.13: Detail Design Phase 4



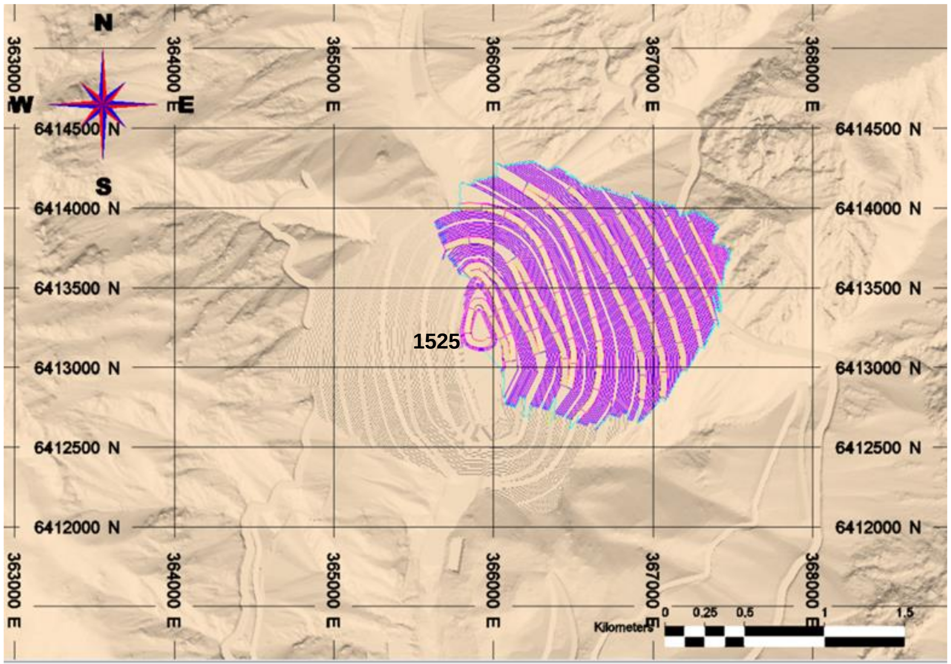
Source: Tetra Tech, 2023

Figure 16.14: Detail Design Phase 5



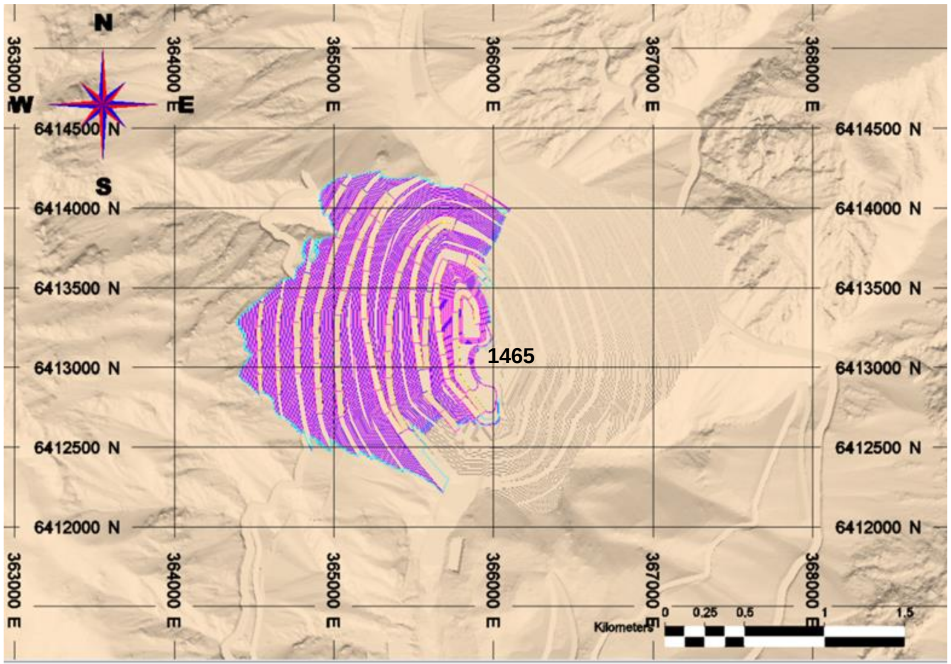
Source: Tetra Tech, 2023

Figure 16.15: Detail Design Phase 6



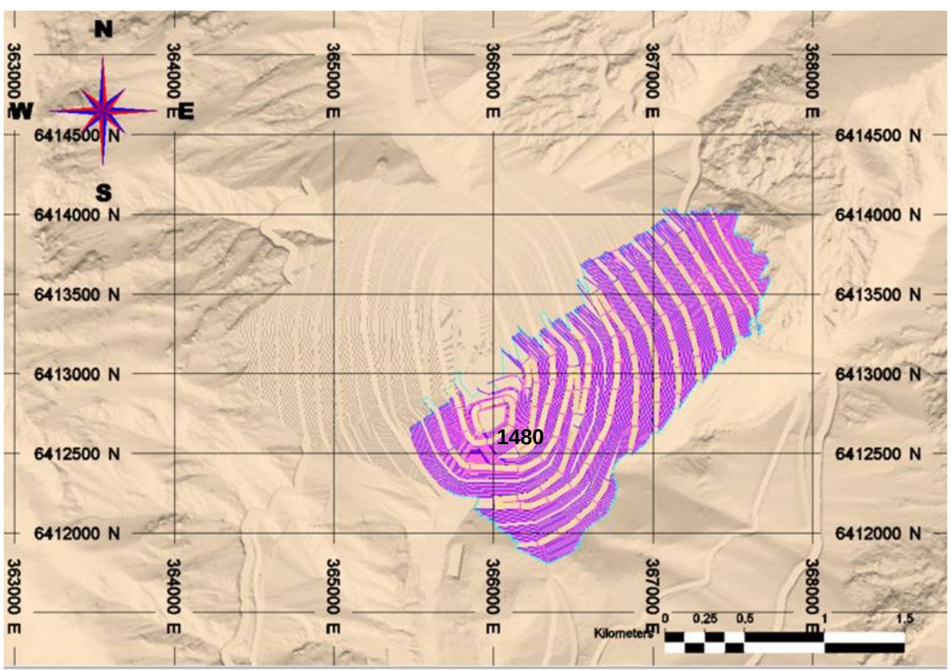
Source: Tetra Tech, 2023

Figure 16.16: Detail Design Phase 7



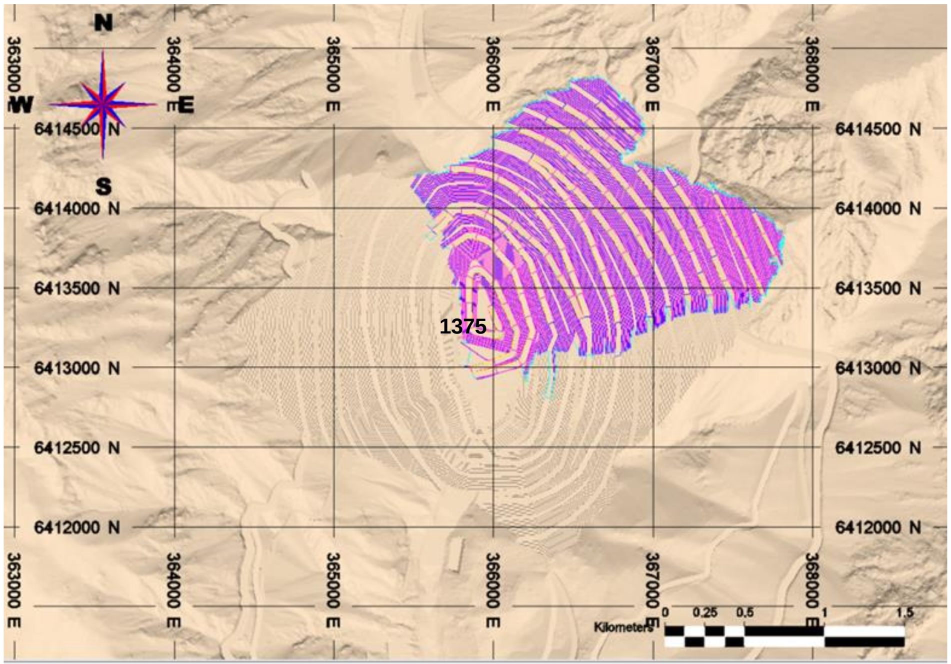
Source: Tetra Tech, 2023

Figure 16.17: Detail Design Phase 8



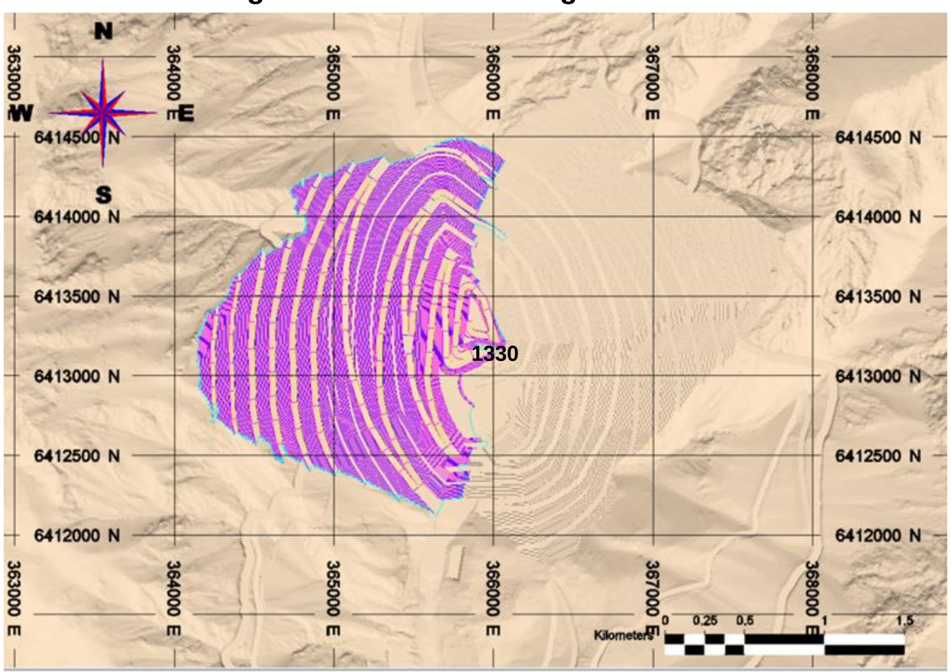
Source: Tetra Tech, 2023

Figure 16.18: Detail Design Phase 9



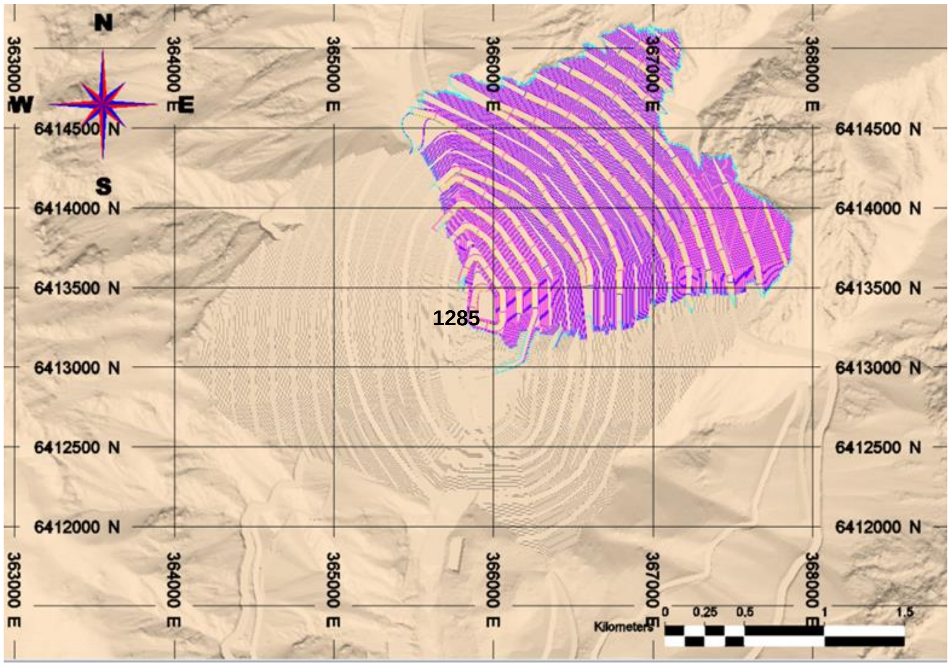
Source: Tetra Tech, 2023

Figure 16.19: Detail Design Phase 10



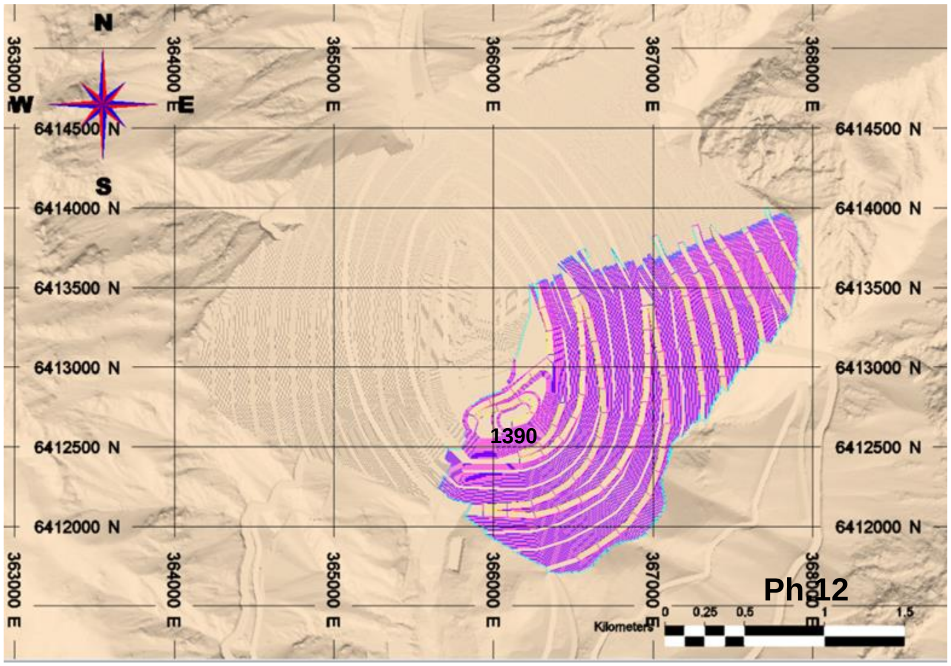
Source: Tetra Tech, 2023

Figure 16.20: Detail Design Phase 11



Source: Tetra Tech, 2023

Figure 16.21: Detail Design Phase 12



Source: Tetra Tech, 2023

16.9 Mine Operations

Estimates for the loading and hauling equipment fleet were based on the mine plan and distances to the mill, waste dump and stockpiles, in addition to operational parameters such as truck speed, availability, operational time and equipment performance.

Subsequently, the drilling and ancillary equipment fleets were estimated using empirical ratios based on criteria such as material movement, number of shovels, number of trucks and number of waste dumps and stockpiles.

Once the fleet requirements were identified, a schedule was defined for equipment increases, replacement, overhaul and removal, to estimate the capital costs for sustaining capital.

Equipment performance, speed and prices were taken from the vendor specification sheets and Tetra Tech's database for similar operations.

16.9.1 Drilling

Three types of equipment were identified for drilling activities: P&H 320XPC, 10-5/8" and KMS ZR122, 10-5/8" for production drilling; and KMS ZT44, 6-7/8" diameter for control (pre-cut) drilling.

16.9.2 Loading and Hauling

Electric shovels, hydraulic shovels, front-end loaders and haul trucks suitable for the loading equipment were considered. The performance of these equipment units was evaluated for medium altitude operations, hence, no equipment de-rating factors for altitude were applied. Loading and hauling equipment capacity and parameters are summarized in Table 16.10 to Table 16.13.

- Loading:
  - 73 yd^3 electric shovel
  - 56 yd^3 hydraulic shovel
  - 31 yd^3 front-end loader
- Hauling
  - Truck (300 t).

Table 16.10: Electric Shovel Parameters

Electric Shovel	Unit	Value
Bucket Capacity	yd^3	73
Fill Factor	%	95
Physical Availability	%	89
Utilization	%	71
Cycle time/pass	min	0.68
N° Pass	pass	3

Table 16.11: Hydraulic Shovel Parameters

Hydraulic Shovel	Unit	Value
Bucket Capacity	yd^3	55
Fill Factor	%	95
Physical Availability	%	82
Utilization	%	64
Cycle time/pass	min	0.80
N° Pass	pass	4

Table 16.12: Front-end Loader Parameters

Front-end Loader	Unit	Value
Bucket Capacity	yd^3	50
Fill Factor	%	95
Physical Availability	%	82
Utilization	%	60
Cycle time/pass	min	1.10
N° Pass	pass	4

Table 16.13: Truck Parameters

Mining Truck	Unit	Value
Hopper Capacity	t	300
Fill Factor	%	95
Speeds		
Up Loaded	km/h	13.9
Down Loaded	km/h	33.0
Up Empty	km/h	40.0
Down Empty	km/h	46.1
Horizontal Loaded	km/h	47.0
Horizontal Empty	km/h	42.3
Operation Time		
Positioning	min	0.50
Load Time	min	2.54
Empty Time	min	0.50
Total Op. Time	min	3.54

16.9.3 Ancillary Equipment

Units that directly support the drilling, loading and hauling operations in the mining operations include bulldozers, wheel dozers, motor graders, water trucks and fuel oil tanker trucks.

16.10 Workforce

The staffing for management, mine operations and mine equipment maintenance activities were estimated.

Shovels and drills are considered critical equipment, therefore, estimates for staffing were based on the number of nominal equipment units. For haul trucks and support equipment, staffing estimates were based on the number of equipment units available. The criteria to estimate the number of mine equipment operators and maintenance staff are summarized in Table 16.14.

Factors of 3.5% and 4.0% were used for absenteeism and vacation, respectively. An average availability of 83% was used to estimate the haul truck and support equipment requirements.

Table 16.14: Operator and Maintenance Staffing Criteria

Item	Staffing Estimate
Critical availability equipment (Drilling and Loading)	No. of equipment x 4 x (1 + % Absenteeism + % Vacations)
Mass Equipment (Non Critical Availability)	(No. of equipment x 4 x (1 +% Absenteeism + % Vacations)) x % Availability
Mine Equipment Maintenance	1.3 x No. Operators

16.11 Mining Plan Options

A long-term mine plan study was conducted which showed the production scenarios for the concentrator for the throughput requirement.

The mine plans are strategic and aimed at optimizing the cut-off grade profiles to obtain the best economic value. As a result, material movements are irregular over time and occasionally fail to fill the mill capacity. This shortfall is mitigated by using stockpiles.

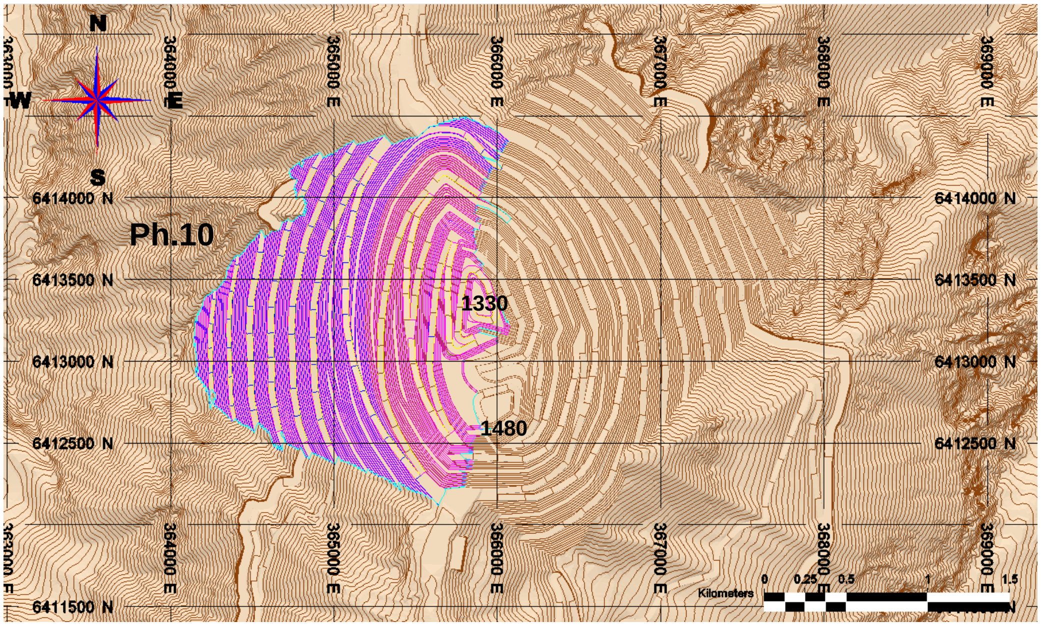
The analysis considered a wide range of mine capacities and adjustments of stockpile cut-off grades. The final plans developed showed the best economic value.

A short-term plan was also developed to ensure compliance with the first year mill feed requirements and mine development. Contractors develop benches at the top of each phase to remove waste rock on narrow mining widths at the start of each phase with smaller equipment.

16.11.1 Ultimate Pit

Operating phase designs were used for this mine plan. Thus, in conducting a marginal phase analysis, the ultimate pit was defined as phase 10, as shown in Figure 16.22. Phases 11 and 12 do not provide additional economic value, mainly because of the high strip ratio.

Figure 16.22: Ultimate Pit (Phase 10)



Source: Tetra Tech, 2023

16.11.2 Mine Production Schedule

Table 16.15 and Figure 16.23 summarize the movement of materials estimated in the mine plan to comply with the production schedule. There is a pre-stripping phase of 3 years followed by a production life of 26 years. Total movement reaches a maximum of 550,000 tpd from Year 10 through Year 19, then starts to decline. In the final periods of the mine plan, the material sent to the mill comes from the stockpiles.

The first 10 years have a maximum movement of 300,000 tpd. From Years 1 to 8, The strip ratio (waste:ore) is 1.54. The total movement over the LOM is 4,211 Mt (including re-handling and waste movement by contractors), with a total feed of 1,220 Mt to the concentrator and a strip ratio LOM (waste:ore) of 2.33.

The material flow to be processed was optimized by separating the mined material into four categories:

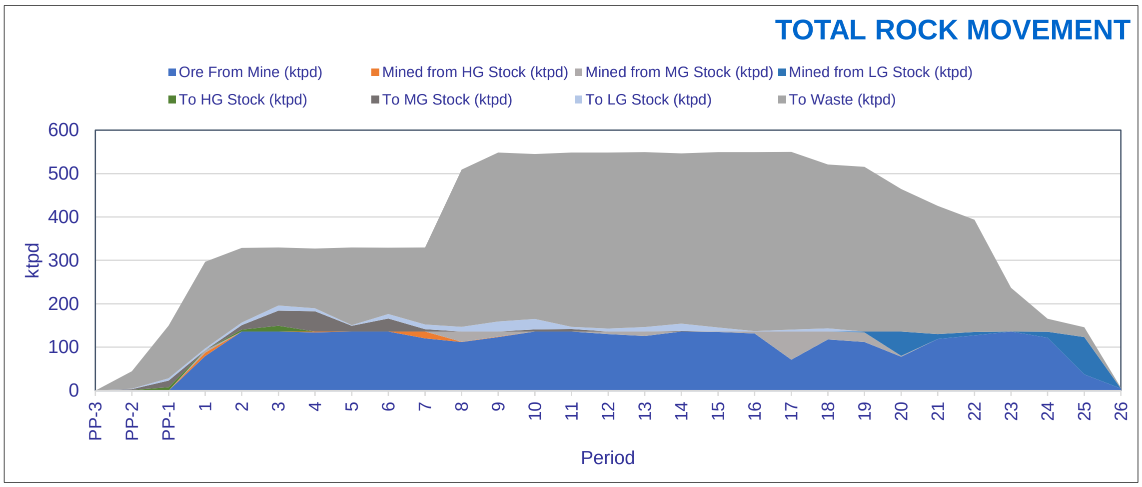
- Mine to Mill: material shipped directly to the mill, with a variable cut-off grade
- High-Grade (HG) Stockpile: material with grade below direct mine to mill grade and above a cut-off grade of 0.34% CuEq
- Medium-Grade (MG) Stockpile: material with grade below HG Stockpile and above a cut-off grade of 0.25% CuEq
- Low-Grade (LG) Stockpile: material with grade below MG Stockpile and above a cut-off grade of 0.20% CuEq.

Table 16.15: Production Schedule

Period (years)	Days	COG	Mineral to Mill					Waste (kt) (including contractors)	Strip ratio (w:o)	Total			ktpd	To Mill: Contained		
			kt	CuEq (%)	Cu (%)	Mo (ppm)	Ag (g/t)			Onsite (kt)	Rehandling (kt)	W/Rehandling (kt)		Cu (t)	Mo (t)	Ag (koz)
- 3	365							15,947				15,947				
- 2	365							46,986		48,514		48,514	133			
- 1	365							68,611		78,881		78,881	216			
1	365	0.25%	33,921	0.53%	0.49%	82.33	1.26	84,561	2.49	115,187	4,827	120,014	329	166,823	2,793	1,378
2	365	0.36%	49,640	0.59%	0.53%	145.35	1.38	74,362	1.50	131,580		131,580	360	262,844	7,215	2,200
3	365	0.37%	49,640	0.52%	0.46%	152.04	1.42	48,832	0.98	120,428		120,428	330	228,443	7,547	2,273
4	365	0.33%	49,640	0.45%	0.40%	138.30	1.20	65,400	1.32	133,864	699	134,562	369	196,723	6,865	1,921
5	365	0.32%	49,640	0.48%	0.42%	167.73	1.21	96,192	1.94	151,190		151,190	414	206,254	8,326	1,930
6	365	0.31%	49,640	0.45%	0.39%	165.46	1.20	86,200	1.74	150,536		150,536	412	191,660	8,214	1,912
7	365	0.26%	49,640	0.42%	0.37%	117.42	1.21	79,526	1.60	129,392	5,737	135,129	370	184,164	5,829	1,928
8	365	0.25%	49,640	0.40%	0.36%	108.67	1.16	146,276	2.95	191,120	8,824	199,944	548	176,272	5,394	1,847
9	365	0.25%	49,640	0.39%	0.35%	97.51	1.22	156,117	3.14	209,627	4,655	214,282	587	174,584	4,840	1,947
10	365	0.26%	49,640	0.38%	0.34%	101.18	1.12	152,757	3.08	212,919		212,919	583	168,577	5,023	1,785
11	365	0.28%	49,640	0.42%	0.35%	186.22	1.23	160,661	3.24	214,269		214,269	587	176,172	9,244	1,956
12	365	0.24%	49,640	0.38%	0.33%	130.22	1.14	148,081	2.98	198,047	2,180	200,227	549	162,918	6,464	1,818
13	365	0.25%	49,640	0.39%	0.35%	114.96	1.06	163,023	3.28	212,653	3,824	216,476	593	172,648	5,706	1,691
14	365	0.25%	49,640	0.44%	0.39%	153.65	1.05	159,259	3.21	215,570		215,570	591	192,951	7,627	1,670
15	365	0.23%	49,640	0.39%	0.34%	127.88	1.08	147,483	2.97	200,307	295	200,602	550	171,010	6,348	1,721
16	365	0.22%	49,640	0.37%	0.31%	186.82	1.07	150,534	3.03	198,950	1,662	200,612	550	151,998	9,274	1,702
17	365	0.23%	49,640	0.32%	0.28%	101.18	0.96	149,486	3.01	177,023	23,694	200,718	550	138,198	5,023	1,529
18	365	0.24%	49,640	0.39%	0.35%	111.77	1.06	137,828	2.78	183,416	6,706	190,122	521	173,889	5,548	1,698
19	365	0.20%	49,640	0.38%	0.34%	118.76	1.19	138,567	2.79	179,473	8,734	188,207	516	166,890	5,895	1,902
20	365	0.18%	49,620	0.32%	0.29%	85.32	0.91	119,952	2.42	148,387	21,185	169,571	465	142,210	4,233	1,449
21	365	0.18%	47,514	0.42%	0.36%	164.60	0.89	107,793	2.27	151,199	4,108	155,307	425	171,240	7,821	1,358
22	365	0.18%	49,372	0.40%	0.33%	206.60	0.85	94,388	1.91	140,844	2,916	143,760	394	161,347	10,200	1,349
23	365	0.19%	49,640	0.39%	0.34%	140.46	1.09	36,775	0.74	86,415		86,415	237	168,230	6,972	1,740
24	365	0.18%	49,498	0.36%	0.31%	158.67	1.04	10,907	0.22	55,357	5,047	60,405	165	151,116	7,854	1,656
25	365	0.19%	45,024	0.27%	0.23%	107.16	0.84	8,162	0.18	21,677	31,508	53,185	146	102,969	4,825	1,221
26	81	0.20%	1,825	0.35%	0.26%	274.36	0.70	706	0.39	2,530		2,530	31	4,744	501	41
Total			**1,219,932**	**0.41%**	**0.36%**	**135.73**	**1.11**	**2,855,370**	**2.33**	**4,059,355**	**136,602**	**4,211,904**				

Figure 16.23 shows the total rock movement for Vizcachitas Project with own equipment. It is observed that there is a mine capacity increase in Year 8 (through to Year 17) which is explained by the increased strip ratio of areas with a higher percentage of Inferred Resource (that is treated as waste rock in this TR). Indeed, approximately 25% of the material to be sent to the waste dumps during the LOM is Inferred Resource.

Figure 16.23: Production Schedule (own equipment only)

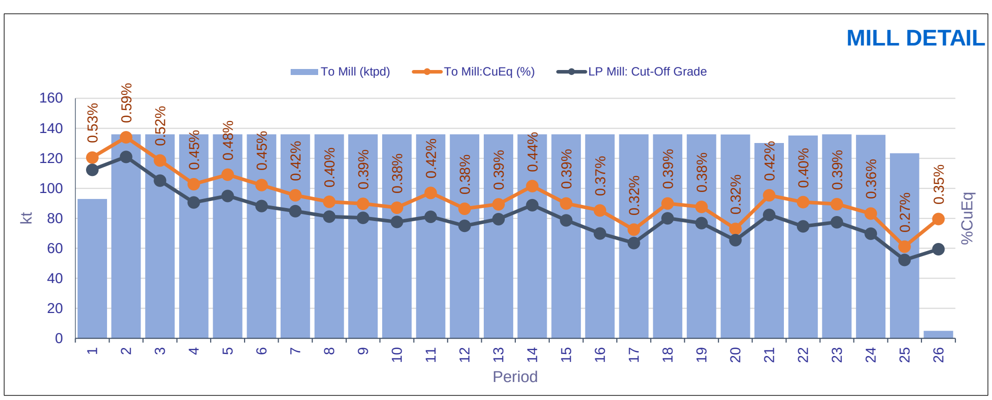


Source: Tetra Tech, 2023

Mine production commences in the first quarter of the fourth year after the start of construction (period 1 in Figure 16.24), reaching the design capacity at the end of that period. From then on, the concentrator operates at full capacity until mine schedule period 25, decreasing in the last year (period 26).

The feed sent to mill is shown in Figure 16.24. During the LOM feed is sent directly to the mill and also sent from the three stockpiles according to the variable cut-off grade strategy applied.

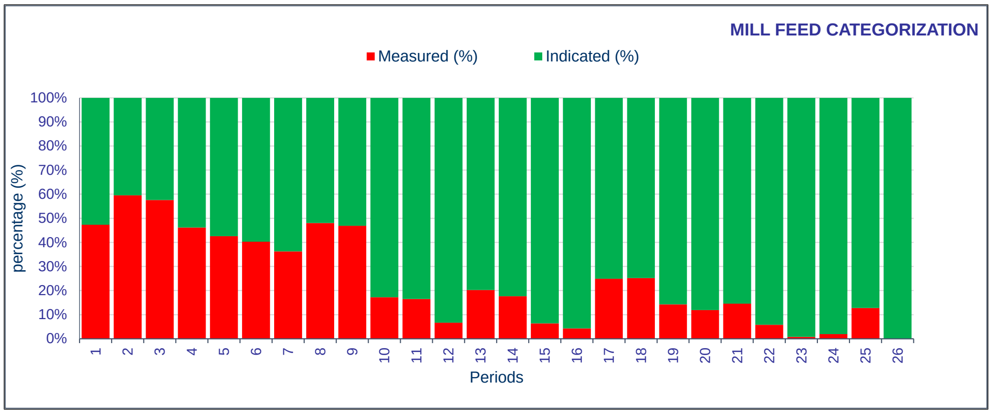
Figure 16.24: Feed to Mill (Mine Schedule)



Source: Tetra Tech, 2023

Figure 16.25 shows the feed to the mill according to the resource category (25% of the total resources are Measured Resources and 75% are Indicated Resources).

Figure 16.25: Mine Schedule Feed Categorization



Source: Tetra Tech, 2023

16.11.3 Equipment Fleet

The main mine equipment to meet the production schedule is listed below:

- Production drill (10-5/8” – 6-7/8’’)
- Electric shovel (73 yd³)
- Hydraulic shovel (55 yd³)
- Front-end loader (50 yd³).

Support equipment is listed below:

- Bulldozer
- Wheeldozer
- Motor grader
- Water truck.

The mine equipment units (excluding contractors’ equipment) required for the mine plan are summarized in Table 16.16.

Table 16.16: Mine Equipment - PFS Mine Plan

Equipment Required		Fleet													
		-2	-1	1	2	3	4	5	6	7	8	9	10	11	12
Shovel P&H 4100XPC - 73 yd^3	un		1	2	2	2	2	2	2	2	4	4	4	4	4
Shovel KMS PC 8000 55 yd^3	un	1	1	2	2	2	2	2	2	2	2	2	2	2	2
Loader P&H L-2350 50 yd^3	un	1	1	1	2	2	2	2	2	2	2	2	2	2	2
KMS 930E-5	un	4	11	17	22	22	22	29	29	29	54	62	65	70	70
Drill P&H320XPC	un	1	1	2	3	3	3	3	3	3	3	3	3	3	3
Drill KMS ZR122	un			1	2	2	2	2	2	2	2	2	2	2	2
Drill KMS ZT44	un	1	1	2	3	3	3	3	3	3	3	3	3	3	3
Bulldozer D475A-5E0	un	2	3	5	6	6	6	6	6	6	6	6	6	6	6
Wheeldozer WD 900-3A	un	2	2	3	4	4	4	4	4	4	4	4	4	4	4
Motor Grader GD 825A-2	un	2	3	5	6	6	6	6	6	6	6	6	6	6	6
Water Truck HD 785-7 WT	un	2	2	4	4	4	4	4	4	4	4	4	4	4	4
Water Tank	un	1	2	3	4	4	4	4	4	4	4	4	4	4	4
CF WA600	un	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Grove	un	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Excavator PC300LC-8	un	1	2	3	4	4	4	4	4	4	4	4	4	4	4
Light Tower Atlas QLT M10	un	3	4	8	9	9	9	9	9	9	9	9	9	9	9
Lowboy Truck	un	1	1	1	1	1	1	1	1	1	1	1	1	1	1

Equipment Required		Fleet													
		13	14	15	16	17	18	19	20	21	22	23	24	25	26
Shovel P&H 4100XPC - 73 yd^3	un	4	4	4	4	4	4	4	4	3	2	1	1	1	1
Shovel KMS PC 8000 55 yd^3	un	2	2	2	2	2	2	2	2	2	2	2	1	1	1
Loader P&H L-2350 50 yd^3	un	2	2	2	2	2	2	2	2	2	1	1	1	1	1
KMS 930E-5	un	71	71	76	76	74	74	70	64	56	56	36	25	19	16
Drill P&H320XPC	un	3	3	3	3	3	3	3	3	3	3	3	3	2	2
Drill KMS ZR122	un	2	2	2	2	2	2	2	2	2	2	2	2	1	1
Drill KMS ZT44	un	3	3	3	3	3	3	3	3	3	3	3	3	2	2
Bulldozer D475A-5E0	un	6	6	6	6	6	6	6	6	5	4	3	3	3	2
Wheeldozer WD 900-3A	un	4	4	4	4	4	4	4	4	3	3	2	2	2	2
Motor Grader GD 825A-2	un	6	6	6	6	6	6	6	6	5	4	3	3	3	2
Water Truck HD 785-7 WT	un	4	4	4	4	4	4	4	4	4	4	2	2	2	2
Water Tank	un	4	4	4	4	4	4	4	4	3	2	2	2	2	1
CF WA600	un	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Grove	un	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Excavator PC300LC-8	un	4	4	4	4	4	4	4	4	3	2	2	2	2	1
Light Tower Atlas QLT M10	un	9	9	9	9	9	9	9	9	8	7	5	5	5	3
Lowboy Truck	un	1	1	1	1	1	1	1	1	1	1	1	1	1	1

16.11.4 Workforce

The estimated staffing required to meet the mine plan production is summarized in Table 16.17 and Table 16.18.

