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TETRA TECH

PRELIMINARY ECONOMIC ASSESSMENT OF THE VIZCACHITAS PROJECT

UNDER NI 43 101 REPORTING STANDARD

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TETRA TECH

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1. SUMMARY

1.1 Introduction

Tetra Tech Chile S.A. (Tetra Tech) was commissioned by Los Andes Copper Ltd. to prepare a technical report (Technical Report, Report) for the Vizcachitas copper-molybdenum porphyry project (the Project, the Property, Vizcachitas,) under the standards required by Canadian National Instrument 43-101 (NI 43-101). The engineering cost estimates contained within this report are in line with AACE International guidelines for a Class 5 study.

The Vizcachitas Project is located in the Andes Mountains, in the Province of San Felipe, Fifth Region of Chile. The Project is 100% owned by Los Andes Copper Ltd. (Los Andes Copper, LAC), a company based in Vancouver and listed on the TSX Venture Exchange. This Technical Report has been prepared for Los Andes Copper under the supervision of qualified persons as defined in NI 43-101 to support the dissemination of scientific and technical information relating to the Project.

A previous Preliminary Economic Assessment (PEA) was completed in December 2013, updated in February 2014, and is now updated with this 2019 PEA.

The most significant changes in the 2019 PEA are:

- A new geological model confirmed the importance of the early diorite porphyry and hydrothermal breccias in controlling the higher-grade mineralization of the deposit. The new geological model also separated a near surface higher-grade supergene enriched mineralization. This supergene zone covers area of 400 by 400 metres where all the drill holes have average grades of greater than 0.5% Cu.
- The resources have increased significantly and include declaring Measured Resources for 46% of the projected mill feed for the first 10 years of the 110 ktpd process plant throughput option.
- The metallurgical test work resulted in the adoption of coarser grind for the rougher flotation circuit, increasing the P_{80} from 180 µm used previously to 240 µm.
- The conditions of the Chilean power market have improved drastically from those present in 2014, with projected long term power prices decreasing from 120 USD/MWh to 45 USD/MWh.

A summary of the resources estimated in this PEA, using a 0.25% Cu cut-off is presented in Table 1.1.

Table 1.1: Summary of Resources Estimated 0.25% Cu Cut-Off

Resource Classification	Tonnage (Mt)	Cu (%)	Mo (ppm)	Ag (ppm)	CuEq (%)	Cu (Mlb)	Mo (Mlb)	Ag (Moz)	CuEq (Mlb)
Measured Resources	254.40	0.439	119.2	1.26	0.489	2,462	67	10.3	2,743
Indicated Resources	1,029.67	0.385	146.9	1.00	0.442	8,740	333	33.1	10,034
Measured and Indicated Resources	1,284.06	0.396	141.4	1.05	0.451	11,202	400	43.4	12,777
Inferred Resources	788.82	0.337	127.0	0.88	0.386	5,861	221	22.3	6,713

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: 3.00 USD/lb Cu, 10.00 USD/lb Mo and 17.00 USD/oz Ag. No allowance for metallurgical recoveries has been considered*
- *Small discrepancies may exist due to rounding errors.*
- *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.*
- *Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability*

The Vizcachitas Project is an open pit mine and concentrator plant that produces concentrate. Three alternatives are presented for process plant throughputs namely, 55 ktpd, 110 ktpd and 200 ktpd, with the 110 ktpd case yielding the highest After-Tax IRR and shortest Payback Period.

As a result of mine plans benefitting from the new geological model, the average head grade to the mill for the first five years of operation improved significantly reaching 0.57% CuEq for the 55 ktpd case, 0.53% CuEq for the 110 ktpd case and 0.49% CuEq for the 200 ktpd case.

The summary of the updated economic results is:

Table 1.2: Key Economic Indicators

Key Economic Indicators				
Description	Unit	55 ktpd	110 ktpd	200 ktpd
After -Tax Net Present Value - 8%	kUSD	931,120	1,797,425	2,198,359
After-Tax IRR	%	16.90%	20.77%	17.37%
Initial Capex	kUSD	1,300,034	1,874,797	2,823,469
C1 Cash Cost w/Mo-Ag Credits (First 8 years operation)(*)	USD/lb	1.30	1.36	1.44
Payback Period from operation (*)	Years	4.3	3.4	4.4
Payback Period from construction (**)	Years	6.3	5.4	6.4

(*) Referred to the first year of mill production

(**) Referred to the beginning of construction

Payback period calculated with nominal cash flows

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

- Geotechnical and geo-mechanical modeling (FF Geomechanics).
- Environmental studies, environmental liabilities, permitting and community issues (Los Andes Copper environmental counsel)
- Mining properties, land tenure, legal access, operational permits, adjacent properties (Los Andes Copper mining counsel)
- Metallurgical test work (SGS Mineral Services and Empirica Consultores)

1.2 Project Location

Vizcachitas property is located on 32° 24' 27" S and 70° 25' 30" W in the Andes Mountains, Chile (see Figure 4.1 Vizcachitas Project Location.)

The central UTM coordinates are 19 H 366.000mE 6.413.500mN. (Datum WGS84).

The Property is located approximately 150 km northeast of Santiago, Chile, and 46 km northeast of Putaendo, San Felipe Province. Of the total distance between the Project and Santiago, approximately 125 km is paved, and 25 km is unimproved dirt and gravel roads. The total travel time from Santiago to the Project site is approximately three hours.

1.3 Property Ownership

Compañía Minera Vizcachitas Holding (CMVH) and Sociedad Legal Minera San Jose Uno de Lo Vicuña, El Tártaro y Piguchén de Putaendo (SLM San Jose) hold favourable and valid title deeds to all the mining properties listed in Table 4.1 and Table 4.2. The location of the mining properties is shown in Figure 4.2 and Figure 4.3. CMVH and SLM San Jose are wholly owned subsidiaries of Los Andes Copper.

The exploration claims allow the owner to assess the land in search of minerals. While active, these claims are valid for two years after which they must become mining properties otherwise, a new application must be filed for an exploration claim. The mining properties are permanent and only subject to the payment of annual mining taxes to the Chilean governmental authorities.

The project includes 52 mining properties covering a surface area of 10,771 ha and 108 exploration claims for a combined total of 30,800 ha. All properties have been granted or are in the process of being granted by the court of Putaendo. Certain exploration claims overlap the mining properties, a practice commonly used in Chile to create an additional layer of protection to the underlying properties over the protections already granted by law.

Tetra Tech is not qualified to issue a legal opinion on the status of the mining properties and has relied on a letter dated May 10, 2019, provided by Ossa Alessandri Abogados, who act as mining attorneys for CMVH in Chile

CMVH has signed a legal agreement with the owner of the land giving access to the Vizcachitas Project, which allows the Company to develop exploration and drilling activities. This is an annual agreement that contains provisions for automatic renewal.

1.4 Property Description

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

The Property is located in the western ridges of the Andes Mountains. Elevations range from less than 1,800 masl to more than 3,400 masl, with an average elevation of near 2,100 masl. The exploration camp at Vizcachitas is located at approximately 1,940 masl.

The weather is warm and temperate with six dry months from late spring to the fall season. Average precipitation is about 300 mm per annum 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.

Vegetation consists of shrubs and trees of low to moderate height, which mainly grow in the bottom of the valley near the river.

The access and topography present certain challenges for the Project and must be addressed in the engineering phase. Other Chilean mines, such as Andina, Los Bronces, and Los Pelambres have been developed in similar terrain.

1.5 Geology and Mineralization

The Vizcachitas Project is a mineralized Cu-Mo porphyry system associated with a complex of hydrothermal breccias and porphyries within Miocene volcanic rocks.

In 2015 all the diamond drill core was re-logged. From this updated information, a new lithological, alteration and mineralization model was developed for the Project. The drilling carried out in 2015–2016 and 2017 was designed to confirm the new exploration concepts developed during the generation of the geological model.

The results from the latest drilling, plus the information provided by historical drilling, show that the core of the Vizcachitas mineralized system comprises a high-grade early diorite intrusive complex and two inter-mineral intrusives namely, one early tonalite intrusive and a later granodiorite intrusive. Associated with these intrusives are hydrothermal breccias and magmatic–hydrothermal breccia bodies; these breccias usually carry higher average copper grades. The final intrusive events comprise post-mineral phreatomagmatic breccias and a series of dacite dikes that cut the former units.

Modeling of the Vizcachitas drilling information showed that:

- Historical drilling (prior to 2015) was generally short and has alteration that is consistent with the higher parts of a porphyry system. The drilling carried out in 2015-2016 and 2017, of which some drill holes were more than 1000 m in length, show that the mineralized system is deep and that it may have a mineralized column that is greater than 1000 m in depth.
- The hydrothermal and/or hydrothermal magmatic breccias are relevant in the geological setting for the Vizcachitas Project and generally show copper grades higher than the porphyry phases. Due to their width and overall volume, they are an important part of the higher copper grade mineral resource (the average grade the hydrothermal breccias is 0.54% Cu). The new drilling has helped enhance the spatial modelling of these bodies and demonstrated their lateral continuity to the north.
- The higher grade early diorite porphyry phase has been confirmed by the new drilling. The average Cu grade for this unit is 0.56% Cu. The spatial distribution of this unit should be further investigated during the Pre-Feasibility Study.
- The copper mineralization is mainly hypogene, predominantly chalcopyrite with some bornite. There is a 100-120 m thick discrete “blanket” of secondary enrichment developed on the northeastern part of the system, comprised predominantly of chalcocite, covellite, pyrite and/or pyrite-chalcopyrite.

The latest drilling has significantly improved the economic potential of the Project. Further drilling is required to test the depth extension of the main hydrothermal breccia bodies, the early diorite porphyry, and the northern extension of the breccia corridor.

1.6 Drilling

Five different drilling campaigns have been undertaken on the property from 1993 to date. A total of 165 diamond drill holes have been drilled, with a total of 52,256 m. The total drilled metres by campaign are summarized in Table 1.3.

Table 1.3: Summary 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	2015-2016	V2015-01 to V2015-08	8	3,610
Los Andes Copper	2017	V2017-01A to V2017-11	11	8,262
Total			**165**	**52,256**

1.7 Mineral Processing and Metallurgical Testing

The Vizcachitas Project has been the subject of a number of physical characterization and metallurgical test programmes to determine the process flow sheet and expected recoveries. Physical characteristics such as the Bond Work and Abrasion Indices have been used in the development of the process estimates in this Technical Report. Leach and flotation test work has also been carried out to further validate process selection and recovery estimations.

The main conclusions of the test work programmes are summarized below:

- Mineralogical analysis showed that the principal copper mineral is chalcopyrite.
- In general, the results of the flotation tests showed both high-grade copper concentrates and high recoveries of both copper and molybdenum are achievable.
- The results suggest that the rougher flotation recoveries are not significantly impacted by the P_{80} on the range analyzed and, on this basis, a coarser primary grind P_{80} of 240 µm is proposed.
- The results of the cleaner flotation tests indicated that three cleaner stages should be considered to achieve a high final concentrate grade.
- Based on the flotation tests, overall recoveries of 91% Cu and 75% Mo can be expected:
    - A copper recovery of 95% in the rougher circuit and 96% in the cleaner circuit

  - A molybdenum recovery of 84% in the collective circuit and 89% in the selective circuit
- The results of the agitated leaching tests showed that the samples tested had a high content of chalcopyrite (78%) and a low acid soluble copper content (<10%). These samples generally showed low copper extractions (< 15%).
- The majority of the mineral zone historically logged as oxide contains a large proportion of sulphide mineralization and metallurgical test work previously completed indicates that leach extractions are uneconomical.

1.8 Mineral Resource Estimate

The mineral resources are contained within an open pit shell to demonstrate the reasonable prospects of eventual economic extraction. Only blocks within the Whittle pit shell are included in the mineral resources. The mineral resources using an open pit mining method are reported below.

Resources estimated for the Vizcachitas Project are shown in Table 1.4 to Table 1.7. The estimate was based on chemical analyses of drill hole samples, the interpretation of an updated geological model and the geostatistical analysis using standard industry methods. Resources were classified according to the CIM standards on Mineral Resources and Mineral Reserves.

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.

The in-pit mineral resources are reported using a 0.25% copper cut-off.

- Measured mineral resources are 254.4 million tonnes grading 0.439% copper, 119.2 ppm molybdenum and 1.26 g/t silver giving a 0.489% copper equivalent.
- Indicated mineral resources are 1,029.67 million tonnes grading 0.385% copper, 146.9 ppm molybdenum and 1.00 g/t silver giving a 0.442% copper equivalent.
- Measured and Indicated mineral resources are 1,284.06 million tonnes grading 0.396% copper, 141.4 ppm molybdenum and 1.05 g/t silver giving a 0.451% copper equivalent.
- The Inferred mineral resources are 788.82 million tonnes grading 0.337% copper, 127.0 ppm molybdenum and 0.88 g/t silver giving a 0.386% copper equivalent.

The tables Table 1.4, Table 1.5, Table 1.6 and Table 1.7 present a sensitivity analysis for the mineral resources under different cut-off grades. The base case for the estimation of resources is 0.25% Cu.

Table 1.4: Measured Resources In-Pit

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 1.5: Indicated Resources In-Pit

Indicated Resources									
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 1.6: Measured and Indicated Resources In-Pit

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 1.7: Inferred Resources In-Pit

Inferred Resources									
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: 3.00 USD/lb Cu, 10.00 USD/lb Mo and 17.00 USD/oz Ag. No allowance for metallurgical recoveries has been considered*
- *Small discrepancies may exist due to rounding errors.*
- *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.*
- *Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability*

1.9 Mineral Reserves Estimate

The Project has no Mineral Reserves; all mineralization is considered as Mineral Resources.

1.10 Mining Methods

The Vizcachitas Project is amenable to conventional large-scale open pit mining methods.

Evaluations were conducted to determine potentially economic pit limits and the mining phase (pushback) development sequence for three process plant throughputs namely, 55 ktpd, 110 ktpd and 200 ktpd.

Mining phases were designed using geotechnical parameters and an extraction sequence defined by the Whittle software. These phases were used to estimate contained mineral resources from which a mine production schedule was developed. Using grade-differentiated stockpiles and preferentially mining the higher-grade supergene, hydrothermal breccias and the early diorite porphyries, the mill feed grade for the first five years of operation is 0.57% CuEq for the 55 ktpd, 0.53% CuEq for the 110 ktpd and 0.49% CuEq for the 200 ktpd cases. Table 1.8 shows the mill feed grade for the first 5 years for the 55 ktpd, 110 ktpd and 200 ktpd cases.

Table 1.8: Mill Feed Summary for First Five Years of Mine Production

Mine Plan Option	Cut-Off Grade	Mineral to Mill					Waste
		Total					
		kt	CuEq (%)	Cu (%)	Mo (ppm)	Ag (g/t)	kt
Case 1: 55 ktpd	0.40%	95,400	0.57%	0.52%	125	1.5	135,498
Case 2: 110 ktpd	0.34%	190,800	0.53%	0.47%	129	1.3	255,524
Case 3: 200 ktpd	0.27%	308,937	0.49%	0.44%	113	1.2	496,110

The slope angles estimated for the sectors of each pit designed were based on the recommendations of the 2018 FF Geomechanics report, "Informe de Modelamiento Geotécnico Preliminar y Análisis de Estabilidad Geomecánica Global para Preliminary Economic Assessment (PEA), Rajo Vizcachitas- Junio – 2018".

Table 1.9 shows the total material movement for each case including the rehandling of the stockpile material

Table 1.9: Material Movement for Each Case

Case	Mine life (Years)	Mill feed (Mt)	Waste (Mt)	W/O (Ratio)	Total Incl. Rehandling (Mt)
55 ktpd	59	1,109	1,102	0.99	2,626
110 ktpd	45	1,665	2,170	1.31	4,263
200 ktpd	30	1,939	2,654	1.37	5,056

Mine fleet, scheduling, manpower requirements and costs were developed for each production scenario as described in detail in Chapter 16 of this Report.

1.11 Process Design and Recovery

The Vizcachitas concentrator operation was analyzed for three alternative throughputs namely, 55 ktpd, 110 ktpd and 200 ktpd. The process plant will process run-of-mine (ROM) material delivered from the open pit and material re-handled from stockpiles utilized to optimize head grade to mill. Copper and molybdenum concentrates as well as tailings will be produced. The proposed process includes crushing and grinding of the mineral, bulk copper-molybdenum rougher and cleaner flotation, regrinding, copper-molybdenum separation, molybdenum flotation, and dewatering of copper and molybdenum concentrates.

The flotation tailings will be thickened to 72% solids before placement in a tailings storage facility (TSF).

Copper and molybdenum concentrates will be shipped by road to a nearby railhead and from there delivered to a port facility for shipment to the final consumer. See Table 1.10 for additional detail.

Table 1.10: Summary of LOM Concentrates and Fine Metal Produced

Case	Total Capacity (ktpd)	Concentrate Produced (kt)	Fine Metal Produced	
			Cu (kt)	Mo (kt)
1	55	11,706	3,512	92
2	110	16,646	4,994	138
3	200	18,666	5,600	156

1.12 Project Infrastructure

The Project is in close proximity to extensive infrastructure, including port facilities, railway lines and high-tension substations. The Project further benefits from a low altitude location permitting year-round exploration and project development.

The Project's location in central Chile means that a significant amount of the infrastructure to support a mine operation is located in relative proximity. The nearby cities of Los Andes and San Felipe are used as a base for many employees and subcontractors who work at the Codelco's Andina mine. Anglo American's Chagres smelter and Codelco's Ventanas smelter are located 90 km and 140 km, respectively, from the project. The port of Ventanas is 140 km away and currently handles copper concentrate from other mining operations in the district. There is an operating railway line in San Felipe with connections to the two smelters and the port of Ventanas. The PEA considers shipping of copper concentrate by rail to Ventanas.

There are several large power substations near to the project site. The Nogales substation is part of the national 220 kV power distribution system and the Las Vegas substation part of the 110 kV system. The Rocin River flows through the project site and is a tributary of the Putaendo and the Aconcagua Rivers. Los Andes Copper currently owns the water rights for a substantial portion of the anticipated water requirements, with an extraction point located along the Aconcagua River approximately 80 km from the project site. To implement the project, Los Andes Copper may have to secure additional water rights.

The Project is a greenfield site and local infrastructure will need to be built. This will include the process water supply, Rocin River diversion tunnel, power supply from the national grid substation to the project site, access road upgrade, concentrate storage and concentrate loading facilities at the railhead.

The power for the project is considered to be provided by the National System (Electric Power Grid - SIC) and connecting to the Nogales substation or Las Vegas substation, 105 km and 74 km away respectively.

1.13 Marketing Studies

The assumptions used for the marketing and handling of copper and molybdenum concentrates have been derived from contacts with smelters, recent data from other projects and information in the public domain.

Commodity Supply and Demand

Most analysts believe that the long negative cycle for copper and other commodities has already ended. It is expected that with anticipated economic growth for China, India, Europe and United States, medium-term copper prices should rise again. In the short term, uncertainty due to the possible trade dispute between US and China is causing slower recovery than expected.

Smelter Capacities and Utilization

For the most part smelter capacities are fixed. The relationship between capacity and utilization dictates a smelter's profitability, hence it's setting of treatment charges (TC), refining charges (RC) and other costs.

In the long term, TCs and RCs are expected to increase to cover additional smelter costs, particularly as environmental legislation becomes more stringent regarding atmospheric and other emissions. The PEA has considered TCs of 102 USD/t and RCs of 10.2 cUSD/lb.

Ocean Freight

Currently, the availability of vessels significantly exceeds demand. Opportunities for reasonable freight costs are available, particularly in relation to the negotiation of long-term freight contracts. The PEA has assumed a cost of 10.09 cUSD/lb for freight and insurance.

Future Metals Pricing

The general industry consensus is for copper prices to stabilise as developing countries such as India and China take up production once again. The PEA has considered using metals values of 3.00 USD/lb copper, 22.0 USD/kg molybdenum and 17.0 USD/oz silver as the base case.

1.14 Capital Cost Estimate

Capital cost estimates are comprised of the following:

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

Contingency estimation was based upon a consideration of direct cost and indirect costs. After incorporating the recommended contingency, the capital cost estimate is considered to have a level of accuracy of +/-35%.

Table 1.11 summarizes initial, sustaining and deferred capital requirements.

Table 1.11: Capital Cost Summary (Nominal values)

Description	Initial			Sustaining and Deferred		
	55 ktpd	110 ktpd	200 ktpd	55 ktpd	110 ktpd	200 ktpd
Direct Costs						
Diversion Rocin River	52,912	52,912	52,912	2,500	2,500	2,500
Access	29,731	29,731	23,557			
Concentrate Transport	29,932	29,932	29,932			
Pipeline Rocin-Chalaco			73,749			
General Facilities	32,746	35,000	40,754			
Operations Platform	29,820	35,145	76,680			
Mine	184,363	277,465	359,328	624,333	1,025,196	1,218,390
Plant	228,440	439,016	687,999			
Tailing Management Facilities	152,290	173,057	230,639		98,250	
Water Reclaim System	2,926	3,653	4,430			
Water Supply System	35,844	47,382	62,132			
Power Supply System	88,125	124,539	168,597			
Total Direct Costs	**867,129**	**1,247,831**	**1,810,708**	**626,833**	**1,125,946**	**1,220,890**
Total Indirect Costs	**164,299**	**242,672**	**361,191**			
Contingency	**268,605**	**384,294**	**651,570**			
Total Capital Cost	**1,300,034**	**1,874,797**	**2,823,469**			

1.15 Operating Costs Estimate

Operating costs have been estimated for Mining, Process Plant, Infrastructure and Administration. Costs are reported under subheadings related to the function of each of the areas identified.

The operating cost estimates are based on long term power prices of 45 USD /MWh and 1.00 USD/l for diesel fuel. Labour costs for mine and process plant consider up to superintendent level and all superior positions are considered as administration costs.

The operating costs are considered to have an accuracy of ± 35%, based on the assumptions listed in this section of the Report.

All estimates have been completed for Life of Mine (LOM).

Table 1.12: Estimation Period by Case

Case 55 ktpd	Case 110 ktpd	Case 200 ktpd
LOM	LOM	LOM
59 years	45 years	30 years

Table 1.13 summarizes the operating costs estimate for the first 8 years and Table 1.14 for LOM.

Table 1.13: Operating Costs Estimate (USD/t plant feed; Nominal values, average first 8 years)

Description	Case 55 ktpd	Case 110 ktpd	Case 200 ktpd
Mine	4.75	4.27	4.90
Plant	5.11	4.92	4.70
Infrastructure	0.18	0.18	0.18
Administration	0.20	0.19	0.20
Total (USD/t)	**10.24**	**9.57**	**9.98**

(*) Mine costs include the waste/mineral ratio for the first 8 year of operation

Table 1.14: Operating Cost Estimate (USD/t plant feed; Nominal values, average for LOM)

Description	Case 55 ktpd	Case 110 ktpd	Case 200 ktpd
Mine	3.59	4.40	4.72
Plant	5.11	4.92	4.70
Infrastructure	0.18	0.18	0.18
Administration	0.18	0.19	0.19
Total (USD/t)	**9.06**	**9.70**	**9.79**

Table 1.15 summarizes the average C1 cash cost for the first 8 years of operation with and without molybdenum and silver credits. Table 1.16 summarizes the average C1 cash cost for the life of mine with and without molybdenum and silver credits

Table 1.15: Average First 8 Years Cash Costs

Description	Unit	55 ktpd	110 ktpd	200 ktpd
Operating Costs	kUSD	1,599,569	2,975,458	5,148,375
NSR	kUSD	86,010	149,521	239,750
Royalty	kUSD	129,807	231,674	328,655
TC/RC	kUSD	5	681,472	1,093,403
Transportation	kUSD	146,771	253,195	406,245
Total Cash Cost w/o Credits	**kUSD**	**2,357,188**	**4,291,322**	**7,216,429**
Molybdenum and Silver Credit	kUSD	333,678	632,903	1,007,854
Total Cash Cost w/ Credits	**kUSD**	**2,023,511**	**3,658,418**	**6,208,574**
Total Copper to be Sold	Mlb	1,561,392	2,693,566	4,321,751
First 8 Years Cash Cost				
Average Cu Cash Cost w/o Mo-Ag Credit	**USD/lb**	**1.51**	**1.59**	**1.67**
Average Cu Cash Cost w/ Mo-Ag Credit	**USD/lb**	**1.30**	**1.36**	**1.44**

Table 1.16: Average Life of Mine Cash Costs

Description	Unit	55 ktpd	110 ktpd	200 ktpd
Operating Costs	kUSD	10,097,016	15,268,600	17,469,645
NSR	kUSD	434,760	620,835	696,754
Royalty	kUSD	641,679	790,004	819,253
TC/RC	kUSD	1,958,779	2,785,373	3,123,334
Transportation	kUSD	727,768	1,034,882	1,160,448
Total Cash Cost w/o Credits	**kUSD**	**13,860,001**	**20,499,695**	**23,269,434**
Molybdenum and Silver Credit	kUSD	2,068,426	3,071,756	3,473,998
Total Cash Cost w/ Credits	**kUSD**	**11,791,575**	**17,427,938**	**19,795,437**
Total Copper to be Sold	Mlb	7,742,210	11,009,381	12,345,195
Life of Mine Cash Cost				
Average Cu Cash Cost w/o Mo-Ag Credit	**USD/lb**	**1.79**	**1.86**	**1.88**
Average Cu Cash Cost w/ Mo-Ag Credit	**USD/lb**	**1.52**	**1.58**	**1.60**

1.16 Economic Analysis

Economic parameters used for the evaluation are shown in Table 1.17.

Table 1.17: Main Economic Parameters

Description	Value	Unit
Cu Price	3.0	USD/lb
Mo Price	22.0	USD/kg
Ag Price	17.0	USD/oz
Energy Costs	45.0	USD/MWh
Inflation	None	---
Currency Fluctuation	None	---

There is no certainty that the PEA results will be realized. Since the analysis is based on a cash flow estimate, it should be expected that actual economic results might vary from the estimates. The PEA has been completed to a level of accuracy of ± 35%. The PEA is not a preliminary feasibility study (PFS) or a feasibility study.

At present, Chilean corporate income tax is 27%. However, there are several factors which impact the overall amount companies pay which may include the profits reinvested in the country, the capital structure of the project and others. Dividends repatriated are subject to additional taxes which can bring the nominal income tax to 35%. Mining companies are also levied with a specific tax on mining operating profits (Mining Royalty Tax). A more detailed summary of the tax regime in Chile is presented in Chapter 22.3 of this Report.

The model includes mine closure costs as required by Chilean legislation. Closure costs for each option considered have been calculated as 5% of the initial direct capital costs and where an expansion has been considered a further 5% of the direct capital has been assigned to process plant capital costs. Further details of this closure cost can be found in Section 22.1 of this Report.

An NSR of 2% has been contemplated in the models, as well as a residual value of 13.75 cUSD/lb of copper on the ground based on remaining copper for each case at the end of the projected operational mine life.

Table 1.18 shows the estimated pre and after-tax net present value (NPV), internal rate of return (IRR) and payback periods for the cases presented. The Mining Royalty Tax is deducted in all cases. This is further discussed in Sections 22.6, 22.7 and 22.8 of this Report.

Table 1.18: Economic Evaluation Summary, Pre and After-Tax

Financial Indicators - Pre-Tax				
Description	Unit	55 ktpd	110 ktpd	200 ktpd
Net Present Value - 8%	kUSD	1,370,914	2,595,839	3,201,879
IRR	5.11	19.73%	24.73%	20.07%
Payback Period (*)	Years	4.1	3.0	4.2
Payback Period construction (**)	Years	6.1	5.0	6.2
Financial Indicators - After-Tax				
Description	Unit	55 ktpd	110 ktpd	200 ktpd
Net Present Value - 8%	kUSD	931,120	1,797,425	2,198,359
IRR	%	16.90%	20.77%	17.37%
Payback Period (*)	Years	4.3	3.4	4.4
Payback Period construction (**)	Years	6.3	5.4	6.4

(*) Referred to the first operation year

(**) Referred to the beginning of construction

Payback period calculated w ith nominal cash flow s

1.17 Conclusions and Recommendations

The Project is a large, medium grade copper deposit which can be exploited using conventional open pit and concentrator technology.

No fatal flaws were identified during the Vizcachitas Project study. The recommendations are based on normal metallurgical and other development test work which would be part of project development.

At the metals prices used, the option which gives the highest NPV and fastest project payback period is that of a mill throughput of 110 ktpd with an after-tax Net Present Value (8% discount rate) of kUSD 1,797,425 and an IRR of 20.77% with a payback of 3.4 years from the first operational year (5.4 years from beginning of construction).

Opex and Capex considerations used for the Project represent those expected for a project of this type exhibiting average mineral abrasiveness and hardness characteristics, and grades and rock type characterizations as indicated in the geological section. Operating costs were generated from first principles and bench marked against other operations. Capital costs were based on quotations for mining equipment, database information and were also benchmarked against similar operations

The mine plan is appropriate to the mineralization and adequately reflects the deposit type, dimensions and host rock characterization

A description of the main risks and opportunities for this Project is presented in Chapter 25.2 of this Report.

Recommendations

Based on the results of the PEA, the Qualified Persons recommend that Los Andes Copper complete a pre-feasibility study (PFS) to further define the Project alternatives to more accurately assess its technical and economic viability and to support permitting activities.

When all additional metallurgical and other test work has been completed, a trade-off evaluation should confirm that the considerations used in selecting the 110 ktpd option as the preferred option are still valid and that it is the preferred option to develop to PFS level. Also, Tetra Tech recommends this alternative as the base case scenario from its own experience in many other projects of similar size and profile. The 55 ktpd alternative seems too small for this size of mineral deposit, while the 200 ktpd alternative demands higher CAPEX and would increase execution risk.

Additional metallurgical studies regarding material characterization and metals recovery should be completed which may provide further input into process plant design and optimization.

Future geotechnical studies based on new geotechnical drilling are also recommended.

2. INTRODUCTION AND TERMS OF REFERENCE

2.1 Purpose of the Technical Report

Los Andes Copper commissioned Tetra Tech to prepare a Technical Report for the Vizcachitas Copper-Molybdenum Project in compliance with the standards required by Canadian National Instrument 43-101 for a Preliminary Economic Assessment. This corresponds with AACE International recommendations for a class 5 study. The operating and capital costs for the project were estimated within an accuracy of ± 35%.

The scope of the work included the evaluation of a range of mill throughputs which sought to maximise resource exploitation and minimise initial capital expenditure. Those cases which reported the highest NPV outcomes are presented in this Technical Report.

The Vizcachitas copper and molybdenum porphyry mineral deposit is located in the Fifth Region of Chile, in San Felipe Province, and is owned by Los Andes Copper Ltd., a Vancouver, B.C. based company listed on TSX Venture Exchange. This Technical Report has been prepared for Los Andes Copper by or under the supervision of qualified persons within the purview of NI 43-101 to support the dissemination of scientific and technical information of Los Andes Copper for the Project.

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

- Geotechnical and geo-mechanical modeling (FF Geomechanics).
- Environmental studies, environmental liabilities, permitting and community issues (Los Andes Copper environmental counsel)
- Mining properties, land tenure, legal access, operational permits, adjacent properties (Los Andes Copper mining counsel)
- Metallurgical test work (SGS Minerals and Empirica Consultores)

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

All units in this Technical Report are based on the International System of Units ("SI"), except for units that are industry standards, such as troy ounces for the mass of precious metals. The currency used is United States Dollars ("USD" or "$"), 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	Litre per second	l/s
American Dollar	USD	Maximum	max
American Dollar Cent	cUSD	Mega Volt Ampere	Mva
Centigrade	*C	Megawatt	MW
Centimetre	cm	Megawatt- hour	MWh
Chilean peso	CLP	Metre	m
Copper	Cu	Metre per hour	m/h
Copper Soluble	CuS	Metre per second	m/s
Copper Total	CuT	Metres above sea level	masl
Copper cyanide	CuCN	Metric tonne	t
Copper equivalent	CuEq	Metric tonne per day	tpd
Cubic foot/feet	ft^3	Metric tonne per hour	tph
Cubic metre	m^3	Microns	µm
Cubic metre per hour	m^3/h	Milligram per litre	mg/L
Day	d	Millimetre	mm
Foot/feet	ft	Million	M
Gram/litre	g/l	Million tonnes per annum	Mpa
Horse power	hp	Minutes	min
Hour	h	Molybdenum	Mo
Insoluble copper	Icu	Part per billion	ppb
Kilo tonne	kt	Part per million	ppm
Kilo tonne per day	ktpd	Percent	%
Kilogram	kg	Pounds	lb
Kilogram per tonne	kg/t	Run of Mine	ROM
Kilometre	km	Short ton	st
Kilovolt	kV	Specific gravity	SG
Kilovolt amp	kVA	Square metres	m^2
Kilowatt	kW	Square metres per tonne per d	m^2/tpd
Kilowatt hour	kWh	Tonnes per day	tpd
Kilowatt hour per cubic metre	kWh/m^3	Tonnes per hour	tph
Kilowatt hour per metric tonne	kWh/t	Troy Ounces	oz
Kilowatt hour per shot tonne	kWh/st	Weight (mass)	wt
Life of mine	LOM	Weight (mass) per cent	%w/w
Litre	l	Work Index	Wi
		Year	y

2.3 Effective Dates (LAC)

The Effective Date of this report is May 10, 2019

The mineral resource estimate and block model was completed on November 6, 2018.

The current personal inspections by qualified persons were completed on January 8, 2016 and May 10, 2017.

The final plans for the PEA mine were issued on November 6, 2018.

The PEA Mineral process engineering and capital cost estimates were completed on January 30, 2019. The cost estimates were reviewed to confirm their effectiveness at the time of publishing the PEA.

The project infrastructure review was completed on January 30, 2019.

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

3.1 Other Independent Expert Persons

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

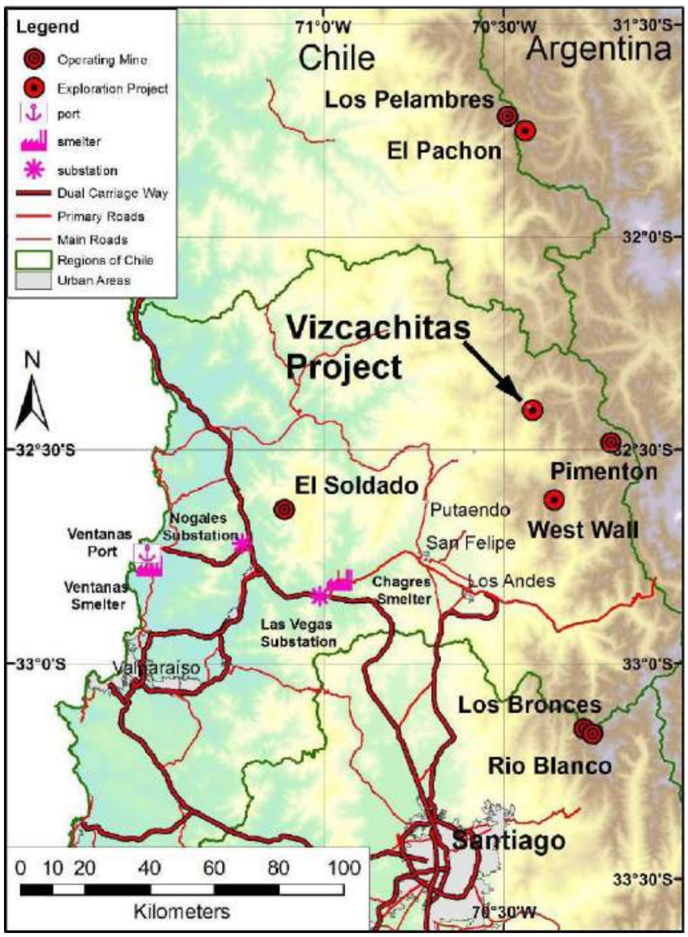
The results and opinions expressed in this Technical Report are accurate and complete as of its issuing date, and no information that may affect the conclusions included herein has been withheld. The authors have the right review this report and the conclusions if they become aware of additional material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

The authors of this Technical Report are not qualified to provide extensive comment on legal issues associated with the Property. For portions of Section 4 dealing with the types and number of mineral tenures and licenses, the nature and extent of Los Andes Copper's title and interest in the Property, the terms of any royalties, back-in rights, payments or other agreements and encumbrances to which the Project is subject, Tetra Tech has relied on the legal opinion of Ossa Alessandri Abogados, lawyers for land tenure, and Los Andes Copper for information related to the NSRs applicable to the project. Copies of these opinions are included as Appendix A.

4. PROPERTY DESCRIPTION AND LOCATION

The Vizcachitas Property is located at 32° 24' 27" S and 70° 25' 30" W in the Andes Mountains of Chile (See Figure 4.1: Vizcachitas Project Location). The central UTM coordinates are 19 H 366.000mE 6.413.500mN. (Datum WGS84).

Figure 4.1: Vizcachitas Project Location (source: LAC, November 2018)



4.1 Land Tenure

Exploration claims entitle the holder to assess the mining potential of the land. As long as the annual tax payments are made to the Chilean Treasury, the exploration claims are valid for a period of two years during which the holder has a preferential right to convert exploration claims to mining properties. Alternatively, a new application must be filed for an exploration claim. The mining properties are permanent and are only subject to payment of the annual mining taxes due to the Chilean Treasury.

The project includes 52 mining properties covering a surface area of 10,771 ha and 108 exploration claims with a combined total of 30,800 ha. All the mining properties are 100% owned by CMVH or by SLM San Jose and have been granted or are in the process of being granted by the court of Putaendo. Part of the exploration claims overlap the mining properties, a practice commonly used in Chile to create an additional layer of protection to the underlying properties over the protections already granted by law. Table 4.1, Table 4.2, Figure 4.2 and Figure 4.3 show the exploration claims and mining properties currently held.

Tetra Tech is not qualified to issue a legal opinion on the mining property and has relied on the letter dated May 10, 2019 provided by Ossa Alessandri Abogados, who act as mining attorneys for CMVH in Chile. See 31 31 APPENDIX III. This letter establishes that:

"We hereby inform you that currently the mining property of Compañía Minera Vizcachitas Holding ("CMVH") is composed of 124 mining concessions: 49 of which are exploitation concessions and 75 exploration concessions. In addition, CMVH currently owns 2 exploitation concessions and 33 exploration concessions that are in the process of being granted.

Sociedad Legal Minera San Jose, a wholly owned subsidiary of Los Andes Copper Ltd., is also the owner of one exploitation concession.

Except for the aforementioned concessions that are still to be granted, all of the foregoing concessions were legally constituted as they were granted by the court of Putaendo and duly registered in the Mining Registry of such city. Additionally, as of this date these concessions do not have any outstanding taxes or permits that need to be paid for."

Table 4.1: Mining Claims as of May 2019

Nº	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 1AL 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 60	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	In Process
52	ROJO 9 1 AL 30	CIA. MRA VIZCAHITAS HOLDING	056040717-3	300	In Process
Total Nº Claims		52	**Total Hectares**	**10,771**	

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Table 4.2: Exploration Claims as of May 2019 (1 of 2)

N°	Exploration Claim Name	Owner	ROL NACIONAL	Hectares	Validity
1	PAOLA 1	CIA. MRA VIZCACHITAS HOLDING	056041943-0	200	2020/01/17
2	PAOLA 2	CIA. MRA VIZCACHITAS HOLDING	056041942-2	200	2020/01/17
3	CHAL 1	CIA. MRA VIZCACHITAS HOLDING	056041964-3	300	2020/08/17
4	CHAL 2	CIA. MRA VIZCACHITAS HOLDING	056041965-1	300	2020/08/17
5	CHAL 3	CIA. MRA VIZCACHITAS HOLDING	056041966-K	300	2020/08/17
6	CHAL 4	CIA. MRA VIZCACHITAS HOLDING	056041967-8	300	2020/08/17
7	CHAL 5	CIA. MRA VIZCACHITAS HOLDING	056041968-6	300	2020/08/17
8	CHAL 6	CIA. MRA VIZCACHITAS HOLDING	056041969-4	300	2020/08/17
9	CHAL 7	CIA. MRA VIZCACHITAS HOLDING	056041970-8	300	2020/08/17
10	CHAL 8	CIA. MRA VIZCACHITAS HOLDING	056041971-6	300	2020/08/17
11	CHAL 9	CIA. MRA VIZCACHITAS HOLDING	056041972-4	300	2020/08/17
12	CHAL 10	CIA. MRA VIZCACHITAS HOLDING	056041973-2	300	2020/08/17
13	CHAL 11	CIA. MRA VIZCACHITAS HOLDING	056041974-0	300	2020/08/17
14	CHAL 12	CIA. MRA VIZCACHITAS HOLDING	056041975-9	300	2020/08/17
15	CHAL 13	CIA. MRA VIZCACHITAS HOLDING	056041976-7	300	2020/08/17
16	CHAL 14	CIA. MRA VIZCACHITAS HOLDING	056041977-5	300	2020/08/17
17	CHAL 15	CIA. MRA VIZCACHITAS HOLDING	056041978-3	300	2020/08/17
18	CHAL 16	CIA. MRA VIZCACHITAS HOLDING	056041979-1	300	2020/08/17
19	CHAL 17	CIA. MRA VIZCACHITAS HOLDING	056041980-5	300	2020/08/17
20	CHAL 18	CIA. MRA VIZCACHITAS HOLDING	056041981-3	300	2020/08/17
21	TOTORA 1	CIA. MRA VIZCACHITAS HOLDING	056042008-0	300	2020/11/28
22	TOTORA 3	CIA. MRA VIZCACHITAS HOLDING	056042007-2	300	2020/11/28
23	TOTORA 4	CIA. MRA VIZCACHITAS HOLDING	056042006-4	300	2020/11/28
24	TOTORA 5	CIA. MRA VIZCACHITAS HOLDING	056042005-6	300	2020/11/28
25	TOTORA 6	CIA. MRA VIZCACHITAS HOLDING	056042004-8	300	2020/11/28
26	MAITEN 1	CIA. MRA VIZCACHITAS HOLDING	056042003-K	300	2020/11/28
27	MAITEN 2	CIA. MRA VIZCACHITAS HOLDING	056042002-1	300	2020/11/28
28	MAITEN 3	CIA. MRA VIZCACHITAS HOLDING	056042001-3	300	2020/11/28
29	MAITEN 4	CIA. MRA VIZCACHITAS HOLDING	056042000-5	300	2020/11/28
30	MAITEN 5	CIA. MRA VIZCACHITAS HOLDING	056041999-6	300	2020/11/28
31	MAITEN 6	CIA. MRA VIZCACHITAS HOLDING	056041998-8	300	2020/11/28
32	MAITEN 7	CIA. MRA VIZCACHITAS HOLDING	056041997-K	300	2020/11/28
33	MAITEN 8	CIA. MRA VIZCACHITAS HOLDING	056041996-1	300	2020/11/28
34	MAITEN 9	CIA. MRA VIZCACHITAS HOLDING	056041995-3	300	2020/11/28
35	MAITEN 10	CIA. MRA VIZCACHITAS HOLDING	056041994-5	300	2020/11/28
36	MAITEN 11	CIA. MRA VIZCACHITAS HOLDING	056041993-7	300	2020/11/28
37	MAITEN 12	CIA. MRA VIZCACHITAS HOLDING	056041992-9	300	2020/11/28
38	MAITEN 13	CIA. MRA VIZCACHITAS HOLDING	056041991-0	300	2020/11/28
39	MAITEN 14	CIA. MRA VIZCACHITAS HOLDING	056041990-2	300	2020/11/28
40	MAITEN 15	CIA. MRA VIZCACHITAS HOLDING	056041989-9	200	2020/11/28
41	MAITEN 16	CIA. MRA VIZCACHITAS HOLDING	056041988-0	200	2020/11/28
42	TOTORA 2	CIA. MRA VIZCACHITAS HOLDING	056041982-1	300	2020/12/20
43	ESPINO 1	CIA. MRA VIZCACHITAS HOLDING	056042009-9	200	2020/12/20
44	ESPINO 2	CIA. MRA VIZCACHITAS HOLDING	056042010-2	200	2020/12/20
45	MAITEN 17	CIA. MRA VIZCACHITAS HOLDING	056042011-0	300	2020/12/20
46	MAITEN 18	CIA. MRA VIZCACHITAS HOLDING	056042012-9	300	2020/12/20
47	VERDE 1	CIA. MRA VIZCACHITAS HOLDING	056042013-7	300	2020/12/20
48	VERDE 2	CIA. MRA VIZCACHITAS HOLDING	056042014-5	300	2020/12/20
49	VERDE 3	CIA. MRA VIZCACHITAS HOLDING	056042015-3	300	2020/12/20
50	VERDE 4	CIA. MRA VIZCACHITAS HOLDING	056042016-1	200	2020/12/20
51	VERDE 5	CIA. MRA VIZCACHITAS HOLDING	056042017-K	200	2020/12/20
52	VERDE 6	CIA. MRA VIZCACHITAS HOLDING	056042018-8	300	2020/12/20

Table 4.3: Exploration Claims as of May 2019 (2 of 2)

N°	Exploration Claim Name	Owner	ROL NACIONAL	Hectares	Validity
53	VERDE 7	CIA. MRA VIZCACHITAS HOLDING	056042027-7	300	2020/12/20
54	VERDE 8	CIA. MRA VIZCACHITAS HOLDING	056042019-6	300	2020/12/20
55	VERDE 9	CIA. MRA VIZCACHITAS HOLDING	056042020-K	300	2020/12/20
56	VERDE 10	CIA. MRA VIZCACHITAS HOLDING	056042021-8	300	2020/12/20
57	VERDE 11	CIA. MRA VIZCACHITAS HOLDING	056042022-6	300	2020/12/20
58	VERDE 12	CIA. MRA VIZCACHITAS HOLDING	056042024-2	300	2020/12/20
59	VERDE 13	CIA. MRA VIZCACHITAS HOLDING	056042025-0	300	2020/12/20
60	VERDE 14	CIA. MRA VIZCACHITAS HOLDING	056042023-4	300	2020/12/20
61	VERDE 15	CIA. MRA VIZCACHITAS HOLDING	056042026-9	200	2020/12/20
62	PEUMO 1	CIA. MRA VIZCACHITAS HOLDING	056042086-2	200	2021/03/25
63	PEUMO 2	CIA. MRA VIZCACHITAS HOLDING	056042085-4	200	2021/03/25
64	PEUMO 3	CIA. MRA VIZCACHITAS HOLDING	056042084-6	300	2021/03/25
65	PEUMO 4	CIA. MRA VIZCACHITAS HOLDING	056042083-8	300	2021/03/25
66	PEUMO 5	CIA. MRA VIZCACHITAS HOLDING	056042082-K	300	2021/03/25
67	PEUMO 6	CIA. MRA VIZCACHITAS HOLDING	056042081-1	300	2021/03/25
68	PEUMO 7	CIA. MRA VIZCACHITAS HOLDING	056042080-3	300	2021/03/25
69	PEUMO 8	CIA. MRA VIZCACHITAS HOLDING	056042079-K	300	2021/03/25
70	PEUMO 9	CIA. MRA VIZCACHITAS HOLDING	056042078-1	300	2021/03/25
71	PEUMO 10	CIA. MRA VIZCACHITAS HOLDING	056042077-3	300	2021/03/25
72	PEUMO 11	CIA. MRA VIZCACHITAS HOLDING	056042076-5	300	2021/03/25
73	PEUMO 12	CIA. MRA VIZCACHITAS HOLDING	056042075-7	300	2021/03/25
74	PEUMO 13	CIA. MRA VIZCACHITAS HOLDING	056042074-9	300	2021/03/25
75	PALMA 2	CIA. MRA VIZCACHITAS HOLDING	056042087-0	300	2021/03/25
76	LOICA 1	CIA. MRA VIZCACHITAS HOLDING	056042110-9	300	In Process
77	LOICA 2	CIA. MRA VIZCACHITAS HOLDING	056042112-5	300	In Process
78	LOICA 3	CIA. MRA VIZCACHITAS HOLDING	056042113-3	300	In Process
79	LOICA 4	CIA. MRA VIZCACHITAS HOLDING	056042114-1	300	In Process
80	LOICA 5	CIA. MRA VIZCACHITAS HOLDING	056042115-K	300	In Process
81	LOICA 6	CIA. MRA VIZCACHITAS HOLDING	056042116-8	300	In Process
82	LOICA 7	CIA. MRA VIZCACHITAS HOLDING	056042117-6	300	In Process
83	LOICA 8	CIA. MRA VIZCACHITAS HOLDING	056042118-4	300	In Process
84	PAICO 1	CIA. MRA VIZCACHITAS HOLDING	S/R	300	In Process
85	PAICO 2	CIA. MRA VIZCACHITAS HOLDING	S/R	300	In Process
86	PAICO 3	CIA. MRA VIZCACHITAS HOLDING	S/R	300	In Process
87	PAICO 4	CIA. MRA VIZCACHITAS HOLDING	S/R	300	In Process
88	PAICO 5	CIA. MRA VIZCACHITAS HOLDING	S/R	300	In Process
89	PAICO 6	CIA. MRA VIZCACHITAS HOLDING	S/R	300	In Process
90	PAICO 7	CIA. MRA VIZCACHITAS HOLDING	S/R	300	In Process
91	PAICO 8	CIA. MRA VIZCACHITAS HOLDING	S/R	300	In Process
92	PAICO 9	CIA. MRA VIZCACHITAS HOLDING	S/R	300	In Process
93	PAICO 10	CIA. MRA VIZCACHITAS HOLDING	S/R	300	In Process
94	PAICO 11	CIA. MRA VIZCACHITAS HOLDING	S/R	300	In Process
95	PAICO 12	CIA. MRA VIZCACHITAS HOLDING	S/R	100	In Process
96	PAICO 13	CIA. MRA VIZCACHITAS HOLDING	S/R	300	In Process
97	PAICO 14	CIA. MRA VIZCACHITAS HOLDING	S/R	200	In Process
98	PAICO 15	CIA. MRA VIZCACHITAS HOLDING	S/R	300	In Process
99	PAICO 16	CIA. MRA VIZCACHITAS HOLDING	S/R	300	In Process
100	PAICO 17	CIA. MRA VIZCACHITAS HOLDING	S/R	300	In Process
101	PAICO 18	CIA. MRA VIZCACHITAS HOLDING	S/R	300	In Process
102	PAICO 19	CIA. MRA VIZCACHITAS HOLDING	S/R	300	In Process
103	PAICO 20	CIA. MRA VIZCACHITAS HOLDING	S/R	300	In Process
104	PAICO 21	CIA. MRA VIZCACHITAS HOLDING	S/R	300	In Process
105	PAICO 22	CIA. MRA VIZCACHITAS HOLDING	S/R	200	In Process
106	PAICO 23	CIA. MRA VIZCACHITAS HOLDING	S/R	200	In Process
107	PAICO 24	CIA. MRA VIZCACHITAS HOLDING	S/R	300	In Process
108	PAICO 25	CIA. MRA VIZCACHITAS HOLDING	S/R	300	In Process
	Total N° Claims	108	Total Hectares	30,800	

Figure 4.2: Mining Properties as of May 2019, (source: LAC)