Table 16.17: Supervisory Staff by Year

Supervision	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12
Mine Manager	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Analyst	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mine Operations Superintendent	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Shift Boss	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Dispatch Engineer	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Dispatch Operator	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Trainer	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Mine Maintenance Superintendent	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Assistant	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Workshop Boss	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Chief Maintenance Planning	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Maintenance Planning Enginner	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Maintenance Supervisor (WorkShop)	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Maintenance Field Chief	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Programmers	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Maintenance Shift Boss	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Engineering Superintendent	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mine Planning Enginner (Long Term / Reserves)	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Geologist (Long Term / Reserves)	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Mine Planning Assistant (Long Term / Reserves)	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mine Planning Engineer (Short Term)	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Surveyor	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Surveyor Assistant	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Mine Geologist (Short Term)	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Geologist Helper	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Core Shed Chief	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Samplers Chief	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Samplers	8	8	8	8	8	8	8	8	8	8	8	8	8	8
Total	59	59	59	59	59	59	59	59	59	59	59	59	59	59

Supervision	13	14	15	16	17	18	19	20	21	22	23	24	25	26
Mine Manager	1	1	1	1	1	1	1	1	1	1	1	1		
Analyst	1	1	1	1	1	1	1	1	1	1	1	1		
Mine Operations Superintendent	1	1	1	1	1	1	1	1	1	1	1	1		
Shift Boss	4	4	4	4	4	4	4	4	4	4	4	4		
Dispatch Engineer	1	1	1	1	1	1	1	1	1	1	1	1		
Dispatch Operator	4	4	4	4	4	4	4	4	4	4	4	4		
Trainer	2	2	2	2	2	2	2	2	2	2	2	2		
Mine Maintenance Superintendent	1	1	1	1	1	1	1	1	1	1	1	1	1	
Assistant	1	1	1	1	1	1	1	1	1	1	1	1	1	
Workshop Boss	2	2	2	2	2	2	2	2	2	2	2	2		
Chief Maintenance Planning	1	1	1	1	1	1	1	1	1	1	1	1		
Maintenance Planning Enginner	2	2	2	2	2	2	2	2	2	2	2	2		
Maintenance Supervisor (WorkShop)	4	4	4	4	4	4	4	4	4	4	4	4		
Maintenance Field Chief	2	2	2	2	2	2	2	2	2	2	2	2	2	
Programmers	2	2	2	2	2	2	2	2	2	2	2	2	2	
Maintenance Shift Boss	4	4	4	4	4	4	4	4	4	4	4	4		
Engineering Superintendent	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mine Planning Enginner (Long Term / Reserves)	1	1	1	1	1	1	1	1	1	1	1	1	1	
Geologist (Long Term / Reserves)	2	2	2	2	2	2	2	2	2	2	2	2	2	
Mine Planning Assistant (Long Term / Reserves)	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mine Planning Engineer (Short Term)	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Surveyor	2	2	2	2	2	2	2	2	2	2	2	2	2	
Surveyor Assistant	2	2	2	2	2	2	2	2	2	2	2	2	2	
Mine Geologist (Short Term)	2	2	2	2	2	2	2	2	2	2	2	2	2	
Geologist Helper	2	2	2	2	2	2	2	2	2	2	2	2	2	
Core Shed Chief	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Samplers Chief	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Samplers	8	8	8	8	8	8	8	8	8	8	8	8	8	
Total	59	59	59	59	59	59	59	59	59	59	59	59	32	7

Table 16.18: Mine Operating Staff by Year

Operating Staff	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12
Loader	12	18	28	33	33	33	33	33	33	44	44	44	44	44
Haulage	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Drilling	18	18	27	27	27	27	27	27	27	27	27	27	27	27
TOTAL OPERATORS	30	36	55	60	60	60	60	60	60	71	71	71	71	71
Bulldozer D375A-6R	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Bulldozer D475A-5E0	8	12	19	23	23	23	23	22	22	22	22	22	22	22
Wheeldozer WD 600-6	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Wheeldozer WD 900-3A	8	8	12	15	15	15	15	15	15	15	15	15	15	15
Motor Grader GD 825A-2	8	12	19	23	23	23	23	22	22	22	22	22	22	22
Water Truck HD 785-7 WT	10	10	19	19	19	19	19	19	19	19	19	19	19	19
Water Tank	4	8	12	15	15	15	15	15	15	15	15	15	15	15
CF WA600	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Grove	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Excavator PC300LC-8	4	8	12	15	15	15	15	15	15	15	15	15	15	15
Light Tower Atlas QLT M10	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Lowboy Truck	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Total Services Equipment	54	70	105	122	122	122	122	120	120	120	120	120	120	120
Total Mine Operator	84	106	160	182	182	182	182	180	180	191	191	191	191	191

Operating Staff	13	14	15	16	17	18	19	20	21	22	23	24	25	26
Loader	44	44	44	44	44	44	44	44	39	28	23	18	18	18
Haulage	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Drilling	27	27	27	27	27	27	27	27	27	27	27	27	27	27
TOTAL OPERATORS	71	71	71	71	71	71	71	71	66	55	50	45	45	45
Bulldozer D375A-6R	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Bulldozer D475A-5E0	22	22	22	22	22	22	22	22	18	15	11	11	11	8
Wheeldozer WD 600-6	0	0	0	0	0	0	0	0	11	11	8	8	8	8
Wheeldozer WD 900-3A	15	15	15	15	15	15	15	15	18	15	11	11	11	8
Motor Grader GD 825A-2	22	22	22	22	22	22	22	22	19	19	10	10	10	10
Water Truck HD 785-7 WT	19	19	19	19	19	19	19	19	12	8	8	8	8	4
Water Tank	15	15	15	15	15	15	15	15	4	4	4	4	4	4
CF WA600	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Grove	4	4	4	4	4	4	4	4	11	8	8	8	8	4
Excavator PC300LC-8	15	15	15	15	15	15	15	15	0	0	0	0	0	0
Light Tower Atlas QLT M10	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Lowboy Truck	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Total Services Equipment	120	120	120	120	120	120	120	120	101	88	68	68	68	54
Total Mine Operator	191	191	191	191	191	191	191	191	167	143	118	113	113	99

17 RECOVERY METHODS

17.1 Summary

The Vizcachitas processing plant is designed for a capacity of 136,000 tpd and is located in two areas within the Rocín River valley: the crushing and grinding circuits on a platform of compacted waste rock obtained from the pre-stripping operation; and the concentrator and filtration plant area, in the valley on a natural plateau next to the access road to the east side of the valley.

The run of mine material is fed to a primary crusher (MK60-110E) where it is reduced to 7 inches, and then to three secondary cone crushers (2,500 HP) in open circuit. The crusher product at 2 inches is stored in a surge bin that feeds the tertiary crushing circuit that includes two HRC3000 high pressure grinding roll (HPGR) crushers arranged in a closed circuit with banana screens.

The undersize from the crushing plant (P80 <3.5 mm) is fed to two 23 MW ball mills arranged in a reverse closed circuit with hydrocyclones to achieve a P80 of 240 µm. The overflow from the hydrocyclones is thickened to 50% solids %wt and then transported via a concrete launder to the flotation stage.

The flotation circuit has two stages: a Cu-Mo bulk flotation and a Cu-Mo selective flotation. Both circuits use mechanical agitated tanks and column cells to produce concentrates. The copper concentrate is filtered, loaded into rotating sealed containers (Rotainers) and then trucked to the port of Ventanas. The molybdenum concentrate is filtered, loaded and stored in bags, and then hauled by truck to the port of Ventanas or to alternative destinations agreed with buyers.

The tailings from flotation are classified using hydrocyclones. The coarse fraction is fed to belt filters and the fine fraction is thickened and then filtered in pressure filters. The filtered cake contains an average moisture of 15% and is of disposed in 2 m layers (co-mingled with mine waste rock) in the bottom and across the valley to shape the dry stack filtered tailings deposit.

The plant design and the equipment selection are backed up by bench-scale testwork and some pilot plant testwork carried out using representative composites of the first years of operation. The concentrator will produce clean concentrates containing approximately 24% Cu. The fresh water consumption is around 271 L/s (with a maximum of 300 L/s) and the power consumption is 16.6 kWh/t.

17.2 Process Design Criteria

The process design criteria consider a 136,000 t/d concentrator that contains a crushing circuit using HPGR followed by a milling stage to achieve a P80 of 240 µm. Copper and molybdenum are recovered in a froth flotation circuit and most of the process water is recovered by tailings filtration.

The primary and secondary crushing stages are arranged in an open circuit configuration to minimize the footprint for the conveyors and the coarse ore stockpile.

The main design criteria are based on results obtained from the PFS testwork carried out on representative samples (SGS, 2021); other sources used in the design are listed in Table 17.1. Table 17.2 to Table 17.4 provide the process design criteria.

Table 17.1: Source Codes for Process Design Criteria

Code	Description
A	Los Andes Copper Basic Design Criteria
B	Benchmark
C	Derived from Los Andes Copper PFS Test Work Program
D	Derived from Design Calculations
E	Engineering Handbook Data
F	Normal Industrial Practice
G	Assumed Values - Represent Best Estimates from Available Data
H	Manufacturer/Vendor - Originated Data
I	Criteria from a Technology Supplier
J	Preliminary Information - Requires Further Evaluation and Confirmation
K	Data from Simulations and Mass Balances

Table 17.2: Process Design Criteria

Description	Unit	Nominal	Design	Source	Comments
General					
Plant Design Capacity	Mt/y	49.64	49.64	A	PFS Mine Plan
Plant Design Capacity	ktd	136,000	136,000	A	
Crushing Plant Runtime	%	70	70	K	Metso BRUNO Simulation
HPGR + Plant Runtime	%	92	92	B	
ROM Particle Size Distribution					
Top Size - F100	in	24	40	B	Based on Nearby Porphyries
Top Size - F100	mm	600	1,016	B	Based on Nearby Porphyries
Size, mm		% Passing			
600	%	100	100	B	Based on Nearby Porphyries
177.8	%	90	90	B	Based on Nearby Porphyries
50.8	%	50.6	50.6	B	Based on Nearby Porphyries
25.4	%	31.4	31.4	B	Based on Nearby Porphyries
Ore Characteristics					
Ore Specific Gravity	t/m^3	2.74	2.69	C	SGS Test Work
Ore Moisture	%	2.5	3	A, B	
Natural pH	-	7.56	7.22	C	
Bond Work Index	kWh/t	11.4	15.3	C	90th Percentile for Design
Bond Abrasion Index	g	0.1918	0.2886	C	90th Percentile for Design
JK SMC Axb	-	38	33.1	C	90th Percentile for Design
HPGR Specific Energy	kWh/t	1.6	1.6	C	Metso PBT Test

Table 17.3: Process Design Criteria (Crushing and Grinding)

Description	Unit	Nominal	Design	Source	Comments
Primary Crushing					
Throughput	t/h	8,100	8,100	K	Metso Simulation
Crusher Type	-		Gyratory	F	
Crusher Feed Opening	in	50	60	F	
Open Side Setting	in	7		K	Metso Simulation
Secondary Crushing					
Throughput	t/h	8100		K	Metso Simulation
Crusher Type	-		Cone	F	
Circuit Type	-		Open	A	
Closed Side Setting	mm	42	42	K	Metso Simulation
Max.Top Size in Product	mm	75	75	I	Metso
Product Size - P80	mm	45	45	K	Metso Simulation
Ore storage					
Material	-		Crushed ore	F	
Live Capacity	t		56,000	J	
Live Capacity	h		9	D	
Tertiary Crushing					
Throughput	t/h	6,159	6,159	D	
Crusher Type	-		HRC	A	
Circuit Type	-		Closed/Direct	F	
Circulating Load	%	49	60	K	Metso Simulation
HPGR Bin Capacity	t	34,000	34,000	F	
Screen Type	-		Banana	F	
Screen Operation	-		Wet	F	
Product Size - P80	mm	3.5	3.5	K	Metso Simulation
Grinding					
Throughput	t/h	6,159	6,159	D	
Equipment Type	-		Ball Mill	F	
Circuit Type	-		Closed-Reverse	F	
Drive	-		GMD	A	
Discharge Type	-		Overflow	I	Metso Calculation
Ball Fill Volume	%	29	32	I	Metso Calculation
Percent Critical Speed	%	78	78.8	I	Metso Calculation
Product Size - P80	micron	240	240	A	
Circulating Load	%	250	350	F	
Ore thickeners					
Throughput	t/h	6,159	7,365	D	
Quantity	-		2	A	
Discharge % Solids	%	50	50	K	Launder Design Study
Ore Launder					
Capacity	kt/d	136	150	K	Launder Design Study
Minimum Throughput	kt/d		88	K	Launder Design Study
% Solids	%		50	K	Launder Design Study
Slope	%		1	A, B	

Table 17.4: Process Design Criteria (Flotation and Dewatering)

Description	Unit	Nominal	Design	Source	Comments
Bulk Flotation					
Throughput	t/h	6,159	6,159	D	
Cu Head Grade	%	0.34	0.42	A, C	Mine plan and test work
Mo Head Grade	ppm	106	107	A, C	Mine plan and test work
Rougher Cells Type	-		Agitated Tank	F	
Residence Time	min		25	C	
Rougher Mass Pull	%	6.81	8.76	C	SGS PFS test work
Rougher Cu Recovery	%		94.1	C	SGS PFS test work
Regrinding Equipment	-		HIG Mill	A, F	
Regrinding Product - P80	micron	25	25	C	SGS PFS test work
Cleaner/Scavenger Stages	-		1	A, C	SGS PFS test work
Cleaner Flotation Cell	-		Column	F	
Cleaner Mass Pull	%	22.2	24.1	C	SGS PFS test work
Cleaner Cu Recovery	%		98.7	C	SGS PFS test work
Overall Mass Pull	%	1.51	1.78	D	
Overall Cu Recovery	%	91.1	92.9	D	Mine Plan
Overall Bulk Mo Recovery	%	84.4	89	D	Mine Plan
Cu Concentrate Grade	%	23.7	22.5	C	SGS PFS test work
Mo Concentrate Grade	%	0.41	0.33	C	SGS PFS test work
Selective Cu/Mo Flotation					
Throughput	t/h	93	110	D	
Moly Rougher Recovery	%	90.6	90.6	C	SGS PFS test work
Moly Plant Cleaner Recovery	%	98	98	B	Benchmark
Overall Mo Recovery	%	74.9	79	D	Mine Plan
Tailings classification					
Tailings Classification Equipment	-		Hydrocyclone	I	WEIR
Classification Cut Size	micron		37	K, C	WEIR Simulation
Hydrocyclone Feed Rate	t/h	5,740		D	
Hydrocyclone Feed Solids	%	30.2		C	
Hydrocyclone U/F Solids	%	65.2		K	WEIR Simulation
Hydrocyclone O/F Solids	%	13.6		K, D	WEIR Simulation
Tailings Thickening					
Fines Thickener Feed	t/h	2,084	2,501	D, A, I	
Tailings Thickener Solids %	%	62	54.8	C	Takraf Test Work
Vacuum Filters					
Vacuum Filtration Feed Rate	t/h	3,983	4,381	D	
Vacuum Filtration Rate	t/h/m²		2.8	C	Takraf Test Work
Cake Moisture (Vacuum)	%	14.7	16.5	C	
Pressure Filters					
Pressure Filters Availability	%		80 – 85	I	Takraf
Pressure Filtration Feed Rate	t/h	2,084	2,501	D	
Pressure Filtration Rate	kg/h/m²		793	C	Takraf Test Work
Total Cycle Time	min		20.25	I	Takraf Sizing Criteria
Total Cycle Filtration Rate	kg/h/m²		117	C, I	Takraf Pilot Test Work
Cake Moisture (Pressure)	%	15.5	16.3	C	SGS Test Work
Concentrate Dewatering					
Concentrate Thickener Solids %	%	64	64	B	
Cu Concentrate Moisture	%	9	10	B	
Mo Concentrate Moisture	%	3	3	B	

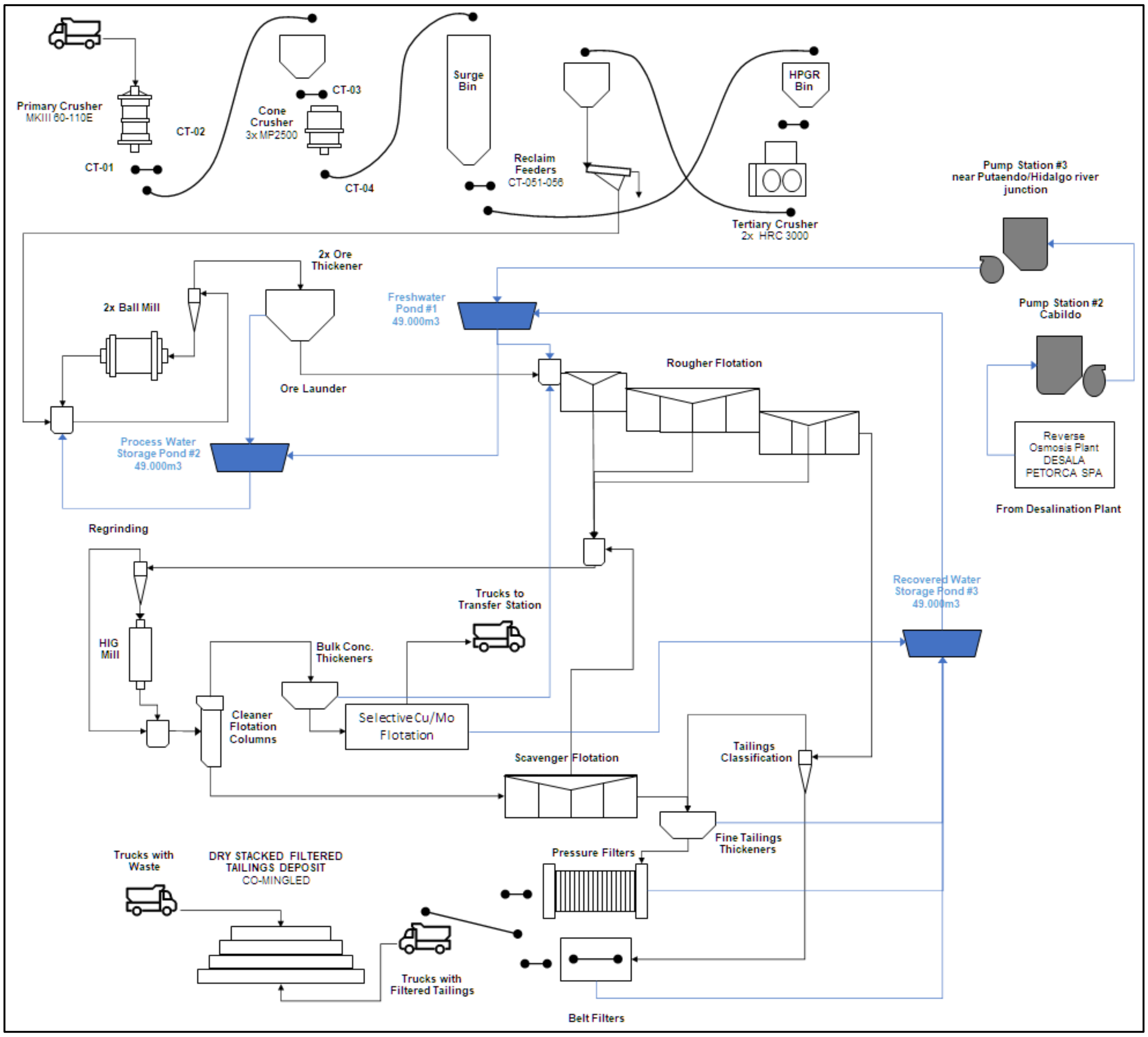
17.3 Process Flowsheet

The Vizcachitas Project design includes the infrastructure, equipment and systems required to support the process facilities. The main process unit operations are:

- Primary and secondary crushing
- HPGR crushing
- Ball milling
- Feed thickening and transportation
- Bulk Cu–Mo flotation
- Selective Cu–Mo flotation
- Tailings dewatering plant
- Concentrate dewatering, storage and loading facilities.

The process flowsheet is shown in Figure 17.1.

Figure 17.1: Vizcachitas Process Flowsheet



Source: Los Andes Copper, 2022.

17.4 Process Description

The process facilities are located in two areas within the Rocín River valley: the crushing and milling circuits, on a platform of compacted waste obtained from the pre-stripping operation; and the concentrator and filtration plant area, downstream in the valley on a natural plateau next to the access road.

17.4.1 Crushing

The run of mine (ROM) feed with an approximate size of 600 mm is fed to a primary crusher (Metso MK60-110E). To ensure operational continuity of the coarse crushing process, there is a ROM stockpile near the primary crusher building that can be used to feed the crusher if required.

The primary crusher reduces the feed to a P80 of 7 inches, this is fed to three secondary standard cone crushers (MP2500) arranged in open circuit to obtain a product 100% -2 inches. In normal operation two of these crushers are operating in parallel. The third crusher can operate when required to guarantee the product size for the downstream process.

17.4.2 Fine Crushing

The secondary crushing product is conveyed and stored in a 56,000 t surge bin. This bin feeds the tertiary crushing circuit that includes two HRC3000 HPGR crushers arranged in a direct closed circuit with six MF2.4 m x 7.3 m double-deck banana screens.

17.4.3 Grinding and Slurry Handling

The undersize from the crushing plant (P80 = 3.5 mm) is fed to two ball mills (28' x 48', 23 MW each) arranged in a reverse closed circuit with the hydrocyclones (800CVX) to achieve a P80 of 240 µm. The overflow from the hydrocyclones is fed to two 70 m diameter thickeners to obtain a slurry of 50% solids (%wt) which then flows through a smooth concrete launder (1.5 m x 1.0 m, 1.0% slope) to the concentrator on the plateau 4.1 km away in the Rocín River valley.

17.4.4 Flotation

The launder discharges the slurry into a concrete box that feeds a conditioning tank (5 min residence time) where flotation reagents are added. The reagent formula is a mixture of sodium isopropyl xanthate (XIPS), isopropyl ethyl thiocarbamate (IPETC), methyl isobutyl carbinol (MIBC), polypropylene glycol (DF400) and diesel.

The flotation circuit has two stages: a Cu-Mo bulk flotation and a Cu-Mo selective flotation. Both circuits use mechanical agitated tanks and column cells to produce concentrates.

The Cu-Mo bulk flotation circuit includes a rougher flotation stage of 24 TankCell e500 cells arranged in two parallel lines that operates at a pH near 7 to avoid the decomposition of xanthates and promote molybdenum recovery. The rougher concentrate is reduced to P80 = 25 µm in three HIG mills (HIG 5000) and then is fed to three 5 m diameter x 13 m cleaner columns operating at a pH of 12.

The cleaner concentrate is fed to the selective Cu-Mo flotation through two 40 m thickeners that adjust the % solids up to 40%wt. The cleaner tailings are fed to 11 TankCell e200 cells that perform the scavenging stage. The scavenger concentrate is recirculated to the HIG mills and the scavenger tailings are sent to the final tailings.

The selective Cu-Mo flotation stage includes rougher flotation and several cleaner stages in counter-current arrangement. Sulphuric acid and sodium hydrosulphide (NaHS) are added to maintain the pH around 9 and the pulp potential near -450 mV.

The copper concentrate is filtered and then loaded and stored in Rotainers, which are hauled by trucks to the port of Ventanas. The molybdenum concentrate is filtered and then stored in bags, which are hauled by truck to the port of Ventanas or other destinations agreed with buyers.

17.4.5 Tailings Dewatering System

The rougher tailings generated in the bulk flotation process are classified in a hydrocyclone plant (20 700CVX hydrocyclones). The coarse fraction (+400 mesh) is fed to 11 belt filters (162 m^2). The fine fraction is fed to two 70 m diameter thickeners and then filtered in 13 pressure filters (FP2500).

The filter cakes contain an average moisture of 15% and are co-mingled with the mine waste rock in the valley to shape the dry stacked filtered tailings deposit.

17.4.6 Ancillary Infrastructure

Ancillary infrastructure includes:

- Crushing circuit dust control
- Reagent preparation plant for SIPX, MIBC, IPETC and DF400 and lime preparation plant
- Air and nitrogen compressors
- Molybdenum plant reagent storage system for sulphuric acid and NaHS.

17.4.7 Water Supply and Management

The concentrator water supply will be supplied by a third party (Desala Petorca SpA) from a desalination plant located in Papudo to a pump station located near Cabildo. The water is pumped to Vizcachitas in a 26 inch diameter, 90.6 km long pipeline by two pump stations, one located near the town of Cabildo and the other near the Project at the confluence of the Rocín and Hidalgo Rivers. The desalination process is designed to obtain water quality that is suitable for the flotation process (Table 17.5).

Table 17.5: Desalinated Water Quality

Item	Value
pH	7.0 – 7.5
Total Dissolved Solids	< 1500 mg/L
Total Organic Carbon	0 mg/L
Ca^{+2}	< 400 mg/L
Mg^{+2}	< 125 mg/L
SO_4^{-2}	< 500 mg/L

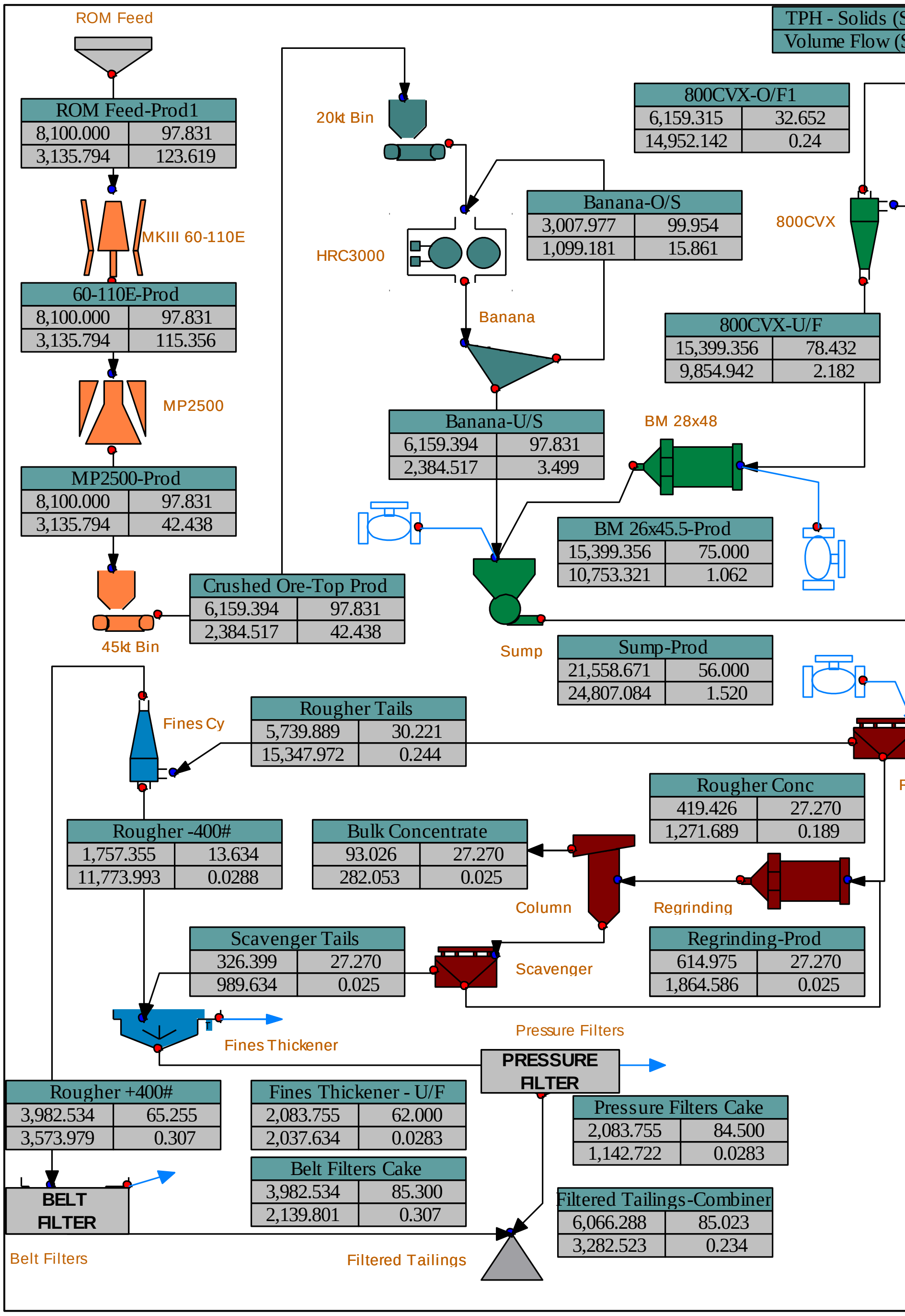
There are three 49,000 m^3 water storage ponds:

- The desalinated water is stored in fresh water storage pond #1 that feeds the reagent plant, the fire extinguishing system, the gland seal water system and process water storage pond #2.
- Process water storage pond #2 also receives recovered water from the feed thickeners; it feeds water to the crushing and grinding circuit.
- The recovered water storage pond #3 receives water from the tailings water recovery systems (tailings thickeners, pressure and belt filters); it feeds water to the flotation process.

17.5 Mass Balances

Mass balances and simulations were calculated in JKSimMet 6.3 using nominal design values. Figure 17.2 shows the summary and mass balances by main process areas.

Figure 17.2: Mass Balance in JKSimMet



MKIII 60-110E	
Feed Rate, t/h	8100.000
Power, kW	286.224
CSS	177.000
MP2500	
Feed Rate, t/h	4050.000
Power, kW	368.115
CSS	42.000
HRC3000	
Capacity, t/h	4583.888
Power, kW	6738.521
Gap, mm	90.000
BM 28x48	
Power,kW	20519.032
Load	0.293
Critical Speed	0.78
Fines Thickener	
%Solids U/F	62.000
Pressure Filters	
%Solids U/F	84.500
Belt Filters	
%Solids U/F	85.300

Source: Los Andes Copper, 2022

17.6 Water Balances

Water balance inputs were obtained from the process design criteria (Table 17.6 and Table 17.7). Results are presented in Table 17.8 and Table 17.9. The water balance flowsheet is shown in Figure 17.3.

Table 17.6: Inputs for Water Balance

Inputs	Value
Solids TPH	6,159
Evaporation Rate, mm/d	5
Ore Moisture	3.00%
Belt Filter Cake Moisture	14.70%
Pressure Filter Cake Moisture	15.50%
Cu Concentrate Moisture	10.00%
Mo Concentrate Moisture	3.00%

Table 17.7: Additional Water Consumption

Water Consumption	m^3/h
Hose Stations	10
Human Consumption	4.2
Reagent Preparation	6
Dust Suppresion	30
Mine Consumption	30
Slurry Maker (Dust Control)	493

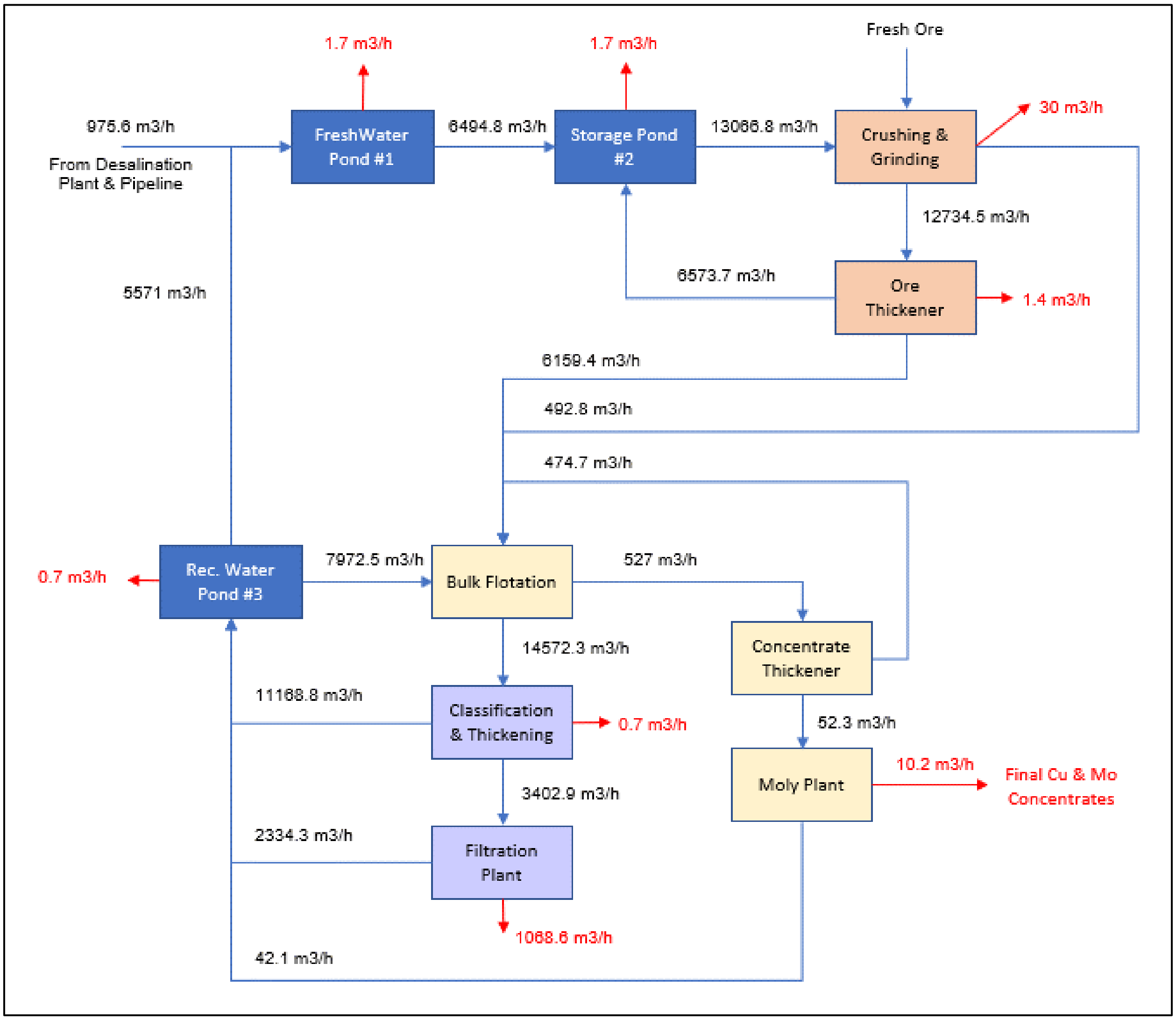
Table 17.8: Water Balance Results

Outputs	Value
Make-Up, m^3/h	975.6
Make-Up, L/s	271
Make-Up, m^3/t	0.16
Inputs, m^3/h	1,166
Losses, m^3/h	1,166
Water in Tailings	14,572
Water Recovered	13,503
Water Recovery	93%

Table 17.9: Water Balance

Stream	Solids, t/h	Water, m³/h	Water, L/s	% Solids
Freshwater from Desalination Plant		976	271	
Water from Rec. Water Pond #3		5,571	1,548	
Storage Pond #1 Evaporation Losses		2	0	
Area 3000 Water Consumption		10	3	
Water to Storage Pond #2		6,495	1,804	
Area 1000+2000 Water Consumption		40	11	
Storage Pond #2 Evaporation Losses		2	0	
Water to Crushing & Grinding		13,067	3,630	
Fresh Ore Feed	6,159	190	53	97.00%
Dust Suppression		30	8	
Slurry Maker		493	137	
Grinding Hydrocylone O/F	6,159	12,735	3,537	32.60%
Ore Thickener Evaporation Losses		1	0	
Ore Thickener Recovered Water		6,574	1,826	
Ore Thickener U/F	6,159	6,159	1,711	50.00%
Water to Bulk Flotation		7,972	2,215	
Bulk Flotation Feed	6,159	14,372	3,992	30.00%
Cu/Mo Bulk Concentrate	93	527	146	15.00%
Cu/Mo Concentrate Thickener U/F	93	52	15	64.00%
Cu/Mo Thickener Recovered Water		475	132	
Cu Concentrate	92	10	3	90.00%
Mo Concentrate	1	0	0	97.00%
Moly Plant Recovered Water		42	12	
Rougher Tailings	5,740	13,267	3,685	30.20%
Scavenger Tailings	326	1,306	363	20.00%
Rougher Tailings +400 mesh	3,983	2,126	590	65.20%
Rougher Tailings -400 mesh	1,757	11,141	3,095	13.60%
Fines (-400 mesh + Scavenger Tails)	2,084	12,447	3,457	16.70%
Fines Thickener Evaporation Losses		1	0	
Fines Thickener Recovered Water		11,169	3,102	
Fines Thickener U/F	2,084	1,277	355	62.00%
Pressure Filter Feed	2,084	1,277	355	62.00%
Pressure Filter Recovered Water		895	249	
Pressure Filter Cake	2,084	382	106	84.50%
Belt Filter Feed	3,983	2,126	590	65.20%
Belt Filter Recovered Water		1,439	400	
Belt Filter Cake	3,983	686	191	85.30%
Filtered Tailings	6,066	1,069	297	85.00%
Recovered Water Pond #3		13,545	3,763	
Rec. Water Pond #3 Evaporation Losses		2	0	

Figure 17.3: Water Balance (Flowsheet)



Source: Los Andes Copper, 2022

17.7 Main Equipment List and Equipment Sizing and Selection

Crushing and grinding equipment were selected and sized by Metso:Outotec based on the PFS testwork (Drop Weight Test, Bond Abrasion Index and Bond Work Index), PBT tests performed in York, PA and subsequent simulations carried out using Metso's proprietary BRUNO simulation software (Metso, 2020; Metso:Outotec 2022). The ball mill requirements were also calculated by Metso:Outotec using the Bond methodology (Rowland, 1982) adjusted using the phantom cyclone methodology.

The conveyor calculations were based on the CEMA methodology (ISO 5048) as detailed in Barfoot, Bennet and Col (Barfoot et al, 2022) (Table 17.10).

Screen calculations were based on the Cost Estimation Handbook published by the AusIMM.

Flotation equipment calculations were based on mass balances performed using JKSimMet and benchmark scale-up relationships (Table 17.11).

Classification and pumping equipment for the grinding and the water recovery stages were sized and selected by Weir Minerals (Weir Minerals, 2020).

The HIG mill sizes were estimated by Metso:Outotec (Metso:Outotec, 2022a) based on the vendor database using similar characteristics and throughput to Vizcachitas.

The tailings water recovery equipment requirements were calculated by Takraf based on laboratory and pilot filtration testwork (Takraf, 2021). Characteristics for sizing the concentrate thickeners and filters were based on vendor databases using similar feeds.

The main equipment is listed in Table 17.12, the installed power column shows the value for each piece of equipment.

Table 17.10: Conveyor Calculations

Belt	Stream	Qty	Belt Length, m	D100, mm	Nominal, t/h	Speed, m/s	Belt Width	Motor Power, kW
CT01	Apron Feeder	1	16	153	8,100	0.30	2,40m	1,215
CT02	Primary Crusher Product	1	344	153	8,100	3.00	84 in	2,517
CT03	MP2500 Feeders	3	15	153	2,700	0.30	1,83m	405
CT04	MP2500 Product	1	300	60	8,100	3.00	84 in	1,775
CT05	Reclaim Feeders	6	15	60	1,027	0.30	1.00 m	154
CT06	HRC Feed	1	300	60	9,165	3.00	84 in	2,004
CT07	HRC Product Feeders	2	15	25	4,583	0.30	2.10 m	687
CT08	Ball Mill Screen Feed	1	300	25	9,165	3.00	84 in	2,004

Table 17.11: Flotation Equipment Calculations

Stage	Units	Rougher	Cleaner	Scavenger
Throughput	t/h	6,159	615	522
Solids SG	-	2.74	3	3
% Solids	%wt	30	26.4	26.4
Pulp Vol. Flow (Nominal)	m^3/h	16,620	2,665	2,262
Pulp Vol. Flow (Design)	m^3/h	19,944	3,198	2,714
Flotation Lab. Time	Min	15	4	20
Scale-Up Factor	-	2	6	2
Residence Time	Min	30	24	40
# Lines		2	2	1
# Cells		24	3	11
Cell Volume	m^3/h	500	-	200

Table 17.12: Main Equipment List

Equipment	Manuf.	Model	Qty	Op	Installed Power Per eq., kW
Primary Crusher	Metso	MKIII 60-110E	1	1	1,193
Apron Feeder CT01	-	2400 mm	1	1	1,215
MK60-110 Prod. Conveyor CT02	-	84 in	1	1	2,517
Feeders CT031-032	-	1830 mm	3	2	405
Secondary Crushers	Metso	MP2500	3	2	1,864
MP2500 Product Conveyor CT04	-	84 in	1	1	1,775
Reclaim Feeders CT051-056	-	1000 mm	6	6	154
HRC Feed Conveyor CT06	-	84 in	1	1	2,004
Feeders CT071-072	-	2100 mm	2	2	687
HRC Product Conveyor CT08	-	84 in	1	1	2,004
Crushed Ore Bin 56kt	-	-	1	-	-
HRC 31kt Bin	-	-	1	-	-
HRC / HPGR	M:O	HRC3000	2	2	11,400
Banana Screen	M:O	MF 2473-2 Double Deck	6	6	55
Ball Mills	M:O	28.0' x 48.0' (overflow)	2	2	23,000
Grinding Hydrocyclones	Weir	800CVX	32	-	-
Mill Pumps	Weir	750 M240 MCR	4	2	2,296
Ore Thickeners	Takraf	ø=70m, altura=4.5m	2	2	20
Rougher Flotation Cells	M:O	TankCell e500	24	24	400
Scavenger Cells	M:O	TankCell e200	11	11	185
Flotation Columns	M:O	5 x 13m	3	3	-
HiG Mill	M:O	HIG5000/35000	3	3	5,000
Tailings Pumps	Weir	650 M200 MCR	2	1	1,594
Rougher Tailings Hydrocyclone	Weir	700CVX	20	-	-
Fine Tailings Thickener (-400#+Scav.)	Takraf	ø=70m, altura=3.5m	2	2	15
Tailings Horizontal Belt Filters	Takraf	162 m2 42B/10-40V	11	9	558
Tailings Pressure Filters	Takraf	FP 2500/118/50/6/M15/O OH	13	11	352
Copper Concentrate Filters	Takraf	FP 1500/96/40/6/M15/O OH	3	2	167
Copper Concentrate Thickeners	Takraf	ø=40m	2	1	15

17.8 Energy and Water Requirements

17.8.1 Energy

The total installed power requirement is estimated to be 139 MW. The overall power consumption for the concentrator is estimated to be 16.6 kWh/t (including a 10% allowance on the flotation and water recovery stages) (Table 17.13).

Table 17.13: Power Consumption

Stage	Specific Energy Consumption, kWh/t	% Total
Crushing + HPGR	2.36	14%
Grinding	7.22	44%
Conveyors	1.18	7%
Flotation	3.43	21%
Water Recovery	1.40	8%
Surplus	0.97	6%
Total	16.6	100%

17.8.2 Water

According to the water balance (Table 17.8) the make-up water consumption is 271 L/s with a maximum of 300 L/s.

17.9 Consumables and Reagents

Consumables and reagents were calculated based on Bond Abrasion Index and flotation tests performed as part of the PFS testwork. The estimated consumptions are shown in Table 17.14 for wear media and Table 17.15 for reagents.

Table 17.14: Wear Media Consumptions

Wear media	Consumption, g/t
Crusher Liners	2
Steel Balls 2.5”	595
Ball Mill Lifters	47
Ceramic Media	10

Table 17.15: Reagent Consumption

Reagent	Type	Addition	Dosage
C3330	Collector (Sodium Isopropyl Xanthate)	Rougher Flotation	20 g/t
MC-C200	Collector (Thionocarbamate)	Rougher Flotation	20 g/t
DF400	Frother (Polypropylene Glycol)	Rougher Flotation	30 g/t
MIBC	Frother (Methyl Isobutyl Carbinol)	Rougher Flotation	5 g/t
Lime	Modifier	Rougher and Cleaner Flotation	259 g/t
Diesel	Modifier	Rougher Flotation	20 g/t
MF 150	Flocculant (polyacrylamide)	Thickeners	10 g/t

18. PROJECT INFRASTRUCTURE

18.1 Summary

The main access road will be 36 km long, from north of Putaendo to the concentrator and filtration plant, avoiding populated areas. The Rocín River will be diverted around the site by a 40 m high dam and a 5 m diameter, 15.965 km long tunnel that will be built in two stages: 10.4 km will serve Years 1 to 5 of the mine production and the additional 5.8 km will serve for the rest of the LOM. A temporary culvert located in the river bed would be built to enable construction of early works and the crushing and grinding circuit until the diversion tunnel is in operation.

The main buildings will be located on platforms of compacted material obtained from the mine pre-stripping. The main buildings have been located in zones that require minor earthworks. Other ancillary on-site infrastructure includes the electrical distribution systems, a truck shop, explosives magazine, general offices, dust suppression systems, a reagent and lime plant, internal roads, rainfall diversion channels and a contact water treatment plant.

In the third quarter of 2022 Los Andes Copper signed a Letter of Intent (LOI) with Desala Petorca SpA, a water supply company, to provide desalinated water to the Project from a desalination plant (to be built on the coast near Papudo) to a storage pond in Cabildo. Vizcachitas will build and operate a 90.6 km long pipeline from Cabildo to the Project site. Alternatively, water could be supplied from a 500 L/s desalination plant owned by Los Andes Copper located near Longotoma. From here the water will be pumped to the Project in a 26 inch diameter, 135 km long pipeline, by three pump stations; one to be located near the coast, one near the town of Cabildo and one near the Project at the Rocín and Hidalgo River confluence.

Power would be supplied via a 61 km long, 220 kV power line connected to the Los Maquis electrical substation near the city of Los Andes.

In addition to ensuring the technical and economic competitiveness of the Project, key drivers were aimed to reduce environmental impact in terms of water and energy consumption, minimize the footprint of the Project, and to secure competitive Capex and Opex. The main differences between the PEA Report (2019) and this TR in this area are:

- Use of desalinated water
- The thickened tailings dam has been replaced by a dry stacked filtered tailings deposit
- All on-site infrastructure is now located in the Rocín River valley, avoiding the use of neighbouring valleys
- The energy consumption has been reduced by using HPGR technology instead of SAG mills
- Overall earthworks have been reduced separating the comminution area from the concentrator and using pre-strip waste to build some platforms
- The power supply has been changed, reducing the length of the power line
- Use of Rotainers to transport copper concentrate.

18.2 Project Infrastructure Design Criteria

Table 18.1 summarizes the general conditions at the Project site.

Table 18.1: Site Conditions

Climate	Mid-mountain
Rainfall	Regular year 120-150 mm (mainly pluvial)
Temperature	-10° to + 35 °C
Site	Alluvial valleys, hill slopes with colluvial and steep deposits requiring extensive blasting
Soil Type	Mainly gravel, localized areas of intrusive rock in the valley
Elevation	Mine: 2,500 masl
	Plant: 1,935 masl
	Tailings Deposit: 1,935 masl
Iso-Keraunic Level	10 (high)

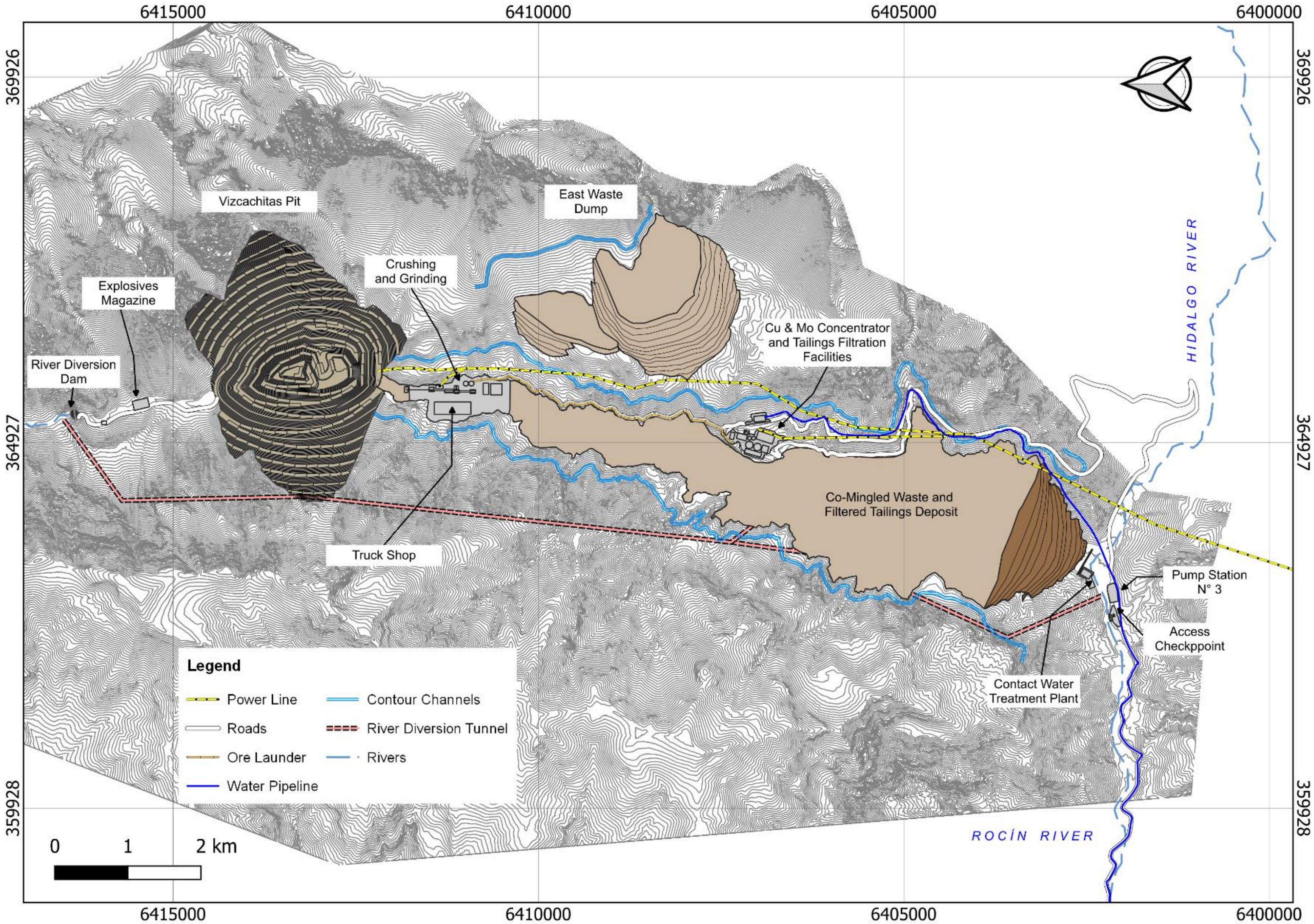
18.2.1 General Infrastructure Design Criteria

The general infrastructure design incorporates the following criteria:

- Minimize footprint and social and environmental impacts by installing the main infrastructure within the Rocín River valley
- Reduce earthworks by installing part of the infrastructure on natural plateaus and on platforms using compacted mine pre-stripping waste
- Avoid the construction of a tailings impoundment, maximizing water recovery and minimizing the footprint using filtered tailings technology
- Water recovered from the open pit dewatering systems and the contact water system will be processed in a water treatment plant and then discharged to the Rocín River
- Rain and spring water captured by the contour channels (non-contact water) will be returned untreated to the Rocín River
- Use overhead cranes for maintenance operations where possible; use mobile lifting equipment in areas with difficult access; and use aisles for good maintenance access to the equipment in the main infrastructure
- Locate main infrastructure at a minimum of 150 m from the final pit boundary
- Minimize roofing for infrastructure buildings
- Incorporate water storage tanks in the firefighting system at each major facility and incorporate dust management systems in the design.

No camp will be required as the site is at commuting distances to towns and cities. An emergency shelter will be provided on site for winter emergencies. The site layout is shown in Figure 18.1.

Figure 18.1: Overall Site Layout



Source: Los Andes Copper, 2022

18.3 Access Road

The main access road is 36 km long, from north of Putaendo to the concentrator, and will provide access to the mine site for personnel, supplies and concentrate hauling. The access road engineering study was prepared by MN Ingenieros. Table 18.2 summarizes the design.

The road design speed is 80 km/h; a 30 km/h speed limit is considered in areas with tight radius and limited visibility.

The geometric design of the road meets the Public Works Ministry of Chile (MOP) road standards. The minimum turn radius required for the design velocities are 25 m for 30 km/h and 250 m for 80 km/h. Slope design considers a maximum of 10% to 12% for 30 km/h and 8% for 80 km/h.

The road is divided in two main sections:

- The first section goes through mountainous terrain, starting at the Vizcachitas concentrator and is 18.5 km long
- The second section goes through the Putaendo River valley, starting at the confluence between the Chalaco stream and the Putaendo River and is 17 km long.

Table 18.2: Main Access Road Characteristics

Km Start	Km End	Road Characteristics
0+000	4+400	Starts at the concentrator with a compacted granular surface and continues along the left wall of the valley (moving in downstream direction). 12 culverts are projected.
4+400	4+500	Bridge crossing the Hidalgo river.
4+500	8+865	Continues in asphalt concrete surface along the left wall of the Putaendo valley. 8 culverts are projected.
8+865	8+900	Bridge crossing the Putaendo river.
8+900	11+135	Continues along the right wall of the Putaendo valley, using the existing path to the Chacrillas' reservoir. 3 culverts are projected. - 3 complementary civil works identified. - Asphalt concrete surface.
11+135	11+520	- Continues along the right wall of the Putaendo valley, using the existing path to the Chacrillas' reservoir. - 1 complementary civil work identified. - Asphalt concrete surface.
11+520	11+619	- Junction with the Chacrillas' bridge.
11+619	16+760	- Continues along the left wall of the Putaendo valley, bordering the Chacrillas reservoir on its southern side, using the existing access road. - 7 complementary civil works identified. - Asphalt concrete surface.
16+760	16+785	- Junction with the bridge downstream from the Chacrillas' dam. - 1 complementary civil work identified .
16+785	16+880	- Continues along the right side of the Putaendo valley, using the existing path to the reservoir. - 1 complementary civil works identified. - Asphalt concrete surface.
16+880	18+500	Continues along the right side of the Putaendo valley. 2 culverts are projected.
18+500	18+520	Bridge for crossing the Putaendo river from the right to the left side.
18+520	33+170	Continues along the left side of the river. 22 culverts are projected.
33+170	35+969	Continues along the left side of the river. 4 culverts are projected.

18.4 River Diversion

The Rocín River will be diverted to begin the construction of facilities in the valley. The river diversion works include: a coffer dam for the construction phase, a dam, an access road, a river intake and the diversion tunnel itself.

18.4.1 River Intake Works

The coffer dam will operate during the construction phase of the main dam and is located 55 m downstream from the river intake (Figure 18.2). The coffer dam will be built from compacted colluvial material it will have a 3:2 slope and will be 8 m wide and 7 m high.

The dam wall will be located 320 m downstream from the river intake and 1.3 km north of the north end of the pit. The dam will be 205 m long, 58 m high with a 5 m wide crest. The MN Ingenieros design includes an allowance for the maximum volume of sediment that could be collected during the LOM (2 Mm^3).

The river intake will be located at UTM coordinates 365,296E and 6,416,537N (WGS84 19S). The river flow is collected by an 8 m wide, 7 m high and 76 m long open concrete channel that discharges into the tunnel. A transition section narrows the end of the open channel from 8 m to 6 m wide.

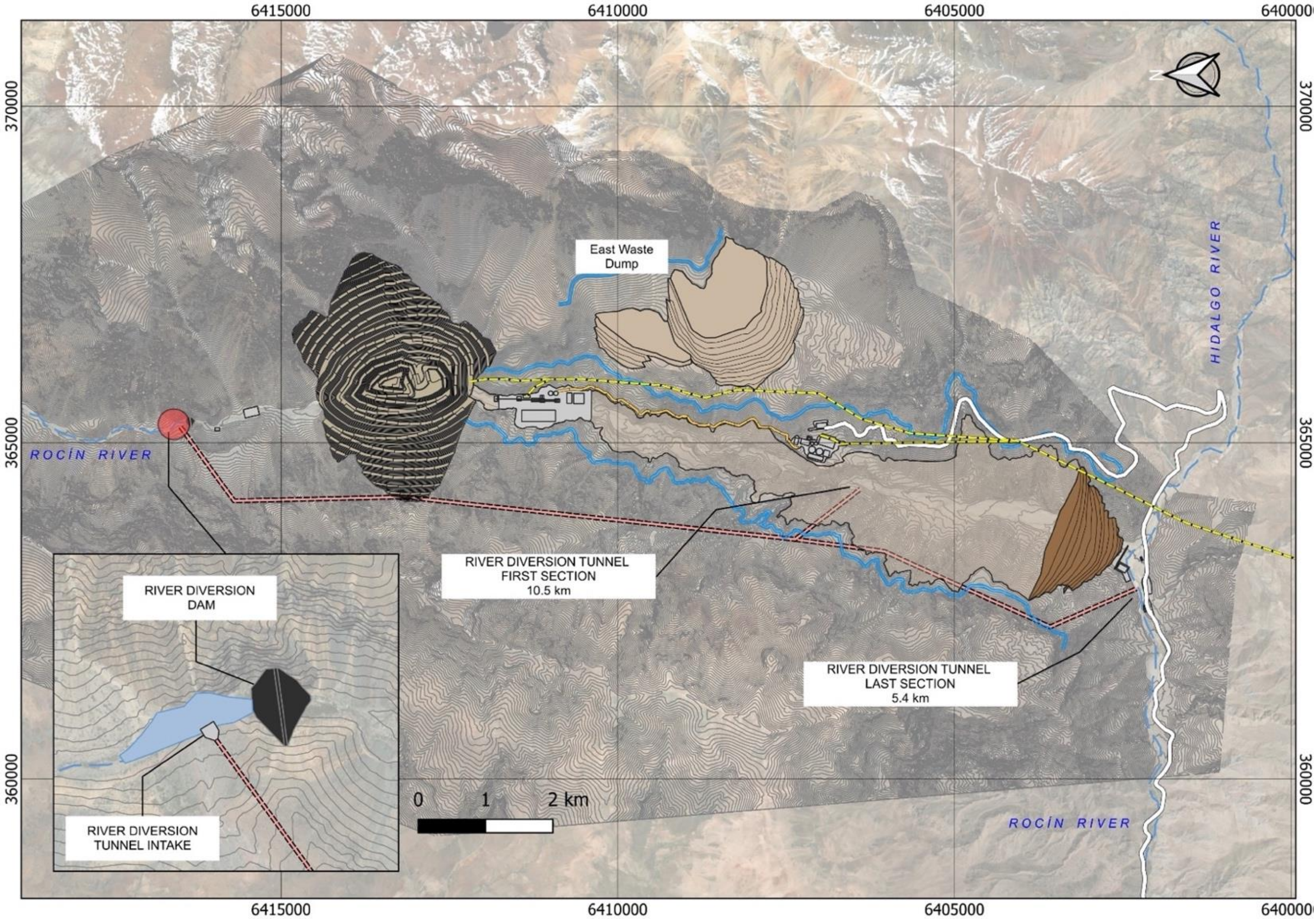
18.4.2 Diversion Tunnel

The diversion tunnel will be 5 m diameter and 15.965 km long, it will be built in two stages using a single tunnel boring machine (TBM). The tunnel will run under the western wall of the Rocín River valley from the intake point upstream to the discharge point downstream near the confluence between the Rocín and Hidalgo Rivers (Figure 18.2). The tunnel and other river diversion works have been designed for a 1,000 year return period or for a maximum flow rate of 200 m^3/s.

The first section of the Rocín tunnel will be 10,518 m long, including an access adit 1,221 m long (needed to start early works) to the west of the concentrator area. The remaining portion of the tunnel is 5,447 m long and will be executed immediately after completing the first section.

For both stages the excavation direction will be from downstream to upstream with a positive slope of 4.1%. It was estimated that 15% of the tunnel will need to be reinforced to control structural discontinuities, faults or weak rock with hydrothermal alteration. The structural support for these sections will use wire mesh, a 10 cm shotcrete layer on the roof and walls and rock bolts (25 mm diameter, 3.0 m long). For the remaining 85% of the tunnel no rock support was considered.

Figure 18.2: River Diversion Works



Source: Los Andes Copper, 2022

18.5 On-Site Infrastructure

18.5.1 Mining Facilities

A truck shop will be located on a single large platform built from compacted waste from pre-stripping. The truck shop will perform maintenance works for Komatsu 930E type trucks, electric shovels, drill rigs and mine service equipment.

A 7 m wide central corridor will provide access to the small bays (minor equipment), the spare parts storage and waste area, allowing traffic in both directions. The layout includes two 28 m x 27 m extended bays and two 20 m x 21 m conventional bays, equipped with 35 t capacity overhead cranes with a 5 t capacity auxiliary trolley.

The building will be equipped with plumbing, electrical, lighting, data/communications, UPS, fire and HVAC systems.

Other mine service facilities include:

- Warehouse for tools, spare parts, and components
- Storage and lubrication service areas
- Wash bay for mobile equipment
- Storage tank for liquid waste
- An area for lubricant reception, storage, and distribution
- Oil storage tanks for engines, transmission, and hydraulic systems
- Welding and repair shop
- Tyre replacement area
- Temporary industrial and hazardous waste storage area
- Fire detection and extinguishing systems
- Water tanks.

The mine fuel will be stored in a 75 m^3 horizontal cylindrical carbon steel tank. The storage facility will be in an area with a spillage containment capacity of 110% of the tank volume. The fuel storage area will be enclosed by a 1.8 m high perimeter fence and will be equipped with fire extinguishers. A third party fuel supplier will manage this facility.

The explosives magazine will be located on a 1.5 ha platform located between the northern end of the pit and the river diversion dam. This area will be managed by a third party explosive provider. Oils and lubricants obtained from larger equipment maintenance will be stored in this area for use in the production of explosives.

18.5.2 Plant Infrastructure

Crushing and Grinding Area

The crushing plant will be located on a platform built with compacted waste rock from pre-stripping. A dust suppression system is included for the feed hoppers and transfer points at each conveyor discharge. The collected dust will be mixed with water to produce a slurry that will be sent to the ball mill discharge pump boxes.

Spillage produced in the milling stage will be collected in a channel, decanted in spillage ponds and then pumped to the mill discharge pump boxes. Coarse spillage will be collected by small mobile equipment (bobcats) and fed back into the process.

The ground product (flotation feed) will be thickened to 50% solids by weight and fed into a smooth concrete feed launder (1.5 m wide, 1.0 m high, 0.3 m thick and 1% slope). This 3,250 m long launder will discharge into a spillway to dissipate the energy and then the slurry will enter a sump that feeds the rougher flotation cells located at an elevation of 1,870 m.

An internal paved road is located next to the feed launder and connects the flotation plant with the mine and crushing/grinding areas.

Flotation Area

The flotation area is downstream from the crushing plant on a natural plateau on the east side of the valley. The following infrastructure is included in this area:

- A reagent plant with xanthate preparation tank, warehouses and four storage tanks for other reagents, frothers and flocculants
- Reagent distribution systems
- Reagent spill collection well which will be cleaned regularly and spillage disposed of in the waste management centre
- A lime storage bin and lime preparation plant
- Mobile crane for flotation equipment and HIG mills
- Blowers
- Slurry spillage pond
- On-line measurement systems and analyzers
- Samplers.

The molybdenum plant includes the following infrastructure:

- Conditioning tanks
- Building for filtration and drying equipment
- Bin loading system
- Reagent plant for NaHS and sulphuric acid
- Nitrogen compressor room

- Electrical room
- Molybdenum process water treatment and recovery system
- On-line measurement systems and analyzers
- Samplers.

18.5.3 Buildings

The administration buildings will provide offices for the administration staff, managers and professional/technical personnel for the plant and mine areas. The administration buildings are smaller than the industrial buildings and can be built in modules. They will be built using prefabricated systems or transportable modules with a light steel structure and wooden panels that can be easily assembled in the field. The modules shipped to site will include electrical and sanitary services.

The service buildings will provide support for the production process. These buildings will be prefabricated modular buildings or buildings with a steel frame, metal cladding and foundations, depending on the size/use.

The Project includes the following administration and services buildings: offices, control room, polyclinic, fire/rescue headquarters, and bus and small vehicle parking areas.

18.5.4 Internal Roads

A network of internal roads will connect the mining, crushing-grinding, flotation, filtering, molybdenum plant and pump stations areas. The design meets the structural civil design criteria and provides access for maintenance and truck traffic for concentrate transport. Table 18.3 lists the internal roads.

Table 18.3: Internal Roads

Type of Road	Width, m	Design Speed, km/h
Backbone road	7	100
Primary plant roads	7	60
Secondary plant roads	6	40
Truck maintenence and inspection	4	30
Access roads	7	80

18.5.5 Fire Protection Systems

The fire protection system will use desalinated water and consists of a gravitational system with water stored in a 1,400 m^3 main tank, and a pressurized system with an electric pump complete with diesel generator back-up for the most critical areas of the plant.

For the mine truck shop area and associated infrastructure on the WBS 1300 area platform, the system will be extended from the plant area and an accumulator tank will be provided for emergencies.

18.5.6 Platforms

Significant earthworks will be required to build platforms for the following infrastructure:

- Mine services infrastructure
- Explosive magazine
- Primary, secondary and tertiary crushing
- Head tanks and fire protection systems
- Ball mills and thickeners
- Fresh and recovered water reservoirs
- Feed launder
- Cu-Mo bulk flotation
- Concentrate thickeners
- Reagent plant
- Tailings thickeners
- Tailings filter building
- Concentrate storage building and loading yard
- Molybdenum building
- Main substation
- Electrical rooms
- Contact water treatment plant
- Residual water treatment plant
- Auxiliary facilities such as compressor yard
- Manoeuvring area for concentrate trucks
- Storage/salvage yards and waste management platforms
- Container storage platform
- Pump station 3 (EB 3) and emergency pond.

The total volume of earthworks required for the main infrastructure areas is shown in Table 18.4

Table 18.4: Summary of Earthworks Quantities

Summary of Quantities	Unit	Mine	Crushing	Flotation	Buildings
Excavation	Million m^3	2.69	0.160	0.56	6.18
Filling	Thousand m^3	51.70	2.142	662.18	6.80
Mine Filling	Million m^3	1.95	13.400	0.00	2.57

18.5.7 Communications System

On-site and off-site communications will use a network of fibre optic links that will connect the electrical rooms, workshops, warehouses, offices and other areas to an integrated operations centre.

18.5.8 Temporary Construction Camp

The camp will be used during the early works and construction phase of the Project. A total requirement of 5,000 beds for the tunneling and plant construction and 500 beds for the water pipeline works have been estimated during the construction stage. The camp includes:

- Bedrooms
- Dining room
- Administration offices
- Warehouse area
- Waste collection area
- Perimeter fence
- Access control
- Parking lot
- Polyclinic
- Water tanks.

The construction camp will be rented from third parties and must comply with the Project standards. This camp will be dismantled and removed after finishing the construction phase, it will not be re-used to house operations personnel.

18.6 Security Gatehouse and Checkpoint

A security gatehouse and checkpoint for controlling the access and egress of personnel and materials will be located adjacent to the main access road near pump station EB3. This area includes a reception office, an area with two X-ray scanners, a small auditorium for conducting HSEC meetings, toilets and parking lots.

18.7 Water Management

The Project water management infrastructure is designed to fulfill the following criteria:

- Water captured above 1,935 masl and whose course is not affected by the Project will be collected through contour channels which will carry the non-contact water downstream to be discharged at the Rocín and Hidalgo River confluence.
- Rain water (contact water) that falls on the Project area below the contour channel (roads, buildings, platforms and concentrator areas) will be channeled through canals located in the contours and low areas and then fed to a central collector. The final destination is the

contact water treatment plant which is located downstream near the Rocín and Hidalgo River confluence.
- Water captured from the mine dewatering system will be channeled through the same central collector to the contact water treatment plant.

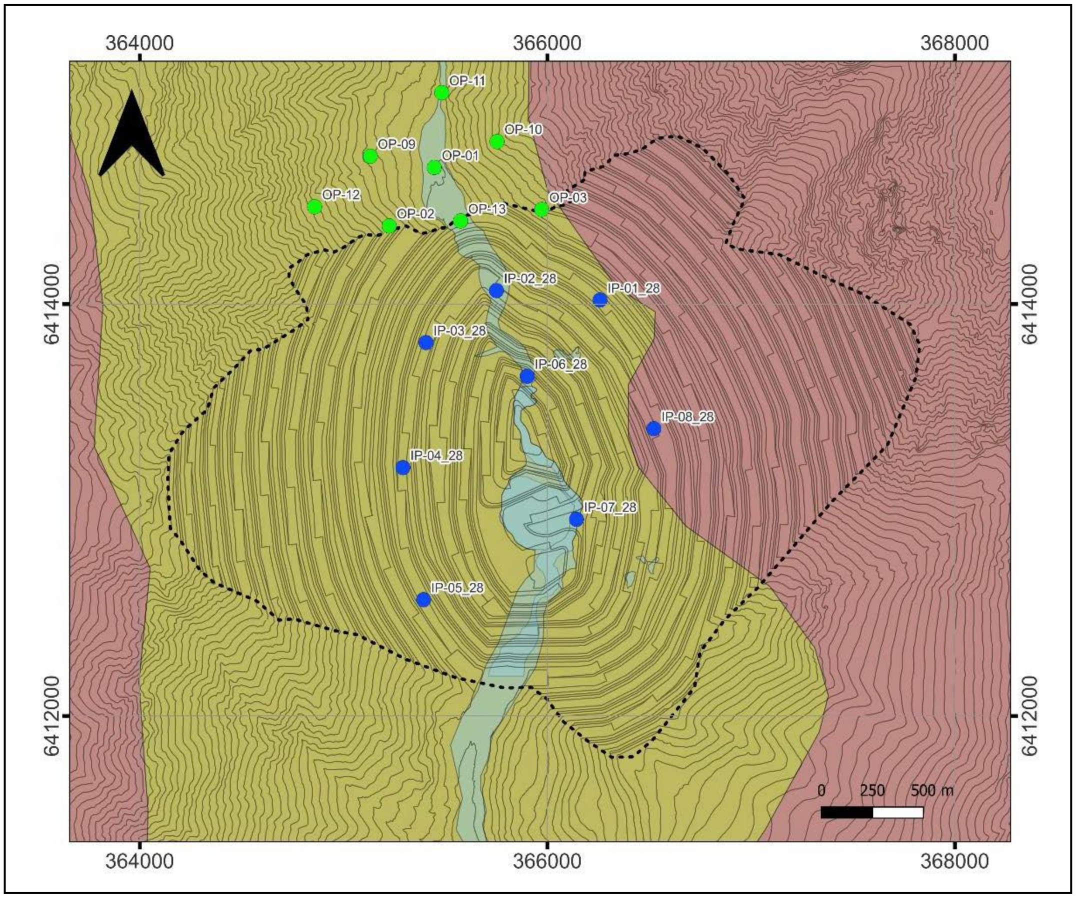
18.7.1 Mine Dewatering System

Water inflow into the pit during operations has been estimated based on analytical equations. The total inflow to the pit has been estimated to be 24 L/s, of which 14 L/s corresponds to an inflow of ground water from the north and 10 L/s corresponds to water drained from storage in the hydrogeological units (HU) that will have to be dewatered.

The Vizcachitas pit will be dewatered by the construction of a hydraulic barrier made up of eight vertical wells located upstream in the northern edge of the pit to capture a ground water flow of 14 L/s from the Rocín River (indicated in green in Figure 18.3). This hydraulic barrier should be in operation at least 1 year before the start of the excavations and should be kept operating throughout the mining process to prevent the entry of this water into the pit.

To control the water stored within the rock mass (10 L/s), eight additional vertical wells are proposed (indicated in blue in Figure 18.3); these will be built according to the pit phases.

Figure 18.3: Mine Dewatering Wells (Final Pit)



HU-1	Alluvial and Colluvial deposits		Out- pit wells
HU-2	Fractured Rock		In-Pit wells
HU-3	Bedrock	·- - - -	Final Open-Pit Boundary

Source: ITASCA, 2023.

18.7.2 Rain Water Management

The rain water management system is designed to collect all the rainfall in the platforms, roofs, roads and confined areas within the crushing, conveying and concentrator areas, and run-off from surrounding areas that enters the concentrator site. Water will be collected and diverted to spillage collection pits and sumps for pumping to the central collector. To achieve the collection and removal of rain water the following infrastructure is proposed:

- Contour channels on platforms or in confined areas discharging at the low point of each area. At the low points the flow will be collected and sent to the main collector. The contour channels will be trapezoidal in shape, with a 500 mm wide base and a variable longitudinal slope depending on the area to be drained.

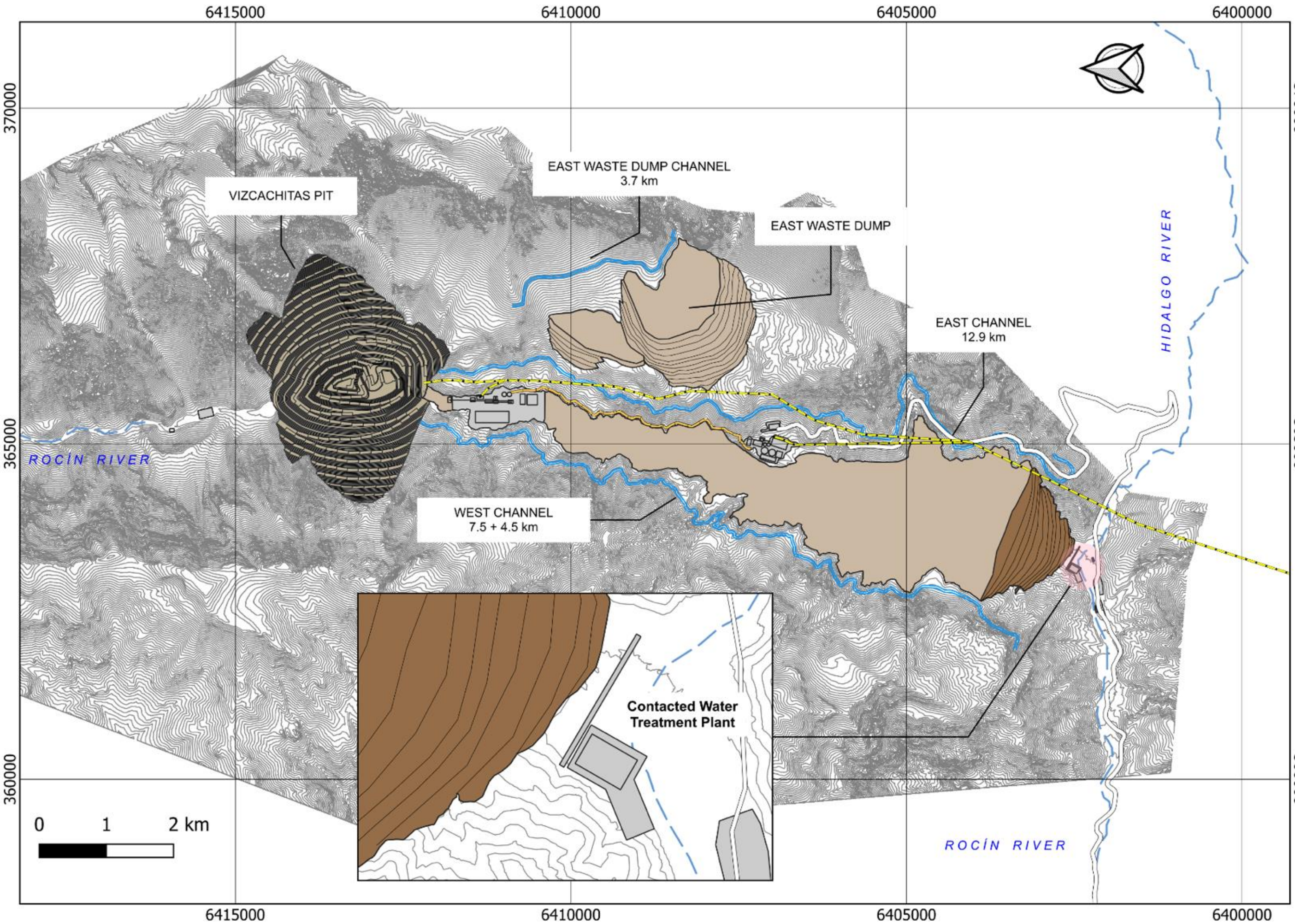
- A central main collector running alongside the main road to the concentrator. The central collector will run from the coarse ore storage area to the process water ponds and will consist of a 900 mm diameter compressed cement pipe with a longitudinal slope varying around 5%. There will be sumps along the central collector to allow for changes in direction and grade to achieve an average slope of 6% to 8%. The platforms in the area converge to this central collector and will discharge rain water into a natural channel near the filtered tailings storage area.
- North and south collector channels will collect run-off from areas around the plant area that slope towards the operation or transportation platforms. This water will be conducted to the process water pond area and then flow into a natural channel that discharges into the filtered tailings storage area. The north and south collector channels will be dug in natural terrain 500 mm wide at the base with variable height and a longitudinal slope of around 1%.