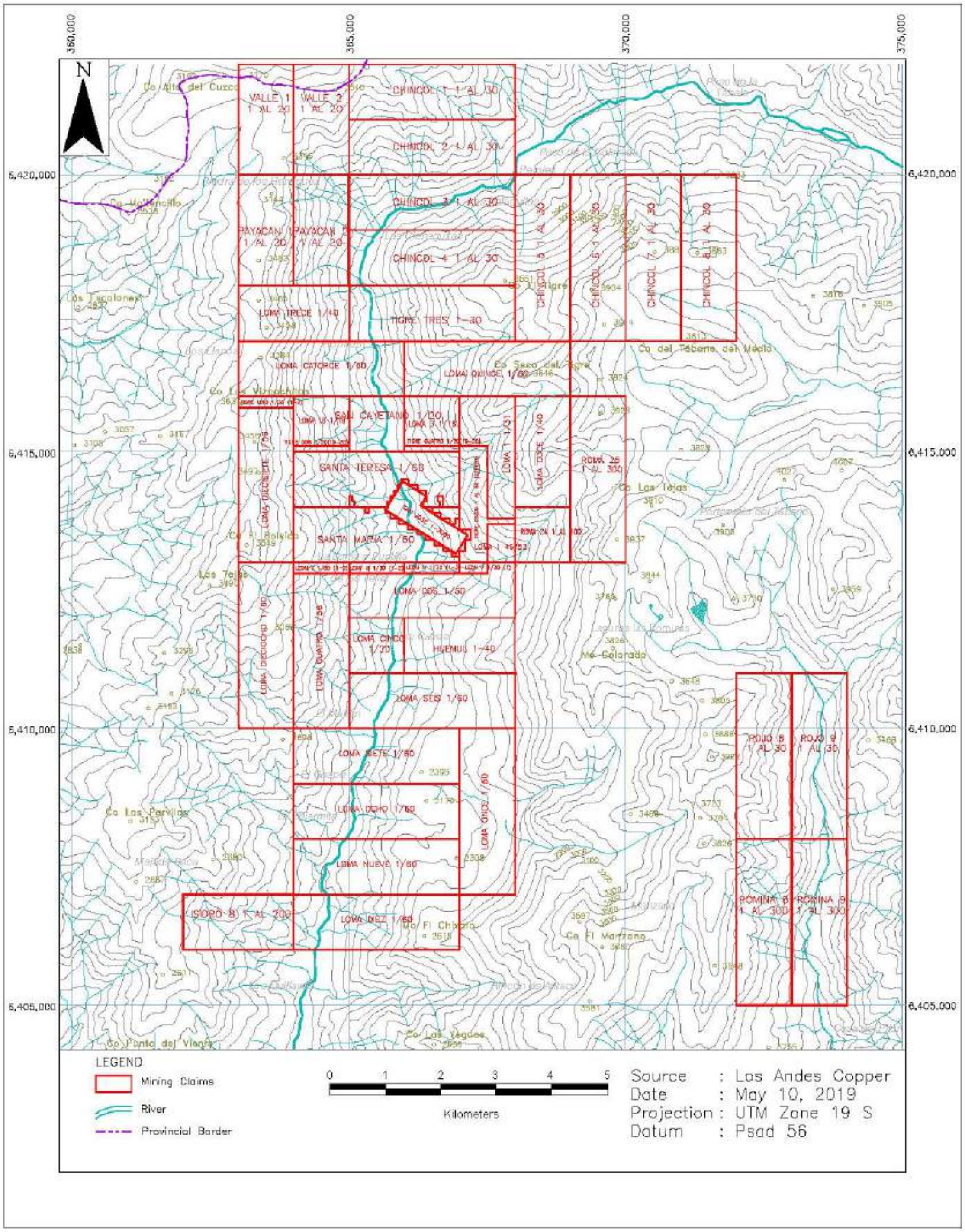
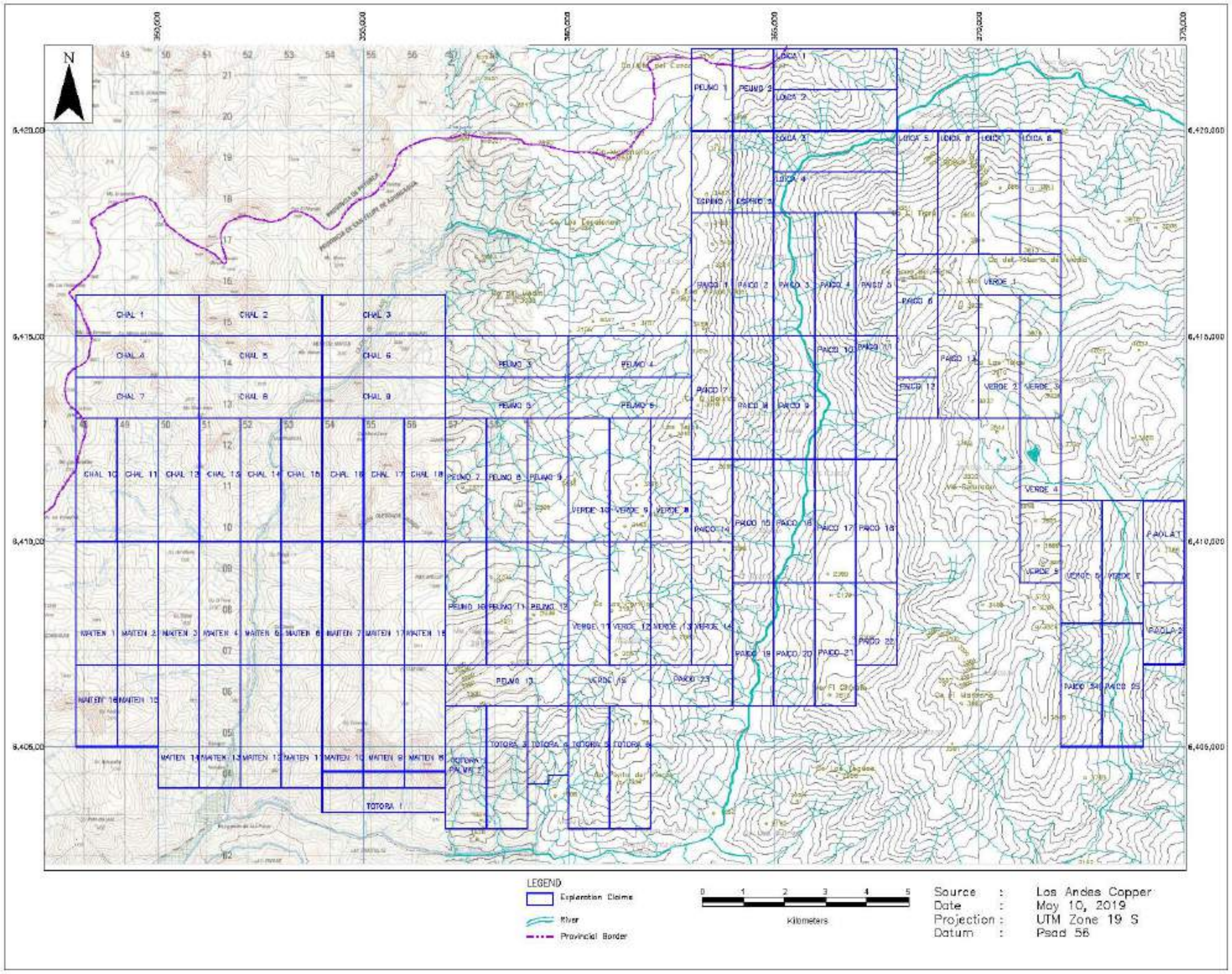


Figure 4.3: Exploration Claims as of May 2019, (source: LAC)



4.2 Title, Surface Rights and Legal Access

CMVH has signed a legal agreement with the owner of the land granting access to the Vizcachitas Project, allowing the Company to carry-out exploration and drilling activities. This is an annual agreement that contains provisions for automatic renewals.

According the Chilean Mining Code, any holder of a mining claim, either for exploration or mining, has the right to establish a right of way over the land as required for the adequate exploration or mining of its claim. Should the owner of the surface property not grant such right of way voluntarily, the holder of the exploration or mining claim may apply for such right of way to the Courts of Justice, which shall issue such access after determining appropriate compensation for potential losses.

4.3 Net Smelter Return

A Net Smelter Return (NSR) of up to 2% is in place over the area where the mineral resources are mined. The economic model reflects a 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 May 10, 2019, confirming the status of the NSRs.

4.4 Environmental Liabilities

Existing environmental impacts are believed to be restricted to exploration-level activities and comprise of disturbances at the drill pads, on access roads and around the exploration camp.

4.5 Operational Permits

CMVH may carry out geological exploration including mapping, surface sampling and geophysics within its claims area.

Future work in the Project will require further infill, metallurgical and condemnation drilling to complete the pre-feasibility and feasibility studies. To carry out this drilling, an Environmental Impact Statement or “Declaración de Impacto Ambiental” (“DIA” for its acronym in Spanish) is required. This process may take between five to nine months to complete.

For the construction and operation of the mine, further environmental studies and permits will be necessary. These would include an Environmental Impact Study or “Estudio de Impacto Ambiental” (“EIA” for its acronym in Spanish), environmental sectorial permits and an approved mine closure plan.

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

5.1 Country Setting

Chile is one of the most developed countries in Latin America. It has had a stable democratic political system since 1990. One third of its 17 million inhabitants live in Santiago, the capital city.

The Chilean economy is one of the strongest on the continent. In the past few years, Chile has had significant and sustained economic growth, where the free-market economy has been maintained through successive governments with additional emphasis on social programmes. The country's international credit ratings have remained in mid-level investment grade for decades.

Chile has the lowest poverty rate in South America. Poverty has progressively decreased, and a strong middle class has arisen. The country has very good universities as well as skilled engineers and administrators. The mining industry benefits from a highly qualified workforce.

It also has the highest copper reserves and is the largest copper producer in the world, satisfying 36% of the global market and holding 28% of worldwide copper reserves. Copper extraction accounts for 30% of Chilean exports. The country has laws favoring the mining industry, although environmental laws have become more stringent in recent years.

Most mining services, from engineering to equipment procurement, may be sourced from within the country.

5.2 Access

The Vizcachitas Project is located about 150 km northeast from Santiago, Chile and 46 km northeast from Putaendo, San Felipe Province. Of the total distance between the property and Santiago (about 150 km), 125 km is paved, and 25 km is unimproved dirt and gravel roads. Travelling time from Santiago to the site is approximately three hours.

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

5.3 Climate

The property is located in the western ridges of the Andes Mountains at an average elevation of 2,100 masl.

The weather is warm and temperate with six dry months from late spring to the fall season. Average precipitation is about 300 mm per annum 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.

5.4 Physiography and Vegetation

The Rocin River valley divides 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 of near 2,100 masl. The exploration camp at Vizcachitas is located at approximately 1,940 masl.

The vegetation consists of shrubs and trees of low to moderate in height, which mainly grow at the bottom of valleys near the rivers.

Figure 5.1: Project Area Topography (source: LAC)

The access and topography present certain challenges for the Project and should be addressed in the engineering phase. Other operations, such as Andina, Los Bronces, and Los Pelambres have been developed in similar terrain.

5.5 Local Resources

There are no major population centres near the Project. The closest town is Resguardo de Los Patos, approximately 25 km away, and the town of Putaendo which is 46 km away. In the cities of San Felipe and Los Andes, 65-85 km from the Project, as well as in Putaendo and other neighbouring towns, there is a significant skilled and semi-skilled labour force. Chile is generally an advanced country in terms of mining technology and infrastructure and provides other countries with high-quality mining experts.

The Rocin River and the Chalaco Stream join together to form the Putaendo River. The Rocin River is dammed downstream of the Project to provide a stable source of fresh water for agriculture in the Putaendo Valley. The Rocin River does not have water consumers between the Project and the dam, which is near Resguardo de Los Patos.

5.6 Infrastructure

Vizcachitas is a greenfield site in which existing site infrastructure is limited to an exploration camp and roads. However, the project benefits from substantial regional infrastructure including a nearby railhead, national power grid and an extensive road network. The project's reasonably low altitude will also be beneficial when site specific infrastructure is developed.

The property is large enough to accommodate an open pit or underground mining operation, although the optimal locations for the infrastructure may overlap with third-party mining properties. As required for the exploration or mining of the claim, the owners of claims have the right to establish a right of way on the surface. Most of this area is owned by a private Chilean company. The process for obtaining right-of-way permits is well established by law in Chile.

The nearby cities of Los Andes and San Felipe are used as a base for many employees and subcontractors who work at Codelco's Andina mine. Anglo American's Chagres smelter and El Soldado mine are located 90 km and 140 km, respectively, from the project. Codelco's Ventanas smelter is located 140 km away from the project. The port of Ventanas is 140 km away and currently handles the copper concentrate from other mining operations in the district. There is an operating railway line in San Felipe with connections to the two smelters and the port of Ventanas. The PEA considers shipping of copper concentrate by rail to Ventanas.

There are several large power substations near the Project site. The Nogales substation is part of the national 220 kV power transmission system and the Las Vegas substation part of the 110 kV system.

The Rocin River runs through the Project site and is a tributary of the Putaendo and Aconcagua rivers. Los Andes Copper currently owns the water rights for a substantial portion of the anticipated water requirements, with an extraction point located along the Aconcagua River approximately 80 km from the project site. To implement the project, Los Andes Copper may have to secure additional water rights.

The Project is a greenfield site and local infrastructure will need to be built. This will include the process water supply, Rocin River diversion tunnel, power supply from the national grid, access road upgrades, concentrate storage and loading facilities at the railhead.

The power for the project is considered to be provided by the National System (Electric Power Grid - SIC) connecting to the Nogales substation or Las Vegas substation, 105 km and 74 km away, respectively.

6. HISTORY

6.1 Description

The central claim, San Jose 1/3000 (San José), was claimed in the 1970's. There is no documentary evidence showing what work was carried out on the Property at that time.

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 Jose claim and entered into an option agreement for the Santa Teresa, Santa Maria, San Cayetano, and Tigre 1 to Tigre 3 claims. They independently claimed the León 1 to León 16 claims. 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), which was subsequently terminated by Boliden in 1998 (Boliden acquired Westmin during the period of the joint venture).

Beginning in 1995, GMC conducted detailed mapping, sampling, geophysics and drilling programmes. Although there is no comprehensive written summary of this work, 61 diamond drill holes were completed through 1998 for a total of 15,815 m. Based on this information, 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 property. Kilborn did an audit of the historic GMC resource and concluded that at a copper price of 1.00 USD/lb, the net present value of the project was USD 201 million at a discount rate of 8% and with a 20% Internal Rate of Return after-tax (Kilborn, 1998).

Shortly after the initial feasibility study was completed, GMC put the project on a care-and-maintenance basis, dropping most of the claims except the central core of concessions.

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

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. During the period CMVH completed a preliminary rehabilitation of the camp and core storage, maintained watchmen at the site, managed the mineral rights and conducted general project orientation for Global management and interested parties.

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 10,400,000 USD and issued to Global 6,280,000 shares and 3,900,000 share purchase warrants in the capital of GHG. After the purchase, GHG decided to focus exclusively on Vizcachitas. GHG was renamed Los Andes Copper Ltd. (Los Andes Copper). No additional field exploration was conducted between 1998 and the date of acquisition by GHG including the period of Global's ownership of the Property.

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

In December 2010, the remaining 49% of the San Jose claim was brought under the control of Los Andes Copper. Since then, all the mining claims have been held by wholly owned subsidiaries of Los Andes Copper.

During the period 2011 to 2012, CMVH systematically compiled and documented the historical data for the Project. The assay certificates for all the samples were located and the pulp samples for the GMC and Los Andes Copper assaying were documented and stored at the project site.

6.2 Historic Resource Estimates

Placer and GMC estimated historic mineral resources at Vizcachitas as part of the exploration work undertaken on the Property. These estimates are not NI 43-101 compliant and are included for historic purposes only and should not be relied upon. The resource estimates prepared by GHG Resources Ltd. in 2007 and Los Andes Copper in 2008, 2013 and 2014 were NI 43-101 compliant.

6.2.1 Placer Dome

This estimate uses five of the six diamond drill holes. The results are shown in Acosta (1992) and Acosta and Zapata (1993). Placer concluded that Vizcachitas contained an inferred 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).

6.2.2 General Minerals

In 1997, using a cut-off grade of 0.3% Cu, GMC estimated a non-NI 43-101 compliant Measured plus 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 estimate was based on 14,370 individual 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 International Incorporated. According to MRA, all resource estimation parameters met or exceeded normal industry standards and met the reporting requirements for Canadian securities commissions at that time (Kilborn, 1998).

In 1998 GMC commissioned Kilborn to complete an "Initial Feasibility Study" on the Vizcachitas Property (Kilborn, 1998). It should be noted that the Kilborn study is no longer valid and was not NI 43-101 compliant. The project, as envisioned by Kilborn, included the following:

- An open pit mine and a conventional crushing, grinding and flotation mill for the recovery of copper and molybdenum concentrates from primary sulphide minerals.
- A partially lined dump leach facility for storage and treatment of secondary enrichment and oxide minerals.
- A solvent extraction and electrowinning plant for the recovery of copper cathode from dump leach material.
- A tailings dam for the storage of mill tailings and recovery of process water.
- Various mine dumps.
- A dam and a 7 km long tunnel designed to divert water from the Rocin River around the project area.
- Administration facilities including offices, change house, cafeteria, truck shop, warehouse and laboratory.
- Electrical supply line from San Felipe, plant site substation, power distribution and access road.

This study detailed all the previous work on the project describing the geology, surface sampling, drilling, resource estimate and metallurgical studies. The conclusions of the report were that using 1.0 USD/lb copper the project had an IRR before tax of 22% and an NPV at an 8% discount rate of 201,000,000 USD (Kilborn, 1998).

6.2.3 GHG Resources Ltd.

In 2007 GHG commissioned A. C. A. Howe International Limited (ACA) to prepare an estimate of Mineral Resources according to NI 43-101. 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 with 0.38% Cu, 0.013% Mo, and 7 ppb Au as an Inferred Resource (Priesmeyer and Sim, 2007).

6.2.4 Los Andes Copper Ltd.

In 2008, prior to the completion of the drilling programme, Los Andes Copper commissioned AMEC and SIM Geological Inc. to prepare a mineral resource estimate in accordance with NI 43-101. The estimate was prepared from 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 area 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 6.1and Table 6.2.

Owing to changes in metal prices during this period, along with the relatively high molybdenum content in the mineral 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 estimated by using the following formula:

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

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

Table 6.1: Sulphide Mineral Resources Estimate, AMEC 2008

Report	AMEC 2008 Sulphide Resources									
Cut-off CuEq % (1)	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

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

Table 6.2: Oxide Mineral Resources Estimate, AMEC 2008

Report	AMEC 2008 Oxide Resources						
Cut-off Cu %	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 Historic Preliminary Economic Assessment

Los Andes Copper published a Preliminary Economic Assessment and an updated NI 43-101 resource estimate on December 12, 2013 (Coffey et al., 2013). This report was published during the non-consumptive water rights consolidation process on a section of the Rocin River, Putaendo, Fifth Region, Chile, along with engineering and other studies and reports for the development of a run-off river power generating facility. This PEA detailed the economic impact of the inclusion and exclusion of a Los Andes Copper hydroelectric plant.

An updated Preliminary Economic Assessment and an updated NI 43-101 resource estimate was published on 18 February 2014 that included the Los Andes Copper hydroelectric plant.

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

Indicated

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

Inferred

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

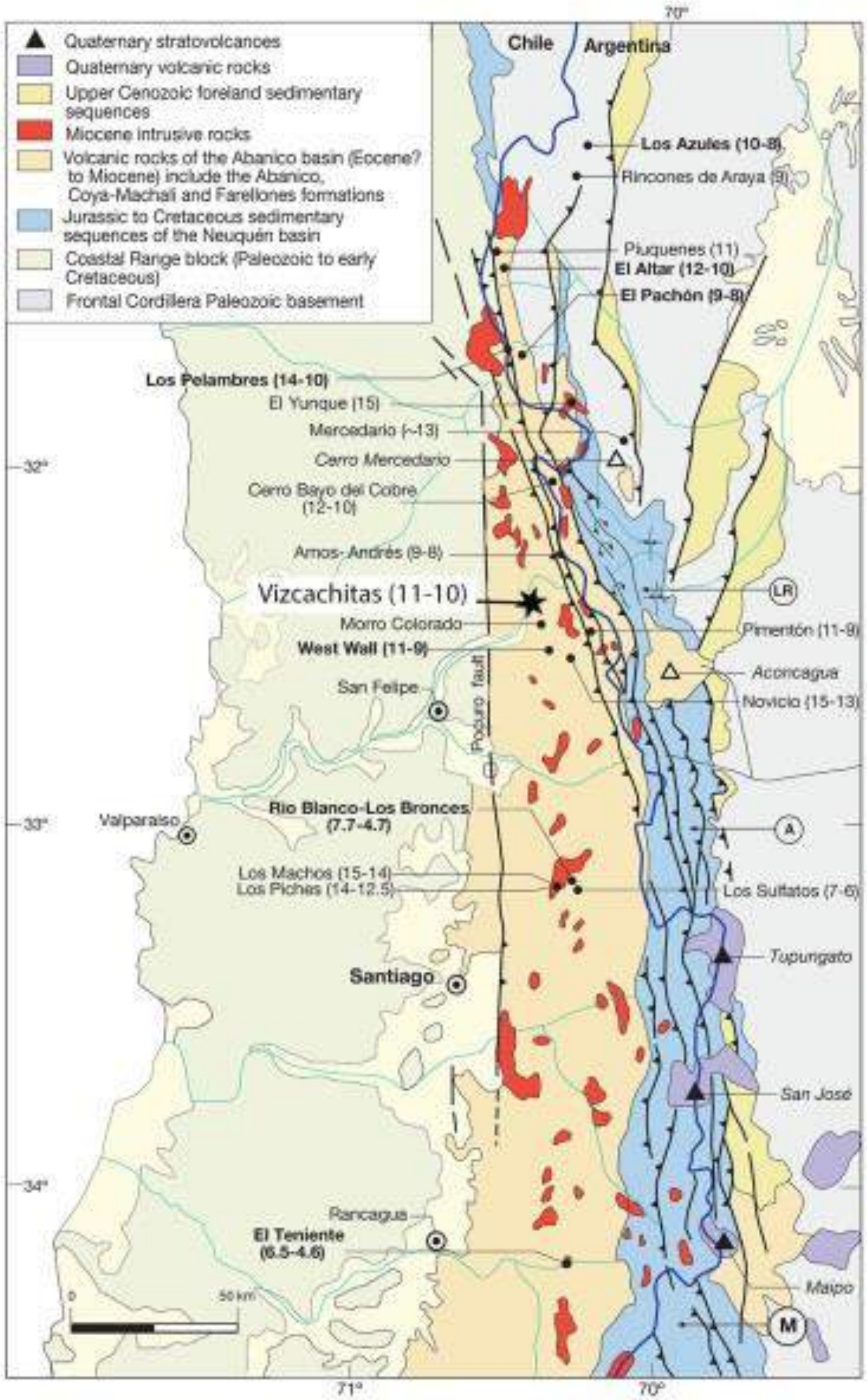
- *Copper equivalent grades (CuEq) were calculated using the following expression: CuEq (%) = Cu (%) + 4.95 x Mo (%), where 4.95 reflects the Mo/Cu price ratio of: 2.75 USD/lb. Cu, 13.6 USD/lb Mo. The quantity and grades are estimates and were rounded to reflect the fact that they are an approximation.*

7. GEOLOGY

7.1 Regional Geology

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

Figure 7.1: Tectonic sketch of the northern end of the Abanico intra-arc basin (31°−34° S) (Mpodozis & Cornejo 2012)



In central Chile, this metallogenic belt includes world class Cu-Mo porphyries, such as Los Pelambres-El Pachón and El Altar, located 75 km north of the Vizcachitas Project, Río Blanco-Los Bronces located 80 km to south of Vizcachitas and El Teniente located 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 less important Au, Cu, and Cu-Au porphyries in the El Indio-Maricunga belt (Davidson and Mpodozis, 1991; Sillitoe, 1991).

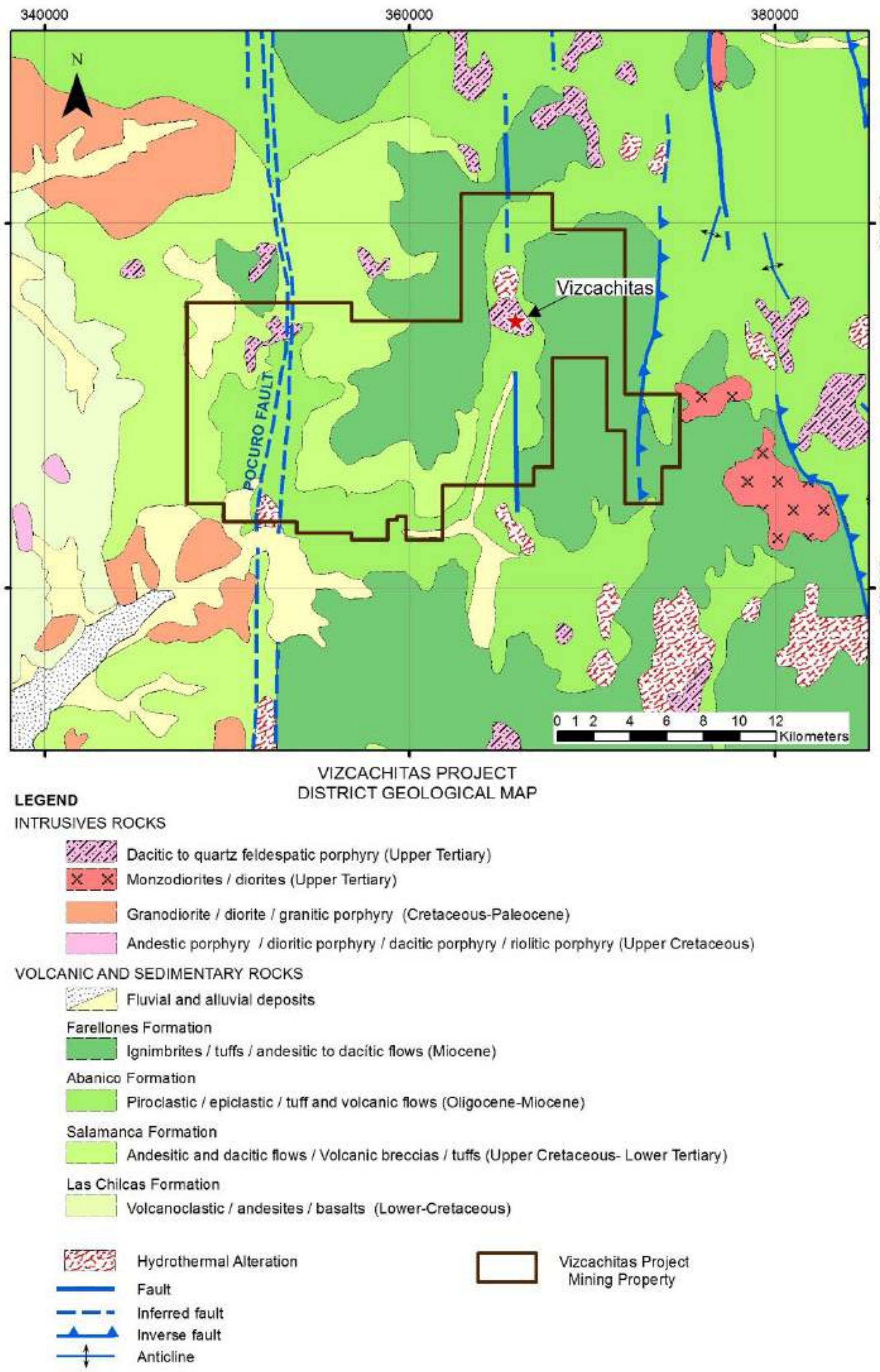
The Neogene metallogenic belt in central Chile coincides with the position of Miocene volcanic centres and associated flat-lying volcanic rocks, sills, and dikes. The Miocene volcanic sequence, with an average thickness of 2,500 m, comprises andesite, basalt (lavas and sills), dacite, and intercalations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

The Neogene porphyry Cu-Mo deposits occur within hydrothermal alteration zones related to multiphase porphyritic stocks with compositions ranging from quartz diorite to granodiorite. These intrusions and their 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 (San Francisco batholith; Serrano et al., 1996) and folded Lower Cretaceous (145-100 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 a sequence of andesitic to basaltic-andesite volcanic flows and volcanic flow breccias that outcrop along the lower half slope of the Rio Rocin valley. The lower andesite sequence is overlain by a well-bedded sequence of interlayered conglomerates, andesite flows and andesite flow breccias that outcrop over the higher half slope of the valley.