The exceptions to the above are the following:

- Process areas that have dedicated spill storage and evacuation systems (such as thickening and grinding). In these areas rain water that exceeds the capacity of the sump recirculation pump system for the process will be channeled to the emergency ponds that capture large spills from the process areas.
- Process areas that have a containment berm such as concentrate and reagent tanks. In these areas spills and rain water will be evacuated using a floor pump and hauled by truck to final disposal or returned to the process, as appropriate.
- Specific areas that will be confined by changes in the slab slope will be provided with floor sumps and pumps. These areas are minimized so that rainfall will not compromise the operation, people or the environment.

Figure 18.4 shows the main rain water management requirements for the Project.

Figure 18.4: Contour Channels and Contacted Water Treatment Plant



Source: Los Andes Copper, 2022

18.7.2.1 Contour Channels

Four contour channels are planned:

- West mine waste dump contour channel with a final length of 3.7 km, discharging into the west contour channel
- East contour channel 12.9 km long discharging into the Hidalgo River
- West contour channel:
  - First section: Years 1 to 6, 7.5 km long discharging into the Rocín River
  - Second section: Years 7 to 26, 4.5 km long discharging into the Rocín and Hidalgo River confluence.

18.7.2.2 Contact Water Treatment Plant

This plant will be at the confluence of the Rocín and Hidalgo Rivers, near the desalinated water pump station EB3, and would treat the contacted waters collected from the following sources:

- Water collected in the wells of the mine dewatering operation
- Contact water from the crushing and grinding area
- Contact water from the flotation, thickening and filtration areas
- Contact water from the co-mingled filtered tailings and waste rock deposit
- Water from other surface areas below the elevation of the contour channels.

For the first 6 years a collection system will be located in the valley close to the concentrator and the retaining wall of the filtered tailings deposit. This will consist of hydraulic barriers, a containment pond and a pipeline to the water treatment plant located at the river confluence.

Based on the hydrological studies carried out in the basin of the Project site (17.3 km^2), the contacted water treatment plant will require a capacity of 1.6 m^3/s.

The contact water treatment plant will be made up of three reactors that will neutralize the contact water using lime. The neutralized water will flow by gravity to a clarifier (54 m diameter). The sludge recovered from the clarifier will be recirculated to the first reactor and the clarified water will be sent to an agitated tank where the pH would be adjusted with CO_2; the water will then be fed into the Rocín River.

18.7.3 Ground Water Seepage Control System

As indicated in the hydrogeological conceptual model prepared by ITASCA (ITASCA, 2022; ITASCA, 2023), there are several underground water sources along the Rocín River basin between the pit and the confluence with the Hidalgo River. The underground water discharge into the Rocín River basin downstream of the Vizcachitas pit is estimated to be a maximum of 80 L/s. Even though this groundwater cannot be diverted to the contour channel system and flows below the river alluvium, a seepage control wall located upstream of the confluence of the rivers will provide the required environmental groundwater control to ensure that the water returned to the river beyond the limits of the Project complies with Chilean regulations.

The Project considers an underground water quality management strategy based on the following principles:

1. Construction of a seepage control wall anchored to the bedrock.
2. Installation a battery of well pumps upstream and along the wall. Each pump is aimed to deplete all the water that could reach the wall.
3. Pumping 100% of the underground water collected by the pumps and deliver the flow to the water treatment plant to ensure that the returned water quality meets Chilean regulations.
4. Construction of monitoring wells downstream of the seepage control wall.
5. Peristaltic pumps will collect samples at the screen level on a continuous basis.

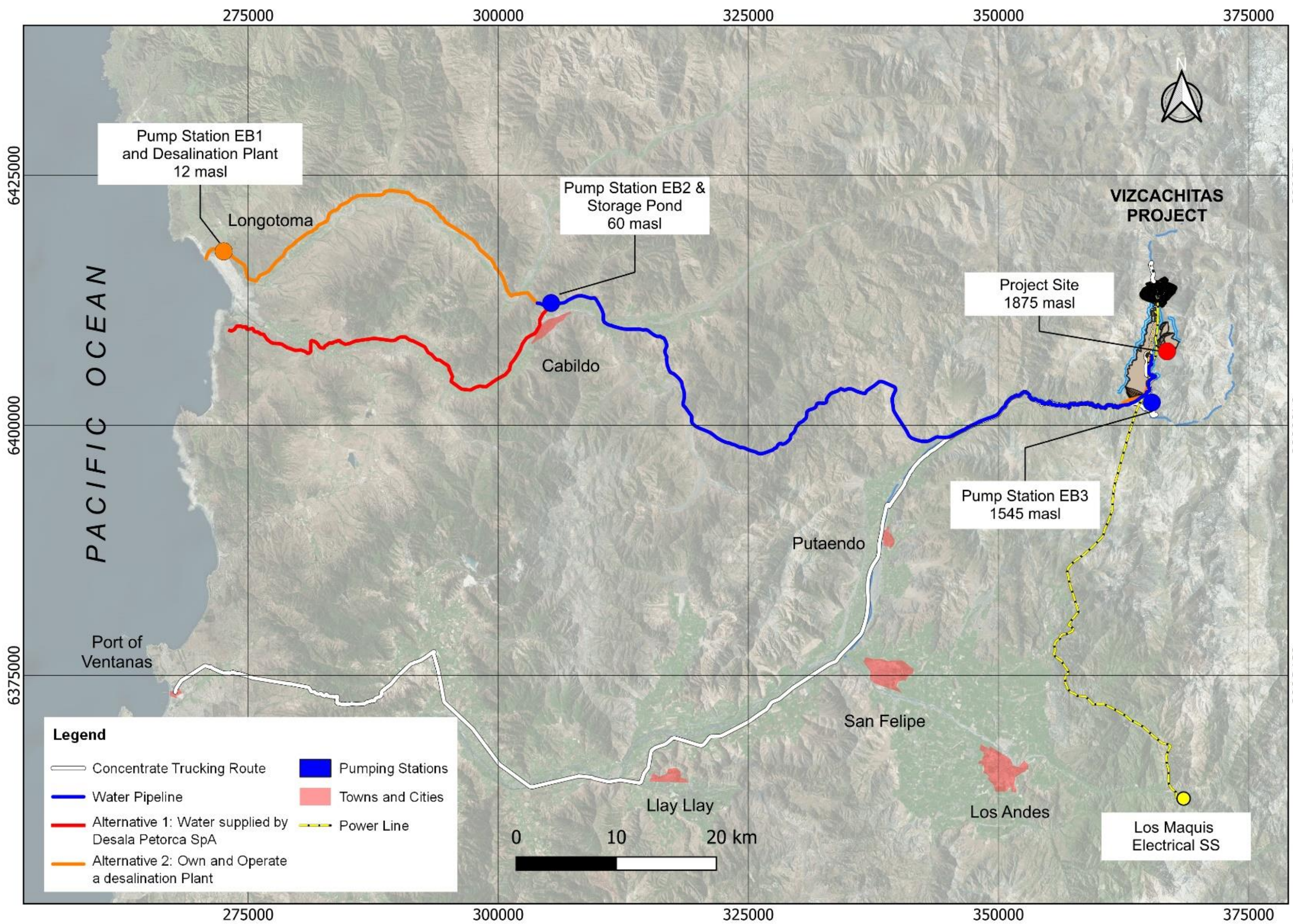
18.8 Water Supply

In the third quarter of 2022 Los Andes Copper signed a Letter of Intent (LOI) with Desala Petorca SpA, a water supply company, to provide desalinated water to the Project from a desalination plant to be built on the coast near Papudo to a storage pond in Cabildo. Vizcachitas will build and operate a pipeline with two pump stations (EB2+EB3) from Cabildo to the Project site (Figure 18.5).

An alternative arrangement would be to supply water from a 500 L/s desalination plant owned by Los Andes Copper located near Longotoma; the water will be pumped to the site by a 26 inch diameter, 135 km long pipeline, using three pump stations (EB1, EB2 and EB3) located near the coast, near the town of Cabildo and near the Project at the Rocín and Hidalgo Rivers confluence.

The water supply system will be located in the Valparaíso Region with facilities in the Petorca and Los Andes Provinces. Figure 18.5 shows the general layout for both alternatives including pump stations EB1, EB2 and EB3.

Figure 18.5: Water and Electrical Supply System



Source: Los Andes Copper, 2023

18.8.1 Desalinated Water Supply System

The desalinated water supply system is designed for 500 L/s with a 26 inch diameter steel pipeline via 2 (or 3) pump stations. The layout and location of the pump stations are shown on Figure 18.5. These locations were defined based on land ownership and proximity to existing electrical substations. The infrastructure engineering studies were developed by CDM Smidth.

Table 18.5 and Table 18.6 show the design specifications for the two pipeline alternatives. The pipeline design criteria are for a pressurized, buried steel pipeline without liners with a booster type drive system without load breakage.

Table 18.5: Pipeline Sections

Section	Alternative	Length (km)	Suction Class Type	Discharge Class Type
EB1 - EB2	1	45	ASME 300	ASME 600
EB2 - EB3	1 + 2	82	ASME 300	ASME 900
EB3 - Terminal Station	1 + 2	7	ASME 300	ASME 600
	Total	134		

Table 18.6: Pipeline Characteristics

Description	Unit	Description
Specification	--	API 5L Grade X65 SAWL
SMYS	Kg/cm^2	4,570
External Diameter	in (mm)	26 (660.4)
Minimum Thickness	mm	10.3 / 12.7 / 14.3 / 19.1 / 22.2 / 23.8
Rugosity	mm	0.1 – 0.5

Each pump station will contain three horizontal 3.99 MW multi-stage BB3 10x16 MSND (API 610 Super Duplex) centrifugal type pumps arranged in parallel (2 operating + 1 stand-by).

Each pump station will have emergency drainage ponds with a holding capacity of 110% of the discharged pipeline. These emergency ponds will be located next to the pumping stations.

Table 18.7 and Table 18.8 show the pump stations and emergency pond characteristics, respectively.

Table 18.7: Pump Stations and Locations

Pumping Station	Alternative	Location	Elevation (masl)	Pumps	Flow per Pump, L/s	TDH, m	Efficiency, %	Power per Pump, kW
N°1 (EB1)	2	Longotoma	12	2 operating + 1 stand-by	150 - 250	706 - 822	76.9 - 83.9	1,390 - 2,474
N°2 (EB2)	1 + 2	Cabildo	640	2 operating + 1 stand-by	150 - 250	1016 - 1247	72.8 - 82.3	2,116 - 3,825
N°3 (EB3)	1 + 2	River Junction	1,545	2 operating + 1 stand-by	150 - 250	515 - 534	80.0 - 83.1	976 - 1,623
Terminal Station (ET)	1 + 2	Project Site	1,875					

Table 18.8: Dimensions of the Emergency Ponds

Pumping Station	Location (km)	Volume to be drained (m^3)	Pool Volume (m^3)
EB 1	7.2	12,978	14,276
EB 2	47.4	16,488	18,137
EB 3	102.2	11,141	12,255

18.8.2 Water Infrastructure Power Supply

The Cabildo substation will supply power to Pump Station EB2. The power to Pump Station EB3, will be supplied from the main Vizcachitas substation. Each substation of the pump station substation will reduce the incoming voltage to 4.16 kV and 0.4 kV. The substations will have 33 kV/ 4.16 kV and 4.16 kV/0.4 kV power transformers (mineral oil insulated).

18.9 Electric Power Supply

The electrical power line will be connected at the Los Maquis substation located 25 km east of the city of Los Andes, Valparaíso Region (Figure 18.5).

The power supply system considers a 220 kV double circuit line to transfer 150 MW, with 0.9 inductive power factor, approximately 61 km from Los Maquis to the Project. The power line will be connected to a 220 kV high voltage yard using gas insulated substation (GIS) technology in a double bar configuration in the Los Maquis substation.

At the Los Maquis 220 kV substation, the high voltage equipment to be installed will include two GIS, six electrical terminal joints, six lightning rods and one control system.

The double circuit transmission line will be built over rugged terrain with elevations ranging from 1,200 masl to 3,500 masl for a total length of 60.8 km. As shown in Table 18.9, 43.4 km of the power line are low zone and 17.4 km are high zone (above 2,200 masl according to the Chilean NSEG 5 En.71 standard).

Table 18.9: Differentiation of Zones by Geographical Altitude

Zone Type	First Station (m)	Final Station (m)	Total Length (m)
Low	0	9,785.62	9,785.62
High	9,785.62	11,847.78	2,062.16
Low	11,847.78	32,697.56	20,849.78
High	32,697.56	48,063.48	15,365.92
Low	48,063.48	60,851.69	12,788.21

It is recommended that AAAC 1200 MCM conductor be used in the low and high zones to handle the altitude-corona effect.

For the low-zone it is proposed to use 16 insulators, calculated for a minimum breaking load of 120 kN. For the complete anchor assemblies, 17 insulators are planned with a minimum breaking load of 120 kN.

For the high-zone, it is proposed to use 18 insulators for a minimum breaking load of 160 kN. For the complete anchor assemblies, 19 insulators are planned with a minimum breaking load of 240 kN.

One guard cable is planned, which should be in the central and highest point of the towers, with a protection angle of 30° or less. To facilitate communications between substations, it is recommended to use OPGW (optical ground wire) 24 fibre guard cable with optical fibre core.

18.9.1 On-Site Electrical Distribution

The electrical distribution system is designed in 33 kV. The power will be supplied from a single main substation that will reduce the voltage from 220 kV to 33 kV, through two 100/120 MVA transformers. The substation will include a 200kV-SF6 GIS, a reactive compensation yard and harmonic filters for the gearless mill drive (GMD) equipment, and an area for diesel emergency generation to support critical loads in case of black-outs.

The 33 kV internal power lines will provide distribution as follows:

- 33 kV feeder cables to the electrical rooms of the main conveyors, secondary crushing plant, tertiary and secondary grinding.
- 33 kV aerial distribution line to the primary crushing area, mine areas (dewatering, explosives plant) with AAAC FLINT conductor, including OPGW guard cable, with a capacity of 25 MVA, and an average length of up to 10 km.
- 33 kV double circuit overhead distribution line to the infrastructure located in the lower zone (flotation plant, filters, molybdenum plant, pump station). This line would run parallel to the water recovery pipeline and the feed launder.

18.9.2 Electrical Rooms

The substations and electrical rooms are located close to the consumption centres. The distribution substations will be unit type, outdoors, installed on concrete foundations, with an oil collection system in case of spills or leaks. Firewalls would be built to isolate the transformers from each other and from the electrical rooms. The electrical rooms will be container type.

Electrical rooms planned for the Project are:

- Main Electrical Room (5220-ER-01)
- Truck Shop Electrical Room (1310-ER-01)
- Mine Services Electrical Room (1320-ER-01)
- Explosives Plant Electrical Room (1600-ER-01)
- Primary Crushing Electrical Room (2100-ER-01)
- Secondary Crushing Electrical Room (2200-ER-01)
- Electrical Room BIN (2300-ER-01)
- HPGR Crushing Electrical Room (2400-ER-01)
- Ball Mill 1 Electrical Room (3110-ER-01)
- Ball Mill 2 Electrical Room (3110-ER-02)
- Grinding Electrical Room (3100-ER-01)
- Feed Thickener Electrical Room (3200-ER-01)
- Reclaimed Water Electrical Room (3160-ER-01)
- Main Electrical Room Lower Zone (5220-ER-02)
- Concentrate Flotation and Thickening Electrical Room (3400-ER-01/3480-ER-02)
- Electrical Room Fresh Water Pump Station (5800-ER-01)
- Molybdenum Plant Electrical Room (3700-ER-01/02)
- Filtering Tailings Plant Electrical Room (4100-ER-01)
- Electrical Room for Contact Water Treatment Plant (5900B-ER-01)
- Electrical Room for Tailings Deposition Conveyor Belts (4100-ER-02)
- Mine Dewatering Electrical Rooms (1450-ER-01, 02, 03)
- Access Control Electrical Room (5110-ER-01).

18.10 Tailings Disposal

Tailings generated by the flotation process will be classified into fine and coarse fractions to be filtered using belt and press filters to obtain a final cake with 15% moisture (2.17 t/m3 density). This filtered material will be stacked in layers co-mingled with mine waste.

Along with reducing the deposit footprint and avoiding the construction of a tailings impoundment, this system has substantial advantages such as:

- Reducing make-up water consumption by 50% compared with the thickened tailings alternative
- Compaction methods and fines content in the tailings avoids infiltration to the bedrock

- Enhanced geotechnical stability in an area with strong seismic activity
- Enhanced chemical stability by reducing oxygen diffusion, potential acid generation is also neutralized by the lime contained in the scavenger tailings.

Approximately 2.2 billion m^3 of co-mingled waste rock (1.5 billion m^3) and filtered tailings (0.7 billion m^3) will be discharged into the tailings storage facility (TSF) located in the valley. The TSF is below 1,935 masl to allow the mine trucks loaded with waste to operate on a downslope into the valley (Figure 18.6).

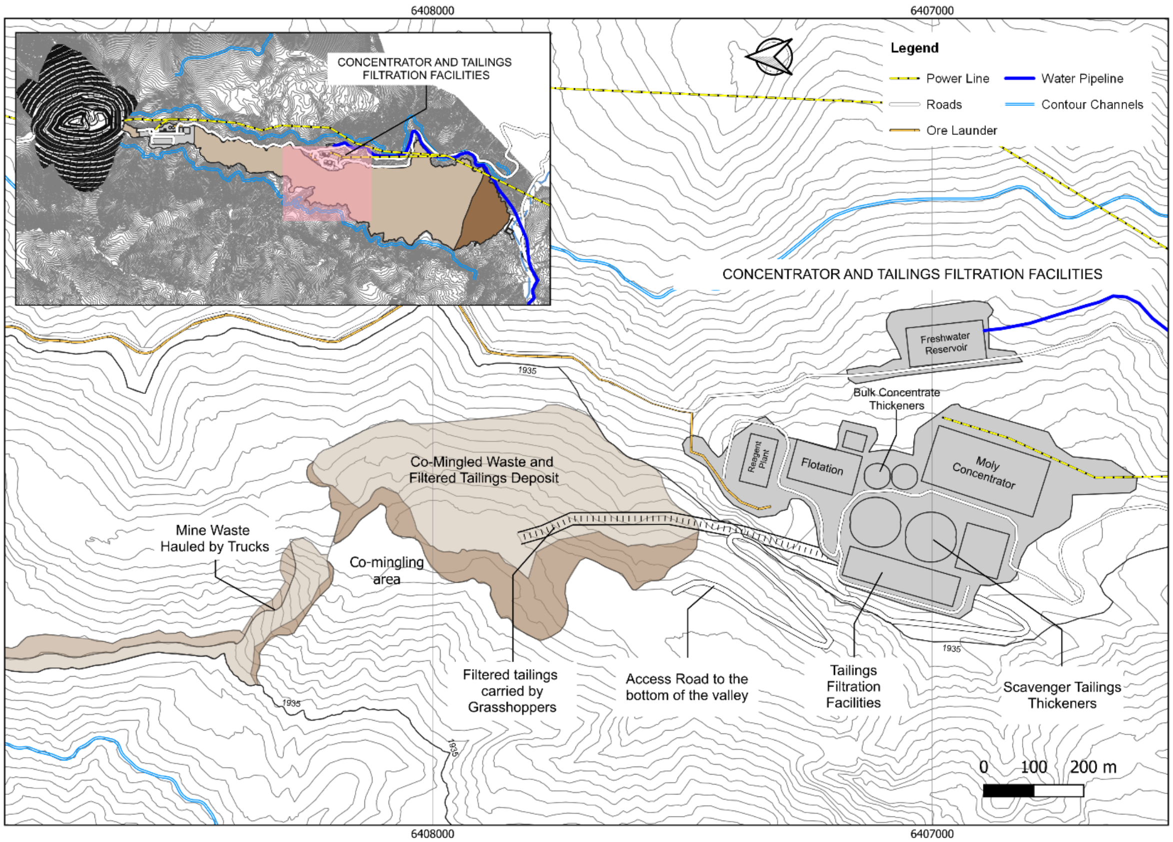
The TSF impoundment strategy requires a starter dam (5 m high) located at the Rocín and Hidalgo River confluence. The starter dam will have a leak prevention barrier (geotextile and jet grouting) that will retain contacted water. This design complies with the design characteristics (top width and leak proofing) indicated in the National Regulation for Approval of the Design, Construction, Operation and Closure of Tailing Dams Projects in Chile (Supreme Decree N° 248/2008, Ministry of Mining).

The foundation conditions in the TSF basin consist of alluvium/colluvium containing gravels and overlying bedrock remaining from the diverted river basin. The thickness of the alluvium/colluvium material varies across the site and is generally thicker (50 m) in the valley bottom compared to the valley slopes.

The filtration plant will be located on a platform at an elevation of 1,860 masl. The area will receive rougher tailings from the bulk flotation and two cyclone clusters will produce a coarse fraction and a fine fraction. The coarse fraction will be pumped to the belt filter building. The fine fraction, along with the scavenger tailings, will be thickened and then fed to 13 pressure filters installed in a nearby building.

The filtered tailings will be collected by conveyor belts located below the belt and pressure filter buildings. The filtered cake will be conveyed to the disposal site in the bottom of the valley using grasshopper conveyors and stackers. A small fleet of mining trucks and bulldozers will prepare layers of mine waste and filtered tailings and stack them to prepare working platforms that will allow the tailings deposit and the conveyor to reach areas further up the valley.

Figure 18.6: TSF Construction at the Start of the Mining Operation



Source: Los Andes Copper, 2022

The growth plan for the tailings deposit considers two stages:

- In the first years the co-mingled deposit will grow from the southern part, near to the filter plant, up to the north towards the crushing and grinding platform. Mine trucks hauling waste will use the west side of the valley, the filtered tailings will be conveyed by grasshopper conveyors and stackers on the east side of the valley (Figure 18.6).
- After the second stage of the diversion tunnel is completed, the co-mingled tailings/waste will be deposited towards the south across the valley.

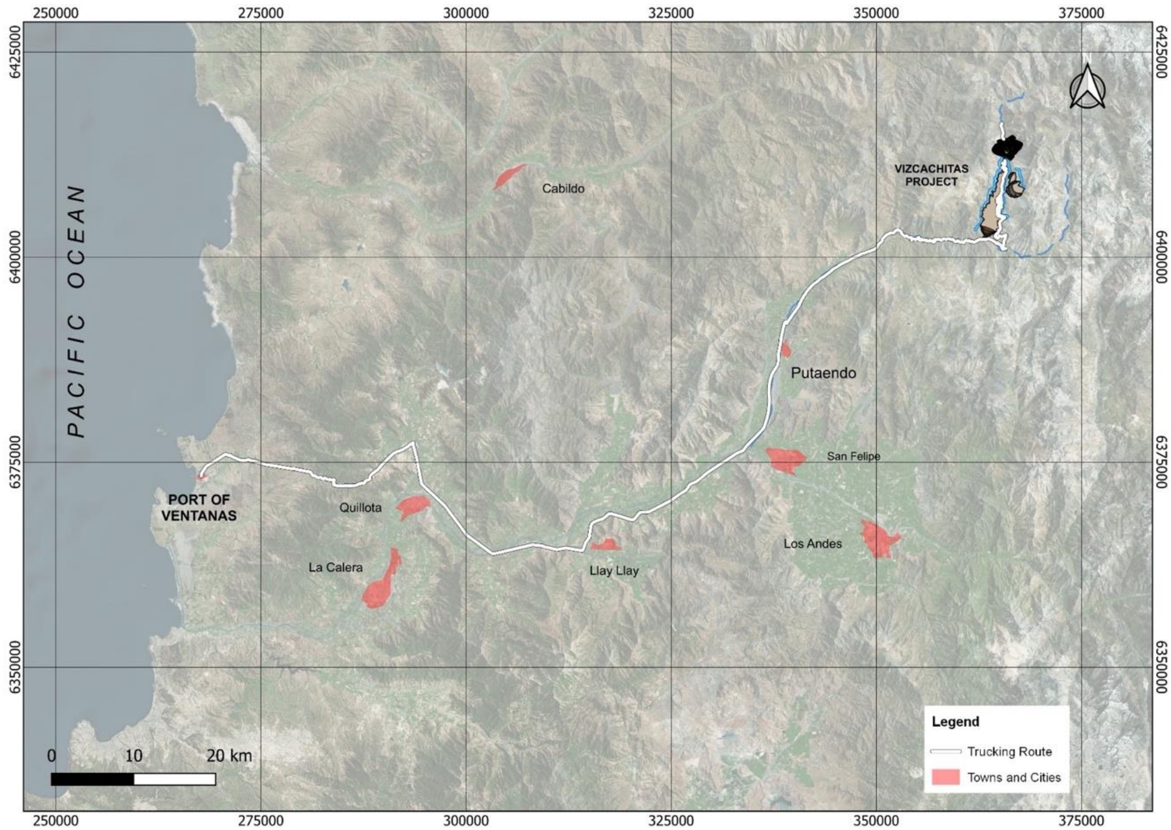
Mobile equipment rehandles mine waste and filtered tailings, building platforms and pads to allow TSF growth.

18.11 Concentrate Handling and Transport

Vizcachitas will produce copper and molybdenum concentrates. The copper concentrate will be loaded into closed Rotainers on site and transported 145 km from the mine to the port of Ventanas. The molybdenum concentrate will be loaded into Maxi-sacs and hauled by truck to the port or to other destinations agreed with buyers.

The copper and molybdenum concentrate trucking operations would be owned and operated by a third-party logistics service provider. The haul truck one way time would be approximately 3 hours. A fleet of approximately 40 trucks per day will be needed at an estimated production rate of 2,400 t/d of copper concentrate.

Figure 18.7: Concentrate Hauling Route to the Port of Ventanas



Source: Los Andes Copper, 2023

18.12 Waste Management Facilities

The waste management centre will require an area of 21 ha. This area would include the infrastructure and services to manage the solid waste that will be generated in the construction, operation and closure of the Project. The main areas are:

- Administration office
- Change room
- Warehouse for temporary collection of hazardous waste
- Recycling shed
- Machinery shed
- Access control
- Washing plant

- Salvage yard
- Controlled deposit for non-hazardous industrial waste
- Sanitary landfill
- Roads
- Perimeter fence
- Perimeter channels.

The waste management centre will be located in the southern area of the site, near the access gatehouse and pump station EB3.

19. MARKETING STUDIES AND CONTRACTS

19.1 Market Studies

The Vizcachitas Project will produce copper and silver contained in copper concentrate, both openly traded commodities, and molybdenum contained in molybdenum concentrate. No deleterious elements (arsenic, zinc, mercury and lead) will be present in the concentrates in amounts that would attract penalties.

The Vizcachitas copper concentrates are forecast to contain approximately 24% Cu over the LOM, and molybdenum concentrates grades are forecast to average 45% to 50% over the LOM. Average silver grades are forecast to be approximately 46 ppm to 65 ppm in the copper concentrate.

The following studies were conducted for this TR:

- CRU+ – Molybdenum Concentrates Market, February 10, 2023
- Synex Consulting Engineers – Projection of Prices for the Power Supply for the Vizcachitas Project, December 1, 2022
- Logsys SpA – Calculation of a Transfer Fee Port for Copper Concentrate through the Use of Rotating Container Technology, February 18, 2023.

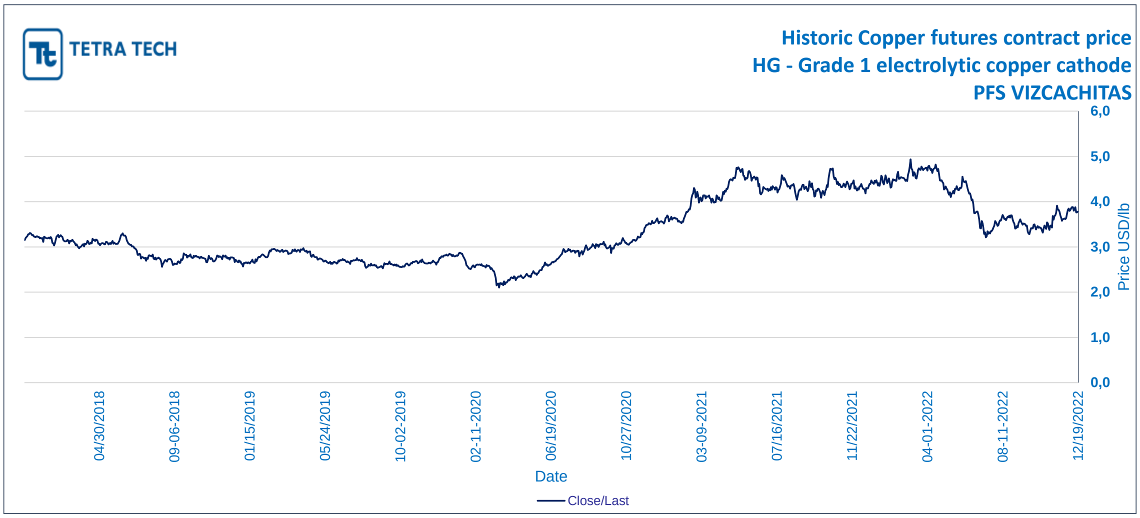
19.2 Commodity Prices and Concentrate Specifications

The economic analysis (Section 22) uses the prices shown in Table 19.1, these are long term consensus copper and silver prices calculated by a leading Canadian bank, and the molybdenum long-term forecast price from CRU. Five year historical prices for these three commodities are shown on Figure 19.1, Figure 19.2 and Figure 19.3.

Table 19.1: Market Consensus Long-Term Commodity Prices

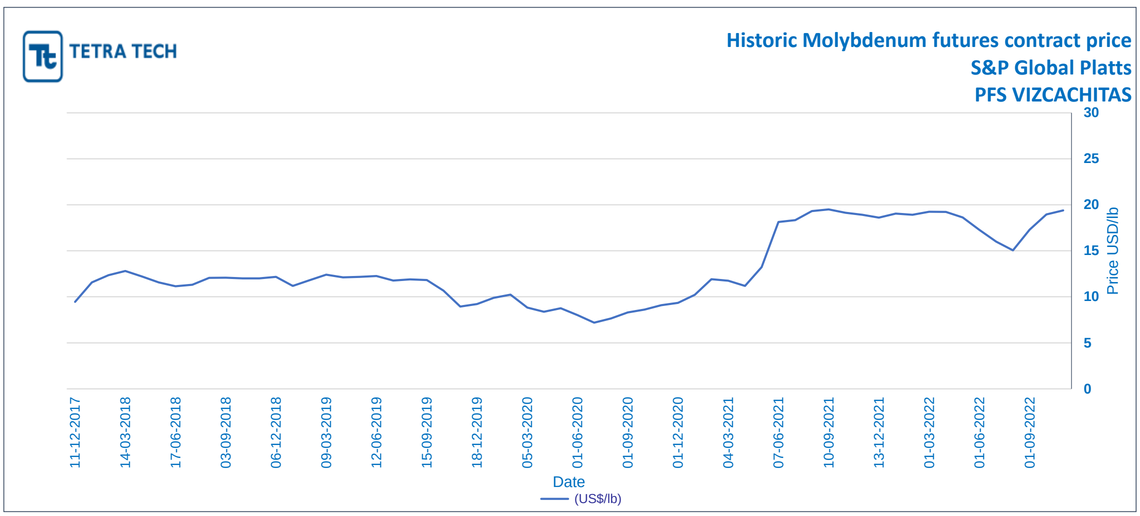
Commodity	Price
Copper	USD 3.68 per pound (lb)
Molybdenum	USD 12.90 per pound (lb)
Silver	USD 21.79 per ounce (oz)

Figure 19.1: Copper Historical Prices, Last 5 Years (HG Futures Contract)



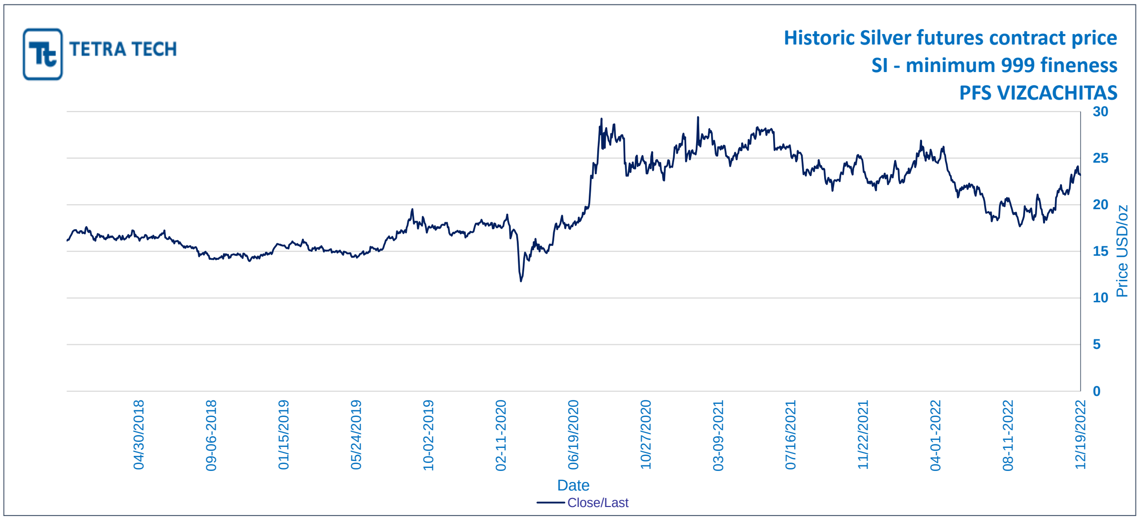
Source: Prepared by Tetra Tech from Nasdaq Historical data, www.nasdaq.com, 2022

Figure 19.2: Molybdenum Historical Prices, Last 5 Years



Source: Prepared by Tetra Tech from Chilean Copper Commission (Cochilco) data, www.cochilco.cl, 2022

Figure 19.3: Silver Historical Prices, Last 5 Years (SI Futures Contract)



Source: Prepared by Tetra Tech using Nasdaq Historical data, www.nasdaq.com, 2022

The selling cost assumptions for copper and molybdenum concentrates are shown in Table 19.2.

Table 19.2: Concentrate Sales Characteristics

Category	Units	Value
Copper Concentrate Freight		
Port	US$/wmt	12.00
Insurance	US$/wmt	0.50
Customs	US$/wmt	0.80
LT Ocean Freight to China	US$/wmt	38.85
Copper Concentrate		
Cu Payability	% of contained	96.50
Ag Payability	% of contained	90.00
Treatment Charges	US$/dmt	80.00
Cu refining Charges	US$/lb Cu	0.08
Ag refining Charges	US$/oz	0.35
Transport Losses	%	0.20
SG&A Costs	US$/lb Cu	0.02
Financing Charges	Days	45
Financing Charges	Rate (annual/360)	3.50
Molybdenum Concentrate		
Mo Payability	% of contained	97.50
Financing Charges	Days	45
Financing Charges	Rate (annual/360)	3.50
Roasting and Leaching Costs	US$/Kg Mo	2.52
Freight and Insurance	US$/Kg Mo	0.13

Copper concentrate will be transported in Rotainers by truck to the Port of Ventanas (145 km); the Ports of Valparaiso and San Antonio are other options. Rail transport from San Felipe to the three ports is an alternative that should be further evaluated. Molybdenum concentrates will be trucked in Maxi-sacs to the port or to other destinations agreed with buyers.

The sales costs used in this TR are either based on the market studies conducted for this TR, or are, in Tetra Tech's opinion, consistent with the range of prices used for existing commercial arrangements in other projects.

19.3 Contracts

There are no contracts in effect at this time.

20. ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Introduction

Los Andes Copper has made significant efforts in the design of the Project to minimize and mitigate environmental impacts, and to comply with environmental regulations. Also it has implemented consistent efforts for community engagement to facilitate the advancement of the Vizcachitas Project. This chapter summarizes the environmental, permitting, and social considerations of the Project and related work conducted to date.

20.2 Permitting Process

Under Chilean law, mining projects must conduct Environmental Impact Assessment (*Estudio de Impacto Ambiental*, EIA) which are evaluated in accordance with the Environmental Impact Evaluation System (*Sistema de Evaluación de Impacto Ambiental,* SEIA) regulations. The steps included in the approval process are described in Laws N° 19.300 and N° 21.417 and Supreme Decree 40/2013 and are summarized below.

- Prepare an EIA, including project description, environmental and social baseline studies, predictive modeling, social assessment, environmental and social management plans, risk assessments, mitigation plans, monitoring plans and an emergency response plan. A description of voluntary commitments should also be included if planned.
- Community consultation is mandatory, in the form of community meetings and open houses which are led and facilitated by the Environmental Assessment Service (*Servicio de Evaluación Ambiental*, SEA).
- Submit the EIA document to the SEA. The project owner uploads the documents online, and the SEA reviews for completeness and admissibility. A summary is posted online and public feedback is open for 60 working days.
- Following the feedback period, the SEA typically releases a request for additional information. This is called the Consolidated Report of Request for Clarifications, Corrections, and Extensions (*Informe Consolidado de Solicitud de Aclaraciones, Rectificaciones y Ampliaciones*, ICSARA). The project owner must then undertake the necessary studies and analysis to respond to the ICSARA.
- The SEA has 15 working days to evaluate the adequacy of the EIA addendum submitted in response to the ICSARA. The SEA may require further information after the addendum is submitted, through a second ICSARA. If not, 15 working days are allowed to publish an online summary of the addendum(s). This is called a Consolidated Report of Assessment (*Informe Consolidado de Evaluación*, ICE).
- Upon completion of the review and provided that there are no further ICSARAs, the Regional Commission of Environmental Assessment (*Comisión de Evaluación Ambiental*, COEVA) issues (or rejects) the environmental licence approving the EIA and

the project and listing approval conditions, in a document called the Environmental Assessment Resolution (*Resolución de Calificación Ambiental*, RCA).

The SEIA system is an online and public process and can be tracked on the https://sea.gob.cl/ website. The assessment of an EIA can take from 2 to 4 years to complete.

20.2.1 Other Environmental Related Regulations

Once the RCA is obtained, the Project must seek several specific permits for construction, operation, and closure from several government entities. Some of the most significant ones are the water-related licences from the Water Authority (*Dirección General de Aguas*, DGA) and mining licence and permits from the National Geology and Mining Service (*Servicio Nacional de Geología y Minería,* SERNAGEOMIN). These licences and permits can be initiated during the EIA review period; however, they cannot be granted until a favourable RCA is issued.

The environmental regulations directly related to the Project include the following:

- Supreme Decree N° 40/2013, Ministry of the Environment, Regulation of the Environmental Assessment System
- Supreme Decree N° 138/2005, Ministry of Health, Establishes the Obligation to Declare Emissions
- Supreme Decree N° 1/2013 modified by Supreme Decree No. 31/2017, Ministry of the Environment, Regulation on Emissions and Transfer of Pollutants (PRTR)
- Supreme Decree N° 75/1987, Ministry of Transport and Telecommunications, Establishes Conditions for the Transportation of Loads
- Supreme Decree N° 4/1994, Ministry of Transport and Telecommunications, Establishes Pollutant Emission Standards Applicable to Motorized Vehicles and Sets Procedures for their Control
- Supreme Decree N° 594/1999, Ministry of Health, Regulation on Basic Sanitary and Environmental Conditions in Workplaces
- Decree with Force of Law No. 725, Sanitary Code
- Supreme Decree N° 148/2003, Ministry of Health, Sanitary Regulation on Hazardous Solid Waste Management
- Supreme Decree N° 78/2009, Ministry of Health, Regulation of Storage of Dangerous Substances, modified by Supreme Decree No. 43/2015
- Supreme Decree N° 298, Ministry of Transport and Telecommunications, Regulates the Transportation of Dangerous Cargo through streets and roads
- Supreme Decree N° 90, Ministry of the General Secretary of the Presidency, Emission Standard for the Regulation of Pollutants associated with Liquid Waste Discharges into Marine and Continental Superficial Waters
- Law N° 20.551 amended by Law N° 20.819 and Supreme Decree N° 41/2012, Law and Regulations on Closure of Mining Works and Facilities
- Law N° 17.288 amended by Law N° 20.021 Law on National Monuments

- Law N° 20.920, Framework Law for Waste Management, Extended Producer Responsibility and Promotion of Recycling
- Supreme Decree N° 38/2011, Ministry of the Environment, Standard for the Emission of Noise Generated by Sources Indicated
- Exempt Resolution N° 144/2020, Ministry of the Environment, Basic Standard for the Implementation of Modification to the Regulation of the Registry of Emissions and Transfers of Pollutants, PRTR.
- Supreme Decree N° 151/2007, Ministry General Secretariat of the Presidency, Formalizes the First Classification of Wild Species according to their State of Conservation
- Supreme Decree N° 23/2019, Ministry of the Environment, Approves and Formalizes Classification of Species according to Conservation Status
- Law N° 19.473 and Law N° 4.601, Hunting Law
- Supreme Decree No. 1/1992, Ministry of National Defence, Regulation for the Control of Water Pollution

20.3 Current Permitting Status

Los Andes Copper has not yet initiated the EIA studies for the Vizcachitas Project.

On May 13, 2021 the Project obtained RCA No. 14 (RCA No. 14/2021) approving exploration activities (DIA, 2019). The Project is also subject to a previous RCA No. 12, April 17, 2019 (RCA No. 12/2019) (Regularization of Mining Drilling Platforms, Las Tejas Area). The approved activities included the drilling of 124 drill holes (diamond drill and reverse air) with an average length of 750 m, over a 4 year period. The permits cover 73 new platforms and 51 existing platforms, with up to 4 drill holes on each platform. These drilling platforms were designed to be in areas of low to zero vegetation density, taking advantage of existing footprints and previously constructed platforms, mainly on rocky hill slopes.

Several administrative and judicial objections have been filed against RCA N° 14/2021. At the administrative level, the objections have been rejected, except for two objections that commenced in January 2023, with resolutions pending as of the Effective Date of this TR. At the judicial level, several objections were filed before the Second Environmental Court in Santiago, with resolutions also pending as of the Effective Date of this TR. The objections seek to revoke and nullify the Vizcachitas RCA N° 14/2021.

On March 18, 2022 the Second Environmental Court issued a preliminary injunction suspending RCA No. 14/2021, effectively stopping the drilling programme, based on potential impacts on the protected species *Leopardus jacobita* (Andean Cat). On July 20, 2022 the same court revoked its previous decision, declaring that the planned work was compatible with the protection of the Andean Cat. The Court also imposed certain conditions for the drilling re-start (with the drilling suspended in the interim), including the implementation of a monitoring plan and the preparation of a draft recuperation, conservation and management plan for the Andean Cat (according to Supreme Decree N° 1/2014, Ministry of Environment).

20.4 Environmental Studies

The Project is located in the Putaendo Municipality, San Felipe Province, Valparaíso Region. Elevations on the Project site range between 900 m and 2,500 m. The Project is within the Rocín River basin, a minor tributary of the Aconcagua River, and the area is defined as part of the Andean Sclerophyllous Scrub and the High Andean Steppe, part of the broader Mediterranean ecosystem in Central Chile.

20.4.1 Meteorology

A macro-characterization of the climate in the study area was defined using the *Köppen* climate classification, (this organizes the climates of the world based on two climatic elements, air temperature and the amount of water available, in relation to the phytogeographical characteristics). The Project area was defined as being in the *Cold Temperate with Winter Rains (Csc) climatic zone,* as summarized in Table 20.1 (DIA, Chapter 2, 2019).

Table 20.1: Description of Climate Classification Csc

<table>
<tr><th>Climate</th><th>Major Climate Group</th><th>Thermal threshold</th><th>Climate</th><th>Vegetation</th></tr>
<tr><td rowspan="5">Csc</td><td>C</td><td>The average temperature of the coldest month is below 18°C and above -3°C.</td><td>Temperate: fall-winter rains and sporadic rains all year round.</td><td>Temperate forest and scrub.</td></tr>
<tr><td>Second Letter</td><td colspan="3">Rainfall Characteristics</td></tr>
<tr><td>s</td><td colspan="3">The rainfall of the driest month in summer is less than one-third that of the wettest winter month.</td></tr>
<tr><td>Third Letter</td><td colspan="3">Thermal Characteristics</td></tr>
<tr><td>c</td><td colspan="3">The average temperature of the coldest month is above -3°C and the warmest month is below 22°C, and less than four months have an average temperature exceeding 10°C.</td></tr>
</table>

Two meteorological stations were identified near the Project area, Resguardo Los Patos and Putaendo in Resguardo Los Patos stations. Both stations take temperature and precipitation measurements. Due to proximity to the Project area, the Resguardo Los Patos records were used for rainfall, and the Putaendo in Resguardo Los Patos records were used for air temperature, relative humidity, wind speed and wind direction. The stations are located within the same macro climate zone of the Köppen classification (Chilean Ministry of Agriculture, National Agroclimatic Network, AGROMET, available at https://www.agromet.cl/reportes-historicos).

The main features of these stations are presented in Table 20.2 (DIA, 2019).

Table 20.2: Monitoring Station Characteristics

Station	Validity	Source	Distance to the Project (km)	Municipality	UTM WGS 84 Coordinates		Elevation masl	Parameter Used
					North	East		
Putaendo in Resguardo Los Patos	In effect	DGA, Ministry of Public Works	33	Putaendo	6,392,440	338,011	830	Air temperature, relative humidity, atmospheric pressure, wind speed and wind direction.
Resguardo Los Patos	In effect	DGA, Ministry of Public Works	16	Putaendo	6,403,111	351,664	1,220	Temperature and rainfall

20.4.1.1 Air Temperature

Table 20.3 shows the air temperature records (in °C) for the Putaendo in Resguardo Los Patos station (AGROMET).

Table 20.3: Average Air Temperature (°C)

Year	Jan	Feb	Sea	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
2016	20.6	21	18.5	13.1	11.5	9.3	9	11.2	14.2	14.7	18.6	19.5
2017	22.8	21	18.2	14.7	8.9	8.9	8.8	8.1	11.5	14.2	17.8	20.3
2018	20.6	20.8	18.6	15.5	12	9.2	7.9	10.9	13.5	15	18.9	19.9
2019	22.8	21.7	18.8	14.6	13.9	n/a	9.2	12.5	12.6	15.3	20.1	21.9
2020	22.7	21.8	22.9	16.4	13.7	7.9	9.1	9.8	13.3	15.8	18.1	n/a
2021	23.5	23.1	19.3	16.3	13.1	10.1	9.7	10.5	12.3	15.7	19.1	20.3

n/a: not available

The temperature records show well-differentiated seasons throughout the year, with June and July as the coldest months and January and February as the hottest months.

20.4.1.2 Wind

Table 20.4 shows the average wind speed (in km/h) and wind direction records for the Putaendo in Resguardo Los Patos station between 2016 and 2021 (AGROMET).

Table 20.4: Speed (km/h) and Wind Direction

Year	Month	Direction	Speed (km/h)
2016	January	Southwest	0.9
	February	Southeast	1.7
	March	Southeast	1.6
	April	North	1.2
	May	Northwest	1
	June	Northwest	1.1
	July	Northwest	1.2
	August	Northwest	1.5
	September	Northwest	1.7
	October	Southeast	1.6
	November	Southeast	1.9
	December	Southeast	1.9
2017	January	Southeast	1.8
	February	Southeast	1.6
	March	Southeast	1.5
	April	Northwest	1.2
	May	North	0.8
	June	North	1.1
	July	Northwest	1
	August	North	1.3
	September	West	1.6
	October	Southeast	1.6
	November	Southeast	1.6
	December	Southeast	1.7
2018	January	South	1.4
	February	South	1.4
	March	Southeast	1.2
	April	North	1
	May	North	0.9
	June	Northwest	1
	July	North	1
	August	Northwest	1.3
	September	Southeast	1.4
	October	North	1.3
	November	North	1.2
	December	North	1.2

Year	Month	Direction	Speed (km/h)
2019	January	South	1.3
	February	South	1.2
	March	North	1.1
	April	North	0.9
	May	North	0.8
	June	s/i	s/i
	July	Northwest	1
	August	Northwest	1.3
	September	Southeast	1.5
	October	North	1.2
	November	North	1.2
	December	North	1.2
2020	January	Southwest	1.4
	February	Southwest	1.3
	March	Southwest	1.3
	April	Southwest	1.1
	May	Southwest	1
	June	Southwest	0.9
	July	Southwest	1.1
	August	Southwest	1.4
	September	Southwest	1.6
	October	Southwest	1.5
	November	Southwest	1.4
	December	Southwest	s/i
2021	January	South	1.5
	February	South	1.4
	March	South	1.2
	April	South	0.8
	May	South	0.9
	June	South	0.9
	July	South	1.2
	August	South	1.3
	September	South	1.6
	October	Southeast	1.7
	November	South	1.2
	December	South	1.4

Prevailing winds for the period 2016 to 2021 are predominantly north, south-east and south with a cumulative frequency of 65% of the time (23%, 21% and 21% respectively). The highest average wind speed (1.9 km/h) was observed in November and December 2016; the lowest wind speed (0.8 km/h) was observed in May 2017 and 2019 and April 2021.

20.4.1.3 Average Relative Humidity

Table 20.5 shows the relative humidity (in %) for the Putaendo in Resguardo Los Patos station between 2016 and 2021 (AGROMET).

Table 20.5: Relative Humidity (%)

Year	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
2016	63.6	57.3	59.3	75.8	75.3	65.8	70.1	59.3	49.6	61.5	55.1	59.5
2017	52.1	57.9	57.2	58.8	59.6	66.7	54.9	53.9	54.1	61.1	57.3	55.3
2018	58.1	58.8	51.8	59.1	54.1	57	65.3	53.5	60.1	56.4	49.6	54.5
2019	53	54.3	51.2	61	55.5	s/i	55.7	43.6	55.3	46	44.8	42.5
2020	49.9	45.9	52.5	54.6	47.5	71.7	66.6	59.3	52.9	51.2	53.7	s/i
2021	50.3	56.1	56.8	55.1	56.3	58.1	44.2	58.3	55.1	50	49.9	52.7

Records indicate higher relative humidity during the fall and winter months of April, May, June and July with the lowest values observed in December and January.

20.4.1.4 Monthly Rainfall (mm)

Table 20.6 shows the rainfall records (in mm) for the Resguardo Los Patos station (DGA, available at https://snia.mop.gob.cl/BNAConsultas/reportes) between 1940 and 2020. Rainfall is mainly concentrated during the months of May through August, with May and June being the wettest months. The driest months are in the summer, from December to March.

Table 20.6: Monthly Precipitation (mm)

Year	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
1940								40	22	18	0	0
1941	0	0	0	155	73.5	105	241	226	0	8	18	0
1942	0	0	0	0	48	98	70.5	143	41	20	35	0
1943	0	0	26	0	25	18	19	35	16	0	0	0
1944	55	26	0	0	33	128	0	162.5	0	35	0	0
1945	0	78	0	0	0	0	53	6	64	0	0	0
1946	0	0	0	9	39	30	33	6	7	0	0	0
1947	0	0	2	3	6	107	32	69	3	30	0	0
1948	0	0	0	44	55	49	185	20	27	7	0	0
1949	4	0	0	0	91.9	10	14	27	0	0	0	0
1950	0	0	0	65	155	0	0	69	27	55	25	0
1951	11	0	0	10	104	83	152	9	27	0	0	0
1952	0	40	0	0	95	104	42	18	56	23	0	0
1953	0	0	0	42	88	55	65	234	48	27	0	0
1954	0	0	0	57	60	68	36	10	0	0	0	0
1955	0	0	10	22	107	38	49	36	16	34	9	16
1956	0	0	25	13	10	14	82	32	15	5	10	10
1957	0	0	0	0	199	8	37	50	14	7	0	23
1958	0	0	7	0	109	89	22	18	12	0	13	0
1959	0	0	11	17	53	112	57	74	0	11	0	0
1960	0	0	0	0	22	153	57	27	10	8	0	0
1961	0	0	24	0	40	106	17	100.5	9.5	14.5	0	3
1962	0	0	0.2	0	55	159	17	11	3	17	0	0
1963	0	10	2	0	28	90	75	80	116	13.3	30	0
1964	0	0	0	0	0	51.5	39.5	59.5	0	0	1	0
1965	0	0	0.5	12	49.5	12.5	139	259	8	18	0	6.5
1966	0	0	0	16	6	145.5	92.5	47.5	0	5.5	12.5	11
1967	0	0	0	0	8	45.5	45	9	95	14.5	0	0
1968	0	0	2.5	17.5	0	9.5	0	20	36.5	0	0	0
1969	0	0	0	16	5.5	65.5	0	32.5	0.5	1	0	0
1970	0	0	3	0	113	0	126.5	12.5	26	19.5	0	0
1971	13	0	0.5	0	2	54	0	26	11	0	0	0
1972	0	0	0	4	27	191.5	20	104	58	10	7.5	0
1973	0	0	0	0.5	54	44.5	92.5	0	0	12.5	0	0
1974	0	0	0	0	0	146.5	0	8	34	0	8	0
1975	0	0	2	5	35	0.5	95	53.5	0	5	0	0
1976	6	4	0	5	17	40	2	25	11.5	33.5	33	0
1977	0	0	0	3	25.5	96	189.5	70	15	28	9.5	0
1978	0	0	0	0	3.5	21.5	242.5	18	28	2	108	0
1979	0	0	0	16.5	7	0	68	51	28.5	0	13	14
1980	0	8	0	119.5	15.5	68	91.5	10	46	20.5	0	3.5
1981	0	6	0	1	96	30	20.5	28	6	0	0.5	0
1982	0	0	25	0	94.5	222	145.5	71	26	8	2	0
1983	16	0	0	32	27	79	133	47.5	30	0	0	0
1984	0	0	0	0	31	14	352	24	38	16.4	3	4.5
1985	0.5	0	15.5	1	27.4	0	75	2	0.6	8.8	0	0
1986	0	7.2	0	1.1	125	111.5	0	39.5	6	2.5	7	0
1987	0	0	15	15.5	10.8	46.2	445.2	239.5	13	20	0	0
1988	0	0	0	0	7.5	6	26	12.9	13.5	0	13	3
1989	0	0	0	3.5	36.5	2.5	97.3	128.5	10.5	0	0	0
1990	0	0	0	1.5	6.5	3.6	20.2	41.5	30	0	0	0.5

Year	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
1991	0	0	0	3.9	44	159.5	107.7	0	73.1	2.8	0	2
1992	0	0	11	28.5	95.5	155	21	24	50	0	12	0
1993	1	0	0	35.2	118	12.9	20.5	17	11	0	0	0
1994	0	0	0	4	23	8.5	65.7	3.8	34	4.5	0	0
1995	18	0	0	11	7	58.5	29	14	22	0	0	0
1996	0	0	0	19.9	4	33.3	22	9.4	2	1.5	0	1.5
1997	0	0	21.5	0	101	388.5	23	0	22.5	47.5	2	0
1998	0	3.5	3	15.5	14	19	0	0	22.1	0	0	0
1999	0	0	11	7	2.5	45.5	6.5	96.5	122.5	0	0	0
2000	0	0	0	20.5	33.5	191.5	65	0	91.5	4.5	0	0
2001	0	0	0	3	24.5	0	199	51.8	10	12.5	0	0
2002	0	0	15	17	154.5	166	119.5	59.5	19	1	0	0
2003	16	0	0	0	92	23	51	2.5	0.5	0	5.5	0
2004	0.1	0	9	46.5	0.8	50.5	54.2	67	24	0	31	0
2005	0	0	24.6	13.5	37	106.3	8.5	128	46	12	10.5	0
2006	0.5	6	0	0	6.5	46	213.5	14.5	1	33.5	0	0
2007	0	0	2	0	6.2	100	30	24	0	0	0	0
2008	0	0	3	16.5	92	76	27	120.5	4.3	0	0	0
2009	0	0	0	0	0	81	13	67	28	0	0	0
2010	0	0	0	0	20	22	35	13	28	7	28	0
2011	0	12	0	2	0	77	43	10	0	7	0	0
2012	0	0	0	29	32	23	0	21.5	4	26.5	0	3.7
2012	0	0	0	29	32	23	0	21.5	4	26.5	0	3.7
2013	0.8	0	0	0	61.2	24.5	2.5	28	2.5	0	0	0
2014	0	0	0	0	0.9	87	5.5	23	14	0	0.6	0
2015	0	0	20	0	0	28.6	54	1.7	37.4	27.2	3	0
2016	0	0	0	48	36	103	55	0	0	16.1	7	0
2017	0	0	0	3	107	49.5	0.4	26	10	27	0	0
2018	0	0	0	0	0	28	24	12	27.5	0	0	0
2019	0	0	0	0	6	0	4	0	9	0	0	0
2020	0	0	0	2	0	82.6	46	0	0	0	0	0
AVG	1.8	3	8.4	28	54.8	65.8	55.6	34.3	16.1	7.8	3.4	1.3

20.4.2 Noise and Vibration

Ambient noise levels in the Project area are generally low, fluctuating between 42 and 58 dB(A) during the day and between 39 and 53 dB(A) during the night, the Rocín River and the wind are the main sources of background noise. Technical monitoring requirements and potential impacts of noise on fauna (sensitive receptors) are regulated under the Evaluation Criteria in the SEIA: Evaluation of Noise Impacts on Native Fauna guide (SEIA, 2022).

20.4.3 Glaciology

Chile does not have special glacier laws or regulations in effect. Evaluation of direct and indirect alterations to glaciers are subject to the general environmental impact assessment legislation. The environmental assessment regulation, Supreme Decree N° 40/2013, Ministry of Environment, defines the required baseline studies in cases of alteration of glacier characteristics, including an analysis of glacier surface, volume, area, geographical location, thickness and surface characteristics, and the assessment of the significance of the potential impact.

The DGA publishes the National Glacier Inventory in Chile, the latest updated edition published in 2022. This inventory is the starting point for the environmental assessment of any project regarding glacier alterations. Los Andes Copper hired glacier expert Geoestudios to conduct a desktop review based on satellite imagery of the relevant areas of the National Glacier Inventory (Geoestudios, 2022). Field studies will be conducted during the feasibility phase to complete the required baseline and assessment work to support the EIA for the Project. Infrastructure and buildings are located at lower elevations, avoiding impacts to inventoried glaciers.

20.4.4 Hydrology

The Project is in the Rocín River basin, which begins on the Chile-Argentina border. The Rocín River is the only water body in the Project area, it joins the Hidalgo River approximately 3 km downstream of the Project site to form the Putaendo River (the Rocín River changes name to Putaendo River at the at the juncture with the Chalaco valley) which is a tributary of the Aconcagua River. In this area, the mountain peaks can exceed 3,500 masl.

The basin has runoff from melting snow generally between October and March and rainfall runoff between April and September. The snow line reaches down to 2,750 masl during winter. The morphological parameters of the Rocín River basin are shown in Table 20.7 (MN Ingenieros, 2022; for reference, the total area of the Aconcagua River basin, of which the Rocín River is a tributary, is 7,333 km^2).

Table 20.7: Morphological Parameters of the Rocín River Basin

Contributor Basin	Total Area (km^2)	Main Channel Length (km)	Slope (m/m)
Rocín River	408	38.67	0.522

A hydrological study was carried out to characterize the flood events potentially impacting the Project area. Data from the fluviometric stations shown in Table 20.8 were used in the study (MN Ingenieros, 2022).

Table 20.8: Coordinates of Nearby Fluviometric Stations

Stations	Total Area (km^2)	Location Geographical Coordinates	
		Latitude S	Length O
Alicahue in Colliguay	265	32°19'50''	70°44'15''
Putaendo in Resguardo Los Patos	885	32°30'05''	70°34'51''

With this information, the flows in Table 20.9 were calculated for a range of return periods (MN Ingenieros, 2022).

Table 20.9: Flood Events, Rocín River Basin

Return Period (year)	Period of Interest	
	Rainfall	Snowmelt
	Apr-Sept	Oct-Mar
	(m^3/s)	(m^3/s)
2	4.79	23.17
5	10.25	48.03
10	15.26	70.31
20	21.18	96.31
25	23.31	105.56
50	30.65	137.25
100	39.21	167.93

20.4.5 Hydrogeology

The Project is located in the mid-section of the Rocín River basin. This sub-basin is formed mainly by intrusive rocks and volcanic deposits and, to a lesser extent, by a filling of unconsolidated fluvial deposits. The fluvial deposits occur in the main channel of the Rocín River basin and are composed of granular materials.

In general, this aquifer has an unconfined behaviour with a predominantly north-south flow direction, following the orientation of the Rocín River, recharge is produced by rainfall and snow melt over the sub-basin and by an inflow of ground water from the north.

20.4.6 Geochemistry

In order to characterize the potential for acid rock drainage and metal leaching in the exposed pit wall and waste, a geochemical programme was initiated as part of the PFS. The results are summarized below.

Six composite samples representing the first 10 years of the mine plan were selected from a quarter-cut of the drill cores and sent to SGS, Metso and Takraf laboratories for chemical and mineralogical characterization, comminution, flotation and tailings filtration tests.

The Project is a primary copper deposit, composed of chalcopyrite and pyrite. Secondary copper sulphides, such as chalcocite and covelline, are present in less than 10% of the sampled core, they are observed only as a surface oxidation process, typical of areas above the water table as oxidation surfaces (patinas) on primary sulphides. Copper sulphides are mainly associated with hard silicates and phyllosilicates. Clays are found in less than 1.3% of the sampled core.

The potential for acid rock drainage and metal leaching was characterized (TR, Section 13) by selecting samples based on geology, spatial and timing criteria. These samples were selected to represent the northern and southern areas of the first 10 years of the pit, identified as North and South. Tailings composites (including scavenger tailings) from the North and South samples were also analyzed using ABA (acid-base accounting), NAG (net acid generation test) and TCLP (toxicity characteristic leaching procedure). Table 20.10 summarizes the results for ABA (an indication of the potential for acid rock drainage).

Table 20.10: Summary of ABA Test Results

Sample	S (%)	S^{-2} (%)	CO_3^{-2} (%)	SO_4^{-2} (%)	pH Paste	NP	AP	Net NP	NP/AP
South	0.5	0.42	0.29	0.04	6.44	12.43	13.13	-0.67	0.95
North	1.51	1.2	0.28	0.66	8.04	12.46	37.63	-20.01	0.47

The values of the neutralization potential (NP) and acid generation potential (AP) were estimated from chemical analysis (carbonates and sulphur). From the net potential neutralization rate (net NP) and the NP/AP ratios it is concluded that the deposit has potential for acid generation (Table 20.11).

Table 20.11: Summary of NAG Test Results

Sample	NAG pH	NAG kg H_2SO_4/t pH 4.5	Type
South	3.3	3.9	Acid Generator
North	2.5	19	Acid Generator

The NAG test results show that both samples have acid generation potential. The North composite sample generates 19 kg of H_2SO_4/t which is considered high.

TCLP is a chemical analysis process used to determine whether there are hazardous elements present in a waste. The test involves a simulation of leaching through a dump and can provide a rating to show whether the waste is dangerous to the environment or not. The TCLP results in Table 20.12 show that there are no hazardous elements or metal content in the leached solution.

Table 20.12: Summary of TCLP Test Results (mg/L)

Sample	As	Ba	Cd	Cr	Hg	Ag	Pb	Se
South	0.01	<0.1	<0.01	<0.05	<0.0005	<0.01	<0.05	<0.001
North	0.01	<0.1	<0.1	<0.05	<0.0005	<0.01	<0.05	<0.001

This geochemical information is preliminary; it will be supplemented with additional static and kinetic testing (humidity cells) as the Project advances to feasibility.

The co-mingling of filtered tailings and mining waste is expected to improve the chemical stability of the stored tailings by reducing oxygen diffusion because the interstices of the coarser mining waste will be filled with filtered tailings particles. The flotation stage rougher tailings (90% of the feed) will have a pH of 8 and the scavenger tailings (8% of the feed) will have a pH of 11.5 (adjusted with lime). Therefore, the remaining water contained in the filter cake will have a high pH which will help to partially neutralize any acid generated.

20.4.7 Air Quality and GHG Emissions

There is no air quality data available for the Project area. The closest official air quality station is located more than 60 km from site. This is not uncommon for rural areas in this part of Chile.

There are currently no air quality restrictions imposed for the Project area. Air quality will be documented during the baseline studies as the Project moves to the feasibility phase.

Green House Gas (GHG) emissions were calculated based on PFS level estimates for on-site fuel consumption (Scope 1) and energy consumption from the national grid (Scope 2). Scope 3 emissions will be calculated when the Project moves to the feasibility stage. GHG emissions (Scope 1 + Scope 2) were estimated to be 1.02 t CO_2e/t CuEq as an annual average for the 26 years of operation (Scope 1 = 1.02 t CO_2e/t CuEq, Scope 2 = 0 t CO2e/t CuEq). These estimates are consistent with those of other mine sites similar in size operating in Chile.

20.4.8 Water Quality and Aquatic Biota

Water Quality

Water is a key component of the environmental studies and the Project made a voluntary commitment to develop a water quality monitoring plan during the PFS drilling campaign, recognizing that this will be an issue for water users downstream (mostly farmers in the Putaendo valley). Monthly campaigns have been conducted with data available from September 2021 through the Effective Date of this TR.

The monitoring campaigns used eight sampling locations for surface water, three sampling locations for ground water, and the water source for the current drilling campaigns (water trucks). Ground water levels were also measured for the three ground water sampling locations.

According to the *Piper* and *Stiff* diagram the water quality results for surface and groundwater, indicate water quality belonging to the calcium sulphated class, with pH in the neutral range.

Water samples were tested for the parameters listed in the Chilean Standard for Irrigation Use (NCh. 1.333/1978 Rev. 1987). Results show that most parameters are below the thresholds (with occasional higher values for aluminum, copper, iron, manganese and sulphates). Fecal coliforms were detected in the surface water samples (Rocín River), although in very low concentrations (<100 NMP vs limit of 1,000 NMP). The pH values for surface waters ranged from 6.6 to 8.6, within the limits.

Aquatic Biota

A desktop review was conducted to characterize the aquatic environment present in the Project area (DIA, 2019). For the continental aquatic ecosystem present in the Rocín River, the study area covered approximately 4.2 km along the river. Over this area the river consists of a straight channel, with little development of substrates, with emerging rocks and sequences of rapids, generating well oxygenated water, but with low diversity of meso-habitats. Most of the substrates present in the river bed are large and emerge from the surface of the water flow. This, in addition to the high speed of the water, generates a turbulent flow and changes in flow direction, hindering the establishment of aquatic vegetation. Some presence of vegetation is limited to banks with shallow waters and sectors with substrates of clasts and sands. The water temperature of the river increases as the elevation decreases.

The aquatic biota consists of the following categories:

- Aquatic plants
- Ichthyofauna
- Macroinvertebrates (Zoobenthos)
- Phytoplankton
- Zooplankton
- Phytobenthic (Periphyton)
- Macrocrustaceans
- Chlorophyll (photosynthetic pigments).

The desktop review identified 31 potential species for the Aconcagua River basin (in conjunction with the Rocín River). Twelve are aquatic and riparian flora, three are macro-crustaceans and sixteen are fish fauna. For the fish fauna species, seven are native species and nine are introduced species. The conservation status of one species was identified as in Danger of Extinction (Northern Shrimp, *Cryphiops caementarius*), seven were Vulnerable species (all species of native fish fauna) and two were Insufficiently Known species.

The desktop review was complemented with field campaigns in the Rocín River. Only the Rainbow Trout (*Oncorhynchus mykiss*), an introduced species, was registered. It presents a migratory behaviour of the facultative type, meaning it can make a river-sea or river-lake migration, but does not require a viable population.

No singularities were observed in the study area, nor were hydrobiological components of exclusive environmental sensitivity observed.

20.4.9 Soils

A total of 23 soil observations were made to describe the soils present in the Project area (DIA, 2019), representing a total area of 1,188 ha. The survey was conducted in accordance with the recommendations in the Guideline for Soil Studies (SAG, 2011), the Environmental Assessment Guide: Natural Soil Resource (SAG, 2018), the Guide for the Description of the Soil, Flora and Fauna Components of Terrestrial Ecosystems in the SEIA (SEA, 2015a) and the Guide for the Evaluation of Adverse Effects on Renewable Natural Resources (SEA, 2015b).

These results indicate that the soils in the Project area are entirely high mountain soils, alluvial valleys, hillsides with colluvial deposits (Entisoles and Inceptisoles) and steep slopes, with little pedogenic development. The area has high slopes, with abundant surface and subsurface stoniness, presenting textures ranging from sandy to visibly rocky without the presence of phreatic levels or water table except for the soils located at the bottom of the river where the water table is between 20 cm and 40 cm deep (i.e. classified as Class VIII of Land Use Capacity, without agricultural, livestock or forestry value, useful only for wildlife, recreation or protection of watersheds).

Due to the strong morpho-dynamics of the landscape, processes of mass landslides, debris slippage, rock fall, mud flows, dry gullies (*arroyadas*), avalanches and strong fluvial erosion are common in the valley, with soils of pink materials, possibly andesites and rhyolites. The Potential Risk of Erosion (CIREN, 2010), determined by the desktop review, was that most of the soils in area are of Severe and Very Severe risk of erosion (total of 863 ha, 72% of the study area).

20.4.10 Flora and Fauna

Flora

The Project area is in the Phyto ecological region of the High Andean Steppe, in the sub-region of the Mediterranean Andes, formation of the Andean Sclerophyllous Scrub and the High Andean Steppe of Santiago (Gajardo, 1994) (Figure 20.1). This Phyto ecological region is characterized by the presence of two main elements typical of the physical environment. First, the climate is characterized by predominantly winter rainfall that increases from North to South. Second, the terrain is characterized by abrupt and mountainous terrain, with high mountains and steep slopes in which lithosols are frequent, with the general aspect of a high altitude desert.

The altitudinal zoning of plant communities is clear, with terrain and altitude as key distribution factors. From a physiognomic standpoint, the dominant flora is low plants, herbaceous or shrubby, although in many places grasses predominate. On the lower floors, sclerophyll elements strongly penetrate in the North and deciduous elements in the South.

Flora surveys were carried out during the field surveys, identifying a total of 89 species of vascular terrestrial flora. By origin, 83 of the floral species detected are native and 6 are adventitious; 19 of the 83 are endemic. Four species are trees, 40 are shrubs (one of them is parasitic), 37 are perennial herbs and 8 are annual herbs. Two species classified in special conservation categories[1] were identified: the sandillon (*Eriosyce aurata*), classified as Vulnerable and the frangel (*Kageneckia angustifolia*) classified as Near Threatened (DIA, 2019).

Figure 20.1: Rocín River Environment

Source: Los Andes Copper, 2019

Fauna

The Project area is within the Mediterranean Zone of Chile, which includes a significant human population. Since the 1600's humans have been the main source of change for natural habitats, with significant portions of land converted into agricultural, livestock and urban and industrial development.

The fauna of the Mediterranean region of central Chile is characterized by a high number of endemic species. Previous baseline studies (DIA, 2019) have identified at least 77 species, 22 of which are classified under some category of conservation. Only 3 of those are considered

[1] Conservation Categories: Extinct (EX); Extinct in the Wild (EW); Critically Endangered (CR); Endangered (E); Vulnerable (VU); Near Threatened (NT); Least Concern (LC); and Insufficient Data (DD).

Vulnerable, these being *Rhinella atacamensis* (Atacameño toad), *Vultur gryphus* (Andean condor) and *Lama guanicoe* (Guanaco).

Other species worth mentioning include amphibians (*Alsodes nodosus* classified as Near Threatened), reptiles (*Liolaemus nitidus, Liolaemus belli, Callopistes maculatus* classified as Near Threatened), birds (*Merganetta armata* – classified as Near Threatened), and mammals (*Puma concolor* and *Leopardus colocolo*, also classified as Near Threatened).

Recent findings made by independent researchers have identified the potential presence of an Endangered species, the *Leopardus jacobitus* (Andean Cat). These and other findings will be reviewed and assessed during the baseline studies to be conducted as part of the future EIA process.

20.4.11 Archaeology

Initial field surveys conducted as part of the drilling program and for the environmental licence determined the absence of archaeological or heritage remains in the Project area. Additional field surveys will be conducted when the Project moves to feasibility. Designs will avoid direct impacts to archaeological sites, if any, where possible. Where impacts cannot be avoided, the identified site will be studied by a professional archaeologist and removed for archiving, if appropriate, in coordination with government authorities (*Consejo de Monumentos Nacionales*, CMN).

Even in the absence of specific archeological sites, the Rocín valley is among several valleys that have historical value because the Liberation Army (the Argentine-Chilean army that fought in 1818 for the Chilean independence from the Spanish empire) crossed from Argentina to Chile through these valleys in the Valparaíso Region, including the Rocín River valley.

20.5 Waste Management and Disposal

The waste rock dump and tailings designs and management approach were developed in the PFS and are described in Section 18 of this TR. From an environmental perspective, the following considerations are significant:

- Dry stacked, filtered tailings at 15% humidity co-mingled with waste rock will provide good physical and chemical stability. This approach will reduce the possibility of acid rock drainage (ARD), will avoid water retention and potential risks of liquefaction and seepage, and will provide a smaller footprint compared to a traditional tailings facility.
- Contour channels will intercept water from contacting the East and West waste rock dumps, significantly reducing the possibility of ARD given the potential acid generating geochemistry described in Section 20.4.6 of this TR.

20.6 Water Management

The water management design and technical approach were developed in the PFS and are described in Chapter 18 of this TR. From an environmental perspective, a philosophy of avoidance and minimization of impacts has been followed resulting in designs for water supply and water management summarized below.

20.6.1 Water Supply

A desalination plant and pipeline owned and operated by a third party will provide 100% of the make-up water required by the Project (estimated to be 271 L/s, with a maximum of 300 L/s). The pipeline will convey desalinated water from Papudo, on the coast in the Valparaíso Region, to Cabildo and then to site. No continental water will be used by the Project and, therefore, there will be no impact on availability of water to downstream users of the Rocín River in the Putaendo valley. A Los Andes Copper owned and operated desalination plant has been studied as an alternative for the same purpose.

20.6.2 On Site Water Management

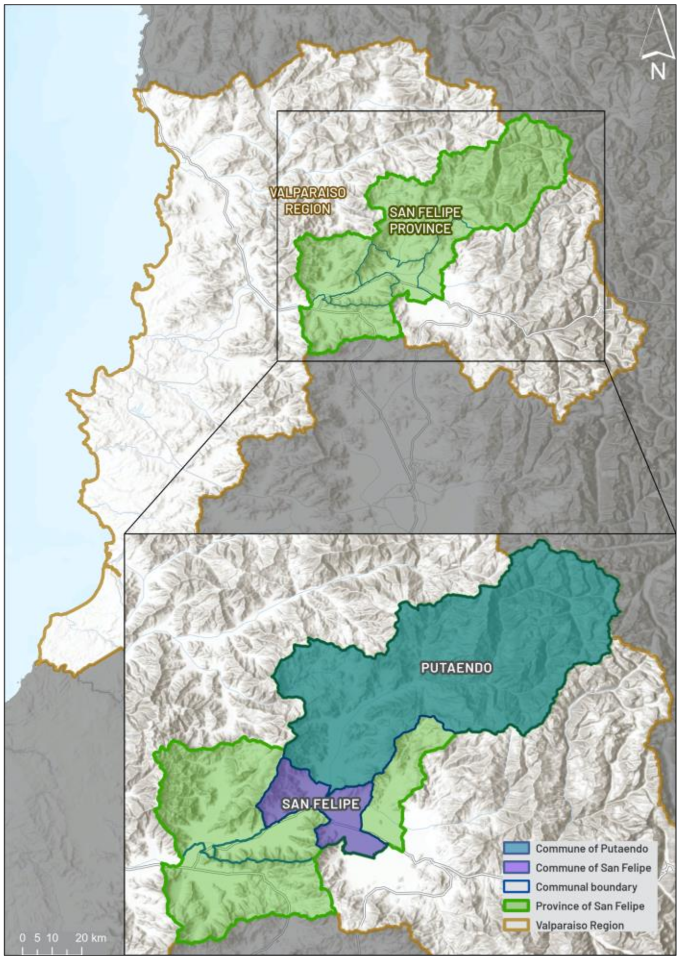
The Project design has planned to avoid and minimize impacts to downstream users. From a water quality perspective, this has resulted in the following design features:

- Water collected above 1,935 masl and whose course is not affected by the Project infrastructure, flows through contour channels and is conveyed downstream to be discharged at the confluence of the Rocín River with the Hidalgo River.
- Rain water that falls on the infrastructure and operations areas (roads, buildings, platforms and concentrator areas) located below the contour channels (i.e. contact water), is conveyed through channels to a central collector whose final destination is the contact water treatment plant located downstream of Project site close to the confluence of the Rocín and Hidalgo Rivers.
- An underground water quality management strategy (described in Section 18.7.2.1) which includes well pumps upstream and along the seepage control wall, and monitoring wells downstream of the wall.