Figure 7.2: District geology (Adapted from Rivano et al, 1993)



This unit has been interpreted in different ways over the years. 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 PhD Thesis from Universidad de Chile (Pamela Paz Jara Muñoz, Doctoral Thesis, 2013) proposed that the lower sequence is in fact part of the Late-Eocene or Early Miocene Abanico Formation. This re-interpretation was derived from regional mapping supported by U/Pb zircon dating, although with no samples from the area surrounding Vizcachitas. The mapping in the area of the project of basaltic andesite in the lower sequence is in agreement with the predominance of mafic volcanic rocks described for the Abanico Formation (Fuentes et. al., 2002).

Irrespective of their exact age and stratigraphic correlation, the lower volcanic and the upper sedimentary-volcanic sequences are pre-mineralization and are cut by fresh and altered intrusives and mineralized porphyries of late Middle Miocene and early Upper Miocene age. According to six available K-Ar and Ar/Ar age datings the age of the Vizcachitas deposit is bracketed between (12 and 10 Ma).

The alteration in the intrusive rocks of the western edge of the deposit is quartz-sericite. There is weak to moderate silica alteration within the volcanic rocks. The intrusive complex generally has weak potassic alteration especially in the deeper parts of the deposit, while the surrounding diorites have weak quartz-sericite and potassic alteration. The breccias have weak chloritic alteration.

The mineralized system of Vizcachitas comprises complex porphyries and breccias with copper and molybdenum mineralization, which intrudes andesitic volcanic rocks. The earliest intrusive rocks are porphyries of diorite composition. The petrographic work identifies this unit as a diorite with biotitized mafics (phlogopite) with deformed quartz veins and others of hyaline quartz/anhydrite veins. This is associated with abundant chalcopyrite and development of sericite haloes where the pyrite is greater than chalcopyrite. This phase has the highest primary grades for both copper and molybdenum.

Hydrothermal breccias have various matrix including: siliceous, silica-K feldspar, silica-sericite, silica-anhydrite, silica-sulphides and silica-tourmaline. These are genetically related to each igneous event. The hydrothermal breccias are normally associated with higher copper grades than their related intrusive phase.

The first inter-mineral intrusive phase is of a tonalitic composition and is followed by a later granodiorite intrusive. The petrographic work describes this as a fine granodioritic porphyry with potassic alteration (biotite with anhydrite) and white quartz B veins with abundant potassic feldspar and anhydrite. This is then cut by anhydrite-gypsum veinlets and sulfides.

A phreatomagmatic breccia or diatreme that is post-mineral has cut through the central part of the project. It is composed or fragments of pre-existing igneous and volcanic rocks in a matrix of rock dust and clays. This unit is barren.

Dacitic porphyries dykes are the last intrusive event and cut the phreatomagmatic breccia and are also barren.

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

The K-Ar dating of a coarse biotite included in the matrix of a polymictic breccia (sulfide Breccia) biotite-anhydrite-muscovite-Cpy-Py matrix provided 11.5 ± 0.3 Ma (Osterman, 1997).

7.3 Mineralization

The latest mapping carried out by Los Andes Copper shows a 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 chrysocolla, are occasionally observed in fractures.

Below the leached zone, there is a secondary enrichment zone or supergene zone of weak to moderate intensity, with presence of chalcocite and covellite, which occurs 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 defined Resource Estimate is 0.501% Cu.

Below the secondary enrichment zone, there is the hypogene or primary mineralization. This mineralization is mainly made up of chalcopyrite, with significant amounts of associated pyrite. Bornite occurs in several of the drill holes below 800 m. In the drill hole V2017-10, located at the northern end of project, bornite accounts for 15% of the total sulphides below 900 m. This indicates that a possible bornite core could be located below the current drilling.

Additionally, chalcographic studies conducted by General Minerals in 1998 show the local presence of gray copper sulphides tennantite, tetrahedrite and enargite in small amounts.

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

8. DEPOSIT TYPES

The Vizcachitas mineral deposit has similar characteristics to other Andean-style porphyry copper and molybdenum mineral deposits. This type of mineralized deposit contains 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 breccia.

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

The surface oxidation commonly modifies the distribution of mineralization in degraded environments. The acidic meteoric waters generated by the oxidation of pyrite 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 that is above a relatively thin zone of supergene enrichment zone. Below the supergene enrichment, there is a thicker zone of primary (hypogene) mineralization.

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 mineralisation. Such enrichment processes result in high grades in the hypogene zone.

9. EXPLORATION

From the beginning of 1990's to date, three companies have carried out exploration on the property, namely: Placer Dome, General Minerals Corporation, and Los Andes Copper.

9.1 Placer Dome Sudamerica S.A.

In 1992, Placer carried out mapping and sampling programmes. These programmes focused on improving the understanding of the geological alteration and mineralization of the deposit with a view 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 recognized the porphyry copper characteristics of the mineralization and alteration and further recognized that the mineralization was centered on two complex breccia pipes.

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

Molybdenum was generally coincident with copper. Anomalies were defined by molybdenum values greater than 50 ppm. Anomalous gold, silver, arsenic, antimony and potassium were also detected in the same area as the copper and molybdenum anomaly.

Placer recognized the importance of the breccia bodies in localizing mineralization. In 1993, Placer completed six diamond drill holes for a total of 1,953 m. The Placer drilling programme is discussed in Chapter 10 of this report.

9.2 General Minerals Corporation

GMC acquired its share in the Vizcachitas Property in 1995. GMC performed the following exploration work:

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

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

Original copies of the drill logs, assay batch dispatch forms and the 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 southeast 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.

Owing to the rugged nature of the terrain, survey lines were initially located along drill roads and then up slopes that could be easily and safely be negotiated by geophysical crews. Most of the area was surveyed with 50 m between stations and with lines 300 m to 500 m apart.

In 1997-1998 General Minerals completed 61 drill holes with a total 15,815 m drilled. The General Minerals drilling programme is discussed in Chapter 10 of this report.

9.3 Global Copper Corporation

Global did not undertake any exploration work on the property.

9.4 Los Andes Copper Ltd.

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. Between July 2007 and October 2008, Los Andes Copper completed a total of 79 diamond drill holes totaling 22,616 m. This drilling is discussed in Chapter 10.

Los Andes Copper did not undertake any geophysical work on the Property.

From 2012 and under a new geology team, Los Andes Copper reviewed and documented all the historical data available on the Property. This included the digital capture of the historical logging and assaying information so that it could be added to the database.

As part of the geological review all the drill core was re-logged. The core from the previous diamond drilling programmes is stored at the project site.

The re-mapping of the drill core:

- Showed that the Vizcachitas Project is a partially eroded hydrothermal system. Most drill holes are from the upper part of the system, the phyllic zone, or within the sericite-chlorite-biotite alteration. These conclusions were supported by the existence of some holes "standing" in potassic alteration when the copper grade was just beginning to increase with depth. It was concluded that the system was open at depth and requires deeper drill holes to confirm the depth potential.
- Identified four types of breccias namely: igneous breccia, magmatic-hydrothermal breccia, hydrothermal breccia, and phreatomagmatic breccia. The phreatomagmatic breccia or diatreme is barren and post-mineral. The hydrothermal and magmatic-hydrothermal breccia usually provided the highest copper grades in the Project.
- Identified a family of productive porphyries that made up the core of the Vizcachitas system. While in general identifying clear contact relationships among the diverse intrusive pulses was not easy, the geological mapping enabled the sequencing in the intrusive events. Thus, it was possible to recognize an early diorite porphyry or intrusive, an early inter-mineral tonalite intrusive, and a late inter-mineral granodiorite intrusive. The early diorite is a diorite complex varying from diorite to quartz diorite, and fine to medium grain, and partly porphyritic. The higher grade copper mineralization (greater 0.5% Cu) is the "early porphyry" or the earliest phase of the porphyry. This idea of an "early porphyry" with a higher grade mineralization was also incorporated into the exploration model because of its relevance to the project's potential.
- The remapping of the core showed that the Vizcachitas mineralized system was open, not only in depth, but also to the west and north.

Los Andes Copper drilled from August 2015 to April 2016 with 8 diamond drill holes, and a total of 3,610 m. The second campaign was carried out from February to July 2017, with 11 diamond drill holes, and a total of 8,262 m.

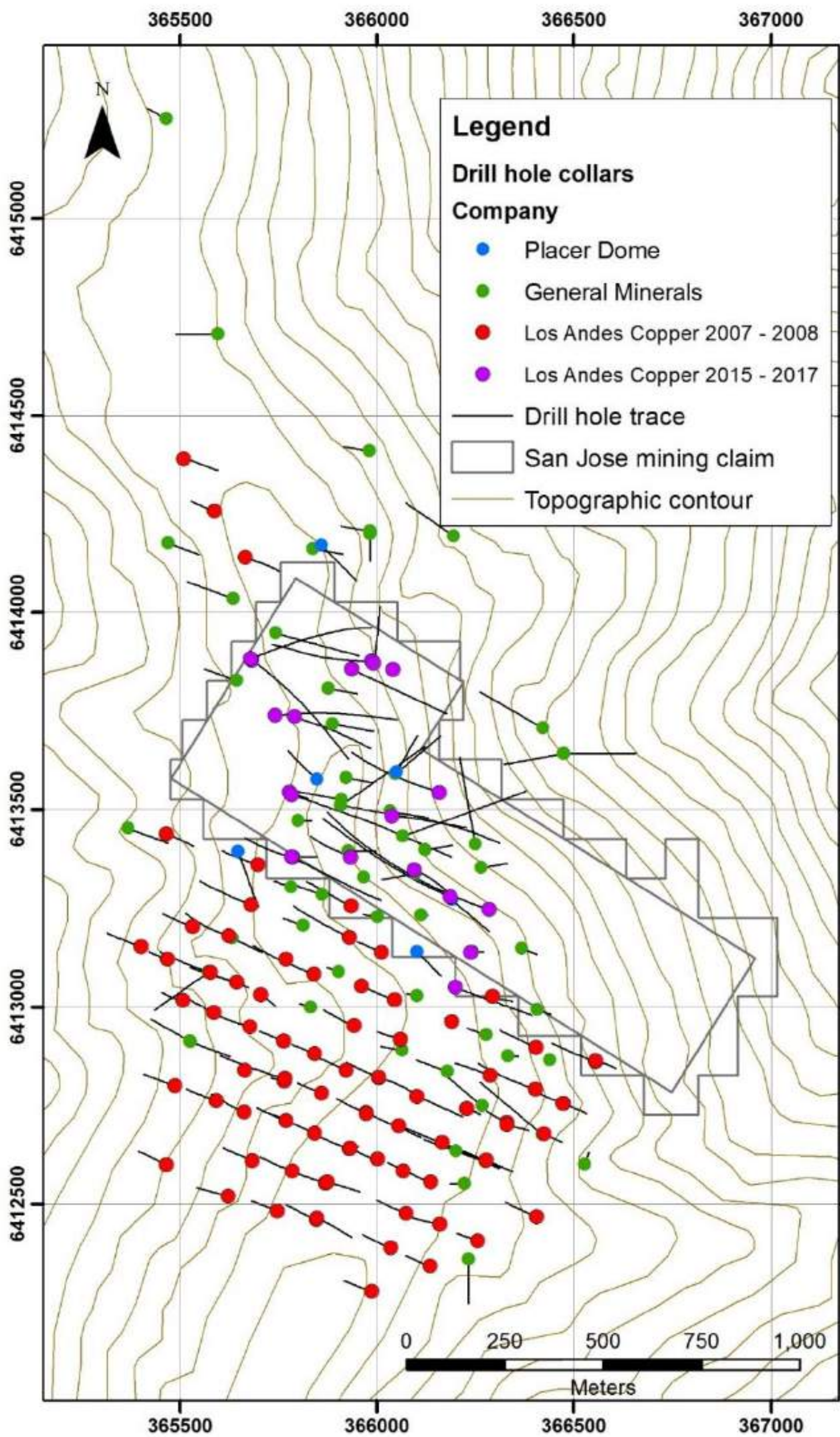
10. DRILLING

Since 1993, a total of 165 diamond drill holes have been drilled in the Property with a total of 52,256 m. The total metres drilled by each company are summarized in Table 10.1. The location of these drill holes is shown in. Figure 10.1. The detailed location, azimuth and inclination of all drill holes are shown in APPENDIX II

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	2015-2016	V2015-01 to V2015-08	8	3,610
Los Andes Copper	2017	V2017-01A to V2017-11	11	8,262
Total			**165**	**52,256**

Figure 10.1: Drill Hole Location (source: LAC, May 2019)



10.1 Placer Dome Sudamerica Limited

The first drilling at the Project, was carried out by Placer Dome in 1993, and consisted of 6 diamond drill holes located in the central-north part of the area of interest, with variable 250 – 500 m lengths and a total of 1,953 m drilled.

The best intercepts obtained by these drill holes are 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 potassium alteration and VP-04 with 30 m @ 0.92% Cu and 160 ppm Mo in a granodiorite to granodiorite porphyry brecciated with potassium alteration.

10.2 General Minerals Corporation

From 1995 to 1998, General Minerals carried out a diamond drilling campaign with a total of 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.

The results obtained in this campaign, which extended over the whole mineralized corridor, helped identify the lithological types that make up the Vizcachitas mineralized system.

The best drilling results are given in Table 10.2 and are mainly related to 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 reported copper mineralization shows chalcopyrite with chalcocite-covellite on the upper levels associated with secondary enrichment.

Table 10.2: General Minerals Drill Holes - Intercepts > 0.5% Cu

Drill Hole	From	To	Section (m)	Cu %	Mo ppm	Litho
V-02	65	139	74	0.8	148	Diorite / Bx magm-hydrot alt QS-KBT
V-03	42	95	53	0.93	56	Bx magm-hydrot alt cl-QS-KBT
V-04	6	344	338	0.67	170	Diorite / Bx magm-hydrot alt KBT-ser-cl
V-05	6	88	82	0.81	123	Diorite alt KBT-QS-cl
V-05	459	488	29	0.81	117	Bx magm-hydrot alt QS
V-06	74	123	49	1.1	99	Andesite alt QS-cl
V-13	20	200	180	0.57	56	Diorite alt KBT
V-15	54	134	80	0.74	66	Andesite / Bx magm-hydrot alt QS-KBT
V-16	96	163	67	0.87	102	Bx magm-hydrot alt sil-QS
V-18	48	201	153	0.77	186	Andesite / diorite alt KBT-QS-cl
V-20	10	135	125	0.65	62	Diorite alt KBT-cl-ser
V-23	11	59	48	0.67	35	Diorite alt KBT-QS
V-23	86	285	199	0.65	118	Diorite / andesite alt KBT
V-32	280	358	78	0.72	134	Quartz diorite / Bx magm-hydrot alt sil-ser
V-46	89	133	44	0.73	102	Andesite alt cl-QS-KBT
V-58	96	185	89	0.66	233	Andesite / Bx magm-hydrot alt KBT-QS-cl

10.3 Los Andes Copper Ltd.

10.3.1 2007-2008 Los Andes Copper Drilling

Between 2007 and 2008, Los Andes Copper conducted an exploration programme at the project with 79 diamond drill holes, with lengths ranging from 150 m to 717 m, with a mean length of 286 m, and a total of 22,616 m drilled. Figure 10.1 shows the locations of the 2007-2008 Los Andes Copper drill holes.

The best intercepts (> 0.5% Cu) obtained in this programme are shown in Table 10.3 and are related to the early diorite complex, magmatic-hydrothermal breccia, andesites, and to a lesser extent, the tonalite intrusive. The alteration continues to show a strong hydrothermal contribution overlaid on earlier potassic alteration.

The copper mineralization always shows chalcopyrite, chalcocite and covellite as secondary enrichment on the upper part of the drill holes.

Table 10.3: Los Andes Copper 2007-2008 - Intercepts > 0.5% Cu

Drill Hole	From	To	Section (m)	Cu %	Mo ppm	Litho
LAV-066	116	178	62	0.56	180	Diorite porphyry alt QS-KBT
LAV-068	78	166	88	0.70	110	Diorite/tonalite porphyry alt KBT-cl
LAV-072	34	140	106	0.69	132	Diorite porphyry/tonalite/igneous breccia alt QS-sil-KBT
LAV-073	28	78	50	0.67	118	Andesite sil-cl-biot
LAV-078	134	180	46	0.70	130	Bx magm-hydrot alt QS-cl
LAV-080	82	120	38	0.74	141	Bx magm-hydrot alt sil
LAV-081	82	260	178	0.57	115	Bx magm-hydrot alt KBT
LAV-082	20	66	46	0.67	129	Andesite alt KBT-cl-ser
LAV-084	92	208	116	0.53	44	Diorite alt KBT-cl
LAV-085	10	144	134	0.65	30	Diorite alt KBT
LAV-088	46	116	70	0.62	48	Tonalite/bx magm-hydrot alt QS-KBT
LAV-088	154	250	96	0.60	100	Bx magm-hydrot alt QS-KBT
LAV-089	34	102	68	0.81	213	Tonalite alt QS-cl
LAV-090	340	412	72	0.64	37	Diorite alt Qs-cl
LAV-091	68	358	290	0.59	116	Diorite alt QS-KBT
LAV-108	42	254	212	0.58	94	Bx magm-hydrot/diorite alt KBT-QS-K Feld
LAV-122	44	162	118	0.57	132	Andesite KBT-cl-ser
LAV-124	238	376	138	0.60	164	Bx magm-hydrot alt KBT-sil
LAV-131	56	266	210	0.68	138	Bx magm-hydrot alt sil-ser-cl

Figure 10.1 shows the location of these drill holes, and Figure 10.2 shows a photograph of the standard of drill hole storage at the project.

Figure 10.2: Main Storage in the Project

10.3.1.1 Procedures

Advisor Drilling and Leduc Drilling Chile S. A. completed core drilling using a Longyear LF-90 truck-mounted platforms. The core diameter was generally HQ diameter (63.5 mm) but NQ diameter (47.6 mm) was used in certain deep holes, and only when strictly necessary for technical reasons. The surface gravels were drilled using a 5-1/2" tricone drill bit and no material was collected. The average depth of drill holes was approximately 286 m, although some drills exceeded 500 m in depth.

The drill core runs were 3 m long. The cores were placed in 1 m long metal boxes and identified with the hole and numbers on the box. The drilling runs were marked with small wooden blocks.

The core recovery was measured by the drillers following the removal from the central tube. The core boxes were collected twice a day and carried by pickup truck to the camp for cutting, sampling and permanent storage. The core boxes were covered and secured during transport. There was ongoing supervision by the geological subcontractor, Geologica, at the drilling site.

As part of the 2008 Technical Report, AMEC observed the core drilling and handling and stated in the report that the current procedures complied with best practices for the mining industry.

10.3.1.2 Surveying

During the 2007-2008 drilling programme, all surveying was carried out using the Universal Transvers 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. In Spanish these are called “Hitos de Mensura.” The triangulation points were previously surveyed with reference to the Chilean Instituto Geográfico Militar (IGM) national grid based on PSAD56.

The collar locations were surveyed using a Total Station prior to drilling and were re-surveyed after drilling was completed. The inclination of the hole was also surveyed.

The drilling contractor conducted the down-hole surveying using the Flexit method, which determines the azimuth by magnetic reading and the inclination of the hole. Readings were taken at 24 m intervals. The drilling contractor delivered digital reports for each surveyed hole. Azimuth measurements were corrected for the local magnetic declination.

10.3.1.3 Geological and Geotechnical Logging

The drill hole mapping consisted of observing and logging the geological information of a drill hole for the lithological, hydrothermal and structural controls of the mineralization.

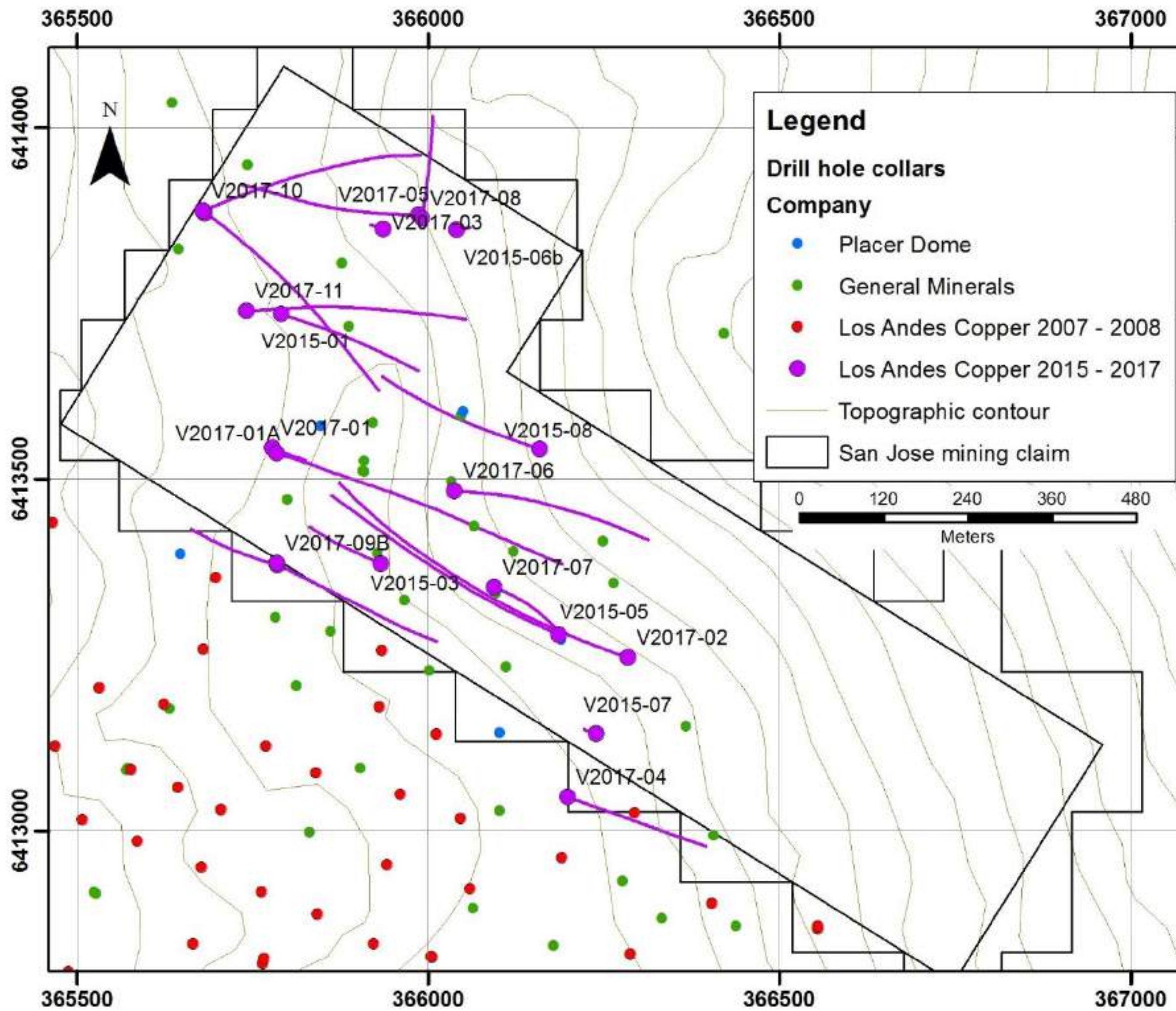
Los Andes Copper geologists conducted detailed geotechnical logging, which included Rock Quality Designation (RQD) data and frequency of fractures usually measured at 2 m intervals. In addition, structure and rock matrix features such as roughness, infilling, wall resistance, filling type, infilling hardness, matrix resistance and weathering grade were recorded using codes.

Geological logging was conducted after cutting the core in half using a diamond saw. The logs were recorded in digital spreadsheet format and contain data of the intervals with their geological descriptions such as lithology, types of veinlets, details on the alteration (including a visual appreciation of the pyrite/chalcopyrite ratio and the presence of gypsum or anhydrite) and mineralization zones using pre-established codes.

10.3.2 2015-2017 Los Andes Copper Drilling

Between 2015 and 2017, Los Andes Copper conducted an exploration programme at the Project with 19 diamond drill holes, with lengths up to 1,030 m, a mean length of 625 m and a total length of 8,262 m.

Figure 10.3: Los Andes Copper 2015-2017Drill Hole Location



10.3.2.1 Procedures

Mr. Humberto Ortega[1], a qualified person under Chilean law N° 20.235, visited the project to verify the drilling operation of the 2015-2016 campaign on January 8th, 2016. At that time, the drilling

[1]*Member of the "Comisión Calificadora de Competencias en Recursos y Reservas Mineras de Chile". Public Register #0067.*

operation had stopped temporally, and Los Andes staff were processing, cutting core and preparing sections and plans of the deposit using the drilling information.

José Luis Fuenzalida[2], a qualified person under Chilean law N° 20.235, visited the site to verify the drilling operation of the 2017 campaign, where he inspected the drilling work, the collection of cores, the marking of the intervals, the sealing of boxes and their transfer to the cutting site. The cutting of cores with a diamond saw was also inspected. These procedures were developed in compliance with best mining industry standards.

10.3.2.2 Surveying

In March 2012, Los Andes Copper carried out a topographic survey of the project's triangulation points to tie them to the national survey grid that uses the SIRGAS/WGS84 and the PSAD56 datum's that are maintained by the Instituto Geográfico Militar (IGM) of Chile.

Additionally, a survey was carried out of some of the existing drill holes in the project area to validate the historical coordinates. A survey of a set of pre-flight targets to be used as ground support for an aerial survey of the area was also conducted. (Topographic Report: Surveys of Drill Holes and Pre-flight Signals, Linkage to the SIRGAS Network, Minera Vizcachitas, Minera Los Andes Copper, March 2012, IT-AC-001_B).