20.7 Social Considerations

The Project is located in the Putaendo Municipality, San Felipe Province, Valparaíso Region, in Central Chile. Figure 20.2 shows the administrative boundaries.

Figure 20.2: Regional Administrative Boundaries



Source: Los Andes Copper, 2023

According to the National Institute of Statistics (INE), the municipality of Putaendo is a mostly rural area with only 40% of the population living in urban areas (INE, 2017). The town of Putaendo is the only urban locality in the area, the town is the centre for the supply of goods and services. It is also a historical centre. The villages located in the Putaendo valley are characteristic of rural areas in the Andean foothills. The essentially rural character of the area is directly related to the importance of agricultural in the local economy. Fruit production is the most important, with livestock also playing a significant role.

No human settlements exist in the area where the mine and plant will be located, the closest village is Los Patos approximately 25 km downstream from the mine site. According to the 2017 census conducted by the INE, the Municipality of Putaendo has a population of 16,754 inhabitants, with a 50/50 female/male ratio. Putaendo has 0.92% of the population of the Valparaíso Region and 0.09% of the country's population. The average age in Putaendo is 37.9 years (35 years is the country average), with 30.5% of the population under 14 years old and 22.9% above 65 years old. Seven percent of the population identify themselves as indigenous and 2% are migrants, similar levels to the Valparaíso Region but significantly lower than the national averages of 13% (indigenous) and 4% (migrants).

Primary and secondary education covers 95% of the targeted population in Putaendo, with 19% of them continuing to college education (compared to 32% regional, and 31% national) and 78% graduating. In terms of employment, 66% of the work force in Putaendo is employed in the tertiary sector (services), 5% in the secondary sector (industry and manufacturing) and 29% in the primary sector (agriculture, mining and livestock) with agricultural production of fruits such as peaches, walnuts and apricots as the main activity (PLADECO, 2015).

Putaendo has a significant part of its population living in conditions of vulnerability. Multi-dimensional poverty in the municipal area is 33.8% (Casen, 2017), significantly higher than multidimensional poverty level at the Region of Valparaíso which is 19.0% (Casen, 2017).

20.7.1 Community Relations and Outreach Activities

The Project team put forward a Vision for the Vizcachitas Project that seeks to insert itself in the territory by reinforcing the compatibility of mining with the development of the local community and its current economic activities. The Project team has a long term view of the territory and seeks to play a key role in the harmonious and sustainable development of Putaendo.

Three key components were identified:

- Development and local value: Los Andes Copper and the Vizcachitas Project are well positioned for the next phases of the environmental permitting process, with high commitment to people, developing a good neighbour reputation, and with the potential to responsibly add significant value to the community.
- Environmental protection: Los Andes Copper and the Vizcachitas Project are focused on developing a sustainable mining activity that ensures the protection of people, biodiversity and ecosystems, and a water management approach with no impact on the basin.
- Participation and social engagement: In addition to carrying out the steps required to facilitate social integration, Los Andes Copper seeks to advance the Vizcachitas Project with the participation of the people in the territory.

The Project has identified a wide range of stakeholders at the local, regional and national level, and maintains a Stakeholders Map (digital and interactive) documenting this process. Key to this process is the permanent presence of the Project team in the area, and a Community Affairs office in Putaendo, available to the public, that is used as a base for outreach activities.

Input and feedback from stakeholders has allowed the Project team to develop and implement a robust Community Relations strategy supported by the following pillars and related actions:

- Water scarcity: In a context of climate change at the global level and a mega drought at the national level. Action: a water management approach that does not alter the quality or quantity of water resources in the basin
- Biodiversity conservation: Concerns about possible alteration of native flora and fauna. Action: monitoring and protection of biodiversity present in the Project area
- Heritage and local way of life: Appreciation of the traditional way of life and local identity. Action: activation of a heritage agenda and cultural identity for the area of impact
- Territorial welfare: High rates of multi-dimensional poverty and vulnerability. Action: promote an inclusive development horizon that allows improvements in the well-being in the area of interest
- Local productive development: Absence of a joint vision on how to put in place an agenda of growth and development. Action: Develop an ongoing dialogue on development together with stakeholders in the community.

Outreach activities occur on a regular basis, including interviews with representatives of organizations, local government, community leaders and members of the public to receive feedback and continually update them on the Project activities. Open houses and community presentations are conducted regularly, including regular interviews on local radios and TV channels.

A formal process of community participation and engagement was carried out in November 2020, with the aim of informing and clarifying doubts about the PFS drilling campaign. Four face-to-face activities were carried out in the towns of Casablanca, El Tártaro, Piguchén and Putaendo (urban), and one activity was carried out by video-conference due to COVID-19 restrictions.

During 2022 the first version of the Vizcachitas Mining Open Call Fund was implemented, with 97 applications from social organizations and entrepreneurs in the municipality of Putaendo. The aim was to promote local development and enhance the participation of female entrepreneurs. Also in 2022 the company's community team worked with the Association of Small Farmers of Putaendo and the Ministry of Agriculture of Chile on the design of a greenhouse project to produce lettuce. The pilot project is expected to be implemented during 2023.

The Project has established several communications channels such as a bi-monthly newsletter, hot line, WhatsApp account, webpage, social media presence (Facebook, Instagram, Twitter and LinkedIn) and a permanent presence in the media (newspapers, radio stations and YouTube).

20.7.2 Indigenous Communities

No Indigenous communities exist or are formally recognized by the National Indigenous Development Corporation (CONADI) in the Project area.

20.8 Closure Planning

In accordance with Chilean Law N° 20.551 modified by Law N° 21.169, a closure plan, cost estimate and financial warranties must be submitted to SERNAGEOMIN. SERNAGEOMIN will approve the closure plan and cost estimates after the approval of the EIA and sectorial permit processes, but prior to the start of construction.

20.8.1 Closure Measures per Installation

The Project closure plan will be designed to ensure long term stability of physical and chemical components of the site. Closure activities for infrastructure owned by third parties such as desalination plant, pipeline and transmission lines, is not considered in this TR.

Specific closure items include:

Mine:

- Access to the truck shop, explosives plant, spare parts warehouse and administration offices will be closed
- De-energization of facilities, dismantling and removal of mechanical and electrical equipment from the remaining works
- Mine infrastructure and buildings, equipment and machinery will be dismantled and or demolished
- For the pit, East and West dumps, dry stack tailings facility, river tunnel and East and West contour channels, dikes will be built and slopes will be stabilized
- Demolition of above ground concrete buildings
- Signage and protection of remaining structures will be installed to warn about the danger of crossing the berms
- Periodic monitoring will be carried out of effluents and infiltrations that may occur
- Leveling and cover of works to restore the topography to as close as possible to the natural, pre-operational environment
- Profiling of access roads and surfaces in general
- Dismantling and removal of electric transmission lines on site
- Monitoring of channels around the waste dumps.

Plant:

- Plant infrastructure and buildings, equipment and machinery will be dismantled and or demolished
- Signage and protection of remaining structures will be installed, to warn about the danger of crossing the berms
- De-energization of facilities, dismantling and removal of mechanical and electrical equipment from the plant, including the desalination pipeline from Cabildo to site
- A cover of sand and borrowed rock material will be placed on top of the plant location
- Wall protection of the dry stacked tailings deposit and options for revegetation
- Ongoing costs will include supplies for a water treatment plant and periodic monitoring of effluents and infiltrations that may occur.

20.8.2 Mine Closure Costs

The closure costs presented in Table 20.13 are a preliminary estimate, calculated in US$, based on reference costs from mining operations of similar size in Central Chile.

Table 20.13: Closure Costs

Area	Activity	Unit	Qty	Unit Price, US$	TOTAL US$
Plant	Profiling of access roads and surfaces in general	ha	120	30,800	3,696,000
Plant	Dismantling of all facilities (buildings, equipment, machinery, removal of materials and spare parts)	m^2	1,200,000	50	60,000,000
Plant	Dismantling and removal of power transmission lines on site	km	61	200,000	12,200,000
Plant	Demolition of facilities that are constructed with surface concrete	ha	120	2,000	240,000
Plant	Periodic (annual) monitoring of biodiversity, physical & chemical stability, effluents and infiltrations	year	10	150,000	1,500,000
Plant	Treatment of dumps and mineral stockpiles	m^3	2,500,000	3	7,718,306
Plant	Removal and final disposal of non-hazardous, household and hazardous waste	t	4,000	500	2,000,000
Plant	Demolition, dismantling and removal of contaminated equipment	m^2	60,000	530	31,800,000
Plant	Tailings – wall protection, revegetation and studies	m^3	718,400	20	14,368,000
Plant	Desal water pipeline decommissioning Cabildo-Site	km	91	40,000	3,620,000
Mine	Pit contour channel	m	3,500	4,500	15,750,000
Mine	O&M costs WTP and effluents monitoring	year	10	150,000	1,500,000
				Total direct costs	154,392,306
				Indirect costs (25%)	38,598,076
				Contingencies (15%)	28,948,557
				Sub total	221,938,939
				VAT (19%)	42,168,399
				TOTAL CLOSURE COSTS	**264,107,338**

21. CAPITAL AND OPERATING COSTS

A variety of sources were used to estimate capital costs, including first principles calculations, quotations for equipment and factoring of construction costs from similar projects. Cost estimates are assumed to have a ±25% margin of error, which is in agreement with the Association for the Advancement of Cost Engineering International (AACEI) Class 4 estimate.

The capital and operating cost estimates (Capex and Opex) were prepared in United States Dollars (US$ or USD). Costs are estimated as of the Effective Date of this Technical Report. Cost projections are presented on a 2023 real dollar basis.

21.1 Capital Cost Estimate

The capital cost estimates are composed of the following:

- Direct cost of construction and assembly: Acquisition of equipment, labour, auxiliary equipment for construction and building materials are considered.
- Indirect project costs: Transportation and equipment insurance, general spare parts, vendor's representatives, engineering, EPCM, start-up and owner costs are considered.
- The contingency estimate was based on Direct Costs plus Indirect Costs.
- Sustaining capital is defined as the costs required to maintain operations and may include capital spent on expansion or new infrastructure items.
- Deferred capital is investment required to complete an expansion of the mine facilities and process plant during the life of the project.

Direct costs were estimated using:

- Material take-offs (MTO) based on layouts, process flow diagrams and topographic information
- Quotes from vendors
- Historical data
- Allowances for similar projects.

Table 21.1 presents the estimated initial, sustaining and deferred capital costs in US$. Estimates were prepared in US$, except for the access roads to mine phases (prepared in CLP) and milling/ classification equipment supply (prepared in EUR).

VAT payable on the Capex can be reimbursed after a pre-determined time. A finance allowance is made for reimbursable VAT over 6 months at a rate of 3.5%.

Table 21.1: Capital Cost Estimate (US$ Million) 2023 Real

	Direct Initial Capital (US$ Thousand)	Sustaining & Deferred (US$ Thousand)
MINE CAPEX		
Rope & Hidr. Shovel (18 months)	36,400	76,026
Front Loader & Perf (12 months)	17,664	64,907
Capex & Ancillary Eq. (rtw)	73,194	355,009
Dewatering	9,511	-
Pre-stripping Contractors	115,902	-
Deferred Stripping Contractors	18,647	305,799
Earlyworks (Main Access Roads)	85,086	-
Access Roads to Mine Phases	79,646	99,852
Sub-total Mine Capex	436,050	901,592
PLANT & INFRASTRUCTURE CAPEX		
Dry Area - 2000		
Primary Crushing	45,108	-
Secondary Crushing	68,480	-
Storage & Reclaim	29,180	-
Tertiary Crushing	75,760	-
Ancillary Facilities Dry Area	26,588	-
Wet Area - 3000		
Grinding - Milling/ Classification	55,257	-
Ore Thickening	14,754	-
Ore Launder to Flotation	7,482	-
Rougher Flotation	39,745	-
Secondary Grinding (regrind)	64,327	-
Cleaner Flotation	26,028	-
Molybdenum Separation	27,165	-
Plant Services	14,022	-
Ancillary Facilities Wet Area	5,358	-
Tailings Filtration/ Reclaim & Water Treatment - 4000		
Tailings Handling	160,131	-
Tailings Waste Area	14,569	-
Ancilliary Facilities Tailings Area	-	-
Leackage Protection Wall & Water Treatm	5,338	-
On-site Infrastructure - 5000		
Buildings	10,130	-
Electrical Supply On-site	28,081	-
Control & Communication Systems	11,392	-
On Site - Urbanization - Site Preparation	75,220	-
Rocin River Diversion Works	102,740	40,391
Services	22,118	-
On-site Ancillary Facilities	29,783	12,581
Mobile Equipment	1,493	-

	Direct Initial Capital (US$ Thousand)	Sustaining & Deferred (US$ Thousand)
PLANT & INFRASTRUCTURE CAPEX (CONTINUED)		
Off-site Infrastructure - 6000		
Desal Water Supply	129,541	3,803
Off-site Power Supply And Power Transmi	75,396	0
Off-site Roads & Water Diversion	33,084	0
Concentrate Transport & Handling	4,900	0
Off-site Systems	1,182	0
Sub-total plant capex	**1,204,353**	**56,775**
TOTAL DIRECT	**1,640,403**	**958,367**
INDIRECT	**454,104**	**311,315**
CONTINGENCY	**346,449**	**224,009**
TOTAL CAPITAL	**2,440,955**	**1,493,691**

Table 21.2 presents the main basis for the estimate by area.

Table 21.2: Estimate Basis by Area

Area	Cost Estimation	
	MTO	Price
Mine		
Mine Works	Calculated	Quotations/ Database
Mine Equipment	Calculated	Quotations
Workshops	Expert Judgement	Database
Infrastructure	Expert Judgement	Calculated/ Factorized
Building	Expert Judgement	Database
Power Supply	Factorized	Database
Process Plant		
Process Equipment	Calculated	Quotations/ Database
Infrastructure	Factorized	Calculated/ Database
Building	Factorized	Calculated/ Factorized
Civil Works	Factorized	Database
Power Supply	Calculated	Calculated/ Factorized
Tailings Storage Facility		
Civil Works	Factorized	Database
Piping	Factorized	Database
Infrastructure	Factorized	Database
Contour Channels	Factorized	Database
Building	Factorized	Database
Support Equipment	Factorized	Database
Power Supply	Factorized	Factorized
Infrastructure		
Power Supply	Calculated	Database
Mine/ Plant Water Supply	Calculated	Quotations/ LOI
Contour Channels	Expert Judgement	Database
Building	Factorized	Database
Support Equipment	Factorized	Database
Infrastructure	Expert Judgement	Expert Judgement

Table 21.3 presents the estimated mine fleet purchase costs.

Table 21.3: Mine Equipment Purchase Costs (US$ Thousand)2023 Real

Item	Cost	Source
Shovel KMS PC 8000 55 yd³	11,643	PC 8000, RTW Vizcachitas Quote - Aug 2022
Shovel P&H 4100XPC 73 yd³	35,938	P&H 4100XPC AC Vizcachitas Quote - Aug 2022
Loader P&H L-2350 50 yd³	13,220	Le Torneau 2350 with Tires & IT technology
KMS 930E-5	6,106	Quote KMS930E5 Conventional + Tires + Hopper
Drilling Drill P&H320XPC	8,865	Quote Vizcachitas Aug-2022
Drilling Drill KMS ZR122	6,281	Quote Vizcachitas Aug-2022
Drilling Drill KMS ZT44	2,162	Quote Vizcachitas Aug-2022
Bulldozer D475A-5E0	1,800	Quote Vizcachitas Aug-2022
Wheeldozer WD 900-3A	1,928	Quote Vizcachitas Aug-2022
Grader GD 825A-2 Engine	2,798	Tetra Tech Database
Water Truck HD 785-7 WT	1,811	Quote Vizcachitas Aug-2022
Water Tank	430	Tetra Tech Database
CFWA600	1,600	Tetra Tech Database
Grove	595	Quote Vizcachitas Aug-2022
Excavator PC300LC-8	405	Quote Vizcachitas Aug-2022
Atlas QLT M10 Lighting Plant	1.5	Tetra Tech Database
Lowboy Truck	2,460	Low Bed Quotation Consultant

Table 21.4 presents the unit construction costs used for the process plant and infrastructure capital cost estimates. The cash flow for the mine fleet Capex considers the contractual terms provided in quotations from suppliers. In most cases, 20% of the purchase cost is required with the order placement with the balance (80%) payable when the equipment is ready-to-work (RTW). For the mine trucks there is a 12 month period between order and RTW, for the rope shovels the period between order and RTW is 18 months.

Table 21.4: All-inclusive Construction Cost Unit Prices

Area	Unit of Measure	Price
Concrete	US$/m³	650
Structural Steel	US$/t	7,000
Piping	US$/kg	6
Rock Excavation	US$/m³	15
Soil Excavation	US$/m³	5
Filling	US$/m³	10

21.1.1 Indirect Costs

Lump sum allowances or factors have been used to calculate indirect costs, as applicable for a PFS. At this level of study many of the sourcing and contract strategies are not fully defined, hence, it is reasonable and customary that assumptions are made based on experience with similar projects. Table 21.5 presents a breakdown of the indirect costs.

Table 21.5: Indirect Costs

Description	Value (US$ Thousand)
Freight & Insurance	41,010
Import Duties	0
Spare Parts	19,685
Vendor Representatives	8,202
PFS-FS-EPCM	327,793
Start Up	8,202
Owners Cost	49,212
Total Indirect Cost	454,104

21.1.2 Contingency

Contingency is an allowance to cover unforeseeable costs that may arise during Project execution which are within the scope of work but cannot be explicitly defined or described at the time of the estimate owing to lack of information. It is assumed that contingency will be spent. Contingency does not cover scope changes or exclusions.

The contingency is based on the level of definition that was used to prepare the estimate. After an assessment by Tetra Tech of Project confidence versus uncertainty by area, a 15% contingency factor has been added to the initial capital cost for all items (direct and indirect).

21.1.3 Accuracy

This estimate has been developed to a level sufficient to assess/evaluate the Project concept, development options and the overall potential Project viability. After incorporating the recommended contingency, the capital cost estimate is considered to have a level of accuracy of ±25%. This is based on the level of contingency applied, the confidence levels of the estimators and engineers in their respective estimates and an assessment comparing this estimate to standard accuracy levels for PFS estimates.

21.1.4 Estimate Exclusions

The following items are not included in the capital cost estimate:

- Foreign currency exchange rate fluctuations

- Interest and financing costs, except VAT financing
- Price escalations beyond 2023 real terms
- Risk due to political upheaval, government policy changes, labour disputes, permitting delays, weather delays, or any other force majeure occurrences.

21.2 Operating Cost Estimate

Operating costs have been estimated for the operating areas of Mining, Process Plant, Infrastructure and Administration. Costs are reported under sub-headings related to the function of each area.

The operating cost estimates are based on energy prices of US$65/MWh for electricity and US$0.60/L for diesel fuel. Table 21.6 summarizes the average unit operating cost by area for the first 8 years of operation and for the LOM. Labour costs for the mine and process plant consider only up to Superintendent level; higher positions are considered to be Administration costs.

The operating costs are considered to have accuracy of ±25%, based on the assumptions listed in this section. All unit operating costs are expressed in terms of tonnes processed.

Table 21.6: Unit Operating Costs (Real$_{2023}$)

Description	UoM	First 8 years	LOM
Mining	US$/$t_{proc}$	3.98	5.02
Processing	US$/$t_{proc}$	3.85	3.90
Surface Infrastructure	US$/$t_{proc}$	1.18	1.20
General & Administration	US$/$t_{proc}$	0.30	0.30
Total Operating Cost	US$/$t_{proc}$	9.32	10.41

In addition to the operating costs above, materials stockpiled and moved to and from stockpiles are subject to an additional charge of US$0.70/$t_{processed}$. The effective rehandling cost over the LOM is US$0.08/$t_{processed}$.

21.2.1 Mine Operating Cost

Mine operating costs are based on owner mining and cover the following:

- Pit operations, drilling, blasting, loading, and hauling
- Construction and maintenance of mine haul roads, sumps, and safety berms
- Operating and maintenance labour
- Mine department supervision and technical services
- Crushing waste rock to supply aggregate for road surfacing, blast hole stemming and other earthworks required for day-to-day mining operations.

The mine production schedule and equipment unit productivity estimates were used to calculate operating shifts and manpower requirements, which in turn were used to derive mine operating costs. Exploration costs are not included in the operating cost estimates.

Unit operating costs for major equipment include labour, power, diesel, lubricant consumption, tyres, materials, spare parts, third party services and other costs. The operating costs were adjusted for local labour rates and supply costs, also tracking recent experience for operations with similar fleets.

Table 21.7 and Table 21.8 provide the mine unit operating costs by activity and by expense item for the first 8 years of operation and LOM.

Table 21.7: Mine Unit Operating Costs by Activity

Description	UoM	First 8 years	LOM
Drilling (c/ Precut)	US$/$t_{proc}$	0.25	0.25
Blasting	US$/$t_{proc}$	0.60	0.65
Loading	US$/$t_{proc}$	0.55	0.65
Transportation	US$/$t_{proc}$	1.57	2.51
Earth Moving Equipment	US$/$t_{proc}$	0.52	0.46
Dewatering	US$/$t_{proc}$	0.01	0.01
Support Services	US$/$t_{proc}$	0.21	0.20
Administration Mine	US$/$t_{proc}$	0.27	0.29
Total	US$/$t_{proc}$	3.98	5.02

Table 21.8: Mine Unit Operating Costs by Expense Item

Description	UoM	First 8 years	LOM
Materials	US$/$t_{proc}$	0.74	0.86
Fuels	US$/$t_{proc}$	0.66	1.00
Third Party Services	US$/$t_{proc}$	0.54	0.71
Labor	US$/$t_{proc}$	0.42	0.39
Electricity	US$/$t_{proc}$	0.03	0.05
Water	US$/$t_{proc}$	0.07	0.06
Maintenance & Repair	US$/$t_{proc}$	1.50	1.91
Dewatering	US$/$t_{proc}$	0.01	0.01
Others & Fixed	US$/$t_{proc}$	0.01	0.01
Total	US$/$t_{proc}$	3.98	5.02

21.2.2 Process Plant, Infrastructure and Administration Operating Cost

Process plant operating costs were developed for the following unit operations:

- Primary crushing and stockpiling
- Grinding
- Copper-molybdenum bulk flotation
- Molybdenum flotation
- Thickening of concentrates and tailings
- Filtration
- Concentrate handling.

Unit operating costs include labour, reagents and consumables, maintenance and spare parts, third party services and other costs. These operating costs were adjusted for local labour rates and supply costs and recent experience for operations with similar equipment. The operating costs were factored as US$/$t_{processed}$ rates for the full capacity steady state operation of the planned process plant. Table 21.9 and Table 21.10 show operating costs for the process plant by activity and expense item, respectively.

Table 21.9: Process Plant, Infrastructure and G&A Operating Costs by Activity

Description	UoM	LOM
Crushing	US$/$t_{proc}$	0.64
Grinding & Flotation	US$/$t_{proc}$	2.32
Tailings	US$/$t_{proc}$	0.94
On-site Infrastructure	US$/$t_{proc}$	0.29
Off-site Infrastructure	US$/$t_{proc}$	0.33
Desalination Plant & Water Pumping	US$/$t_{proc}$	0.58
G&A	US$/$t_{proc}$	0.30
Total	US$/$t_{proc}$	5.40

Table 21.10: Process Plant, Infrastructure and G&A Operating Cost by Expense Item

Description	UoM	LOM
Labor	US$/$t_{proc}$	0.52
Energy	US$/$t_{proc}$	1.39
Reagents & Consumables	US$/$t_{proc}$	1.37
Maintenance	US$/$t_{proc}$	0.63
Services	US$/$t_{proc}$	1.49
Total	US$/$t_{proc}$	5.40

21.2.3 Cash Cost Metrics

Cash cost metrics, as defined below, have been calculated per pound copper produced and are presented in Table 21.11. C1 cash costs include site operating costs (mining, processing), but exclude indirect, head office G&A and exploration expenses. C3 cash costs include C1 costs, plus all surface, infrastructure, site G&A, royalty and indirect expenses. The All-in Sustaining Cost (AISC) includes all cash costs, sustaining capital and product selling expenses, but excludes head office G&A and exploration expenses.

Table 21.11: Average First 8 Years and LOM Cash Costs

Category	UoM	First 8 years	LOM
Sales Income	US$/lb Cu	4.10	4.16
Selling Expenses	US$/lb Cu	(0.60)	(0.61)
Gross Revenue	**US$/lb Cu**	**3.50**	**3.55**
Mining Cost	US$/lb Cu	(0.47)	(0.71)
Processing Cost	US$/lb Cu	(0.46)	(0.54)
C1 Cost	**US$/lb Cu**	**(0.93)**	**(1.25)**
Surface Infrastructure	US$/lb Cu	(0.14)	(0.17)
Indirects	US$/lb Cu	(0.04)	(0.04)
Royalty	US$/lb Cu	(0.09)	(0.10)
C3 Cost	**US$/lb Cu**	**(1.19)**	**(1.56)**
Sustaining Capex	US$/lb Cu	(0.34)	(0.17)
All-in Sustaining Costs	**US$/lb Cu**	**(2.13)**	**(2.35)**
AISC Margin	%	48%	44%

Notes: Sales income include by-products. AISC include all cash costs, sustaining capital and selling costs, but excludes head office G&A, exploration expenses.

22. ECONOMIC ANALYSIS

This PFS economic evaluation is based on a production plan that includes Mineral Resources in the Measured and Indicated categories only. The mine scheduling software applied a zero grade to all Inferred Resources.

This section contains information and assertions that are forward-looking in nature, as defined under Canadian securities law. Forward-looking statements pertain to the Project economic and physical parameters, estimates of the Mineral Resources and Mineral Reserves, the price and timing of the Project development, the proposed mine plan and mining methods, mining recoveries due to dilution, processing method and production rates, and anticipated metallurgical recoveries, among other things.

The calculated net present value (NPV) results from the above forward-looking statements as well as estimates of infrastructure requirements; capital, operating and sustaining costs estimates; commodity prices applied; and the Project location and jurisdictional factors such as taxation and royalty.

22.1 Basis of the Financial Model Estimate

The NPV was calculated using a discounted cash flow model in MS Excel 365. Using the mine plan as input, the model calculated annual quantities of metal production, the associated revenues and the capital, operating and other costs to sustain production in annual periods. The financial model is in 2023 US$ constant rates (real terms).

22.1.1 Schedule

The LOM is 26 years, with 3 years of pre-production activities. The annual period when capital expenditures are initiated is defined as Year -3. The NPV has been calculated annually from the year of initial capital expenditures.

A construction period of 3.25 years is expected; mined material stockpiling starts after the first year of construction in Year -2. The first 3 years are denoted as Years -3, -2 and -1, respectively. The first concentrate production is in Year 1 commencing in quarter 2 (Q2).

The financial model serves as an indication of the economic potential of the Project to help with investment decisions and does not place the Project inside a projected calendar schedule.

22.1.2 Financial Model Parameters

The base case discount rate applied is 8%. Chile is a politically stable country and the Vizcachitas Project has the same technical features as many other projects and operations in Chile.

The exchange rate is not a direct input in the financial model since essentially all estimates were developed in US$.

The analysis is based on a cash flow estimate; the actual economic results might vary from the calculated results based on the prevailing economic conditions. Economic parameters used for the evaluation are shown in Table 22.1.

Table 22.1: Main Economic Parameters

Description	UoM	Value
Cu Price	US$/lb	3.68
Mo Price	US$/kg	12.9
Ag Price	US$/oz	21.79
Currency Fluctuation	---	n/a
NSR	%	2

22.1.3 Working Capital

Receivables outstanding were based on contract terms provided with a standard 30-day payables rate applied to applicable operating costs. The receivables outstanding pipeline was financed at an annual rate of 3.5%, as summarized in Table 22.2.

Table 22.2: Working Capital Assumptions

Description	UoM	Value
Receivables Outstanding	days	45
Payables Outstanding	days	30
Receivables Finance Rate	% per annum	3.5%

22.1.4 Taxes and Royalties

The following section summarizes the Chilean income tax code applicable to mining companies.

22.1.4.1 Income Tax

The corporate income tax legislation was modified in 2014, creating two tax treatment options: a semi-integrated system and an attributed income system. The semi-integrated system is applicable to the Project.

Companies and individuals with residence or domicile in Chile are subject to income tax on their worldwide income; non-resident entities and individuals are taxed only on their Chilean source income. Companies organized under Chilean Law are deemed residents of Chile.

Corporate Income Tax (*Impuesto de Primera Categoría*, in Spanish) is imposed on income derived from investments, commercial, industrial and mining activities. The corporate tax rate is 27%, except where the company is classified as a small or medium company, in which case the applicable tax rate is 25%. The 27% corporate income tax rate will be applicable to the Project.

This annual tax is generally applied to the net taxable income determined from full accounting records. The net taxable income is equal to the gross income less direct costs and necessary expenses incurred to produce that income.

Withholding Tax (*Impuesto Adicional*, in Spanish) operates as a withholding tax and affects, among other incomes, Chilean-source income withdrawn or remitted abroad to non-residents or non-domiciled individuals, companies or other entities organized abroad with or without a permanent establishment in Chile in the form of branches, offices, agencies or representatives. Dividends paid to shareholders not domiciled or resident in Chile are subject to withholding tax at a rate of 35%.

The Chilean income tax system is considered to be an integrated system; the Corporate Income Tax paid by Chilean companies can be used as a credit against Final Taxes (Individual Tax and Withholding Tax) applicable to dividends received by the final shareholders (Chilean individuals or non-Chilean residents). Dividends distributed between Chilean companies are not subject to taxation.

Following the tax reforms passed in Chile in 2014 and 2020, as a general rule, the amount of Corporate Income Tax creditable against the Individual Tax and the Withholding Tax is limited to 65%. However, in the case of foreign shareholders or partners that have residence in a Tax Treaty country, the corporate tax is 100% creditable against the Withholding Tax applicable on dividend distributions. The 100% credit applies even if the relevant Tax Treaty is not still in force, provided it was signed prior to January 1, 2020.

The Chilean Congress is currently discussing a tax reform that may disintegrate the Corporate Income Tax system. According to this proposal, the Corporate Income Tax will not be a tax credit against Final Taxes and dividend distribution would be subject to a 22% Withholding Tax. However, if the beneficiary of the dividend distribution is a resident of a Tax Treaty country, the integration will be maintained.

The financial model for this PFS was based on the laws currently in effect. The after-tax economic evaluation shown in this PFS assumes that only the 27% Corporate Income Tax would be applicable.

22.1.4.2 Mining Royalties

All mining properties are subject to a Mining Royalty Tax (*Impuesto Específico a la Minería*, in Spanish). This tax was introduced in 2006 and amended in 2010 and is applied against the collective operating (mining) profits of all the operating units. The tax rate is calculated on a step scale based on fine copper equivalent sales:

- 0 t to 12,000 t copper equivalent: No tax applied
- 12,001 t to 50,000 t copper equivalent: 0.5% to 4.5% of the Mining Operating Income according to the scale shown in Table 22.3.

- More than 50,000 t copper equivalent: A scale applies that starts at 5% of the Mining Operating Income for Mining Operating Margins less than or equal to 35%, and increases to 34.5% for Mining Operating Margins up to 85%. This scale is shown in Table 22.4. A tax of 14% is applicable for Mining Operating Margins that exceed 85%.

Table 22.3: Mining Royalty Tax Scale (Up to 50,000 t of Equivalent Copper)

CuEq (t)		Marginal tax
From	To	%
0	12,000	0.0
12,001	15,000	0.5
15,001	20,000	1.0
25,001	30,000	2.0
30,001	35,000	2.5
35,001	40,000	3.0
40,001	50,000	4.5

Table 22.4: Mining Royalty Tax Scale (Over 50,000 t of Equivalent Copper)

Operating Profit (%)		Marginal tax
From	To	%
0	35	5.0
35	40	8.0
40	45	10.5
50	55	15.5
55	60	18.0
60	65	21.0
65	70	24.0
70	75	27.5
75	80	31.0
80	85	34.5

The Mining Operating Income on which this tax is applied is determined following specific rules. Certain expenses such as losses from past periods and accelerated depreciation of fixed assets are not allowed for this purpose.

The Mining Operating Margin is determined as a ratio of the Mining Operating Income to the mining operational revenues.

The Chilean Congress is currently discussing a tax reform that may modify the Mining Royalty Tax for large copper mines (over 50,000 t).

The financial model for this PFS was based on the laws currently in effect and does not consider potential modifications to mining taxation.

22.1.4.3 Depreciation

Depreciation on fixed assets, except for land, is tax deductible by the straight-line method based on the useful life of the asset in accordance with the Chilean Tax Service guidelines (*Servicio de Impuestos Internos*, SII) computed on the re-stated value of the assets. However, the taxpayer may opt for accelerated depreciation for new assets when acquired locally, or new or used assets when imported, with a useful life of over 5 years. Accelerated depreciation considers the useful life equivalent to one-third of the normal useful live, eliminating fractions of months. Taxpayers may discontinue the use of the accelerated method at any time but may not return later to the accelerated method. The difference between the accelerated and straight-line methods is that the latter method does not deduct the taxable profit that can be withdrawn by partners or distributed to shareholders. The public document available from SII includes the straight line and accelerated depreciation schedules for different asset categories.

No allowance is made for amortization of intangible assets such as goodwill, patents and trademarks. Depletion is not tax deductible.

A summary of the depreciation periods per capital category is presented in Table 22.5.

Table 22.5: Depreciation Profiles

Capital category	UoM	Value
Normal Depreciation		
Mining Equipment (Shovels, Loaders, Trucks, etc)	years	9
Pre-Stripping, Dewatering, Early Works	years	6
Deferred Stripping & Access Roads to Phases	years	2
Plant & Infrastructure	years	5
River Diversion & Off-Site Infrastructure	years	20
Accelerated Depreciation		
Mining Equipment (Shovels, Loaders, Trucks, etc)	years	3
Pre-Stripping, Dewatering, Early Works	years	2
Deferred Stripping & Access Roads to Phases	years	2
Plant & Infrastructure	years	1
River Diversion & Off-Site Infrastructure	years	6

22.1.4.4 Stock/Inventory

The costing of goods sold and supplies consumed are based on the first-in, first-out (FIFO) basis, although the average method may be elected. The method adopted determines the basis for the valuation of the closing inventory. The valuation method used is, however, adjusted in the manner stipulated for the annual monetary correction procedures.

22.1.4.5 Dividends

Dividends received from Chilean corporations are exempt from First Category Tax. There is no distinction in Chile between dividends and inter-company dividends. A dividend in kind does not exist. Dividends must be expressed in cash, notwithstanding the fact that the company may distribute certain assets corresponding in value to the dividend amount. Stock dividends in the form of bonus shares or increases in the par value of existing shares are not considered income for tax purposes.

22.1.4.6 Interest Deductions

Generally, interest accrued or paid in a financial year is a deductible expense, providing that it has been incurred in connection with loans related to the business.

22.1.4.7 Losses

Losses incurred in the fiscal year are deductible. There is no limit on the carrying forward of losses. If the enterprise has taxable retained profit, losses must be carried back first. There are no loss carry-back provisions, nor is it possible to group profitable and unprofitable affiliates for tax purposes.

22.1.4.8 Foreign Sourced Income

Non-domiciled or non-resident corporations are only subject to income taxes on their Chilean-sourced income. If the domestic corporation receives amounts that exceed the book value of an investment when a foreign subsidiary is liquidated, these funds are considered income subject to regular taxes. From 2012 on, income received or accrued from derivatives such as forwards, futures, swaps and options, by persons or entities without domicile or residence in the country, is not subject to income tax, except income arising from derivatives that are settled by physical delivery of shares or rights in companies incorporated in Chile.

Interest payments to financial institutions not domiciled in Chile are subject to an additional withholding tax of 4%.

22.1.5 After-Tax Analysis

The preparation of a comprehensive after-tax model results from tax planning modelled with the advice of taxation specialists.

22.1.6 Closure Costs and Salvage Value

As explained in Section 20.8.2, the closure costs (presented in Table 20.13), are a preliminary estimate, calculated in US$, based on reference costs from mining operations of a similar scale in central Chile.

Conservatively, no salvage value was assumed for residual infrastructure. No reduction in closure cost provisions were made as a result of a possible residual value. For the mining fleet the sustaining capital was optimized to net off the salvage value to zero.

22.1.7 Residual Value In-Situ

The Measured and Indicated Resources were included in the mine schedule and Inferred Resources were zero-rated. Therefore, no residual value is applicable, as per NI43-101 specification.

22.1.8 Financing

The financial model cash flows are unlevered on the Vizcachitas asset level at 100% equity ownership. Project financing considerations are not included in this valuation.

22.1.9 Inflation

No inflation factors were modelled. The model is presented in 2023 US$ real terms.

22.2 Production Summary

Data from the mine production schedule were used as the basis for the concentrate production as presented in Table 22.6.

22.3 Cash Flow Model

The output from the cash flow model for the LOM (starting in Year -3) is shown in Table 22.7.

Table 22.6: Physical Production

Description	UoM	Total	-3	-2	-1	1	2	3	4	5
Physicals										
Ore Processed	kt	**1,219,932**	-	-	-	33,921	49,640	49,640	49,640	49,640
Cu Head Grade	%	**0.36%**	0.00%	0.00%	0.00%	0.49%	0.53%	0.46%	0.40%	0.42%
Mo Head Grade	g/t	**136**	0	0	0	82	145	152	138	168
Ag Head Grade	g/t	**1.1**	0.0	0.0	0.0	1.3	1.4	1.4	1.2	1.2
Cu Recovery	%	**91.1%**	0.0%	0.0%	0.0%	91.0%	91.4%	90.9%	91.0%	91.2%
Mo Recovery	%	**74.8%**	0.0%	0.0%	0.0%	73.6%	74.8%	73.5%	75.4%	73.8%
Ag Recovery	%	**75.0%**	0.0%	0.0%	0.0%	75.0%	75.0%	75.0%	75.0%	75.0%
Cu In Cu Concentrate	kt	**3,975**	-	-	-	151.87	240.36	207.66	178.94	188.18
Ag In Cu Concentrate	koz	**32,712**	-	-	-	1,033.2	1,650.0	1,704.9	1,440.4	1,447.1
Mo In Mo Concentra	kt	**124**	-	-	-	2.05	5.40	5.55	5.17	6.15
Payable Cu	kt	**3,836**	-	-	-	146.56	231.95	200.39	172.68	181.60
Payable Ag	koz	**29,441**	-	-	-	929.8	1,485.0	1,534.4	1,296.3	1,302.4
Payable Mo	kt	**121**	-	-	-	2.00	5.26	5.41	5.04	5.99

	UoM	Total	6	7	8	9	10	11	12	13
Physicals										
Ore Processed	kt	**1,219,932**	49,640	49,640	49,640	49,640	49,640	49,640	49,640	49,640
Cu Head Grade	%	**0.36%**	0.39%	0.37%	0.36%	0.35%	0.34%	0.35%	0.33%	0.35%
Mo Head Grade	g/t	**136**	165	117	109	98	101	186	130	115
Ag Head Grade	g/t	**1.1**	1.2	1.2	1.2	1.2	1.1	1.2	1.1	1.1
Cu Recovery	%	**91.1%**	91.0%	89.8%	89.2%	89.8%	90.6%	91.2%	91.3%	91.9%
Mo Recovery	%	**74.8%**	73.6%	73.2%	73.3%	74.5%	76.3%	76.5%	75.1%	73.4%
Ag Recovery	%	**75.0%**	75.0%	75.0%	75.0%	75.0%	75.0%	75.0%	75.0%	75.0%
Cu In Cu Concentrate	kt	**3,975**	174.50	165.35	157.26	156.74	152.76	160.69	148.71	158.62
Ag In Cu Concentrate	koz	**32,712**	1,433.9	1,446.2	1,385.3	1,460.2	1,338.7	1,466.6	1,363.2	1,267.9
Mo In Mo Concentra	kt	**124**	6.05	4.27	3.95	3.61	3.83	7.07	4.85	4.19
Payable Cu	kt	**3,836**	168.39	159.57	151.76	151.25	147.42	155.07	143.50	153.07
Payable Ag	koz	**29,441**	1,290.5	1,301.5	1,246.8	1,314.1	1,204.9	1,320.0	1,226.9	1,141.1
Payable Mo	kt	**121**	5.89	4.16	3.85	3.52	3.74	6.90	4.73	4.08

	UoM	Total	14	15	16	17	18	19	20	21
Physicals										
Ore Processed	kt	**1,219,932**	49,640	49,640	49,640	49,640	49,640	49,640	49,620	47,514
Cu Head Grade	%	**0.36%**	0.39%	0.34%	0.31%	0.28%	0.35%	0.34%	0.29%	0.36%
Mo Head Grade	g/t	**136**	154	128	187	101	112	119	85	165
Ag Head Grade	g/t	**1.1**	1.0	1.1	1.1	1.0	1.1	1.2	0.9	0.9
Cu Recovery	%	**91.1%**	91.8%	91.2%	91.3%	89.9%	89.7%	91.0%	92.2%	92.7%
Mo Recovery	%	**74.8%**	74.0%	75.8%	75.4%	74.4%	73.6%	75.5%	75.2%	75.1%
Ag Recovery	%	**75.0%**	75.0%	75.0%	75.0%	75.0%	75.0%	75.0%	75.0%	75.0%
Cu In Cu Concentrate	kt	**3,975**	177.22	156.02	138.85	124.24	156.01	151.85	131.08	158.82
Ag In Cu Concentrate	koz	**32,712**	1,252.0	1,290.8	1,276.2	1,146.4	1,273.4	1,426.5	1,086.6	1,018.3
Mo In Mo Concentra	kt	**124**	5.65	4.81	7.00	3.74	4.09	4.45	3.18	5.88
Payable Cu	kt	**3,836**	171.02	150.56	133.99	119.89	150.55	146.54	126.49	153.27
Payable Ag	koz	**29,441**	1,126.8	1,161.7	1,148.6	1,031.7	1,146.0	1,283.8	978.0	916.4
Payable Mo	kt	**121**	5.51	4.69	6.82	3.64	3.98	4.34	3.10	5.73

	UoM	Total	22	23	24	25	26
Physicals							
Ore Processed	kt	**1,219,932**	49,372	49,640	49,498	45,024	1,825
Cu Head Grade	%	**0.36%**	0.33%	0.34%	0.31%	0.23%	0.26%
Mo Head Grade	g/t	**136**	207	140	159	107	274
Ag Head Grade	g/t	**1.1**	0.8	1.1	1.0	0.8	0.7
Cu Recovery	%	**91.1%**	92.6%	91.0%	91.5%	91.4%	92.7%
Mo Recovery	%	**74.8%**	76.0%	75.4%	77.0%	75.1%	75.0%
Ag Recovery	%	**75.0%**	75.0%	75.0%	75.0%	75.0%	75.0%
Cu In Cu Concentrate	kt	**3,975**	149.34	153.06	138.23	94.16	4.40
Ag In Cu Concentrate	koz	**32,712**	1,011.4	1,304.5	1,241.9	915.8	30.8
Mo In Mo Concentra	kt	**124**	7.75	5.25	6.05	3.62	0.38
Payable Cu	kt	**3,836**	144.11	147.71	133.39	90.87	4.25
Payable Ag	koz	**29,441**	910.3	1,174.0	1,117.7	824.2	27.7
Payable Mo	kt	**121**	7.55	5.12	5.90	3.53	0.37

Table 22.7: Cash Flow Model Output

Description	Total (US$ Thousand)	-3	-2	-1	1	2	3	4	5
Cash Flow									
Copper Sales, Metal Value	32,248,820	–	–	–	1,232,158	1,950,077	1,684,761	1,451,780	1,526,726
Silver Sales, Metal Value	712,790	–	–	–	22,512	35,953	37,149	31,386	31,532
Molybdenum Sales, Metal Value	3,526,063	–	–	–	58,424	153,566	157,724	147,133	174,786
Selling & Payability Expenses	(4,590,036)	–	–	–	(164,881)	(268,953)	(236,332)	(205,040)	(218,069)
Finance Advance	(137,546)	–	–	–	(4,951)	(8,066)	(7,086)	(6,146)	(6,533)
Project Net Smelter Return	(635,202)	–	–	–	(22,865)	(37,252)	(32,724)	(28,382)	(30,169)
Gross Revenue	**31,124,889**	**–**	**–**	**–**	**1,120,396**	**1,825,324**	**1,603,491**	**1,390,730**	**1,478,272**
Mining Opex	(6,240,694)	–	(39,106)	(76,392)	(129,057)	(170,281)	(171,952)	(179,105)	(197,429)
Stockpile Rehandling Cost	(95,622)	–	–	–	(3,379)	–	–	(489)	–
Processing Opex	(4,752,280)	–	–	–	(121,287)	(188,345)	(193,374)	(193,374)	(193,374)
Surface Infrastructure	(1,461,026)	–	–	–	(37,288)	(57,904)	(59,450)	(59,450)	(59,450)
G&A	(371,607)	–	–	–	(10,333)	(15,121)	(15,121)	(15,121)	(15,121)
Total Operating Costs	**(12,921,229)**	**–**	**(39,106)**	**(76,392)**	**(301,343)**	**(431,651)**	**(439,897)**	**(447,540)**	**(465,375)**
Royalty	(876,751)	–	–	–	(17,480)	(63,080)	(45,900)	(28,773)	(32,510)
Operating Cash Flow (pre-tax)	**17,326,910**	**–**	**(39,106)**	**(76,392)**	**801,573**	**1,330,593**	**1,117,694**	**914,417**	**980,388**
First Category Tax	(3,644,623)	–	–	–	–	(16,466)	(241,528)	(192,643)	(215,928)
Operating Cash Flow (post-tax)	**13,682,287**	**–**	**(39,106)**	**(76,392)**	**801,573**	**1,314,127**	**876,166**	**721,774**	**764,459**
Initial Capital	(2,440,955)	(475,138)	(585,753)	(926,981)	(453,083)				
Deferred Capital	(1,493,691)					(151,395)	(84,924)	(50,572)	(229,612)
Closing Costs	(264,107)	–	–	–	–	–	–	–	–
Free Cash Flow (pre-tax)	13,128,156	(475,138)	(624,859)	(1,003,373)	348,490	1,179,198	1,032,770	863,845	750,776
Free Cash Flow (post-tax)	**9,483,533**	**(475,138)**	**(624,859)**	**(1,003,373)**	**348,490**	**1,162,732**	**791,242**	**671,202**	**534,848**
Discounted Cash Flow (pre-tax)		(457,202)	(556,732)	(827,757)	266,199	834,027	676,352	523,819	421,534
Discounted Cash Flow (post-tax)		**(457,202)**	**(556,732)**	**(827,757)**	**266,199**	**822,381**	**518,178**	**407,004**	**300,298**

	Total (US$ Thousand)	6	7	8	9	10	11	12	13
Cash Flow									
Copper Sales, Metal Value	32,248,820	1,415,706	1,341,518	1,275,854	1,271,609	1,239,361	1,303,714	1,206,484	1,286,874
Silver Sales, Metal Value	712,790	31,244	31,512	30,186	31,817	29,171	31,958	29,704	27,628
Molybdenum Sales, Metal Value	3,526,063	171,924	121,344	112,441	102,620	109,053	201,179	138,069	119,070
Selling & Payability Expenses	(4,590,036)	(203,728)	(187,853)	(178,297)	(176,673)	(173,157)	(193,568)	(172,875)	(180,248)
Finance Advance	(137,546)	(6,102)	(5,634)	(5,348)	(5,301)	(5,194)	(5,792)	(5,180)	(5,404)
Project Net Smelter Return	(635,202)	(28,181)	(26,018)	(24,697)	(24,481)	(23,985)	(26,750)	(23,924)	(24,958)
Gross Revenue	**31,124,889**	**1,380,863**	**1,274,869**	**1,210,140**	**1,199,591**	**1,175,249**	**1,310,741**	**1,172,277**	**1,222,961**
Mining Opex	(6,240,694)	(194,860)	(197,509)	(261,171)	(282,446)	(312,837)	(328,709)	(308,050)	(325,185)
Stockpile Rehandling Cost	(95,622)	–	(4,016)	(6,177)	(3,259)	–	–	(1,526)	(2,677)
Processing Opex	(4,752,280)	(193,374)	(193,374)	(193,374)	(193,374)	(193,374)	(193,374)	(193,374)	(193,374)
Surface Infrastructure	(1,461,026)	(59,450)	(59,450)	(59,450)	(59,450)	(59,450)	(59,450)	(59,450)	(59,450)
G&A	(371,607)	(15,121)	(15,121)	(15,121)	(15,121)	(15,121)	(15,121)	(15,121)	(15,121)
Total Operating Costs	**(12,921,229)**	**(462,805)**	**(469,470)**	**(535,293)**	**(553,650)**	**(580,782)**	**(596,655)**	**(577,521)**	**(595,807)**
Royalty	(876,751)	(41,464)	(37,919)	(28,509)	(26,122)	(22,748)	(32,721)	(24,953)	(28,857)
Operating Cash Flow (pre-tax)	**17,326,910**	**876,594**	**767,480**	**646,337**	**619,819**	**571,719**	**681,365**	**569,803**	**598,297**
First Category Tax	(3,644,623)	(175,146)	(163,175)	(145,322)	(128,654)	(110,983)	(140,599)	(130,977)	(146,948)
Operating Cash Flow (post-tax)	**13,682,287**	**701,448**	**604,304**	**501,015**	**491,165**	**460,736**	**540,767**	**438,826**	**451,349**
Initial Capital	(2,440,955)								
Deferred Capital	(1,493,691)	(126,859)	(51,725)	(291,894)	(109,156)	(57,705)	(101,335)	(17,312)	(69,969)
Closing Costs	(264,107)	–	–	–	–	–	–	–	–
Free Cash Flow (pre-tax)	13,128,156	749,734	715,754	354,443	510,664	514,014	580,030	552,491	528,327
Free Cash Flow (post-tax)	**9,483,533**	**574,589**	**552,579**	**209,121**	**382,010**	**403,031**	**439,431**	**421,515**	**381,379**
Discounted Cash Flow (pre-tax)		389,767	344,539	157,978	210,747	196,417	205,225	181,001	160,264
Discounted Cash Flow (post-tax)		**298,714**	**265,992**	**93,207**	**157,653**	**154,007**	**155,478**	**138,092**	**115,688**

	Total (US$ Thousand)	14	15	16	17	18	19	20	21
Cash Flow									
Copper Sales, Metal Value	32,248,820	1,437,821	1,265,782	1,126,451	1,007,986	1,265,734	1,231,983	1,063,474	1,288,545
Silver Sales, Metal Value	712,790	27,282	28,126	27,809	24,979	27,747	31,083	23,678	22,188
Molybdenum Sales, Metal Value	3,526,063	160,589	136,912	198,936	106,278	116,194	126,638	90,532	167,105
Selling & Payability Expenses	(4,590,036)	(204,567)	(179,984)	(170,545)	(143,268)	(177,234)	(174,745)	(148,030)	(186,090)
Finance Advance	(137,546)	(6,128)	(5,394)	(5,100)	(4,295)	(5,314)	(5,239)	(4,440)	(5,570)
Project Net Smelter Return	(635,202)	(28,300)	(24,909)	(23,551)	(19,834)	(24,543)	(24,194)	(20,504)	(25,724)
Gross Revenue	**31,124,889**	**1,386,697**	**1,220,532**	**1,154,000**	**971,846**	**1,202,584**	**1,185,525**	**1,004,710**	**1,260,454**
Mining Opex	(6,240,694)	(315,671)	(318,169)	(322,658)	(310,484)	(308,955)	(306,566)	(282,738)	(258,214)
Stockpile Rehandling Cost	(95,622)	–	(207)	(1,163)	(16,586)	(4,694)	(6,114)	(14,829)	(2,876)
Processing Opex	(4,752,280)	(193,374)	(193,374)	(193,374)	(193,374)	(193,374)	(193,374)	(193,301)	(185,765)
Surface Infrastructure	(1,461,026)	(59,450)	(59,450)	(59,450)	(59,450)	(59,450)	(59,450)	(59,428)	(57,111)
G&A	(371,607)	(15,121)	(15,121)	(15,121)	(15,121)	(15,121)	(15,121)	(15,115)	(14,473)
Total Operating Costs	**(12,921,229)**	**(583,616)**	**(586,321)**	**(591,767)**	**(595,016)**	**(581,595)**	**(580,626)**	**(565,410)**	**(518,439)**
Royalty	(876,751)	(43,520)	(28,251)	(24,416)	(15,696)	(33,759)	(33,206)	(21,526)	(50,529)
Operating Cash Flow (pre-tax)	**17,326,910**	**759,561**	**605,961**	**537,817**	**361,134**	**587,231**	**571,694**	**417,774**	**691,486**
First Category Tax	(3,644,623)	(189,988)	(147,422)	(130,688)	(94,514)	(154,300)	(154,597)	(115,415)	(186,231)
Operating Cash Flow (post-tax)	**13,682,287**	**569,573**	**458,539**	**407,129**	**266,620**	**432,930**	**417,097**	**302,359**	**505,255**
Initial Capital	(2,440,955)								
Deferred Capital	(1,493,691)	(64,497)	(50,556)	(8,118)	(2,661)	(4,783)	(7,956)	(1,058)	(4,231)
Closing Costs	(264,107)	–	–	–	–	–	–	–	–
Free Cash Flow (pre-tax)	13,128,156	695,064	555,405	529,699	358,473	582,447	563,738	416,716	687,256
Free Cash Flow (post-tax)	**9,483,533**	**505,076**	**407,983**	**399,011**	**263,959**	**428,147**	**409,140**	**301,301**	**501,025**
Discounted Cash Flow (pre-tax)		195,224	144,442	127,553	79,927	120,246	107,762	73,757	112,632
Discounted Cash Flow (post-tax)		**141,862**	**106,103**	**96,083**	**58,854**	**88,391**	**78,210**	**53,329**	**82,111**

	Total (US$ Thousand)	22	23	24	25	26	27	28	29	30
Cash Flow										
Copper Sales, Metal Value	32,248,820	1,211,560	1,241,806	1,121,428	763,937	35,693	–	–	–	–
Silver Sales, Metal Value	712,790	22,038	28,425	27,060	19,955	670	–	–	–	–
Molybdenum Sales, Metal Value	3,526,063	220,349	149,416	172,064	103,041	10,676	–	–	–	–
Selling & Payability Expenses	(4,590,036)	(183,351)	(178,640)	(166,325)	(111,632)	(5,950)	–	–	–	–
Finance Advance	(137,546)	(5,479)	(5,351)	(4,977)	(3,343)	(177)	–	–	–	–
Project Net Smelter Return	(635,202)	(25,302)	(24,713)	(22,985)	(15,439)	(818)	–	–	–	–
Gross Revenue	**31,124,889**	**1,239,816**	**1,210,943**	**1,126,265**	**756,518**	**40,095**	**–**	**–**	**–**	**–**
Mining Opex	(6,240,694)	(247,595)	(165,237)	(123,579)	(91,678)	(14,426)	(636)	–	–	–
Stockpile Rehandling Cost	(95,622)	(2,041)	–	(3,533)	(22,056)	–	–	–	–	–
Processing Opex	(4,752,280)	(191,735)	(193,288)	(192,865)	(176,823)	(20,929)	(584)	–	–	–
Surface Infrastructure	(1,461,026)	(58,946)	(59,424)	(59,294)	(54,362)	(6,434)	(179)	–	–	–
G&A	(371,607)	(15,039)	(15,121)	(15,078)	(13,715)	(556)	–	–	–	–
Total Operating Costs	**(12,921,229)**	**(515,356)**	**(433,070)**	**(394,348)**	**(358,633)**	**(42,346)**	**(1,399)**	**–**	**–**	**–**
Royalty	(876,751)	(50,027)	(59,448)	(57,374)	(27,964)	–	–	–	–	–
Operating Cash Flow (pre-tax)	**17,326,910**	**674,432**	**718,425**	**674,543**	**369,921**	**(2,251)**	**(1,399)**	**–**	**–**	**–**
First Category Tax	(3,644,623)	(181,469)	(193,406)	(182,598)	(105,626)	–	–	–	–	–
Operating Cash Flow (post-tax)	**13,682,287**	**492,964**	**525,020**	**491,945**	**264,295**	**(2,251)**	**(1,399)**	**–**	**–**	**–**
Initial Capital	(2,440,955)									
Deferred Capital	(1,493,691)	(984)	(172)	(726)	(4,514)	(977)	–	–	–	–
Closing Costs	(264,107)	–	–	–	–	–	–	(88,036)	(88,036)	(88,036)
Free Cash Flow (pre-tax)	13,128,156	673,449	718,253	673,817	365,407	(3,228)	(1,399)	(88,036)	(88,036)	(88,036)
Free Cash Flow (post-tax)	**9,483,533**	**491,980**	**524,848**	**491,218**	**259,781**	**(3,228)**	**(1,399)**	**(88,036)**	**(88,036)**	**(88,036)**
Discounted Cash Flow (pre-tax)		102,193	100,919	87,662	44,017	(360)	(145)	(8,418)	(7,795)	(7,218)
Discounted Cash Flow (post-tax)		**74,656**	**73,744**	**63,907**	**31,293**	**(360)**	**(145)**	**(8,418)**	**(7,795)**	**(7,218)**

22.4 Financial Valuation Results

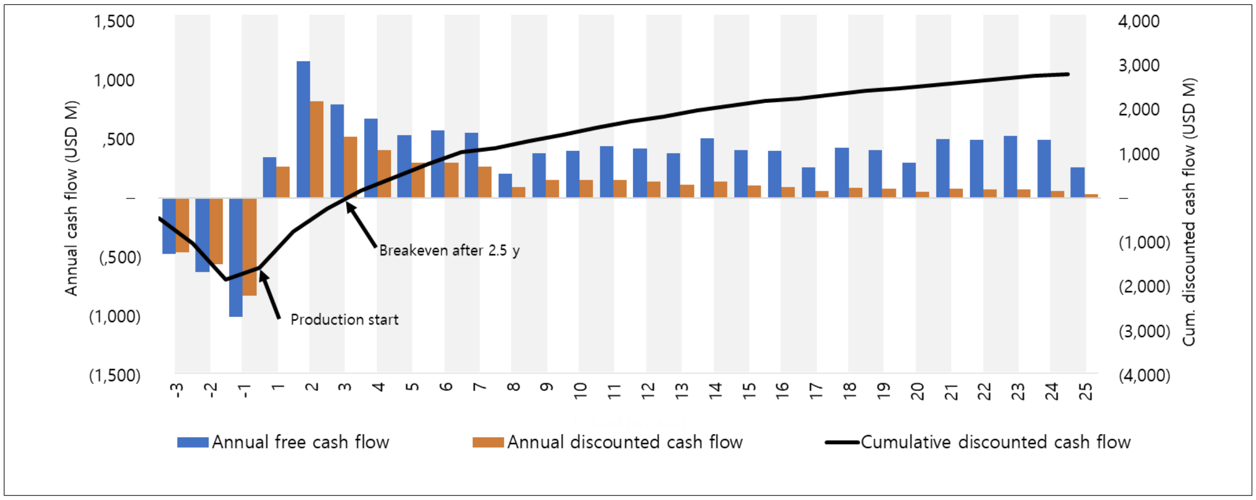
Based on the projections from the financial model, the pre-tax and after-tax NPV, IRR and payback periods are shown in Table 22.8.

Table 22.8: Summary of Financial Results

Metric	UoM	LOM
Net Present Value (8%, 2023_{real}), Pre-tax	US$ Million	3,999
Net Present Value (8%, 2023_{real}), After-tax	US$ Million	2,776
Internal Rate of Return, Pre-tax	%	28.5%
Internal Rate of Return, After-tax	%	24.2%
Undiscounted Post-tax Cash Flow (LOM)	US$ Million	9,484
Payback Period from First Production	Years	2.5
Initial Capex	US$ Million	2,441
LOM Sustaining Capex (excluding Closure)	US$ Million	1,494
LOM C1 Cash Costs	US$/lb Cu	1.25
Nominal Process Capacity (Annual)	ktpa	49,640
Nominal Process Capacity (Daily)	tpd	136,000

The after-tax cash flow over the LOM is shown in Figure 22.1.

Figure 22.1: Free Cash Flow (After-tax) Over LOM



Source: Fraser McGill, 2023

Copper contributes 88% of the net revenue, molybdenum contributes 10%, and the balance are silver credits in copper concentrate.

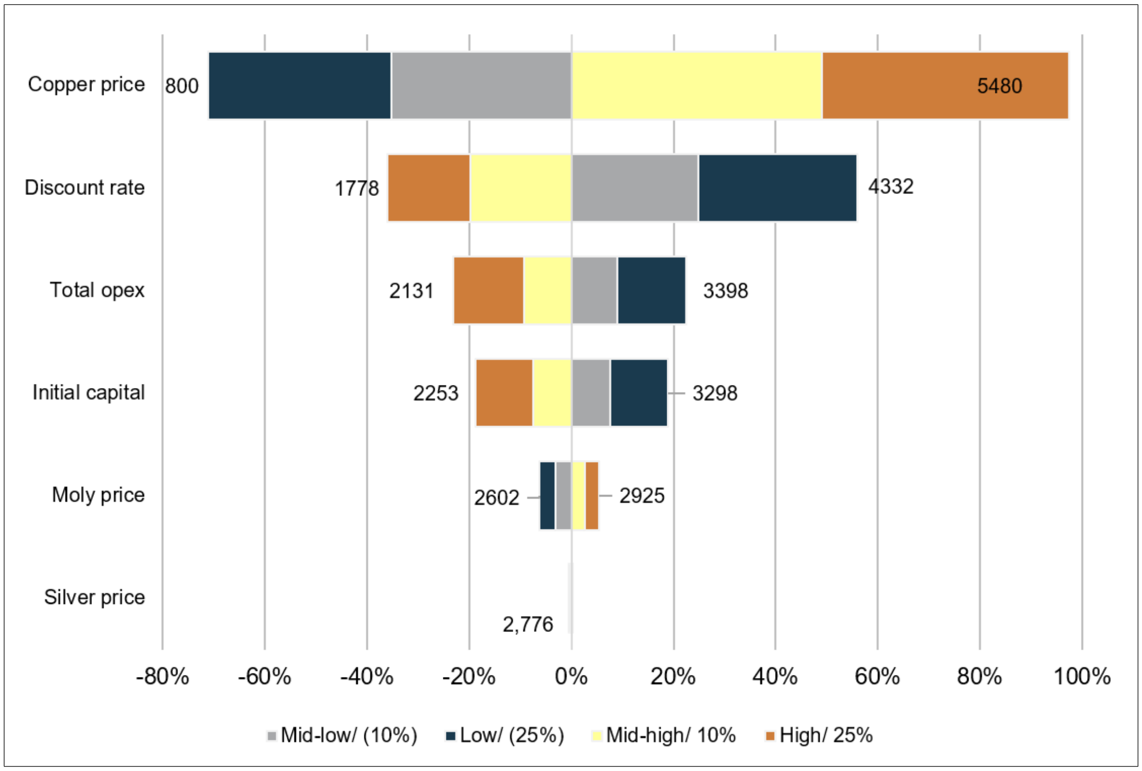
22.5 Sensitivity Analysis

The sensitivity analysis provides a range of outcomes for the Project when the key parameters are varied from the base case values to account for possible market changes and the impacts of capital and operating cost changes. The NPV estimate is most sensitive to the copper price, followed by the discount rate applied and the total operating cost.

In the single variation scenarios considered, the Project NPV (after-tax) remains robustly positive, as shown in the tornado sensitivity diagram presented in Figure 22.2 and in Table 22.9. The following is a summary of the results:

- The after-tax NPV ranges from US$800 million to US$5,480 billion as the copper price is varied between US$2.75/lb Cu and US$5.00/lb Cu. For the same input range, the IRR ranges between 13% and 36%. This indicates that the Project can generate positive returns even in a low Cu price environment.
- The after-tax NPV ranges from US$ 1,788 billion to US$4,332 billion when the discount rate varies between 11% and 5%.
- The after-tax NPV ranges between US$2,131 billion and US$3,398 billion when the operating cost ranges between US$13.5/t processed and US$8.7/t processed. The IRR ranges between 21% and 27%.
- The after-tax NPV ranges between US$3,298 billion and US$2,253 billion with the initial capital range between US$1,831 billion and US$3,051 billion. The IRR ranges between 19% and 33%.

Figure 22.2: NPV (After-tax) Tornado Sensitivity Diagram (US$ Million)



Source: Fraser McGill, 2023

Table 22.9: Sensitivity Analysis Supporting Table

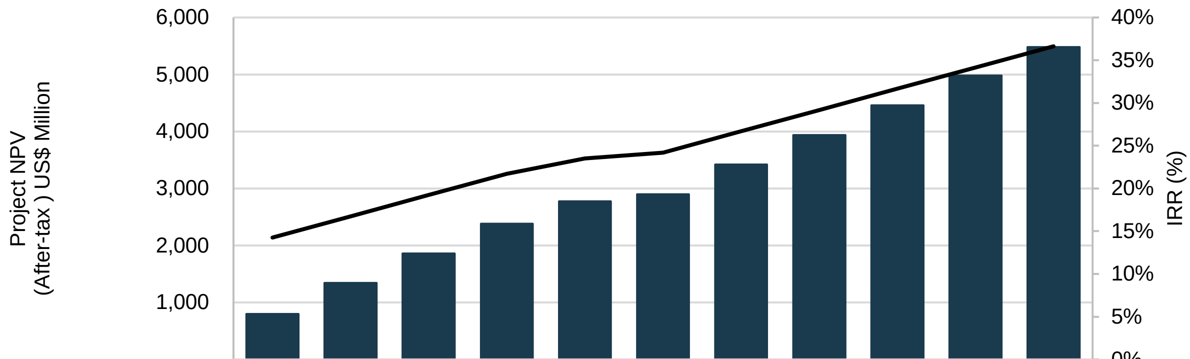
Range	UoM	Low	Low-mid	Base case	Mid-high	High
Copper price	USD/lb	2.75	3.22	3.68	4.34	5.00
	NPV (USD M)	800	1,796	2,776	4,137	5,480
	IRR (%)	13%	19%	24%	30%	36%
Discount rate	%	5%	6.50%	8%	9.50%	11%
	NPV (USD M)	4,332	3,464	2,776	2,224	1,778
Total opex	USD/t_{proc}	8.7	10.2	11.1	12.1	13.5
	NPV (USD M)	3,398	3,027	2,776	2,521	2,131
	IRR (%)	27%	25%	24%	23%	21%
Initial capital	USD M	1,831	2,197	2,441	2,685	3,051
	NPV (USD M)	3,298	2,985	2,776	2,567	2,253
	IRR (%)	33%	27%	24%	22%	19%
Moly price	USD/lb	22	25.2	28.4	31.2	34
	NPV (USD M)	2,602	2,689	2,776	2,851	2,925
	IRR (%)	23%	24%	24%	25%	25%
Silver price	USD/oz	20	20.9	21.79	22.4	23
	NPV (USD M)	2,763	2,769	2,776	2,780	2,784
	IRR (%)	24%	24%	24%	24%	24%

Note: The base case is the Project NPV point estimate

22.5.1 Copper Price Variation

Figure 22.3 presents the one-factor effect of changing the copper price input assumption over the LOM. The range applied is conservative based on a consensus long term real forecast from a leading Canadian bank.

Figure 22.3: Copper Price Sensitivity Analysis



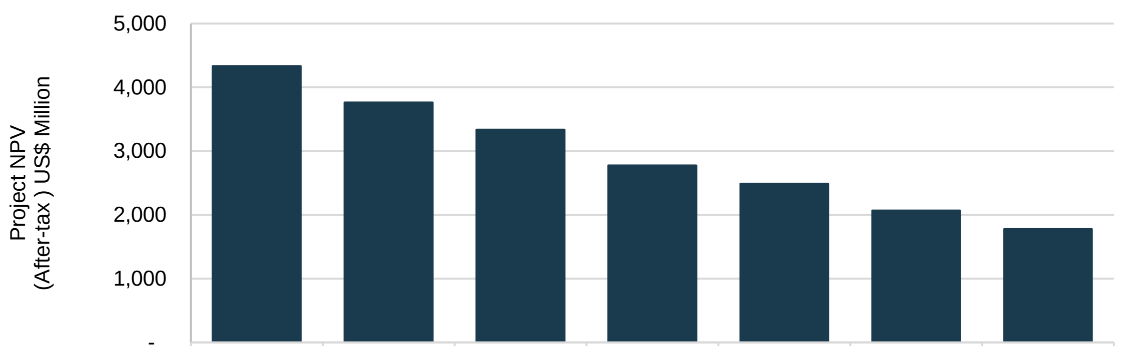
Copper Price (US$/lb)	2.75	3.00	3.25	3.50	3.68	3.75	4.00	4.25	4.50	4.75	5.00
NPV (After-tax)	800	1,342	1,861	2,381	2,776	2,900	3,420	3,939	4,459	4,979	5,480
IRR (%)	14%	17%	19%	22%	24%	24%	27%	29%	32%	34%	37%

Source: Fraser McGill, 2023

22.5.2 Discount Rate Variation

Figure 22.4 presents the one-factor effect of changing the Discount Rate.

Figure 22.4: Discount Rate Sensitivity Analysis



Discount rate (%)	5.0%	6.0%	7.0%	8.0%	9.0%	10.0%	11.0%
NPV (After-tax)	4,332	3,761	3,338	2,776	2,491	2,068	1,778

Source: Fraser McGill, 2023

Table 22.10 presents the two-factor sensitivity analysis when the copper price and discount rate are varied simultaneously. With the most deterministic view applied, a positive NPV is still achieved.

Table 22.10: Two Factor Sensitivity on NPV: Copper Price and Discount Rate

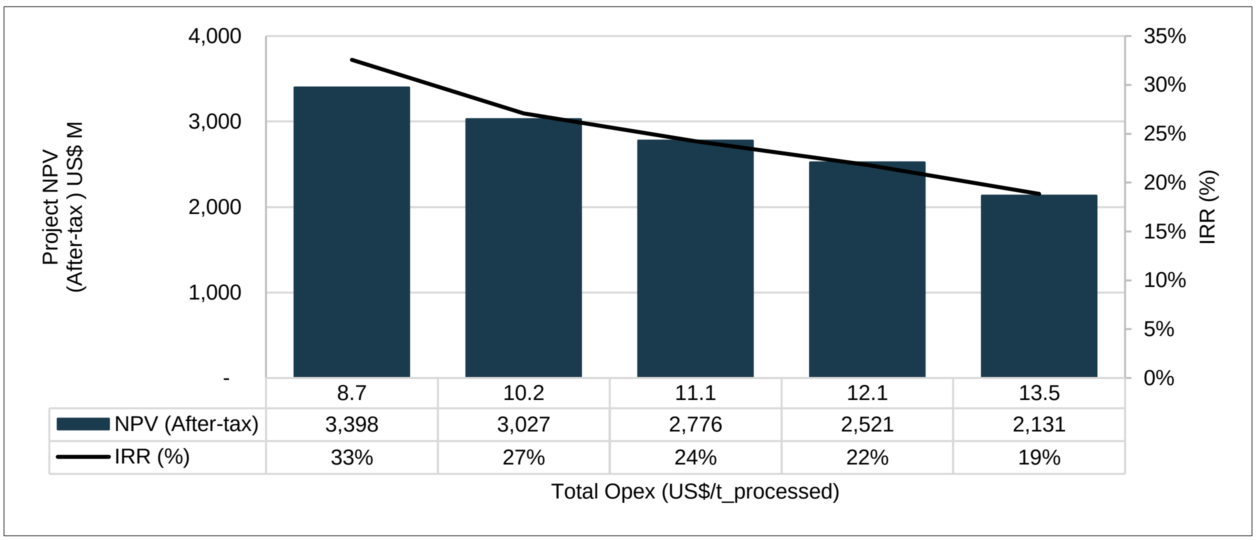
NPV (After-Tax) US$ Million		Copper Price				
		2.75	3.22	3.68	4.34	5.00
Discount Rate	5.0%	1,604	2,978	4,332	6,216	8,076
	6.5%	1,155	2,319	3,463	5,056	6,628
	8.0%	800	1,796	2,776	4,137	5,480
	9.5%	516	1,378	2,224	3,399	4,559
	11.0%	288	1,039	1,778	2,801	3,811

Note: Cu price of US$3.68/lb and 8% discount rate are the Project Base Case

22.5.3 Total Opex Variation

Figure 22.5 presents the one-factor effect of varying the Opex over ±25%.

Figure 22.5: Total Opex Sensitivity Analysis

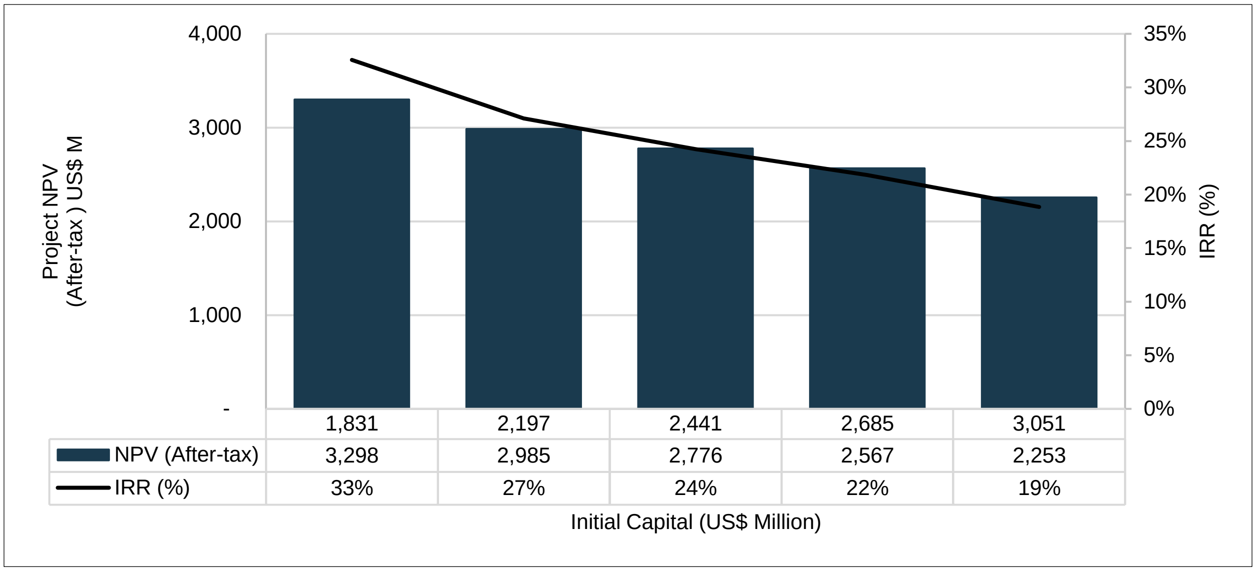


Source: Fraser McGill, 2023

22.5.4 Initial Capital Variation

Figure 22.6 presents the initial capital cost sensitivity with the capital expensed in the first 4 years of the valuation period ranging over ±25%. The total initial capital cost varied includes the allowances for indirect costs and contingencies (this is a conservative approach).