Additionally, in March 2016 Los Andes Copper carried out a topographic survey of all the drill hole collars on the Property, this included the historic drill hole and the recent drill campaign. A total of 115 drill collars were surveyed (Vizcachitas Project Drill Hole Survey, March 2016). This survey was carried out using professional topographic instruments to ensure a high degree of accuracy and in the UTM Datum WGS84 coordinate system.

10.3.2.3 Geological and Geotechnical Logging

Los Andes Copper geologists kept detailed geological records logging inter alia the lithology, structures, mineral zones and alteration, mineralization of oxides, sulphides, gangue and alteration/mineralization ratio.

The geotechnical logging includes RQD and fracture frequency, which was generally measured at intervals of 2 m. In addition, the characteristics of the structure and the rock matrix, such as roughness, type of fill, hardness of fill, types of small veins, faults and others were surveyed. The

[2] *Member of the "Comisión Calificadora de Competencias en Recursos y Reservas Mineras de Chile". Public Register #0171.*

recording of this information was logged on paper sheets and then entered into Excel spreadsheets.

10.3.2.4 Summary of drill hole results

During 2014, a complete review of the historical information was performed to better understand the project, including re-logging the 146 drill holes located within the property. This detailed review showed that the historical logging and geological model had not properly identified the importance of the higher-grade early diorite porphyry and hydrothermal breccias. The re-logging showed that these higher-grade geological units extend over a distance of 1,400 m north-south and 700 m east-west. The mapping showed that these breccias have grades increasing with depth and demonstrates the potential for higher grades below the historical drilling.

In 2015, Los Andes Copper began a drill programme to confirm the new geological model and to demonstrate the extent of the central core mineralization. A first stage of this exploration campaign was completed in 2015-2016 with eight diamond drill holes totaling 3,610 m. During 2017 Los Andes Copper carried out a second stage of this campaign with eleven drill holes totaling 8,262 m.

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 intersected the near surface higher grade supergene enriched mineralization outlining an area of 400 by 400 m where all the drill holes have average supergene grades of greater than 0.5% Cu. Some examples of this near surface supergene mineralization are:

V2015-03 – 17.9 m @ 0.78% Cu from 44.1 m down-hole

V2015-05 – 78 m @ 0.56% Cu from 68 m down-hole

V2015-08 – 92 m @ 0.71% Cu from 92 m down-hole

V2017-06 – 92 m @ 0.66% Cu from 46 m down-hole

V2017-07 – 98 m @ 0.61% Cu from 44 m down-hole

V2017-09B – 77 m @ 0.52% Cu from 64.4 m down-hole

The drilling also demonstrated that the early diorite porphyry and hydrothermal breccias extend 250 m to the north, further than previously shown. The northernmost drill holes had the following intersections:

V2017-10 – 506 m @ 0.57% Cu from 486 m down-hole, V2017-05 – 90 m @ 0.49% Cu from 170 m down-hole,

This mineralization remains open to the north.

In the core of the Project, the drilling demonstrated good mineralization to the west and south of the diatreme. The drill hole V2015-05 intersected 52 m @ 0.81% Cu from a down-hole depth of 544 m. The drill hole V2017-02 was drilled underneath V2015-05 and intersected 88 m @ 0.60% Cu from a down-hole depth of 680 m, demonstrating the vertical continuity of this mineralization over a distance of 250 m. The 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.

Some results of the drilling campaigns carried out by Los Andes Copper in the project area are shown in Table 10.4.

Table 10.4: Highlights of Los Andes Copper Drilling Campaigns

Hole Number	Length (m)	Depth Downhole From (m)	Cu %	Mo ppm	Ag g/t	CuEq %*
V2015-08	502.0	130.0	0.63%	209	1.3	0.70%
including	54.0	130.0	1.02%	128	1.4	1.07%
including	37.0	198.8	0.92%	132	2.0	0.98%
including	396.3	235.8	0.57%	233	1.2	0.64%
V2015-03	39.1	44.1	0.74%	145	1.9	0.80%
V2015-05	52.1	492.2	0.81%	190	2.0	0.89%
V2015-05	120.0	72.0	0.54%	169	1.4	0.60%
V2015-02	52.0	142.0	0.60%	170	1.8	0.66%
V2015-01	64.0	322.0	0.60%	258	1.2	0.68%
V2017-01A	302.1	71.9	0.55%	115	1.4	0.59%
including	134.0	226.0	0.60%	150	1.5	0.65%
V2017-04	90.0	92.0	0.51%	127	1.6	0.56%
V2017-05	90.0	170.0	0.49%	231	1.0	0.56%
V2017-05	80.0	798.0	0.53%	285	1.9	0.63%
V2017-06	440.0	64.0	0.51%	164	1.1	0.56%
including	56.0	76.0	0.81%	72	1.6	0.85%
V2017-07	102.0	44.0	0.62%	157	1.5	0.66%
V2017-09B	77.6	64.4	0.52%	153	1.3	0.57%
V2017-10	506.0	486.0	0.57%	357	1.1	0.67%
including	76.0	514.0	0.69%	522	1.5	0.84%
including	88.0	684.0	0.70%	278	1.4	0.78%
including	60.0	924.0	0.73%	341	1.5	0.83%

All thicknesses from drill holes are down-hole drilled thicknesses. True widths cannot be determined from the information available.

11. HISTORIC SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 Quality Assurance and Quality Control Definitions and Protocols

The NI 43-101 and CIM (Exploration Best Practice Guidelines; 2010, 2003b and 2005) state that a data verification programme must accompany an exploration programme to confirm the validity of exploration data. In addition, the CIM's Best Practice Guidelines for Mineral Resource and Mineral Reserve Estimate require that a quality assurance and quality control programme be used to certify that analytical accuracy and precision are adequate to support a resource estimate (CSA, 2011a and 2011b).

Some basic concepts related to a quality assurance and quality control (QA/QC) programme are clarified below, which are applicable to all programme elements:

- Precision: The ability to consistently reproduce a measurement in similar conditions;
- Accuracy: The closeness of these measurements to the "true" or accepted value;
- Contamination: The inadvertent transfer of material from one sample to another.

As a rule, two laboratories should be used during a sampling campaign namely, a primary laboratory, where all original samples are analyzed and a secondary (or referee) laboratory, where a representative portion of the samples from the primary laboratory is re-examined.

The QA/QC programme states that the analysis of samples at the primary laboratory must be accompanied by a proportion (usually 5%-10%) of blind control samples and the shipment of a portion of the original coarse waste sample from the primary laboratory to the secondary laboratory, also accompanied by a proportion (usually 5%-10%) of blind control samples.

The purpose of blind insertion of control samples is to prevent the laboratory from identifying control samples, or at least the nature and equivalence of control samples. All accredited laboratories have internal QA/QC procedures and the test certificates generally include internal QA/QC results. However, some laboratories will only disclose those controls that pass their internal controls not those that fail.

For this reason, the internal laboratory QA/QC programme should not replace the client’s QA/QC programme.

A QA/QC programme should monitor several essential elements of the sampling and analysis sequence to control or minimize the potential total error in the sampling, division and analysis sequence, namely:

- Collection and division of samples (sampling variance or sampling precision)
- Preparation of sample and sub-sampling (sub-sampling variance or sub-sampling precision, contamination during the preparation)
- Analytical precision, analytical accuracy, and analytical contamination
- Accuracy in data transfer (from paper to digital).

11.2 Historic Data

While there is little documentation for the procedures used by Placer and GMC during their drilling programmes, Los Andes Copper has maintained the original documentation of all drill holes including the original logs, assay batch forms, and analysis certificates. The drill hole cores are well preserved and carefully stored at the project site.

The average length of the Placer drill hole sample is 1.1 m with numerous samples less than 1.0 m in length (Sim, 2005). This sampling scale is small in relation to the type of mineralization being sampled and the typical extraction scale of high tonnage and relatively low-grade mineral deposits.

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

All drill cores were recorded and divided by using a manual core splitter or diamond saw. All drill cores are stored at Vizcachitas project site and are in good condition.

Sampling and drilling programme sample analyses are available with rock chips from Placer and GMC drilling campaigns. The drilling programmes were sent to ACME Analytical Laboratories (Chile) Limited (ACME), in Santiago for analysis.

During the Placer and GMC drilling campaigns, QA/QC programmes were implemented by introducing duplicate samples and two different standards for analysis within each lot. These samples were not introduced at regular intervals but were sent every 20 to 40 samples. At that time no blind sample shipment system was implemented, and the laboratory entered the blind samples as duplicates in the test records.

Los Andes Copper compiled and documented all available data for the sampling and analysis conducted by Placer and GMC. The final report prepared by Placer included the hand-drawn geological records and photocopies of ACME test certificates (Acosta, 1993). Los Andes Copper has the original test shipment forms and test certificates of all the surface and drilling sampling.

The original test certificates were scanned, and the data collected by using optical character recognition. The data from the test certificates were imported into a Micromine GBIS database. The scanned certificates were compared with historical database values, the differences were compared with the original certificates and the data was corrected to match the certificates. The number of errors identified was minimal, showing that both the original database and the optical character recognition database were compiled accurately.

A significant number of duplicate samples were sent to secondary laboratories, namely, Lakefield Research Chile S.A. (Lakefield), CIMM Tecnologías y Servicios SA (CIMM), and ALS Geolab S.A. (ALS), as summarized in Table 11.1.

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 samples sent to the three laboratories show a good correlation and the trend line as drawn demonstrates that there is a small bias between them. Figure 11.1, Figure 11.2 and Figure 11.3 illustrate this bias for each laboratory.

Figure 11.1: GMC Secondary Laboratory Duplicates ACME v/s CIMM

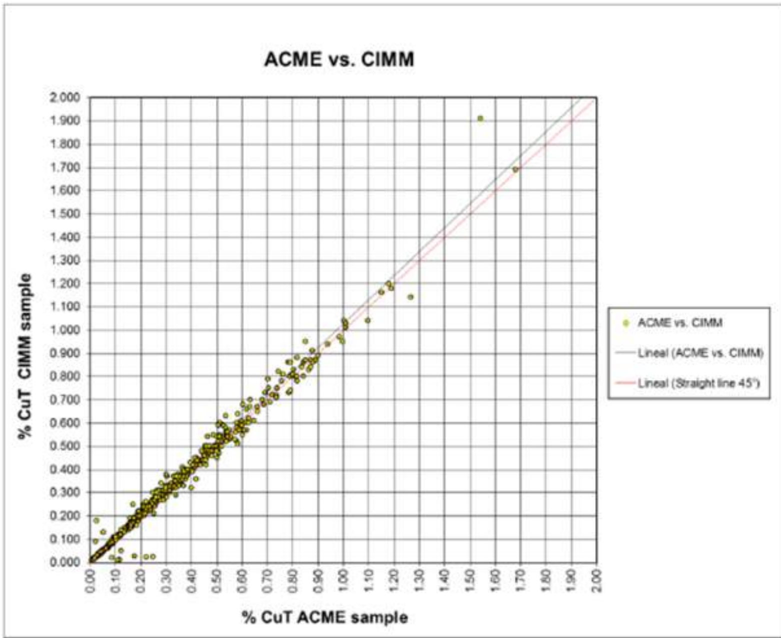


Figure 11-1: GMC Secondary Laboratory Duplicates ACME vs. CIMM

Figure 11.2: GMC Secondary Laboratory Duplicates ACME v/s Lakefield

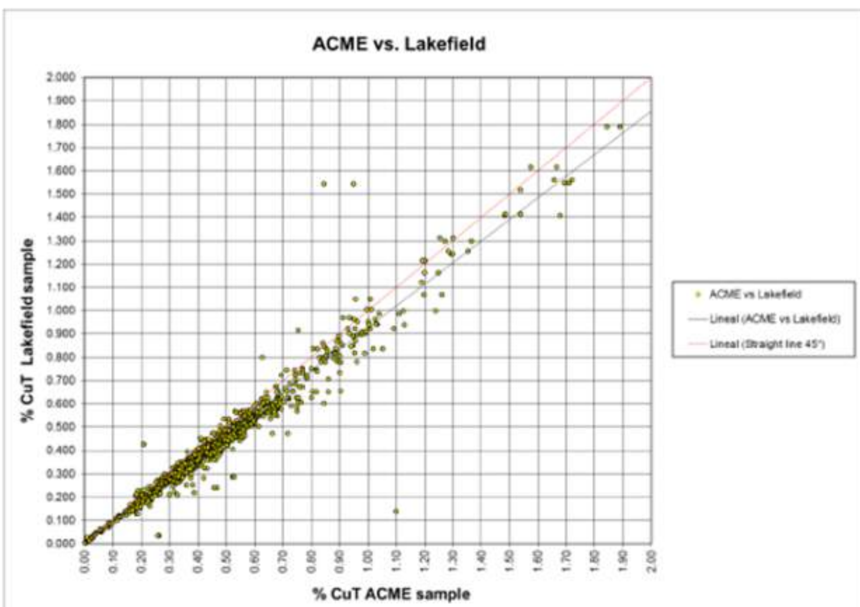


Figure 11-2: GMC Secondary Laboratory Duplicates ACME vs. Lakefield

Figure 11.3: GMC Secondary Laboratory Duplicates ACME v/s ALS

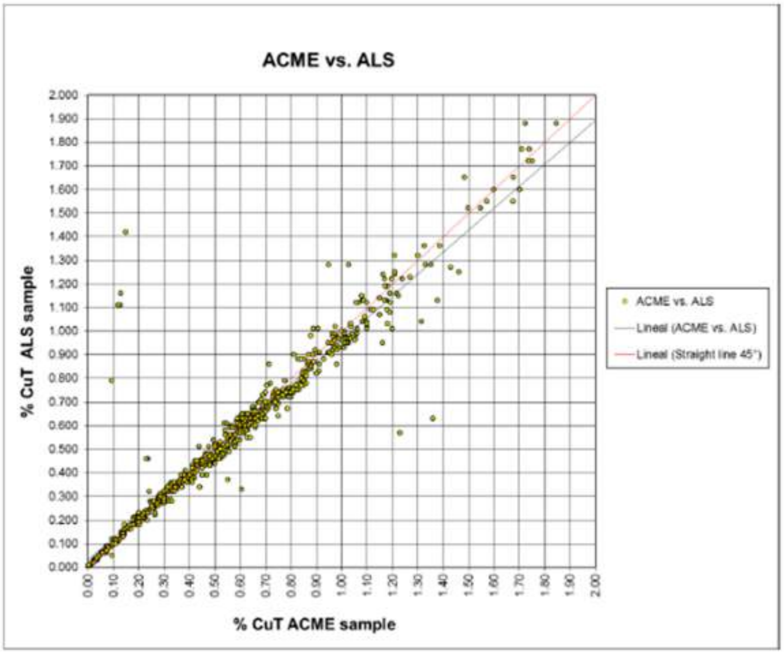


Figure 11-3: GMC Secondary Laboratory Duplicates ACME vs. ALS

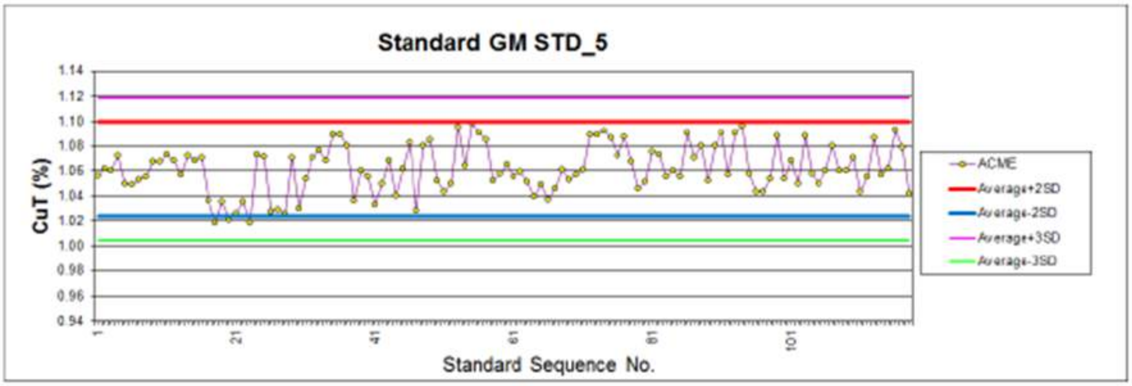
The copper assays from the drill holes V-28 to V-37 were used by MRA as part of the Kilborn study (Kilborn, 1998). According to MRA, the primary reference laboratory was ACME and controls were performed by CIMM. Both ACME and CIMM were commercial laboratories in Chile and were certified under ISO 9001.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

GMC used 6 different CRMs discontinuously and without an established procedure and ACME added its own reference material. This information shows that the results are consistent with little variance and no change in 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 11.4.

Table 11.2: ACME Reference Material Results

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 11.4: Acme Reference Material STD_5



Only portion of the leached and supergene zone samples were assayed to determine the soluble copper content and of those only a portion were assayed with sequential analysis. This type of analysis is an important tool to define with analytical precision the limits of mineralization, between the leached, supergene and hypogene mineralization.

Following the AMEC QA/QC review as part of the 2008 resource estimate, Los Andes Copper has taken actions to improve the level and accessibility of information related to sample preparation, analytical procedures and quality control protocols. Most of the information is now in Los Andes Copper's physical and electronic database, which includes 90% - 95% of test certificates and nearly 100% of the drilling logs, resulting in an improved standard level of QA/QC for the project.

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

During the 2007-2008 drilling, Los Andes Copper implemented a quality control (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 corresponding to the previously analyzed batches stored in the camp facilities
- 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 (CRMs) in a proportion of four per batch (8%). Two samples correspond to CRM Oreas 43P (for Mo) and two samples correspond to 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 shipping batches on site before sending them to SGS. Los Andes Copper also assigned 5% of the routine samples as verification samples (reject of duplicates) to the Actlabs laboratory (Actlabs) in La Serena. Los Andes Copper quality control programme did not include twin samples or pulp duplicates and AMEC recommended that such control samples be inserted in the same batches as the originals. AMEC also recommended that pulp duplicates should be sent to a secondary laboratory to externally verify the analytical precision of the test laboratory.

All samples from Los Andes Copper were sent to the SGS Lakefield research laboratory in Santiago. SGS prepared and analyzed all samples in Chile. The methodology applied in this certified laboratory is in accordance with the procedures accepted by the industry.

During the drilling campaign conducted by Los Andes Copper between 2007 and 2008, SGS analyzed 10,092 copper samples by using industry standard procedures. Copper grades ranged from less than the detection limit of 0.001% to 1.74%. A total of 10,088 samples were analyzed by SGS for molybdenum and grades varied from 0.0005% to 0.5%.

As part of the QA/QC review for the 2008 Technical Report, AMEC reviewed 100 coarse duplicates, 714 certified reference materials and 356 blanks taken during the drilling campaign.

At the end of the 2007-2008 drilling campaign, Los Andes Copper had inserted a total of 159 coarse duplicates, 940 certified reference materials and 469 blanks into the samples sent for analysis.

11.3.1 Duplicates

During the 2007-2008 drilling campaign, a total of 159 coarse duplicates were added to the assay sample batches. The copper failure rate was 4% with a total of 6 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 are prepared from the coarse reject sample from a previous sample batch that is inserted into the batch.

Coarse duplicates were checked during sample preparation and sub-sampling for variance, sub-sampling precision and contamination. Pulp duplicate tests were evaluated for analytical accuracy, precision and contamination.

During the QA/QC review for the 2008 Technical Report, AMEC found that the sub-sampling variance was within acceptable limits. However, AMEC suspected that some of the failing duplicates may have been mixed up and that these samples should be re-examined. Subsequently, a detailed review and validation of the assay database was completed by Los Andes Copper.

Figure 11.5: Los Andes Copper Coarse Duplicates Analysis

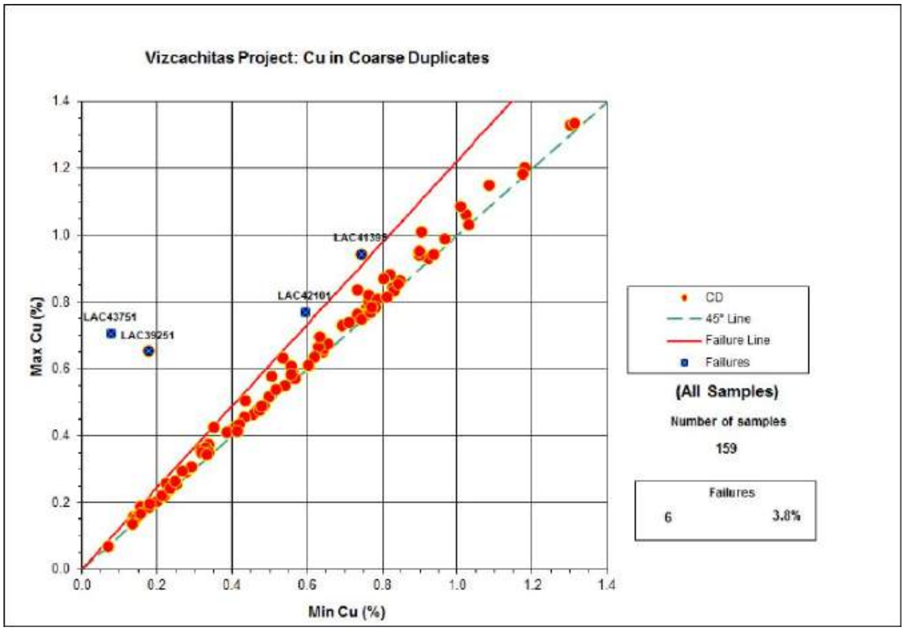


Figure 11.5 and Table 11.3 present the error variation of the duplicate samples. Los Andes Copper submitted a total of 1,291 duplicates in the 2007-2008 drill programme. Statistical analysis and comparison of the Los Andes Copper and GMC results (2,321 samples) show that the results in both cases are acceptable when considering the average, median, standard deviation, error and variance.

Table 11.3: Statistical results of Duplicate Samples from the Vizcachitas Project. (LAC, June 2013)

Laboratory	ACME - GMC		SGS - LAC	
Parameters	Cu (%)_Rou	Cu (%)_Dup	Cu (%)_Rou	Cu (%)_Dup
Mean	0.320	0.319	0.333	0.333
Standard error	0.006	0.006	0.007	0.007
Median	0.267	0.267	0.301	0.303
Mode	0.016	0.016	0.319	0.243
Standard deviation	0.283	0.283	0.239	0.238
Sample variance	0.080	0.080	0.057	0.057
Kurtosis	6.547	7.110	2.173	2.172
Asymmetry coefficient	1.848	1.897	1.153	1.154
Range	2.397	2.412	1.683	1.684
Minimum	0.001	0.000	0.001	0.001
Maximum	2.398	2.412	1.684	1.695
Count	2,321	2,321	1,291	1,291
Mayor (1)	2.398	2.412	1.684	1.695
Low (1)	0.001	0.000	0.001	0.001
Confidence level (95.0%)	0.012	0.012	0.013	0.013

ACME – GMC samples are shown to the left and the SGS - Los Andes Copper samples to the right, all showing the original and duplicate values.

The diagrams for correlation coefficient, relative error and dispersion for the ACME and SGS results are shown in APPENDIX I.

11.3.2 Certified Reference Materials (CRM)

Los Andes Copper established an adequate programme to insert systematically 4 CRM types. Table 11.4 illustrates the certified grades and results for the CRM's used.

Table 11.4: 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 11.4 clearly shows that the bias detected is low for Oreas 93, 94 and 95 (copper standards). Conversely, the molybdenum standard (Oreas 43P) shows a high negative bias (-18.46%) indicating molybdenum analysis should be monitored closely should be monitored in future drilling programmes.

The assay results indicate the copper values for the standards 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 real molybdenum values in the deposit.

APPENDIX I graphically presents the assay results and statistical parameters for Oreas 92 (Figure 29.2) Oreas 93 (Figure 29.3), the medium copper grade standard and Oreas 94 (Figure 29.4) the high copper grade CRM.

11.3.3 Blank Samples

Los Andes Copper inserted quartz rock blank samples into each batch. The statistics for 469 blank samples used by Los Andes Copper during the 2007-2008 drill campaign are shown in APPENDIX I (Figure 29.5).

Tetra Tech believes the results give an acceptable level of confidence to the work completed by SGS at this level of project maturity. The results support the Vizcachitas data base to an acceptable level of precision.

11.3.4 Samples to Check Laboratory

Los Andes Copper sent 440 coarse reject check samples to Actlabs, a second laboratory for independent validation. The results indicated an acceptable coefficient of correlation between both SGS and Actlabs and are as presented in APPENDIX I (Figure 29.6). The confirmation analyses showed good correlation with error variations no greater than ± 20%. This error may be explained as a result of the use of coarse rejects.

11.3.5 Los Andes Copper Assay Procedures

Lakefield Research (SGS) was the primary analytical laboratory. SGS prepared and assayed all samples from the 2007-2008 drill hole campaign. SGS is certified under ISO 9001-2000. All assays were conducted in Chile.

Tetra Tech did not review the SGS preparation and assaying protocols, but according to 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 sample using rotary splitter to obtain a nominal 1,000 g subsample for pulverizing;
- Pulverizing the nominal 1,000 g split with a 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 backup.

Samples were subsequently analyzed by Atomic Absorption Spectrometry (AAS) for total copper and molybdenum following three-acid digestion (Table 11.5). Occasionally, sulphuric acid-soluble copper and citric acid-soluble copper were also determined. The methods used by SGS were AAS023G for copper and Mo, AAS051D for sulphuric acid soluble copper and AAS067D for citric acid-soluble copper. SGS reported that the detection limit for all of these methods was 0.001%.

Table 11.5: SGS Analytical Methods

Element	Digestion	Determination	Limits	Lab Code
Total Cu, Total Mo	A prepared sample (1.0 g aliquot) is digested with nitric, hydrochloric and perchloric acids	The resulting solution is analyzed by atomic absorption spectrometry (AAS) in AA-Varian Spectra spectrometer	Cu: 0.001% to 10% Mo: 0.001% to 10%.	AAS023
Soluble Sulphur-acid Cu	A prepared sample (1.0 g aliquot) is digested with 25 ml of sulfuric acid, 5% v/v	The resulting solution is analyzed by atomic absorption spectrometry (AAS) in AA-Varian Spectra spectrometer	Cu: 0.001% to 10%	AAS051D
Soluble Citric-acid Cu	A prepared sample (1.0 g aliquot) is digested with 25 ml of citric acid, 1M	The resulting solution is analyzed by atomic absorption spectrometry (AAS) in AA-Varian Spectra spectrometer	Cu: 0.001% to 10%	AAS067D

11.4 Los Andes Copper 2015-2017 Drilling QA/QC and Re-analysis of Historical Pulp Samples

A total of 5,270 routine core samples from the 2015-2017 drilling campaign and a total of 16,225 historical pulp samples were assayed. A full Quality Assurance and Quality Control (QA/QC) programme was implemented to ensure the accuracy and precision of the assay results.