Figure 22.6: Initial Capital Sensitivity Analysis



Source: Fraser McGill, 2023

The payback period impact of the initial capital cost sensitivity ranges from 9 months (low capex) to 4 years and 3 months (high capex).

Due to annual periods applied in financial modelling and the first revenue being generated before the final capital is expensed in Year 4, the initial capital cost presents a more deterministic view of the financed capital requirements compared to the calculated maximum negative drawdown.

Table 22.11 presents the two-factor Sensitivity analysis on Project NPV by varying total Opex and initial Capex together.

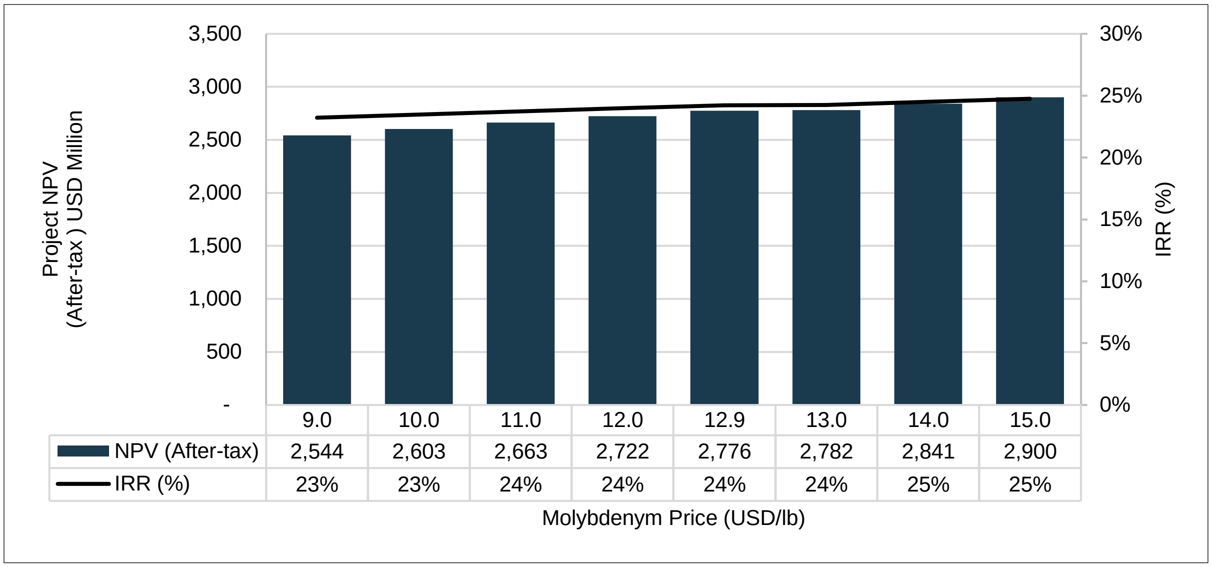
Table 22.11: Two Factor Sensitivity on NPV: Opex and Capex

NPV (After-Tax) US$ Million		Opex				
		(25%)	(10%)	0%	10%	25%
Capex	(25%)	3,920	3,607	3,398	3,189	2,875
	(10%)	3,549	3,236	3,027	2,818	2,504
	0%	3,298	2,984	2,776	2,567	2,253
	10%	3,043	2,730	2,521	2,312	1,998
	25%	2,654	2,340	2,131	1,922	1,609

22.5.5 Molybdenum Price Variation

Figure 22.7 presents the impact of molybdenum price fluctuations on the sensitivity of the Project. The range applied is a view based on the CRU+, Molybdenum Concentrates Market report, commissioned by Los Andes Copper.

Figure 22.7: Molybdenum Price Sensitivity Analysis (After-tax)



Source: Fraser McGill, 2023

23. ADJACENT PROPERTIES

West Wall is a copper porphyry exploration project located approximately 20 km south-east of the Vizcachitas Property. Glencore plc and Anglo American plc each own a 50% interest in the project comprising 141 mining concessions totalling 44,199 ha.

The estimated resources according to the 2012 Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (JORC Code) as published by the owners in their 2022 reports for Resources & Reserves (Anglo American 2022 and Glencore 2022), are:

- Indicated Resources of 861 Mt @ 0.51% Cu, 0.05 g/t Au and 0.008% Mo
- Inferred Resources of 1,072 Mt @ 0.42% Cu, 0.05 g/t Au and 0.006% Mo.

Note: Mineral Resources are quoted above a 0.20 %Cu cut-off within an optimized pit shell.

The geology in the area is dominated by Tertiary pre-mineral stratified volcanics locally intercalated with clastic lacustrine sediments. Copper mineralization is associated with sub-volcanic porphyry intrusive bodies of dioritic to quartz-monzonitic composition. The porphyry intrusive bodies at Lagunillas have been grouped into inter-mineral and late inter-mineral phases, with the main mineralizing events associated with the inter-mineral phases. Post-mineral covers include unconsolidated glacial terraces, colluvial sediments and alluvial deposits.

Exploration was focused in the south of the prospect at Lagunillas and West Wall North. During the 2011-2012 drilling programme, 24,000 m of infill drilling were completed and incorporated into the geological models and mineral resource estimate.

24. OTHER RELEVANT DATA

The Qualified Persons are not aware of any other data or information that would be relevant to this Technical Report which is not already contained in one of the existing sections of this Technical Report.

25. INTERPRETATIONS AND CONCLUSIONS

Based on a review of the data available for this TR, the QPs note the following interpretations and conclusions in their respective areas of expertise.

25.1 Mineral Tenure, Surface Rights and Royalties

Los Andes Copper holds exploitation concessions over the entirety of the Vizcachitas Project and exploration concessions that overlap and extend beyond the deposit, providing an appropriate level of property protection and additional exploration opportunities.

Los Andes Copper does not have surface rights to develop the Vizcachitas Project. Chilean Mining Law provides legal rights to obtain the easements needed to explore, develop and operate the property.

The economic model includes a third-party NSR at the 2% level.

25.2 Geology and Mineralization

The Vizcachitas Project is within the north-south trending Neogene metallogenic belt that extends along the slopes of the Andes Mountains in Chile and Argentina. This belt includes world-class copper-molybdenum porphyries, such as Los Pelambres-El Pachón 75 km north of the Vizcachitas Project, Río Blanco-Los Bronces 80 km to the south, and El Teniente 180 km to the south.

The geological team at Vizcachitas has developed a detailed geological model derived from surface mapping, multi-element geochemical sampling, age dating and drill core mapping. The geological model is the basis for the Mineral Resource estimate, metallurgical recoveries and the mine plan. The Vizcachitas Project is a complex set of porphyries and hydrothermal breccias intruded into a sequence of andesitic and daciandesitic volcanic rocks of the Abanico Formation of Eocene-Oligocene age.

25.3 Exploration and Drilling

From the beginning of the 1990s to date, three companies have explored the Property: Placer Dome, General Minerals Corporation (GMC) and Los Andes Copper. The geological information has confirmed that Vizcachitas is a zoned porphyry copper system with early central potassic alteration zoned to distal propylitic alteration and up into a halo of phyllic alteration.

The surface mapping and sampling have identified prospective exploration targets at Breccia Roja and Breccia Sericita that warrant further work.

A 3D Induced Polarization/Resistivity and Magneto-Telluric survey programme conducted in July 2020 shows good correlation between IP/Resistivity and copper mineralization. The higher grade mineralization, lower grade diatreme and inter-mineral intrusive are reflected in resistivity data.

The survey identifies an untested conductive zone extending 750 m north from the northernmost drill hole and conductive zones to the east of the diatreme and along the Campamento Fault to the east.

Los Andes Copper conducted drilling campaigns in 2007-2008, 2015-2017 and 2021-2022, drilling 115 drill holes for a total of 43,156 m. The 2007-2008 campaign drilled in the southern part of the Project, extending the mineralization to the Campamento Fault. The 2015-2017 drill campaign validated the new geological model confirming the importance of the early diorite porphyry and hydrothermal breccias in controlling the higher grade mineralization of the deposit: the deeper drilling, demonstrated potential at depth. The drilling also confirmed the near-surface, higher-grade supergene-enriched mineralization that can be targeted in the first years of the mine schedule.

In the 2021-2022 campaign one hole was drilled from the top of the bedrock down to a depth of 1,048 m within mineralized rock, demonstrating the vertical continuity of the system. Also, the first deep drill hole on the eastern side of the diatreme intersected 0.62% CuEq (0.55% Cu, 212 ppm Mo and 1.3 g/t Ag) from a depth of 560 m until the end of the drill hole at 891 m. This drill hole has shown that the mineralization to the east of the diatreme is similar to the mineralization to the south and west, demonstrating the potential on the eastern side.

The drill campaigns have extended the Mineral Resource; however, the Project remains open to the west, north and east. A new exploration programme should be carried out to demonstrate the potential for growth of the Mineral Resource.

25.4 Sample Preparation, Analysis and Security

A complete QA/QC system and set of protocols were implemented covering the sampling and assaying procedures for the drilling programmes carried out by GMC and Los Andes Copper at the Vizcachitas Project. Sampling and assaying procedures are well documented, with diamond core for all the drill holes and certificates for all the assays. In addition, the Company has 100% of the pulp samples it prepared and 80% of the Generals Minerals Corporation pulp samples.

The QA/QC system implemented for the sampling, mechanical preparation and analytical procedures meets or exceeds mining industry best practices. Independent QPs visited the Project site during each drill campaign to review the drilling procedures, sample preparation and implementation of the QA/QC procedures.

In the opinion of the QP, the geological and geochemical data reviewed are an adequate and accurate reflection of the geology of the Vizcachitas Project. The data reviewed meets the standards of an NI 43-101 Technical Report and for use in the Mineral Resource estimation.

25.5 Mineral Processing and Metallurgical Testing

The Vizcachitas Project is a sulphide deposit primarily composed of chalcopyrite and pyrite. Chalcocite and covellite are observed only as a superficial oxidation process typical of zones above the water table, as patinas over the primary sulphides. Copper sulphides are mainly associated with hard silicates and phyllosilicates.

The Vizcachitas deposit has less than 1.3% of clays (kaolinite and montmorillonite). Minerals that contain deleterious elements such as Pb, As and Sb are present at less than 1% and are mainly associated with pyrite.

Comminution parameters show that Vizcachitas is medium hard with Axb and Bond Ball Work Index (BWi) of 38.0 and 11.38 kWh/t, respectively.

It is planned to use desalinated water in the process plant. A strong frother must be used in the reagent formula to ensure stable bubble swarms and a steady froth that can carry coarser particles (larger than 240 µm). Operating the rougher stage with a pH between 7.0 and 7.5 is recommended to enhance molybdenum recoveries.

The main association of copper sulphides is with silicates, lower regrinding target sizes must be achieved (25 µm) to liberate the sulphides and produce a clean copper concentrate.

A single stage of cleaner/scavenger flotation is recommended. Expected recoveries for this stage are 22.2% for mass, 98.5% for Cu and 95.8% for Mo. No pyrite activation was observed in the LCTs at a laboratory scale.

The PFS metallurgical testwork demonstrated good overall LOM recoveries for copper at 91.1%, molybdenum at 74.8% and silver at 75.0%.

Vizcachitas can produce clean concentrates. No deleterious elements are present in amounts that could lead to penalties.

Vizcachitas tailings are suitable to be filtered and dry stacked. SGS and Takraf tests showed excellent filtration rates compared with current operations worldwide. This is possible due to the coarser grind size (240 µm) and the lack of clays and ultra-fine particles. The expected cake moisture is 15%.

25.6 Resource Estimation

The Mineral Resource drilling database for the Vizcachitas Project uses 168 diamond drill holes from the 1996 to 2022 drilling campaigns, totalling 58,628 m.

Los Andes Copper geologists created a comprehensive geological model that defines how mineralization is controlled and produced a robust Mineral Resource estimate. The geological model provides a level of understanding sufficient for the declaration of Measured Resources.

The statistical analysis and exploratory data analysis defined the geological estimation units based on a combination of lithological units, veinlet intensity and mineral zones.

To assess reasonable prospects for eventual economic extraction, a Whittle pit shell was prepared to constrain the estimated resource blocks using the same general costs and technical parameters used for the PFS.

The Vizcachitas Mineral Resources have an Effective Date of February 7, 2023. Several factors may affect the geological interpretation or the Whittle pit shells and, therefore, may positively or negatively impact the Mineral Resources. These factors include:

- Changes in interpretations of mineralization geometry and continuity of mineralization zones
- Input parameters used in the Whittle shells that constrains Mineral Resources amenable to open pit mining methods
- Metallurgical and mining recoveries
- Operating and capital cost assumptions
- Metal price and exchange rate assumptions
- Concentrate grade and smelting/refining terms
- Delays or other issues in reaching agreements with local or regulatory authorities and stakeholders
- Changes in land tenure requirements or in permitting requirements from those discussed in this TR.

The Tetra Tech QPs concluded that the database and geological models are in good order and auditable. In addition, the database (assay and geological data) has been validated and is of sufficient quality to support this Mineral Resource estimate. Therefore, the Mineral Resource estimate is in line with good industry practices.

25.7 Mineral Reserves Estimates

Estimations of Mineral Reserves for the Project conform to industry best practices. The Mineral Resources were classified using the CIM Definition Standards (2014) and the CIM Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines (2019). Reviews of the environmental, permitting, legal, title, taxation, socio-economic, marketing, political factors and constraints for the operation support the declaration of Mineral Reserves using the assumptions outlined.

Tetra Tech perceives little risk in the Mineral Reserves, as stated, but there may be room for further optimization. Various factors may affect the Mineral Reserves estimate including dilution; metal prices; metallurgical recoveries and geotechnical characteristics of the rock mass; capital and operating cost estimates; and effectiveness of surface and ground water management.

Vizcachitas is a large, near surface, bulk mineable deposit that is well suited for extraction by conventional open pit methods. The mine plan is based on reasonable long term metal prices, supported by a Mineral Resource and PFS level geotechnical and metallurgical inputs.

The TR results indicate that the Vizcachitas Project is robust at this stage of development, demonstrating favourable economic potential that warrants further work and the development of feasibility studies.

25.8 Mining Methods

An open pit mining method has been selected for the Vizcachitas deposit mainly based on the copper and molybdenum grades and the continuity of the mineralization occurring near the surface which results in a low strip ratio for open pit mining. The mine has been scheduled to operate 365 days per year. The annual operating period for the mine considers negligible snow related downtime.

The final pit was optimized using the existing resource model following the economic and geotechnical parameters defined for the Project. This exercise was performed using the Lerchs-Grossman algorithm introduced into the Whittle4X software. The mine plan schedule was prepared using the COMET strategic planning software. The mine plan options were assessed at a detailed level, determining the hauling distances for mineralized material and waste and then estimating the equipment fleet and the purchasing requirements over time.

The study considered the best set of mine phase designs suitable for 136,000 tpd. Each phase was designed with its own access roads to enable rock haulage from the loading area to the destination (primary crusher or waste dump). The access roads for the top benches were designed with a width of 25 m to operate smaller trucks in narrow areas. Earthmoving contractors will mine those areas. Other roads were designed with a width of 38 m to operate high capacity AHS mining trucks.

The operating mine design includes 12 phases totalling 1,421 Mt of mineral with an average Cu grade of 0.35% and 4,325 Mt of waste (including waste movement by contractors), using a cut-off grade of ≥0.18% of Cu. The ultimate pit was defined as phase 10. Phases 11 and 12 do not provide additional economic value, mainly because of the high strip ratio. This ultimate pit results in 1,220 Mt of feed to the concentrator with an average Cu grade of 0.36% and 2,855 Mt of waste (including waste movement by contractors), using a variable cut-off grade.

25.9 Recovery Methods

The process flowsheet incorporates knowledge from current practices, vendors and results from testwork conducted in 2020 using representative samples to define a robust plant design to achieve the desired throughput and recovery targets. The process is well-known and consists of a three stage crushing using HPGR, grinding using ball mills, bulk flotation using conventional cells, copper–molybdenum separation, concentrate thickening and reagent facilities. The tailings dewatering processes designed for the classification, thickening and belt and pressure filtration were validated and supported by specific testwork and are robust enough to achieve the designed filtration rates and filtered cake moisture.

25.10 Project Infrastructure

This TR aimed at reducing the environmental impacts, including reduction of water and energy consumption and minimizing the footprint of the Project. The Project design has also looked at reducing earthworks by installing infrastructure on natural plateaus and on platforms using compacted mine pre-stripping material, by avoiding the construction of a tailing impoundment, and by eliminating a permanent on-site camp.

Major infrastructure considered in this TR are:

- A 36 km long access road from north of Putaendo to the concentrator and filtration plant
- A 15.965 km diversion tunnel for the Rocín River
- On-site facilities including mine service facilities, plant facilities, buildings, internal roads
- Water management systems to capture non-contact water and return it to the Rocín River, and to capture contact water and treat it before returning it to the Rocín River
- A 90 km long desalinated water pipeline connecting the point of delivery in Cabildo to the site
- Tailings disposal facilities for dry stacked filtered tailings.

Further studies may contribute to optimization of this infrastructure and layout.

25.11 Environmental, Permitting, and Social Considerations

Los Andes Copper has conducted several environmental studies for the Project as part of its current exploration activities. Los Andes Copper holds an environmental permit to conduct explorations activities, which is currently suspended until certain conditions are met. An EIA for the Vizcachitas Project has yet to be prepared and submitted for approval before the Chilean environmental impact assessment system (SEIA). An EIA can take 2 to 4 years for approval after it enters the SEIA.

The Project will be designed and built to meet Chilean Environmental Laws, practices and standards. The Project will be designed to meet current social and environmental management practices. Provisions have been made within the mine plan and operating costs to account for the environmental protection and rehabilitation of the Project when mining has been completed. No known environmental issues existing or anticipated could materially impact the ability to develop the Vizcachitas Project.

An experienced community and corporate affairs team, based on an office in Putaendo, has worked for a number of years in implementing engagement programmes and plans in the Putaendo area, including enabling and empowering social organization, small entrepreneurs and women who lead small businesses. The team has constant communications with the community on the contributions that the Vizcachitas Project can provide for the sustainable development of the county of Putaendo, Province of San Felipe and Region of Valparaíso.

There are political risks and uncertainties surrounding legislation, regulatory requirements and general business climate including:

- Potential changes in existing laws and approval of more onerous tax laws
- Increased costs or financing hurdles

- Shortage of skilled labour owing to competing demand from the mining industry in general and other mines in Chile in particular
- Capital and operating cost escalation as Project plans and parameters change or are refined
- Failure to obtain or maintain, or a delay in obtaining, permits or approvals from government authorities.

25.12 Markets and Contracts

The Vizcachitas Project will produce copper and silver contained in copper concentrate, both openly traded commodities, and molybdenum contained in molybdenum concentrate. No deleterious elements will be present in the concentrates in amounts that would attract penalties.

Following consensus long term copper and silver prices calculated by a leading Canadian bank, and molybdenum long term forecast price from CRU, the TR uses the following prices:

- Copper : US$3.68/lb
- Molybdenum : US$12.90/lb
- Silver : US$21.79/oz.

No contracts are currently in place. The marketing approach will be consistent with existing industry practices.

25.13 Capital Cost and Operating Cost Estimates

The capital cost estimate was developed in accordance with AACEI Class 4; with the applied contingency the estimate is assessed to be accurate to within ±25%. The estimate is presented in real 2023 US$ terms.

The initial capital cost, including indirect costs and contingency, is estimated to be US$2.441 billion and will be expensed over a period of 3.25 years.

The sustaining and deferred capital cost which includes capital to sustain operations and deferred infrastructure capital, is estimated to be US$1.494 billion over the 26 year LOM.

Operating costs have been estimated for the areas of mining, process plant, infrastructure and administration at an accuracy of ±25%. At the steady state processing rate of 49,640 ktpy, mining costs are estimated to be US\$5.02/$t_{processed}$, processing costs are estimated to be US\$3.90/$t_{processed}$, infrastructure costs are estimated to be US\$1.20/$t_{processed}$ and general and administration costs are estimated to be US\$0.30/$t_{processed}$. In addition, the cost for rehandling stockpiled plant feed, moved to and from stockpiles, as per the production schedule, is estimated to be US$0.70/t rehandled, the tonnage per year is variable based on the production schedule.

The total operating costs are estimated to be US\$10.41/$t_{processed}$ over LOM, with rehandling over the LOM contributing an additional US\$0.08/$t_{processed}$.

25.14 Economic Analysis

The Project generates a US$2.776 billion after-tax Net Present Value (US$3.999 billion pre-tax) using an 8% discount rate and an after-tax internal rate of return (IRR) of 24% (29% pre-tax) at US$3.68/lb Cu, US$12.90/lb Mo and US$21.79/oz Ag.

Copper contributes 88% to the net revenue, followed by molybdenum with 10%, the balance being silver credits in copper concentrate (2%). A payback period of 2.5 years from first production was calculated.

The economic analysis demonstrates that the mine plan generates favourable returns at the base assumptions, as well as over a wide range of key input sensitivities:

- Copper price variation from US$2.75/lb Cu to US$ 5.00/lb Cu results in an NPV variation between US$800 million and US$5.480 billion, respectively.
- A discount rate variation between 5% and 11% results in an NPV variation between US$4.332 billion and US$1.778 billion, respectively.

25.15 Overall Conclusions

There are no known factors related to metallurgical, environmental, permitting, legal, title, taxation, socio-economic, marketing or political issues which could materially impact the ability to develop the Vizcachitas Project.

Based on the information, interpretations and conclusions contained in this TR, Tetra Tech confirms that the Vizcachitas Project has technical and economic merit. The Vizcachitas Project has the potential to return a significant net present value and internal rate of return. There are no fatal flaws that could put the Project at risk.

26. RECOMMENDATIONS

The QPs consider that several opportunities should be explored to continue adding value to the Vizcachitas Project in the next phases, including an optimization phase and initiation of feasibility studies (2023-2024). The recommended work programme has an estimated cost of US$18,500,000 (Table 26.1).

Resource Estimate and Geology

Additional infill drilling should be carried out to convert Inferred Mineral Resources to Indicated Resources and to find the extent of the mineralization where the resource is still open.

The base of the Indicated Resources limits the depth of the PFS open pit in the southern part of the resource. Deeper infill holes should be drilled in this area to extend the Indicated Resource to determine the maximum extent the open pit could reach. The mineralization at Vizcachitas has a sub-vertical control, hence drilling deeper in this area should have a positive impact on the Project.

Infill drilling on the eastern side of the resource could convert Inferred Mineral Resources to Indicated Resources within the current PFS open pit. These Inferred Resources within the PFS are currently treated as waste in the PFS mine plan.

The resource is still open to the east, north-west and north. The 2015-2017 and 2021-2022 drilling demonstrated potential in these areas which should be drilled to determine the limits of mineralization.

The first deeper drill hole (CMV-012B) east of the diatreme intersected good mineralization over 330 m. Further drilling should be completed to the north and south of this drill hole to test this area. The eastern side of the diatreme has the potential for similar mineralization to that found to the west of the diatreme.

The Induced Polarization/Resistivity anomaly extends from the northern part of the current resource to Breccia Roja. These targets should be tested by drilling; they are shallow targets near the main Project area.

Drill holes CMV-001B, CMV-011 and V2017-05 intersected good mineralization leaving the northwestern side of the resource open. This side of the resource should also be tested by drilling.

Geotechnical, Hydrology and Hydrogeology

It is recommended that the slope stability, the location of the strata boundaries between the surface material, the regolith type material and the subjacent layers of rock material, and the qualities of these materials be assessed. Seismic profiling or another geophysical technique should be used to interpret the stratigraphy, in particular the thickness of the surface covering material with lower geotechnical competence. The results of these investigations would further the geotechnical model in these areas, increasing the confidence level in slope stability analyses and in the pit wall geotechnical design inputs.

It is also recommended that Los Andes Copper continues the hydrological and hydrogeological studies for the Vizcachitas Project including:

- Create a hydrogeological model for the Project area to enable modeling of the sub-surface water flows around the proposed open pit and the water diversion tunnel
- Conduct water balance studies within the infrastructure area to size the systems for managing non-contact water and contact water.

Metallurgy and Processing

There are several opportunities that should be developed through further studies and testwork:

- Analyze coarser grinding for the rougher stage flotation. The Project currently contemplates grinding to a P80 of 240 µm. Preliminary testwork has shown the potential for a P80 of 280 µm to 300 µm. This has the potential to further reduce power consumption and maintain efficient recovery levels in the rougher flotation.
- Conduct HPGR/HRC pilot plant testwork and carry out a trade-off with different vendors (Metso, Weir, Thyssenkrupp) to confirm equipment sizing, performance and wear rates.
- Explore flotation alternatives for the cleaning/scavenging stages at pilot scale to avoid fines losses in the scavenger tailings.
- Confirm LCT results with more samples and with a mini or full scale pilot plant. Explore the alternative of returning the scavenger concentrate to the first cleaner stage (without regrinding) to avoid generation of fines that could decrease recoveries.
- Study the full circuit for molybdenum flotation at the laboratory scale using pilot plant concentrates.
- Future laboratory testwork should consider a control sample that will be tested by flotation on a monthly basis to assess the effect of sample ageing on the Cu/Mo recoveries. In the sample preparation stage the 10 mesh samples must be purged with nitrogen, vacuum sealed and stored in freezers.
- Conduct humidity cell tests to simulate weathering at the laboratory scale to further understand the acid generation potential of the Vizcachitas tailings.
- Validate the sizing and selection of the main equipment by performing mini or pilot scale testwork using samples representative of the definitive mine-plan.
- Perform a reliability analysis of the entire circuit, including process variability and geometallurgical features defined by the mine plan.

Infrastructure

Los Andes Copper should continue to optimize the off-site infrastructure, including the power line route and connection points, and the design of the main access road to increase safety and minimize exposure to geohazards.

A cost-benefit analysis should be prepared to compare trucks vs rail transport of concentrate to the ports.

Environmental, Permitting and Social

Environmental studies should be progressed during the next optimization phase, including:

- Geohazard assessment of areas with significant infrastructure/operations to optimize the design and minimize the effect of natural hazard events such as avalanches, alluvial flows and seismic events.
- Field studies to analyze the small rock glaciers recently added to the Water Agency (DGA) Glacial Inventory to confirm the absence of impact by the Project.

It is recommended that baseline studies for the Vizcachitas Project are initiated as soon as possible, together with the development of an overall strategy for preparing and organizing the EIA to ensure that the EIA properly covers all relevant social and environmental matters.

It is recommended that the current social and engagement programmes be continued and advanced as the Project design evolves.

Work Programme

The recommended work programme has an estimated cost of US$18,500,000 as summarized in Table 26.1.

Table 26.1: Recommended Work Programme Cost Estimate

Item	Value US$
Resource Estimate and Geology	8,500,000
Geotechnical, Hydrology and Hydrogeology	3,000,000
Metallurgy and Processing	1,500,000
Infrastructure	2,000,000
Environmental and Permitting	1,500,000
Initiating of Feasibility Studies	2,000,000
Total	18,500,000

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28. CERTIFICATES AND SIGNATURES

The undersigned prepared this technical report titled Pre-Feasibility Study Vizcachitas Project National Instrument 43-101 Technical Report, Effective Date February 20, 2023.

CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled: Pre-Feasibility Study Vizcachitas Project, with Effective Date February 20, 2023, Under NI 43-101 Reporting Standards

I, Severino Modena, do hereby certify that:

1) My position at Tetra Tech Sudamérica S.A. (Chile office) is General Manager.
2) I hold the professional degree of Civil Mining Engineer obtained at Universidad de Santiago, Chile. I have practiced my profession continuously since 1979. I have estimated and audited projects for reserve estimates, open pit mine planning, analysis of operational productivity, due diligence and expert revisions, preparation of reports for banks and financial institutions and support in the analysis and definition of mine infrastructure and processes.
3) I am a Competent Person from the Chilean Mining Commission, with Registration No. 134. I am registered at the Institute of Chilean Mining Engineers (IMCH), and with the Australian Institute of Mines and Metallurgy (AusIMM), Licence No. 301987;
4) I visited the Vizcachitas Project on October 13, 2017;
5) I have read the definition of Qualified Person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfill the requirements to be a Qualified Person for the purposes of National Instrument 43-101 and this technical report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1;
6) As a Qualified Person, I am independent of Los Andes Copper as defined in Section 1.5 of National Instrument 43-101. As per Exchange Policy requirement (Appendix 3F) I am also independent of the Property. There is no scenario outside of my scope as a qualified person that I would not satisfy the requirement of independence;
7) I am the co-author of this report and responsible for Sections 2, 3, 4, 5, 15, 16, 19, 20, 21, 22, 23, 24, 27, and parts of Sections 1, 25 and 26 of this report and accept professional responsibility for those sections of this technical report;
8) I have read National Instrument 43-101 and confirm that this technical report has been prepared in compliance therewith;
9) Severino Modena (Chile) was retained by Los Andes Copper to visit the subject property and to contribute to a technical report on the Vizcachitas project in accordance with National Instrument 43-101 and Form 43-101F1 guidelines;
11) I have not received, nor do I expect to receive, any interest, directly or indirectly, in the securities of Los Andes Copper.;
12) That, as of the date of this certificate, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

["Original signed and sealed"]

Santiago, Chile
March 30, 2023

Severino Modena (Mining Engineer)
Mining Chilean Commission (Registration No. 134)
Tetra Tech Sudamérica S.A.

CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled: Pre-Feasibility Study Vizcachitas Project, with Effective Date February 20, 2023, Under NI 43-101 Reporting Standards

I, Sergio Alvarado Casas, do hereby certify that:

1) I am a Consultant Geologist, a member of Tetra Tech Sudamérica S.A. (Chile office);
2) My professional title is Geologist, with the degree of Geology obtained in 1991 at Universidad Católica del Norte, Chile, with post graduate studies in resource assessment at the Universidad de Chile, in 1997. I have practiced my profession continuously since 1985. I have estimated and audited mineral resources for a variety of early and advanced international base and precious metals projects. I have worked in the mining industry on several underground and open pit mining operations and held various senior operational and corporate positions;
3) I am a Competent Person from the Chilean Mining Commission, with Registration No. 004. I am registered at the Institute of Chilean Mining Engineers (IMCH), Licence No. 1939, and with the Canadian Institute of Mines (CIM), Licence No. 144015;
4) I visited the Vizcachitas Project during March 22-23, 2022;
5) I have read the definition of Qualified Person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfill the requirements to be a Qualified Person for the purposes of National Instrument 43-101 and this technical report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1;
6) As a Qualified Person, I am independent of Los Andes Copper as defined in Section 1.5 of National Instrument 43-101. As per Exchange Policy requirement (Appendix 3F) I am also independent of the Property. There is no scenario outside of my scope as a qualified person that I would not satisfy the requirement of independence;
7) I am the co-author of this report and responsible for Section 6, 7, 8, 9, 10,11 and 12 and parts of Sections 1, 25 and 26 of this report and accept professional responsibility for those sections of this technical report;
8) I have read National Instrument 43-101 and confirm that this technical report has been prepared in compliance therewith;
9) Sergio Alvarado Casas (Chile) was retained by Los Andes Copper. to visit the subject property and to contribute to a technical report on the Vizcachitas project in accordance with National Instrument 43-101 and Form 43-101F1 guidelines;
11) I have not received, nor do I expect to receive, any interest, directly or indirectly, in the securities of Los Andes Copper.;
12) That, as of the date of this certificate, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

["Original signed and sealed"]
Sergio Alvarado Casas (Geologist)
Mining Chilean Commission (Registration No. 004)
Tetra Tech Sudamérica S.A.

Santiago, Chile
March 30, 2023

CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled: Pre-Feasibility Study Vizcachitas Project, with Effective Date February 20, 2023, Under NI 43-101 Reporting Standards

I, Mario Riveros, do hereby certify that:

1) I am a Consultant Engineer, a member of Tetra Tech Sudamérica S.A. (Chile office);
2) My professional title is Civil Chemical Engineer, with the degree of Chemical Engineering obtained in 1971 at Universidad de Santiago, Chile. I have practiced my profession continuously since 1971. I have estimated and audited several projects in the mineral processing area, in Chile and other countries;
3) I am a Competent Person from the Chilean Mining Commission, with Registration No. 129 and with the Australian Institute of Mines and Metallurgy (AusIMM), Licence No. 306810;
4) I have not visited the Vizcachitas Project.
5) I have read the definition of Qualified Person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfill the requirements to be a Qualified Person for the purposes of National Instrument 43-101 and this technical report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1;
6) As a Qualified Person, I am independent of Los Andes Copper as defined in Section 1.5 of National Instrument 43-101. As per Exchange Policy requirement (Appendix 3F) I am also independent of the Property. There is no scenario outside of my scope as a qualified person that I would not satisfy the requirement of independence;
7) I am the co-author of this report and responsible for Sections 13, 17, 18 and parts of Sections 1, 25 and 26 of this report and accept professional responsibility for those sections of this technical report;
8) I have read National Instrument 43-101 and confirm that this technical report has been prepared in compliance therewith;
9) Mario Riveros (Chile) was retained by Los Andes Copper. to visit the subject property and to contribute to a technical report on the Vizcachitas project in accordance with National Instrument 43-101 and Form 43-101F1 guidelines;
11) I have not received, nor do I expect to receive, any interest, directly or indirectly, in the securities of Los Andes Copper.;
12) That, as of the date of this certificate, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading. .

["Original signed and sealed"]

Santiago, Chile
March 30, 2023

Mario Riveros (Civil Chemical Engineer)
Mining Chilean Commission (Registration No. 129)
Tetra Tech Sudamérica S.A.

CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled: Pre-Feasibility Study Vizcachitas Project, with Effective Date February 20, 2023, Under NI 43-101 Reporting Standards

I, Ricardo Muñoz, do hereby certify that:

1) I am a Consultant Engineer, a member of Tetra Tech Sudamérica S.A. (Chile office);
2) My professional title is Civil Mining Engineer, with the degree of Civil Mining Engineering obtained in 1995 at Universidad de La Serena, Chile. I have a postgraduate, MBA in Mining Management from the University of Chile in 2014. I was also in the "Citation in Applied Geostatistics" programme of the extension faculty of the University of Alberta of Canada in 2005. I have practiced my profession continuously, for 40 years, in activities associated with the entire geological mining value chain. The experience acquired in these 40 years of practice comes from having participated in various projects and mines, specifically in Chile, Mexico, Argentina, Peru and Brazil. These projects included copper, gold, lead and zinc deposits in different stages of development, from exploration to production where, depending on the state of progress of the project, I have participated in database activities, ore resource estimation, ore control and estimation of resources and reserves.
3) I am a Competent Person from the Chilean Mining Commission, with Registration No. 005 and with the Australian Institute of Mines and Metallurgy (AusIMM), Licence No. 312131;
4) I visited the Vizcachitas Project during March 02-03, 2023;
5) I have read the definition of Qualified Person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfill the requirements to be a Qualified Person for the purposes of National Instrument 43-101 and this technical report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1;
6) As a Qualified Person, I am independent of Los Andes Copper as defined in Section 1.5 of National Instrument 43-101. As per Exchange Policy requirement (Appendix 3F) I am also independent of the Property. There is no scenario outside of my scope as a qualified person that I would not satisfy the requirement of independence;
7) I am the co-author of this report and responsible for Section 14 and parts of Sections 1, 25 and 26 of this report and accept professional responsibility for those sections of this technical report;
8) I have read National Instrument 43-101 and confirm that this technical report has been prepared in compliance therewith;
9) Ricardo Muñoz (Chile) was retained by Los Andes Copper. to visit the subject property and to contribute to a technical report on the Vizcachitas project in accordance with National Instrument 43-101 and Form 43-101F1 guidelines;
11) I have not received, nor do I expect to receive, any interest, directly or indirectly, in the securities of Los Andes Copper.;
12) That, as of the date of this certificate, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading. .

["Original signed and sealed"]
Ricardo Muñoz (Civil Mining Engineer)
Mining Chilean Commission (Registration No. 005)
Tetra Tech Sudamérica S.A.

Santiago, Chile
March 30, 2023

CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled: Pre-Feasibility Study Vizcachitas Project, with Effective Date February 20, 2023, Under NI 43-101 Reporting Standards

I, María Loreto Romo, do hereby certify that:

1) I am a Mineral Resource Estimate Senior Engineer, a member of Tetra Tech Sudamérica S.A (Chile office);
2) My professional title is Civil Mining Engineer, with the degree of Civil Mining Engineering obtained in 2012 at Universidad de Santiago, Chile. I have estimated and audited projects for reserve estimates, due diligence, geologic modeling, and resource estimates management for projects located in Chile and other countries;
3) I am a Competent Person from the Chilean Mining Commission, with Registration No. 371;
4) I have not visited the Vizcachitas Project;
5) I have read the definition of Qualified Person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfill the requirements to be a Qualified Person for the purposes of National Instrument 43-101 and this technical report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1;
6) As a Qualified Person, I am independent of Los Andes Copper as defined in Section 1.5 of National Instrument 43-101. As per Exchange Policy requirement (Appendix 3F) I am also independent of the Property. There is no scenario outside of my scope as a qualified person that I would not satisfy the requirement of independence;
7) I am the co-author of this report and responsible for Section 14 and parts of Sections 1, 25 and 26 of this report and accept professional responsibility for those sections of this technical report;
8) I have read National Instrument 43-101 and confirm that this technical report has been prepared in compliance therewith;
9) María Loreto Romo (Chile) was retained by Los Andes Copper to contribute to a technical report on the Vizcachitas project in accordance with National Instrument 43-101 and Form 43-101F1 guidelines;
11) I have not received, nor do I expect to receive, any interest, directly or indirectly, in the securities of Los Andes Copper;
12) That, as of the date of this certificate, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading. .

["Original signed and sealed"]
María Loreto Romo (Civil Mining Engineer)
Mining Chilean Commission (Registration No. 371)
Tetra Tech Sudamérica S.A.

Santiago, Chile
March 30, 2023