Los Andes Copper has stored most of the pulp samples from the 1997-1998 drill campaign and all the pulp samples from the 2007-2008 drill campaigns. Historically these samples had only been analysed for copper and molybdenum using atomic absorption. In 2017, these historical pulp samples were analysed using the ALS geochemical procedure ME-MS61, using ultra trace level ICP-MS analysis. All ME-MS61 copper assays with values of greater than 6,000 ppm Cu were also analysed using the ALS geochemical procedure ME-OG62, which uses a 4 acid near-total digestion and ICP finish.

11.4.1 Core Cutting and Sample Preparation

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

- The drilling contractor would lay out the drilled core into core boxes provided by Los Andes Copper. The drillers would then insert the wooden tags into the core boxes indicating the end of each run and its depth.
- Los Andes Copper personnel transported the core from the drilling rig in secure covered boxes to the project camp site.
- The core was checked, and the geotechnical logging carried out including core recovery and RQD.
- The core was photographed.
- The core was logged by the geologist for lithology, alteration, veining, mineralization and structure.
- The sample intervals were marked by the geo-technician. The interval was 2 m except for conforming to contacts of rock types.
- The geologist marked the centre line along which the core was to be cut. This was to ensure that as far a possible the core was cut diagonal to the veining.
- All geotechnical logging and geological logging data was 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 second half was stored in the core box. When the core was highly fractured and it could not be cut by the core saw, it was separated by hand into the material that was sent to laboratory for analysis and material that was stored in core box.
- The samples were stored at the project site in a secure location until shipment.

- The samples were shipped to the laboratory in a truck accompanied by Los Andes Copper personnel. At the laboratory, the samples bags were counted to verify any possible discrepancies between the samples shipped and samples received.
- Once the samples were assayed, the coarse rejects and the pulp samples were shipped to the project site for storage.

11.4.2 QA/QC Programme

For the 2015-2017 drilling campaign, Los Andes Copper implemented a protocol consisting of the insertion of approximately 1 control sample for every 6 routine samples, giving an insertion frequency of 16%. The following control samples were inserted:

- Twin Samples at a proportion of approximately one every 50 samples. A twin sample is where both halves of the cut core were analysed.
- Coarse blanks in a proportion of approximately one every 40 samples. The coarse blank was prepared from 2-inch cubes of quartz provided by a certified assay laboratory.
- Coarse duplicates at a proportion of approximately one every 60 samples prepared from the coarse rejects corresponding to the previous sample.
- Fine duplicates at a proportion of approximately one very 60 samples, these were prepared from the pulverised sample corresponding to the previous sample.
- Fine blanks in a proportion of approximately one every 50 samples. These were prepared from pulverised quartz samples provided by a certified assay laboratory.
- Certified Reference Material or Standards in a proportion of approximately one every 15 samples. The Standards were provided by ORE Research & Exploration Pty Ltd, Bayswater North, 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 that were to be used for the preparation of the coarse blanks and the coarse duplicates were identified in each batch. Once ALS had prepared all the samples, including the coarse blanks, coarse duplicates and fine duplicates, Los Andes Copper personnel inserted the standards and the fine blanks into the sequence of pulp samples. The samples numbers were then changed so that the laboratory would not know which samples were QA/QC samples. This task was carried out at the ALS Laboratory in Santiago. The re-numbered samples were then sent by ALS to their laboratory in Lima, Peru for analysis.

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

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

- Fine duplicates at a proportion of approximately one very 60 samples. These were prepared from the pulverised sample corresponding to the previous sample

- Fine blanks at a proportion of approximately one every 50 samples. These were prepared from pulverised quartz sample provided by a certified assay laboratory.
- Certified Reference Material or Standards at a proportion of approximately one every 15 samples. The Standards were provided by ORE Research & Exploration Pty Ltd, Bayswater North, Victoria, Australia.

The type and number of control samples that were used as part of the QA/QC programme are summarised in the table below.

Table 11.6: 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>Fine 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>Fine duplicate</td><td>79</td><td>66</td><td>480</td><td>33</td></tr>
<tr><td>Twin duplicate</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>

11.4.3 ALS Sample Preparation and Analysis of the 2015-2017 Core Samples

All core was cut to in half using a diamond saw. The half-core was sent to the ALS laboratory in Santiago, Chile, where the samples were prepared using the ALS PREP-31B procedure. The half-core sample was logged into the ALS laboratory tracking system, weighed, dried and the entire sample was crushed to produce a crushed product with 70% of material less than 2 mm diameter. One kilogram was then split and pulverized to more than 85% passing 75 µm.

All samples were analysed using the ALS geochemical procedure ME-MS61 using ultra trace level ICP-MS analysis. The prepared sample (0.25 g) is digested with perchloric, nitric, hydrofluoric and hydrochloric acids (four acid digestion) and analyzed by inductively coupled plasma-atomic emission spectrometry. The detection limits for the main elements is as follows: Cu 0.2 ppm, Mo 0.05 ppm and Ag 0.01 ppm. All copper assays with values of greater than 6000 ppm Cu were analysed using the ALS geochemical procedure ME-OG62 which uses a 4 acid near-total digestion and ICP finish.

11.4.4 Twin and Duplicates Samples

Twin and duplicate samples were added to the sequence of samples. These samples assisted in the evaluation of the sampling variance.

Three types of duplicates were inserted:

Twin Samples – The second half of the cut core was sent for analysis. These samples helped assess the sampling variance.

Coarse Duplicates – These samples were a split of the samples prepared at the laboratory once they had been crushed.

Fine Duplicates – These samples were a second split from a prepared sample. These samples were indicators of the analytical precision at the laboratory.

The copper and molybdenum results for each sample are summarised in the tables below.

Table 11.7: Twin and Duplicate samples - Absolute difference between samples

Cu	Twin Samples		Coarse Duplicates		Fine Duplicates	
Absolute Difference Range	No of Samples	% of Samples	No of Samples	% of Samples	No of Samples	% of Samples
0 - 10%	75	61%	65	82%	75	95%
10 - 20%	32	26%	10	13%	1	1%
> 30%	15	12%	4	5%	3	4%
	122		79		79	
Mo	**Twin Samples**		**Coarse Duplicates**		**Fine Duplicates**	
Absolute Difference Range	No of Samples	% of Samples	No of Samples	% of Samples	No of Samples	% of Samples
0 - 10%	76	62%	45	57%	64	81%
10 - 20%	8	7%	14	18%	6	8%
> 30%	38	31%	20	25%	9	11%
	122		79		79	

Twin Samples

A total of 122 twin samples were added to the sample sequence. These samples demonstrated that the variability between the two halves of the core was greater than 30% for 12% of the samples. For the molybdenum assays 31% of the samples had a variability of greater than 30%. No systematic bias was identified between the left and the right-hand sides of the samples.

In view of the high variability for the assay results for the twin samples, the photographs of the twin samples and their neighbouring samples were reviewed to identify the source of this variability. The samples with higher variability were normally the samples that had a high proportion of the mineralization associated with veining. This was especially true for the molybdenum where the molybdenite can be seen on the fracture surfaces. To ensure that the core was cut to minimise any variability, the cut line was marked by the geologist once he had inspected the core and reviewed the direction of the veining and the intensity of the mineralization. While this procedure reduced the variability, it did not eliminate it. This high variability shows the importance of the sample selection and the cutting of the core. The variability generated from the cutting of the core is far greater than any other action taken during the sample preparation.

Coarse duplicate

During the 2015-2017 drilling campaign, a total of 79 coarse duplicates were added to the sample sequence. Eighty two percent of the samples had an absolute difference of less than 10% for the copper assay and 57% for the molybdenum assay. These results are within the limits expected for this type of sample.

Fine duplicates

During the 2015-2017 drilling campaign, a total of 79 fine duplicates were added to the sample sequence. Ninety five percent of the copper assays had an absolute difference of less than 10% while 81% had and absolute difference of less than 10% for the molybdenum assay. The results show good correlation between the twin duplicates and are within the expected limits for this type of duplicate.

A total of 479 fine duplicates were added to the pulp sampling programme. Ninety nine percent of the samples had an absolute difference of less than 10% for the copper assay, while for the molybdenum assay 72% had and absolute difference of less than 10%.

11.4.5 Blanks

Two types of blank samples were inserted in the sample sequence.

Coarse Blanks - Barren quartz rock in 5 cm lumps. This rock passes through all the sample preparation processes. The coarse-blank samples check for contamination during the sample preparation procedure and identify any mistakes made in the numbering or during the handling of the samples.

Fine Blanks – Barren quartz rock that has been pulverised. This sample checks for contamination during the analysis or for mistakes made during the numbering or handling of the analysis.

Table 11.8: Blank Samples

Coarse Blanks						
Element	Nº Blanks	Minimum	Maximum	Average	Variance	Standard Deviation
Cu	126	5.50	46.20	14.92	44.89	6.70
Mo		0.47	9.73	1.66	2.09	1.44
Ag		0.01	0.05	0.02	0.00	0.01

Fine Blanks Drilling						
Element	Nº Blanks	Minimum	Maximum	Average	Variance	Standard Deviation
Cu	100	4.50	64.90	8.73	37.01	6.08
Mo		1.15	17.50	2.55	5.03	2.24
Ag		0.01	0.23	0.05	0.00	0.03

Fine Blanks Pulp Reassay						
Element	Nº Blanks	Minimum	Maximum	Average	Variance	Standard Deviation
Cu	470	5.00	35.50	11.54	10.58	3.25
Mo		0.89	5.05	1.40	0.26	0.51
Ag		0.01	0.44	0.04	0.00	0.06

The initial results from batch 1 had elevated assay values for the blank samples. This was reviewed by the laboratory and by checking the colour of the pulp samples. It was evident that there had been an issue during sample preparation. The blank samples are white in colour and when checking the pulp samples, it was clear that the samples had been mixed up. Re-analysis of the whole 500 sample batch was not undertaken, only those samples between the blank samples that did not have any errors were re-analysed.

The results of the remaining blanks were within the limits expected. None of the blank samples had values approaching the lowest background values of the normal samples.

11.4.6 Mineral Grade Analysis

All samples were initially analysed using the ALS geochemical procedure ME-MS61 using ultra trace level ICP-MS analysis. All copper assays with values of greater than 6,000 ppm Cu were then re-analysed using the ALS geochemical procedure ME-OG62 which uses a 4 acid near-total digestion and ICP finish. This cut-off was selected after discussion with ALS related to the precision of the analysis verses the copper grade. The correlation coefficient between the two analysis methods is 0.97.

The result from the ME-OG62 analysis is the assay result that has been used in the resource estimation.

11.4.7 Certified Reference Material

Certified Reference Materials (CRM, also known as Standards) were inserted into the sample sequence. These Standards were purchased from Ore Research & Exploration Pty Ltd (ORE) of Australia. These samples have defined assay values for multiple elements including copper, molybdenum and silver. The Standards are prepared under special conditions by a certified commercial laboratory and are used to estimate the assay accuracy.

Each CRM is supplied with a certificate indicating the Certified Value for each element, the 1st, 2nd, and 3rd Standard Deviations, 95% Confidence Limits and the 95% Tolerance Limits. This data enables the user of the CRM to evaluate the accuracy of the assaying. If there are multiple assays outside the 2 standard deviations limit, the batch is reviewed. A value outside the 3 standard deviations limit is considered a failed assay. The ALS QA/QC for each batch is also available for review via the online system Webtrieve.

The 8 different standards cover the full grade spectrum for the copper, molybdenum and silver mineralization that is found at the Vizcachitas project.

The following Standards were used during the 2015-2017 drilling campaign and for the re-analysis of the pulp samples.

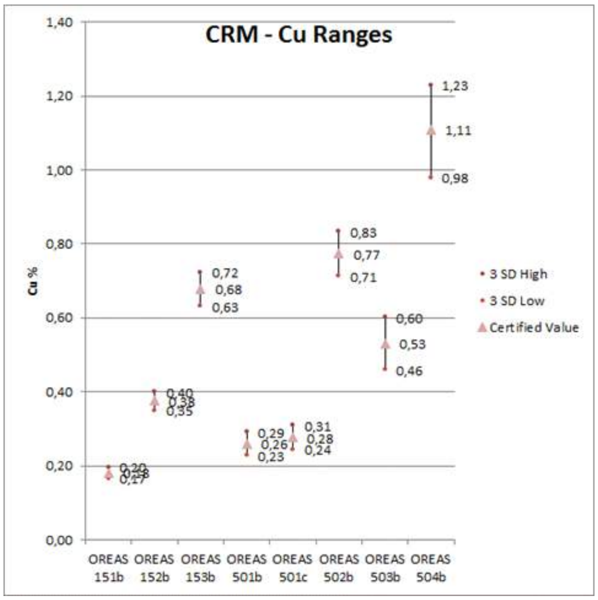
Table 11.9: CRM used during the drill programme

	Cu			Mo			Ag		
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

Out of a total of 788 Standards that were inserted into the sample sequence, a total of 18 samples (2.3% of the Standards) had assay results outside of 3 standard deviations for either copper, molybdenum or silver. Twelve of the assays outside the 3 standard deviations were for the molybdenum assay of the standard Oreas 501c and Oreas 151b. These are low grade molybdenum assays with narrow standard deviations. The 3-standard deviation limit for Oreas 151b is 62 ppm and the Certified Value is 55.

For each of the batches where there were assay results outside of the 3-standard deviation limit, the results were reviewed in detail. The ALS standards results were downloaded from their Internet site and the results of the batches discussed with ALS Incident Team. It was decided after this review not to re-analyse these batches.

Figure 11.6: CRM Cu and Mo 3 Standard Deviation Ranges



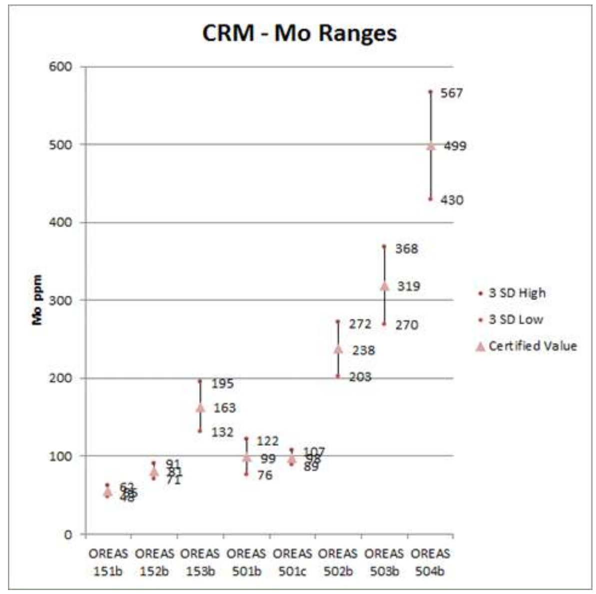


Figure 11.7: CRM Ag 3 Standard Deviation Ranges

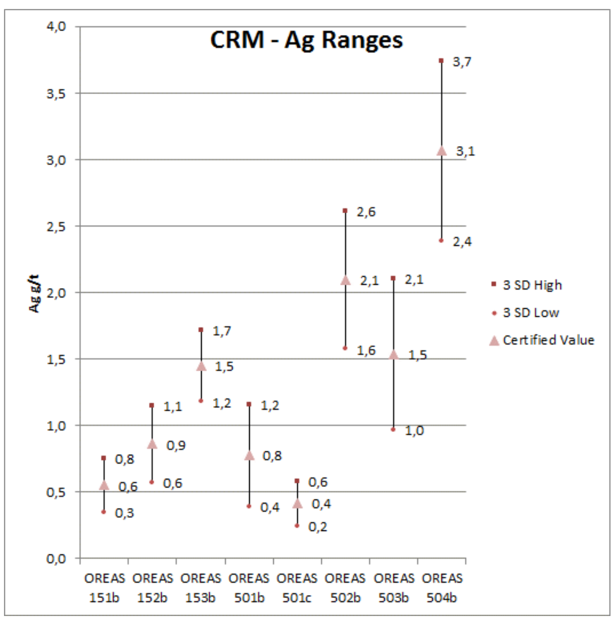


Table 11.10: Summary of Drilling and Pulp re-assay CRM Cu

RM	N	Cu wt.%		Observed Cu wt.%		Percent of Accepted
		Accepted	Std. Dev.	Average	Std. Dev.	
Oreas 504b	49	1.110	0.042	1.105	0.024	99.5%
Oreas 503b	130	0.531	0.023	0.528	0.015	99.5%
Oreas 502b	114	0.773	0.020	0.762	0.022	98.6%
Oreas 501c	85	0.276	0.008	0.276	0.007	100.0%
Oreas 501b	76	0.260	0.011	0.261	0.007	100.4%
Oreas 153b	88	0.678	0.015	0.678	0.017	100.0%
Oreas 152b	105	0.375	0.008	0.380	0.010	101.5%
Oreas 151b	141	0.182	0.005	0.185	0.005	101.6%
Total	**788**				**Weighted Average**	**100.2%**

Table 11.11: Summary of Drilling and Pulp re-assay CRM Mo

RM	N	Mo ppm		Observed Mo ppm		Percent of Accepted
		Accepted	Std. Dev.	Average	Std. Dev.	
Oreas 504b	49	499.000	23.000	480.980	16.649	96.4%
Oreas 503b	130	319.000	16.000	311.331	13.037	97.6%
Oreas 502b	114	238.000	11.000	234.158	8.918	98.4%
Oreas 501c	85	163.000	10.000	161.903	5.299	99.3%
Oreas 501b	76	99.000	7.500	101.509	3.812	102.5%
Oreas 153b	88	98.000	3.000	100.306	3.846	102.4%
Oreas 152b	105	81.000	3.400	83.432	2.792	103.0%
Oreas 151b	141	55.000	2.200	58.062	2.456	105.6%
Total	**788**				**Weighted Average**	**101.0%**

Table 11.12: Summary of Drilling and Pulp re-assay CRM Ag

RM	N	Ag ppm		Observed Ag ppm		Percent of Accepted
		Accepted	Std. Dev.	Average	Std. Dev.	
Oreas 504b	49	3.070	0.220	3.114	0.116	101.4%
Oreas 503b	130	2.090	0.170	2.083	0.096	99.6%
Oreas 502b	114	1.540	0.190	1.539	0.054	100.0%
Oreas 153b	88	1.450	0.090	1.519	0.057	104.7%
Oreas 501b	76	0.861	0.096	0.914	0.039	106.2%
Oreas 501c	85	0.778	0.128	0.749	0.032	96.3%
Oreas 152b	105	0.551	0.068	0.556	0.033	100.8%
Oreas 151b	141	0.417	0.053	0.438	0.053	105.1%
Total	**788**				**Weighted Average**	**101.7%**

11.4.8 Second Laboratory

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

The results for the copper and molybdenum assays showed good correlation. The silver results for the re-assayed samples showed a negative bias.

The QA/QC for the batch showed that of the 21 Standards that were inserted into the batch, one Standard had a value greater than the 3-standard deviation limit for molybdenum. For Silver, out of the 21 Standards inserted the Standard Oreas 501b, 3 had assays with low values outside of the 3 Standard Deviation limit. The second laboratory used atomic absorption with a lower detection limit of 0.2 g/t Ag. The original samples were analysed 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 reason the ALS ME-MS61 was used is because of the low detection limit for the silver analysis, which should mean that the precision at the level of 0.5 g/t to 2.0 g/t should be better than that for the AA analysis.

Further work should be carried out to investigate the silver grade. Analysis with a lower detection limit should be used to verify the ALS ME-MS61 silver assays.

Table 11.13: Summary of Second Laboratory CRM Cu

RM	N	Cu wt.%		Observed Cu wt.%		Percent of Accepted
		Accepted	Std. Dev.	Average	Std. Dev.	
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 11.14: Summary of Second Laboratory CRM Mo

RM	N	Mo ppm		Observed Mo ppm		Percent of Accepted
		Accepted	Std. Dev.	Average	Std. Dev.	
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 11.15: Summary of Second Laboratory CRM Ag

RM	N	Ag ppm		Observed Ag ppm		Percent of Accepted
		Accepted	Std. Dev.	Average	Std. Dev.	
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%**

11.5 Opinion on the Adequacy of Sample Preparation and Assay Quality

Three types of duplicates were inserted in each batch namely, twin samples (the second half of the cut core is sent for analysis), coarse duplicates and fine duplicates. Tetra Tech found the sample preparation to be adequate and comply with standard industry practice.

Los Andes Copper implemented a quality control programme which included the insertion of coarse duplicates, CRMs and coarse blanks. The precision of sub-sampling is within acceptable limits for both copper and molybdenum. The accuracy for copper is good and the accuracy for molybdenum has improved after marking of the centre line of the core by a geologist before being cut, however there is still room for improvement.

11.6 Los Andes Copper Core Sampling

Tetra Tech verified that these procedures were reliable and carried out according to industry standards.

The survey of collar coordinates was carried out by a topographic surveyor who sent a report (certificate) to the chief of geology who reviewed and validated it. The coordinates are stored in a Micromine database which identifies the source of the coordinates.

The downhole trajectory of the drill hole was recorded by a contractor for the drilling company. The data was sent to the geologists in an Excel file.

Collar coordinates, name of the logger, orientation, start and completion dates, were not recorded on the log sheets in most cases. The completeness and legibility varied from one drill file to another. However, the relevant information could be identified in all reviewed files.

Tetra Tech was able to verify that these procedures were reliable and performed according to industry standards and was able to verify compliance with the observations made by AMEC.

11.7 Specific Gravity Sampling and Determination

For drilling campaigns prior to 2015, Los Andes Copper took specific gravity (SG) measurements on drill core on site using the common water displacement method. Trained Los Andes Copper personnel conducted SG determinations every 40 m down-hole or more frequently if major lithological changes occurred within that interval. The bulk density 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 air and under water. The method measures the volume of water displaced by the sample.

The specific gravity of drilling samples from campaigns in 2015-2016 and 2017 was calculated with the "geometric density" method, using the following formula:

Where,

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

v = volume of the core segment, calculated by $v = \Pi r^2$

11.8 Opinion on the Adequacy of Sample Preparation and Assay Quality

Tetra Tech found the sample preparation to be adequate and meet with standard industry practice. Los Andes Copper implemented a quality control programme which included the insertion of coarse duplicates, CRMs and coarse blanks. The precision of sub-sampling is within the acceptable limits for both copper, molybdenum and silver.

The precision for molybdenum assays for the twin sample is adequate, having implemented the marking of the centre line by a geologist prior to the cutting process. This issue should continue to be monitored as there may be room for improvement.

Regarding the calculation method of the specific gravity for campaigns in 2015-2016 and 2017, Tetra Tech suggested returning to the water displacement method and coating of the sample with paraffin wax.

12. DATA VERIFICATION

12.1 Qualified Person Current Site Visit

On January 8th, 2016 Mr. Humberto Ortega, a qualified person under Chilean law N° 20.235, visited the Vizcachitas Project.

At that time, the drilling operation had stopped temporally. The Los Andes Copper staff were mapping the core, cutting the core, preparing the samples and working on the cross sections of the deposit using recent information.

The site visit included:

- Review of the overall project geology.
- Inspections of the current drill platforms.
- Detailed examination of representative cores from several holes drilled at the Project site.
- Collection of five quarter core samples.

On May 10, 2017, Mr. Jose Luis Fuenzalida, a qualified person under Chilean law N° 20.235, visited the Vizcachitas Project.

During the field visit to the Project site, the drilling process was verified in the field, observing:

- Proper recovery of the drill core.
- The core at the drilling site after it has been removed from the drill pipe, was measured and marked with small wooden blocks and finally recorded in Excel.
- Transfer of the drill core to the samplers processing area in covered trays according to protocol.
- Placing the core trays on the logging table and review of the length and systematic regularization every 2 m.
- Geotechnical mapping of the core.
- Core marking by a geologist seeking an equal division of the mineralization in both halves of the core.
- Digital photographs of core in groups of three trays.
- Core cut by diamond saw.
- Detailed geological mapping of the: rock type, alteration, mineralization and main structures.
- Storage of the core trays in roofed racks.
- Batches of 50 samples in numbered bags sent to the laboratory. The transportation of the samples was carried out by a contractor that was governed by a protocol for the transfer

and reception of samples using forms that were signed by a laboratory representative on receiving the batch.

In opinion of Mr. Jose Luis Fuenzalida, the activities reviewed during the visit were in accordance with standards for a technical report. The geological information was of a level that would support a resource estimate process.

12.2 Data Validation

The historical information was validated by AMEC as part of the 2008 Resource Estimate Report. AMEC compared the sample intervals, the drill hole logs and the sectional geological interpretation for differences. This procedure was carried out on nine drill holes, accounting for 6.9% of the total drill holes as of that date.

Coffey confirmed that the GMC logging was completed in handwritten records which were then digitized. The spreadsheets included all variables normally considered for Cu-Mo porphyry mineral deposits. Los Andes Copper drill hole core log was entered on the PC tablets and complies with the logging information standard.

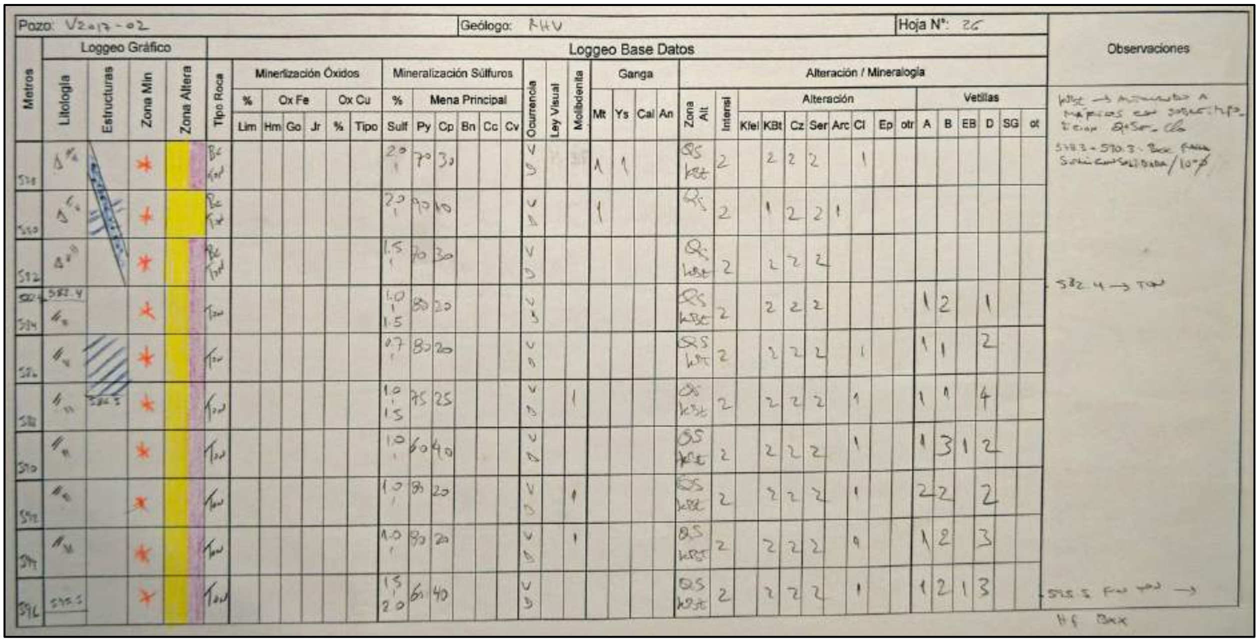
Tetra Tech validated the information from the 2015-2016 and 2017 drill hole campaigns by comparing some of the chemical analysis certificates with the digital database. During the visit, it verified the matching between the currently developed drill hole mapping (V2017-05) and the core. The geological mapping was also validated by comparing the mapping sheet with the digital database logs.

12.2.1 Log Files

The main files reviewed are related to certified documents and paper records (Figure 12.1), which were then entered into a documentary database and Excel including:

- Type of drill hole (DDH, diameter).
- Geological mapping chart containing depth, lithology, structures, mineral zones and alteration, mineralization of oxides, sulphides, gangue, and alteration/mineralization ratio, among others. Figure 12.1 shows a typical logging sheet.
- Logging of sample recovery by drilling intervals.
- Selection and logging of sampling intervals to send the samples for cutting, preparation and analysis.

Figure 12.1: Typical Drill Hole Mapping Sheet



The structure, organization and contents of the database supporting the resource model helped ensure the reliability of the information used in the mineral resource estimate and geological modeling.

12.2.2 Drill Hole Collars

AMEC verified the coordinates and elevation of the borehole as part of the 2008 Resource Estimate Report and determined that they are generally reliable. Coffey confirmed this conclusion but recommended that the topography be re-evaluated in the future.

Coffey did not review in detail the positions of the collars and accepted AMEC's conclusions. Coffey physically verified Los Andes Copper drill collars from the 2007-2008 drill campaign, confirming that they are in good condition and in accordance with industry practice.

Tetra Tech reviewed some of the collar files from recent drilling campaigns, comparing the collar coordinate certificates with the information contained in the database concluding that they show no differences. The reviewed information also included coordinate and trajectory certificates, survey type, depth, tilt, azimuth and trajectory.

12.2.3 Down-Hole Survey

In 2008, AMEC reviewed 61 drill holes and found no significant deviations from the reviewed drilled trajectories. However, according to AMEC's opinion, the drill holes deeper than 100 m should be systematically measured down to the bottom of hole.

Tetra Tech randomly reviewed the coordinate and trajectory data of the 61 drill holes contained in the database, graphic comparison of collar trajectories and coordinates, confirming that the database information is adequate and does not show unacceptable inconsistencies and is in compliance with the requirements of this Technical Report. No due diligence revision was performed, i.e. recovery of the trajectory's measurement.

12.2.4 Database Review

The review of the database carried out by Tetra Tech, shows that there are currently 165 drill holes in the database with a total of 52,256 m of drilling. Four of these drill holes have been discarded because they have no associated lithology or mineral zone and 161 drill holes were used in the resource estimate, as they have complete geology and copper grades.

The database review considered the structure, organization and contents of the unaudited data, to ensure the reliability of the information used in the construction of the geological and mineral resource estimate model.

Geological Data

The geological database adequately stores the data for the main mappable elements, such as lithology, structures, alteration, mineralization, observations, and assay values of the elements (copper, molybdenum, silver, and others). This information was well supported in paper and digital format. Tetra Tech believes that the contents of the main geological elements in the repository were well represented and protected.

Assay Data

Los Andes Copper has a complete paper and digital record of all drilled holes and surface sampling. This includes information from all campaigns that have been developed on the property by Placer, GMC and Los Andes Copper.

The information validated by Tetra Tech is mainly related to the 2015-2017 campaigns. The chemical analysis information was complete and reliable for the purposes of estimating the mineral resources and compilation of this technical document.

12.3 Interpretation of Geology, Alteration and Mineralization

The lithological model contained information from the remapping of historical drill holes (approx. 38,000 m but does not include the Placer Dome drill holes) and mapping of drill holes from the 2015-2017 campaigns (approx. 11,500 m).

This model was constructed from the modeling of 20 cross-sections, oriented N 110°E, separated by 100 m intervals. (Figure 12.2). These sections were used for 3D modeling using the Leapfrog software tool.

The model was reviewed in detail by Tetra Tech and became the basis for defining the estimate units (UE) used in the mineral resource estimate.

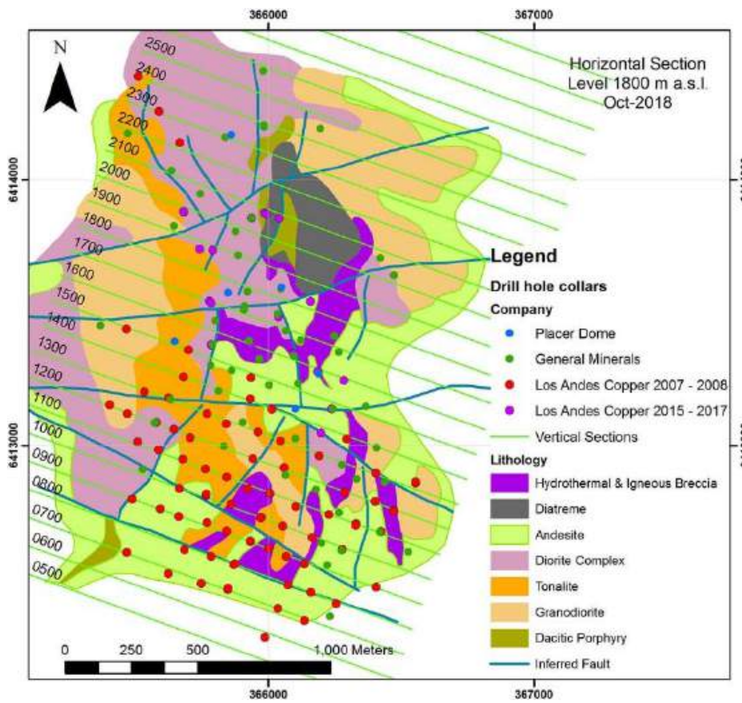
This new geological model has a family of productive porphyries intruding the andesitic host rocks that make up the core of Vizcachitas system. The main mineralising intrusive phases are the early diorite porphyry, an early inter-mineral tonalite intrusive, and a late inter-mineral granodiorite intrusive. The initial the early diorite porphyry has an average grade of 0.559% Cu, the inter-mineral tonalite intrusive has an average grade of 0.273% Cu and the inter-mineral granodiorite intrusive has an average grade of 0.170% Cu. Associated with each of these intrusive phases are hydrothermal breccias. The average copper grade of the most important hydrothermal breccia, BXH-1 is 0.54 % Cu. See Table 12.1

Table 12.1: Average Copper grade for Main Intrusive Phases

Description	Lithology	Mean Cu%	Max Cu%
Supergene		0.501	
GRD	105	0.170	1.167
TON	109	0.273	1.684
DEP-1	123	0.559	3.080
BXH-1	112	0.540	2.530

The importance of the early diorite porphyry and hydrothermal breccias in controlling the higher-grade mineralization of the deposit is the main difference from the previous geological models. This separation of the units means that the mine plan can be designed to extract these units in the early phases of the project.

Figure 12.2: Lithological Model Sections (LAC 2017)



The results were as follows:

- At present, the mineralized system has a recognized extension of approximately 2,000 m in the north-south direction, and 700 m in the east-west direction.
- At the southern end, a post-mineral trans-pressure NW fault with vertical movements places the porphyry and breccia complex to the north in contact with the andesites to the south. It is likely that as a result of this fault the southern block has descended, so some breccia and porphyries might be found at depth in this sector.
- At the northern end, the productive porphyry and breccia system is open. Drill hole V2017-10, the northernmost drill hole of the recent campaign intersected mineralization of > 0.5% Cu.
- The model integrated the hydrothermal breccias element into the system showing its true importance within the mineralized complex. Breccias are present with different dimensions along almost the whole mineralized corridor, from section 800 to 1900, and are open to the north.
- The geological modeling helped recognize two main mineralized breccia bodies: one at the southern end of the mineralized zone and the other in the central-northern part near the phreatomagmatic breccia. These breccia bodies are separated by 400 m and are 400 x 400 m in plan. The width of the breccias generally increases with depth.

- In addition, the model showed the extent of the diorite complex. This complex includes the early diorite porphyry, which is linked to the central-western part of the mineralized corridor and the phreatomagmatic breccia environment.

Figure 12.3 (section 1600) and Figure 12.4 (section 2100), provide a WNW-ESE view of the sectional model.

Figure 12.3: 1600 Section (LAC 2017)

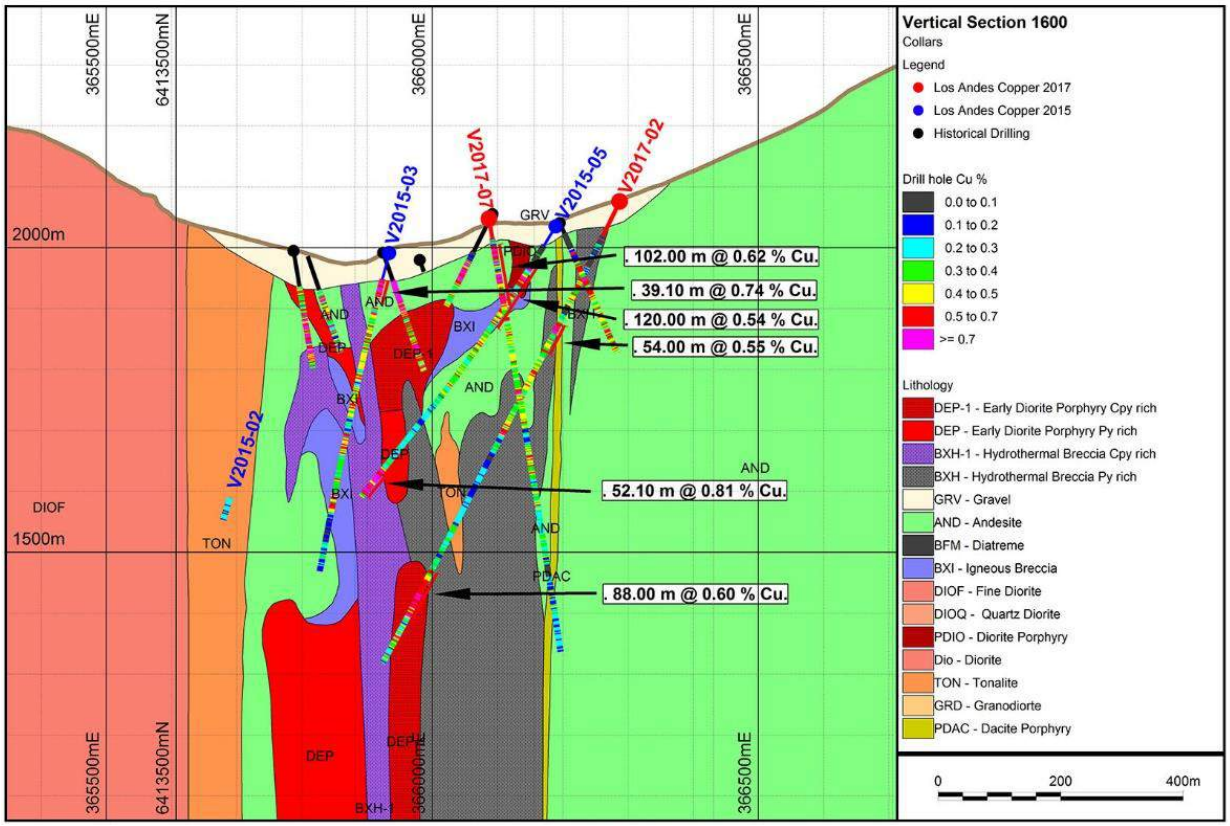
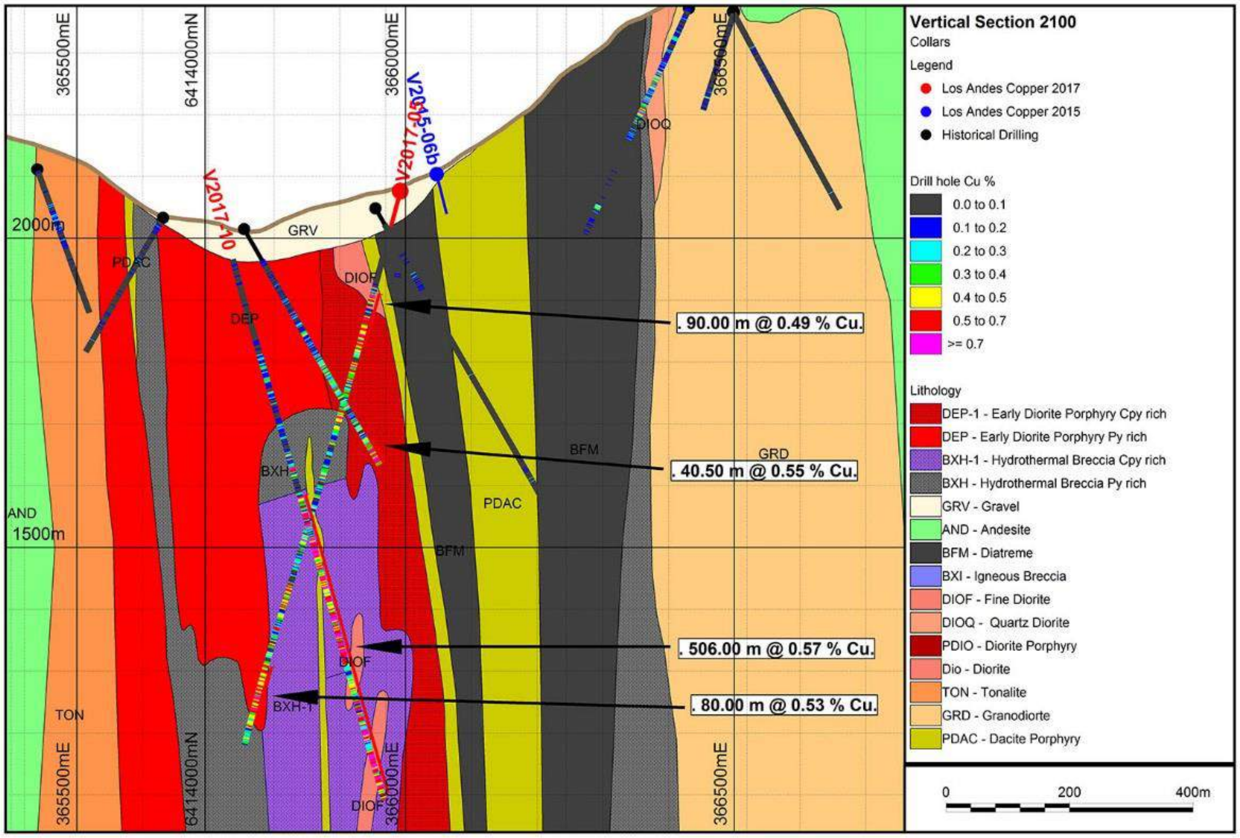


Figure 12.4: 2100 Section (LAC 2017)



12.4 Specific Gravity

GMC and Los Andes Copper conducted 938 specific gravity (SG) tests covering all types of rocks in different mineralization zones (leached, supergene, hypogene and overburden). The samples were taken from the drill cores.

In the opinion of Tetra Tech, Los Andes Copper has compiled a reasonably complete specific gravity database and considers the SG data to be reliable for the resource estimate.

Table 12.2: Number of Specific Gravity Samples by Mineral Zone

Mine Zone	N°. Samples
Hypogene	714
Leached	66
Overburden	1
Supergene	157
Grand Total	**938**

Table 12.3: Number of Specific Gravity Samples by Lithology

Lithology	Range		Average	N° Samples
	Minimum	Maximum		
AND	2.261	2.876	2.603	363
BFM	2.387	2.654	2.544	3
BXH	2.301	2.851	2.676	36
BXH-1	2.351	2.818	2.568	71
BXI	2.367	2.852	2.580	61
DEP	2.369	2.868	2.687	59
DEP-1	2.281	2.932	2.634	41
DIOF	2.496	2.678	2.618	18
DIOM	2.503	2.720	2.593	5
DIOQ	2.343	2.555	2.499	4
GRD	2.345	2.720	2.539	58
GRV	2.377	2.720	2.575	5
PDAC	2.161	2.778	2.541	22
PDIO	2.466	2.665	2.533	12
TON	2.235	2.783	2.539	180
All Sample	**2.161**	**2.932**	**2.589**	**938**

12.5 Opinion of Adequacy of the Database

The review of the database considered the structure, organization and contents of the data related to the NI 43-101 technical report ensuring the reliability of the information used in the construction of the geological and mineral resource estimate model.

The review was focused on:

- The Excel database. This contains the drill hole coordinates, trajectories, geology, sampling, grades, recovery, QA/QC and re-analysis (duplicates, standards, blanks).
- The geological information collection and storage process. Its management was reviewed, and it may be stated that the data backup and maintenance of this database is adequate. However, supporting a database in Excel does not comply with accepted industry standards, as this platform is easily modifiable and therefore unsafe. For the next stages

of the project Tetra Tech suggests using a drilling data base management software platform, such as GeoBank, aQuire, etc.

- The documentary database contains the information of documents in pdf, dwg, and jpg format. This database is quite complete, as it contains most of the historical documents generated by the project and adequately supports the database.

13. MINERAL PROCESSING AND METALLURGICAL TESTING

The Project has been the subject of a number of physical characterisation and metallurgical test programmes to determine the anticipated process route and expected recoveries. Bond work and abrasion indices have defined the mineral physical characteristics used in the development of the study and both leach and flotation test work have been carried out on Vizcachitas samples to determine likely process routes and recovery estimations.

During 2017, a number of metallurgical tests were carried out on 40 individual drill hole samples identified as PVMM-01 to PVMM-40 from the Vizcachitas mineralized body.

The tests carried out correspond to head mineral characterization, grinding tests and rougher circuit flotation tests at feed grain sizes and closed cycle tests, among others. The tests were conducted at SGS Minerals Services laboratories.

13.1 Comminution Test Work

13.1.1 1998: Ball Bond Work Index

In October 1998, Lakefield Laboratories (Lakefield Research Chile S.A., 1999a) determined standard Bond Ball Work indices (Wi) for five of the composite samples using standard laboratory procedures. Table 13.1 shows the results of the Bond tests.

Table 13.1: Summary of Bond Ball Mill Work Index (source: Lakefield Research Chile S.A., 1999a)

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

The work index values of the different composites exhibited large variations ranging from a low of 10.1 kWh/st to a high of 13.5 kWh/st.

13.1.2 2008: SMC Test and Bond Abrasion Test

A total of 6 samples for comminution tests and 6 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 mineral samples were regarded as hard to moderately hard. The results of the abrasion tests showed that the samples were regarded as moderately abrasive.

Table 13.2 and Table 13.3 show the results of SMC Test and abrasion index respectively

Table 13.2: Summary of SMC Test (source: SGS Minerals Services, 2009)

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

Where:

- DW_i: Drop weight index (drop weight test)
- M_{ia}: Comminution energy for coarse particles (down to 750 µm)
- AxB: Parameters obtained from SMC Test
- A_i: Abrasion index

Table 13.3: Summary Abrasion Index Test (source: SGS Minerals Services, 2009)

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

13.1.3 2017: Ball Bond Work Index

In 2017 Bond Index tests were conducted on the 40 drill hole samples, which were required for the sizing of the Ball Mills, gave average results of Wib of 13.1 kWh/t (11.9 kwh/st), ranging between 7.4 and 18.6 kWh/t (6.7 - 16.8 kwh/st), which is slightly higher than the that determined in the tests conducted in 1998 which gave a Wib of 12.7 kWh/t.

13.1.4 2017: Starkey Test

A Starkey test was conducted, measuring the time required to reach a given P_{80}. 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 that relates the SEC with grinding time (Starkey Test) was known.

The average value of grinding time found for the 40 samples was 66 minutes with a range between 32 minutes and 103.9 minutes.

13.2 Flotation Test Work

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 test work campaigns.

The database of flotation tests performed included the following programmes:

- 1996 Test work (Lakefield Research Chile S.A., 1996)
    - Chemical analysis
    - Mineralogical analysis

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

13.2.1 1996 Test Work

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 (source: Lakefield Research Chile S.A., 1996)

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 amount of chalcocite and minor amounts of covellite. Pyrite values varied between 1% and 6% by weight.

Table 13.5 shows the mineralogical characterization and identifies the species present in the samples, which was completed by automated quantitative mineralogy (QEMSCAN):

Table 13.5: Mineralogy (source: Lakefield Research Chile S.A., 1996)

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 tests and open-circuit cleaning tests were also completed on the five samples. 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; time 12 min (source: Lakefield Research Chile S.A., 1996)

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

Molybdenum recoveries, with a few exceptions, were generally unsatisfactory, this was mainly owing to the fact that diesel which is commonly used as a promoter for molybdenite flotation was not used during these tests.

Ten open-cycle tests were developed with three cleaning stages for different grind and regrind product sizes.

Table 13.7 presents the results obtained. Copper recoveries on third cleaner stage of 98% with a regrind size of 20 µm - 30 µm and final concentrate grades between 30% and 38% were obtained. A finer regrind product produced a lower grade copper concentrate. Recoveries were calculated on each individual test.

Table 13.7: Cleaner Flotation Summary (source: Lakefield Research Chile S.A., 1996)

Test	Sample	Conditions		Head				Rougher Concentrate				1st Cleaner Concentrate				2nd Cleaner Concentrate				3rd Cleaner Concentrate			
		P_{80}		Head, Calc.		Head, Dir.		Grade		Recovery		Grade		Recovery		Grade		Recovery		Grade		Recovery	
		Grinding [µm]	Regrind [µm]	Cu [%]	Mo [%]	Cu [%]	Mo [%]	Cu [%]	Mo [%]	Cu [%]	Mo [%]	Cu [%]	Mo [%]	Cu [%]	Mo [%]	Cu [%]	Mo [%]	Cu [%]	Mo [%]	Cu [%]	Mo [%]	Cu [%]	Mo [%]
S1-3	Saco 1	92	30	1.19	0.011	1.17	0.012	9.40	0.056	97.70	65.400	26.40	0.050	97.90	31.100	34.20	0.037	98.60	55.500	37.70	0.026	98.90	63.300
S1-4		92	53	1.18	0.010	1.17	0.012	9.70	0.048	89.50	57.400	17.90	0.063	98.00	69.900	21.40	0.059	98.60	77.900	23.80	0.042	98.80	62.400
S3-3	Saco 3	92	19	0.44	0.007	0.50	0.007	5.70	0.048	97.80	47.200	22.70	0.164	96.30	83.200	27.30	0.190	98.50	94.800	29.90	0.200	98.70	94.800
S3-4		92	19	0.43	0.007	0.50	0.007	9.30	0.179	98.70	47.300	22.60	0.150	83.70	5.200	22.60	0.140	83.70	5.200	22.60	0.121	83.70	5.200
S5-3	Saco 5	100	24	1.04	0.013	0.95	0.012	9.80	0.105	98.30	79.900	28.00	0.130	97.90	42.200	34.60	0.125	98.80	76.800	36.40	0.110	99.40	83.400
S5-4		100	37	0.95	0.012	0.95	0.012	9.40	0.103	97.90	86.300	21.60	0.172	97.30	70.400	26.80	0.175	98.10	80.700	28.30	0.150	98.90	80.200
S6-3	Saco 6	135	25	0.74	0.016	0.64	0.012	9.30	0.179	97.80	87.100	24.30	0.412	98.20	86.800	28.10	0.415	99.00	86.200	29.90	0.340	98.80	76.000
S6-4		135	39	0.67	0.014	0.64	0.012	9.50	0.169	98.20	81.700	20.80	0.314	98.40	83.800	25.40	0.340	98.80	87.900	27.90	0.290	98.30	76.100
S7-3	Saco 7	114	20	0.73	0.033	0.67	0.028	10.10	0.313	98.30	66.900	23.60	0.327	98.20	43.900	29.00	0.259	99.10	63.800	31.50	0.180	98.40	63.000
S7-4		114	35	0.66	0.035	0.67	0.028	8.40	0.276	98.30	61.300	21.40	0.335	98.20	47.000	25.50	0.278	99.40	69.000	27.80	0.190	99.00	62.100
		Average		0.80	0.016	0.79	0.014	9.08	0.148	97.30	68.000	22.90	0.212	96.40	56.400	27.50	0.202	97.30	69.800	29.60	0.165	97.30	66.700
		Maximum		1.20	0.035	1.20	0.030	10.10	0.313	98.70	87.100	28.00	0.412	98.40	86.800	34.60	0.415	99.40	94.800	37.70	0.340	99.40	94.800
		Minimum		0.40	0.007	0.50	0.010	5.70	0.048	89.50	47.200	17.90	0.050	83.70	5.200	21.40	0.037	83.70	5.200	22.60	0.026	83.70	5.200
		Standard		0.28	0.010	0.25	0.010	1.25	0.090	2.74	15.110	2.86	0.130	4.51	26.830	4.31	0.120	4.77	25.540	4.79	0.100	4.78	24.210

13.2.2 1998 Test Work

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 of up to 20%, which normally adversely affects the overall recovery of copper minerals. Table 13.8 illustrates the description and chemical analysis of the samples.

Table 13.8: Sample Description and Chemical Analysis (source: Lakefield Research Chile S.A., 1999a)

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 principal 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.

Table 13.9: Summary of Cu grade (wt %) and Percent Distribution of different Cu-bearing minerals (source: Lakefield Research Chile S.A., 1999a)

Sample	Cu	Chalcopyrite	Chalcocite	Covellite	Bornite	Tetrahedrite/ Tennantite
	[%]	[%]	[%]	[%]	[%]	[%]
A	0.95	58	20.3	20	1.4	-
B	0.55	66.4	21.3	10.9	1.4	-
C	0.57	99.8	-	-	0.2	-
D	0.96	73.3	18.5	4.8	0.2	2.1
E	0.59	79.6	2.7	13.3	1.6	2.8
F	0.5	97	-	-	-	3
G	0.78	59	18	23	-	-
H	0.54	73	12	15	-	-
I	0.43	98	1	-	1	7.0 (enargite)
J	0.53	66	4	7	14	-
K	0.59	78	3	7	12	-

Rougher flotation tests were carried out at two different grind sizes with P80´s of 100 μm and 150 μm. The best results were obtained at the finer grind using Aeropromoter 3477 and diesel as a molybdenite promoter. Table 13.10 shows the results for Composite A.

Table 13.10: Summary of Rougher Flotation Test Composite A (source: Lakefield Research Chile S.A., 1999a)

Test	Conditions						Calculated Head		Rougher Concentrate			
	Reagent [g/t]					P_{80} Grinding			Grade		Recovery	
	Collector			Frothers			Cu	Mo	Cu	Mo	Cu	Mo
	3477	SIPX	Diesel	MIBC	DF-250 /MIBC	μm	[%]	[%]	[%]	[%]	[%]	[%]
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

It should be noted that this was a relatively high-grade composite sample at over 0.9% Cu.

The open cycle test was composed of rougher flotation stages, first cleaner, scavenger and second cleaner. The grind size P_{80} was 100 μm.

Table 13.11 shows the results of the tests for the composites J and K (two tests for each composite). It was noted that the finer regrind product (P_{80} 60 µm) generated a lower grade copper concentrate at the second cleaning stage. Tests with regrind size P_{80} of 28 µm obtained concentrate copper grade of 30%. The table also incorporates an estimate of recoveries for locked cycle test work.

Table 13.11: Summary of Cleaner Flotation Result – Samples J and K (source: Lakefield Research Chile S.A., 1999a)

Test	Sample	Conditions	Head, Calc.		Rougher Concentrate				1st Cleaner Concentrate				2nd Cleaner Concentrate			
			Grade		Grade		Recovery		Grade		Recovery		Grade		Recovery	
		P_{80} Regrind [µm]	Cu [%]	Mo [%]	Cu [%]	Mo [%]	Cu [%]	Mo [%]	Cu [%]	Mo [%]	Cu [%]	Mo [%]	Cu [%]	Mo [%]	Cu [%]	Mo [%]
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 (%)			Cu	97.30							
					Cleaning circuit (locked cycle)			Mo	89.40							

Five locked cycle tests using the same test circuit described in the open cycle test work were completed. These tests were carried out for four samples obtained from mixing the 11 previous composites. The composites of the four samples tested are as presented in Table 13.12.

Table 13.12: Composites used in Locked Cycle Test

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 very similar to those used for the open circuit test work. The main difference in operating conditions was that a small amount of sodium cyanide (NaCN) was added to the roughing stage to suppress pyrite in two of the composite samples (JK and CFI). The regrind size P_{80} was 28 µm.

Table 13.13 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 13.13: Summary of Locked Cycle Tests (source: Lakefield Research Chile S.A., 1999a)

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 the first composite sample, the head grades for these 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 are poor.

Impurity analyses of the final concentrates showed that there were some impurities which were above the limit normally permitted without incurring penalties (0.024% Sb, 0.24% As). This should be reviewed in any future test work.

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

13.2.3 2008 Test Work

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; all of these samples were supplied for flotation test work. These samples were coarse rejects from drilling campaigns.

Table 13.14 presents the chemical analysis of the eleven samples received. The mineral had a copper grade between 0.34% and 0.76% and had a molybdenum grade between 0.008% and 0.023%.

Table 13.14: Chemical Analysis (source: SGS Minerals Services, 2009)

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 P_{80}'s of 100 µm, 130 µm and 160 µm.

Table 13.15 and Table 13.16 show the summary of results obtained in the rougher kinetics flotation test.

The tests carried out in 2008 show that very good rougher recoveries in the range of 94% to >98% were achieved for all of the composite samples. The grind had significantly less of an effect on the rougher recovery when compared to previous test work. The recoveries were less than one percent lower for the coarser grind than for the finer grind.

Rougher grades were lower than those from previous test work 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%), while the rougher grades were relatively low due to the lower head grades.

Table 13.15: Cumulative Cu Recoveries and Grades at different P_{80} – 12 minutes flotation residence time (source: SGS Minerals Services, 2009)

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 13.16: Cumulative Mo Recoveries and Grades at different P_{80} – 12 minutes flotation residence time (source: SGS Minerals Services, 2009)

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	0.16
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	0.15
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 P_{80}'s of 35 µm and 50 µm and included two stages of cleaning. An initial grind size P_{80} of 160 µm was defined 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 13.17 and *Table 13.18*. The tables also incorporate an estimate of recoveries for the locked cycle circuit.

Recoveries are calculated for each individual test.

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 13.17 and Table 13.18. The tables also incorporate an estimate of recoveries for the locked cycle circuit.

Recoveries are calculated for each individual test.

Table 13.17: Copper and Molybdenum Recovery, regrind at 35 µm – two cleaner stages (source: SGS Minerals Services, 2009)

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 13.18: Copper and Molybdenum Recovery, regrind at 50 µm – two cleaner stages (source: SGS Minerals Services, 2009)

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					

The conditions for cleaner stages were changed and tested on four samples, a finer regrind with a P80 of 25 µm and three stages of cleaning were tested. There was generally an improvement in recovery and concentrate grade as shown in Table 13.19

Open-circuit cleaner tests were carried out on the five samples delivered to SGS in October 2008. These were completed at a regrind size P80 of 35 µm and three stages of cleaning. The results of these tests are shown in Table 13.20

Table 13.19: Copper and Molybdenum Recovery, regrind at 25 µm – three cleaner stages (source: SGS Minerals Services, 2009)

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							

Open-circuit cleaner tests were carried out on the five samples delivered to SGS in October 2008. These were completed at a regrind size P_{80} of 35 µm and three stages of cleaning. The results of these tests are shown in Table 13.20.

Table 13.20: Copper and Molybdenum Recovery, regrind at 35 µm – three cleaner stages (source: SGS Minerals Services, 2009)

Test	Sample	Regrind of 35 µm								Regrind of 35 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
17	43522	95.40	96.80	94.70	85.60	3.70	14.70	23.60	28.50	92.10	97.00	93.20	85.30	0.20	1.00	1.50	1.80
18	43523	95.00	95.30	95.70	84.40	7.70	26.50	31.00	33.60	91.10	94.20	95.00	83.90	0.13	0.44	0.51	0.55
19	43524	96.00	95.80	92.50	87.00	3.70	17.00	26.70	30.60	90.70	93.80	88.50	81.30	0.08	0.36	0.54	0.58
20	43525	97.00	94.60	75.40	57.00	5.70	17.00	26.10	30.30	91.70	92.40	68.60	51.50	0.15	0.44	0.61	0.64
Average		95.60	95.10	86.50	73.70	5.70	20.90	28.70	32.20	91.80	94.10	83.00	71.40	0.15	0.57	0.79	0.88
Maximum		97.00	96.80	95.70	87.00	7.70	29.20	36.10	37.80	93.50	97.00	95.00	85.30	0.24	0.96	1.52	1.83
Minimum		94.50	93.30	74.00	54.40	3.70	14.70	23.60	28.50	90.70	92.40	68.60	51.50	0.08	0.36	0.51	0.55
Standard dev.		0.90	1.30	10.80	16.50	2.00	6.50	4.90	3.60	1.10	1.80	12.80	16.80	0.06	0.25	0.42	0.54
Calculated recovery (%) Cleaning circuit (locked cycle)		98.8								96.4							

13.3 Leach Test Work

The database of leach tests completed included the following programmes:

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

13.3.1 1999 Test Work

13.3.2 1999-2000 Test Work

The second set of leach tests were also carried out in Lakefield Research in Santiago in 1999-2000. (Lakefield Research Chile S.A., 2000).

A column leach test work programme was carried out on five different samples from Vizcachitas. The objective of the study was to characterize the metallurgical response of these minerals when processed under bacterial leaching conditions.

The received diamond drill rejects were crushed to 100% - 3/4" with a P_{80} of ½" and analysed for chemical and mineralogical content. Three of the mineral samples contained variable amounts of chalcocite with copper grades ranging from 0.6% Cu_T to 0.8% Cu_T. The other two samples were low grade containing approximately 0.4% Cu_T, mainly present as chalcopyrite.

Preliminary acid consumption tests were carried out to determine the amount of acid to add to agglomeration.

Five 2 m high by 0.10 m diameter columns were charged and leached in April 1999 using an open circuit with an irrigation rate of 6 l/h/m^2. The columns were continuously inoculated with a mixture of typical ferro-oxidising bacteria for about 180 days. After this period, air injection and recycling of solutions and a solvent extraction (SX) operation on a weekly basis was instituted. The overall leach period was 310 days.

The characterization of samples and results of the tests are shown in Table 13.24.

Table 13.24: Leach Test Summary (source: Lakefield Research, 2000)

Column identification	Unit	Column 1	Column 2	Column 3	Column 4	Column 5	Average	Maximum	Minimum
Head grade									
CuT	%	0.588	0.798	0.795	0.431	0.404	0.603	0.798	0.404
CuT (seq)	%	0.613	0.786	0.796	0.425	0.406	0.605	0.796	0.406
Cu (AS)	%	0.103	0.035	0.078	0.040	0.100	0.071	0.103	0.035
Cu (CN)	%	0.259	0.353	0.205	0.056	0.066	0.188	0.353	0.056
Cu(rec)	%	0.347	0.394	0.283	0.097	0.165	0.257	0.394	0.097
Mineralogy									
Chalcopyrite	%	52.40	58.80	80.00	99.70	82.80	74.70	99.70	52.40
Chalcocite	%	30.50	20.30	14.90	0.20	12.60	15.70	30.50	0.20
Bornite	%	0.01	0.00	1.50	0.01	1.49	0.60	1.50	0.00
Covellite	%	17.20	19.50	3.60	0.00	1.40	8.30	19.50	0.00
Column leach test results									
Cu recovery (*)	%	42.50	34.90	17.30	4.00	8.40	21.42	42.50	4.00
Total acid consumptions	kg/t ore	74.80	40.80	69.90	61.70	51.20	59.68	74.80	40.80
Net acid consumptions	kg/kg Cu	30.60	13.90	55.60	2021.50	259.60	476.24	2021.50	13.90

(*) Based on head-tailings copper balance.

Better recoveries were obtained with samples with higher proportions of chalcocite.

SGS reached the conclusion that the chalcopyrite samples from Vizcachitas are not amenable to conventional heap leach process using bacterially assisted ferric leaching. They also commented that secondary-enriched mineralization could be treated by this process, but further work needed to be undertaken to improve copper dissolution kinetics and final copper extraction.

13.3.3 2001 Test Work

The third set of leach tests was carried out at Little Bear Laboratories, Golden, Colorado, U.S.A. (Little Bear Laboratories, Golden, Colorado, 2001). This was completed in 1999-2000 using samples received from Lakefield Research in Santiago.

All the samples were blended into a single sample, with a head grade of 0.421% Cu occurring almost totally (97%) as chalcopyrite. The purpose of this test was to determine if thermophilic microorganisms and/or control of redox potential improved the rate and total extraction of copper from the mineral.

Small 3 kg samples were agglomerated with dilute sulphuric acid and loaded into 5 columns, 4” in diameter and 24” tall.

The columns, to be operated at 50 °C and 65 °C, were wrapped with heat tape. Temperature was controlled with a temperature probe and thermistor.

The columns operated at 35 °C were placed inside a temperature-controlled cabinet.

The initial leach solution composed of 400 ml of a modified medium (at pH 1.6 with sulphuric acid) containing 4.0 g/l of Fe^{2+} (ferrous sulphate) plus 100 ml of a starter culture composed of mixed cultures of iron and sulphur oxidizing mesophilic, moderately thermophilic and extremely thermophilic organisms.

The results of these tests are presented in Table 13.25.

Table 13.25: Bacterial Leach Test Results (source: Little Bear Laboratories, Inc., 2001)

Column	Operating temperature [°C]	Inoculation with	Aeration	Days of operation	Copper extraction [%]
1	65 range 64 - 70	Moderate + extreme thermophiles	Yes	215	52
2	50 range 49 - 53	Moderate + extreme thermophiles	Yes	215	38
3	35 range 34 - 36	Mesophiles	Argon	172	26
4	35, then to 65 after 172 days	Mesophiles, then thermophiles at 172 d	Yes	256	39
5	35 range 34 - 36	Mesophiles	No	172	14

13.4 2017 Flotation Test Work

During 2017 a number of flotation metallurgical tests were conducted with 40 drill hole samples at SGS Metallurgical Laboratories.

The main purpose of this study was to analyze the behavior of copper and molybdenum recovery at a coarser grind and with a grain size larger than established in the base case of 180 micron in the feed to the Rougher flotation circuit.

The metallurgical results showed a good recovery for a grain size with P_{80} of 240 µm, which would allow forecasting of an increase in the treatment capacity in the grinding circuit, considering the historical baseline configuration established for the treatment capacity of 44 ktpd at a P_{80} of 180 µm.

Alternatively, closed-cycle tests were also conducted confirming good recoveries, which allowed an estimate of the overall recovery and the final copper concentrate grade at the new grain size of 240 µm for the feed to the Rougher circuit.

According to this information and the closed-circuit test, overall recovery of 91% and a concentrate grade of 30% can be considered.

13.4.1 Metallurgical Tests

A total of 40 drill hole samples were sent to SGS for metallurgical tests.

The tests were as follows:

- Sample characterization
- Rougher flotation tests at coarser grain sizes
- Cleaner flotation tests
- Closed-circuit tests.

13.4.2 Chemical Characterization

The analyzed average copper and molybdenum grades showed the following:

The average copper grade was 0.48% Cu_T, slightly lower than obtained in the tests conducted in 2009 (0.51%).

The average molybdenum grade was 0.022%, higher than reported in the tests carried out in 2009.

13.4.3 Mineralogical Characterization

The mineralogical characterization of the 40 samples showed that the main copper species is chalcopyrite, which accounts for 95% of the copper sulphides and 35% of sulphide species. A significant presence of pyrite was noted, reaching 62% of the total sulphides, as shown in the following table.

Table 13.26: Mineralogical Characterization

	% Chalcopyrite ($CuFeS_2$)	% Bornite (Cu_5FeS_4)	% Covellite (CuS)	% Chalcocite (Cu_2S)	% Pyrite (FeS_2)	% Molybdenite (MoS_2)
Mean	1.29	0.038	0.003	0.018	2.30	0.045
Typical error	0.07	0.006	0.002	0.010	0.26	0.004
Median	1.28	0.026	0.000	0.002	1.97	0.043
Standard dev.	0.47	0.038	0.010	0.064	1.66	0.026
Variance	0.22	0.001	0.000	0.004	2.74	0.001
Minimum	0.62	0.002	0.000	0.000	0.16	0.007
Maximum	2.57	0.173	0.064	0.385	8.86	0.133
Counts	40	40	40	40	40	40

13.4.4 Rougher Circuit Metallurgical Tests

The Rougher Circuit Metallurgical tests helped analyze and assess the effect on the copper and molybdenum recovery when varying the feed grain size to the rougher circuit. In the 2014 PEA a P_{80} of 180 µm was used.

As shown in Table 13.27, when compared to the base case for copper of P_{80} of 180 micron there was a 2-point decrease when operating at a grain size P_{80} of 240 µm and a 1-point decrease when operating at a P_{80} of 210 µm. For molybdenum there was a 3-point decrease for both cases when compared to the P_{80} of 180 µm. The Rougher concentrate grade variation is slightly lower.

These results show that even with a coarser grain size the copper and molybdenum recovery is reasonable, which is important when considering the potential for optimization of the comminution circuit when operating at a P_{80} of 240 µm.

Table 13.27: Rougher Flotation Results

Rougher flotation results																			
Caracterizacion		Head grade		Rougher Concentrate Cu grade				Cu recovery				Mo grade				Mo recovery			
Rock type	Mineralization	Cu head	Mo head	150	180	210	240	150	180	210	240	150	180	210	240	150	180	210	240
Andesite	Primary (HG)	0,52	0,01	7,0	6,7	6,5	6,2	93,1	90,7	88,4	88,9	0,14	0,15	0,13	0,13	91,4	91,5	82,9	83,1
Breccia	Primary (HG)	0,69	0,01	8,7	8,9	9,2	8,9	97,8	97,2	96,4	95,0	0,10	0,11	0,10	0,10	89,3	89,2	88,6	87,9
Diorite	Primary (HG)	0,48	0,03	8,3	7,7	7,2	7,9	97,5	96,9	96,1	94,0	0,56	0,49	0,50	0,49	96,9	93,8	91,7	87,3
Diorite	Primary (HG)	0,57	0,01	9,3	9,5	9,3	8,7	98,3	97,7	97,3	96,8	0,11	0,11	0,10	0,10	87,6	86,6	86,5	86,7
Breccia	Primary (HG)	0,30	0,03	6,5	6,2	6,4	5,3	97,2	96,0	96,0	93,6	0,75	0,70	0,74	0,56	97,0	94,1	91,9	90,7
Diorite	Primary (HG)	0,34	0,02	9,5	8,1	10,1	8,2	97,9	97,6	94,7	95,0	0,75	0,63	0,70	0,54	96,3	96,2	90,8	90,8
Diorite	Primary (HG)	0,33	0,04	9,0	7,2	6,6	7,7	97,3	96,4	95,0	95,0	0,72	0,73	0,63	0,77	79,5	96,8	93,1	91,0
Diorite	Supergene (SG)	0,60	0,02	9,3	8,8	8,8	9,6	96,6	96,1	94,3	92,5	0,33	0,31	0,30	0,32	91,4	91,2	86,9	86,5
Diorite	Primary (HG)	0,41	0,02	7,9	8,2	7,9	8,7	97,8	96,9	95,8	94,0	0,34	0,36	0,34	0,35	94,5	94,3	94,3	87,9
Breccia	Primary (HG)	0,26	0,01	3,6	3,4	3,3	3,4	97,6	97,2	94,8	95,5	0,08	0,08	0,07	0,07	84,8	84,4	83,5	84,6
Diorite	Primary (HG)	0,49	0,03	12,8	13,9	13,4	12,5	98,3	97,4	96,8	96,8	0,71	0,70	0,69	0,65	96,2	92,0	92,1	92,1
Breccia	Primary (HG)	0,52	0,02	9,1	9,3	9,2	7,9	97,8	97,0	95,7	95,9	0,30	0,30	0,26	0,24	94,6	94,4	87,9	89,1
Tonalite	Primary (HG)	0,28	0,02	8,2	7,8	8,5	6,3	96,7	96,5	94,3	94,7	0,47	0,47	0,44	0,33	93,8	93,7	86,6	87,7
Andesite	Supergene (SG)	0,73	0,02	6,2	6,0	6,2	6,4	96,9	96,2	94,5	93,7	0,04	0,04	0,04	0,04	84,9	84,6	83,1	84,0
Breccia	Primary (HG)	0,33	0,03	7,8	7,8	7,3	7,0	97,2	95,3	94,6	94,0	0,59	0,58	0,55	0,52	95,9	91,9	81,9	89,2
Andesite	Primary (HG)	0,33	0,03	6,6	6,5	6,4	6,0	96,2	95,2	93,0	93,0	0,53	0,53	0,51	0,45	92,4	89,5	85,2	89,1
Tonalite	Primary (HG)	0,19	0,04	8,8	8,6	8,5	7,2	98,0	97,1	96,0	95,6	1,76	1,85	1,72	1,39	97,5	95,6	92,8	95,0
Diorite	Primary (HG)	0,51	0,02	10,3	10,1	10,2	9,3	97,7	97,4	96,3	95,0	0,39	0,37	0,36	0,34	85,9	82,5	81,2	79,0
Breccia	Primary (HG)	0,52	0,03	7,0	6,6	6,7	6,5	96,9	95,6	93,4	92,6	0,26	0,25	0,27	0,26	91,0	83,3	90,8	87,1
Andesite	Supergene (SG)	0,86	0,02	10,0	10,0	9,0	8,3	94,5	91,4	90,5	87,2	0,13	0,17	0,17	0,13	85,1	86,7	88,4	80,6
Diorite	Primary (HG)	0,41	0,04	9,0	8,9	9,0	8,0	97,6	96,9	95,9	95,2	0,95	0,96	0,55	0,65	95,6	95,6	53,9	79,1
Breccia	Primary (HG)	0,79	0,01	12,7	12,0	11,4	11,1	98,9	98,4	98,4	97,6	0,25	0,23	0,20	0,18	94,5	94,4	88,6	93,1
Diorite	Primary (HG)	0,58	0,02	8,8	8,4	8,6	7,2	97,9	97,0	95,7	95,3	0,21	0,20	0,20	0,16	88,6	88,4	78,3	82,7
Tonalite	Primary (HG)	0,39	0,05	9,1	9,0	8,1	7,6	98,3	98,0	97,6	96,8	1,13	1,10	0,99	0,90	98,1	96,1	94,4	96,1
Diorite	Primary (HG)	0,57	0,02	12,4	11,7	10,8	9,7	98,4	97,9	97,3	96,8	0,33	0,30	0,27	0,21	94,2	88,5	88,6	87,1
Andesite	Primary (HG)	0,36	0,01	6,7	6,0	5,8	5,1	97,9	97,3	96,4	96,3	0,07	0,07	0,06	0,06	79,7	80,5	79,4	81,6
Andesite	Supergene (SG)	0,57	0,03	10,3	10,1	10,4	9,6	98,1	97,1	95,8	93,8	0,47	0,42	0,43	0,42	90,2	92,7	86,4	80,3
Breccia	Primary (HG)	0,43	0,02	3,8	3,8	3,7	3,3	98,4	98,1	97,5	96,4	0,18	0,18	0,16	0,17	89,0	88,8	84,7	86,3
Breccia	Primary (HG)	0,59	0,07	12,1	11,7	12,2	10,0	98,5	98,2	97,7	96,9	1,53	1,31	1,31	1,01	95,1	91,9	84,9	82,7
Diorite	Primary (HG)	0,20	0,01	2,0	1,7	1,9	1,9	95,7	93,8	92,2	89,8	0,09	0,08	0,08	0,08	82,1	89,7	80,3	79,9
Breccia	Primary (HG)	0,63	0,03	16,1	16,7	15,1	12,5	99,0	98,7	98,5	98,0	0,88	0,86	0,83	0,70	95,0	94,8	95,2	95,1
Diorite	Primary (HG)	0,23	0,02	6,1	5,2	4,7	5,4	97,7	96,5	94,8	93,7	0,45	0,39	0,33	0,36	86,2	82,4	80,6	79,8
Diorite	Primary (HG)	0,41	0,01	7,5	6,8	6,7	5,7	97,9	97,4	96,6	96,3	0,13	0,14	0,10	0,10	88,9	90,1	87,5	88,6
Diorite	Primary (HG)	0,33	0,01	8,2	7,8	7,2	6,6	99,2	97,6	96,8	96,6	0,26	0,24	0,22	0,20	92,1	91,8	84,9	91,0
Tonalite	Primary (HG)	0,34	0,02	9,9	9,4	9,0	9,5	98,2	98,7	98,5	96,6	0,64	0,60	0,58	0,54	96,1	96,0	96,1	90,6
Andesite	Supergene (SG)	1,06	0,01	12,1	11,8	10,8	11,1	91,8	86,3	79,9	81,6	0,08	0,08	0,07	0,07	87,9	87,5	67,0	73,9
Andesite	Primary (HG)	0,41	0,01	6,0	5,6	4,7	5,1	96,8	95,8	94,4	93,5	0,09	0,09	0,07	0,08	76,4	87,0	87,1	86,4
Tonalite	Primary (HG)	0,26	0,02	8,0	8,6	8,4	8,8	98,6	98,5	98,9	94,4	0,52	0,58	0,53	0,56	94,7	94,8	94,5	94,1
Breccia	Primary (HG)	0,38	0,04	9,4	8,8	8,6	8,2	99,0	98,3	97,7	96,1	0,88	0,84	0,57	0,56	94,9	95,0	89,4	89,5
Diorite	Primary (HG)	0,54	0,02	11,5	11,0	11,2	9,5	98,2	97,9	97,4	96,7	0,34	0,32	0,32	0,29	94,2	94,1	88,7	89,3
Promedio		0,47	0,02	8,7	8,4	8,2	7,7	97,4	96,5	95,3	94,4	0,46	0,45	0,41	0,38	91,0	90,8	86,3	86,9
Minimo		0,19	0,01	1,97	1,70	1,86	1,86	91,83	86,34	79,86	81,60	0,04	0,04	0,04	0,04	76,39	80,50	53,89	73,90
Maximo		1,06	0,07	16,09	16,67	15,06	12,51	99,17	98,74	98,92	98,03	1,76	1,85	1,72	1,39	98,10	96,78	96,08	96,14
Desviacion standard		0,18	0,01	2,65	2,77	2,64	2,32	1,46	2,33	3,24	3,04	0,39	0,38	0,35	0,29	5,42	4,37	7,62	5,03

13.4.5 Closed-Circuit Metallurgical Tests

Flotation circuit closed-circuit tests were conducted to analyze the effect of operating at a grain size with a P_{80} of 240 µm on the overall recovery and final concentrate grade of the copper. The results are summarized in Table 13.28

Table 13.28: Closed-Circuit Tests

Locked Cycle Test (Rougher Circuit P_{80} 240 µm)						
Test	Composite Characteristic	Grade %		Recovery %		
		Cu	Mo	Cu	Mo	
TCC-14	Andesitic/Surface SG/HG composite	31.9	0.16	94.5	36.5	
TCC-15	Andesitic/Surface SG/HG composite	31.7	0.29	92.8	66.3	
TCC-16	High Fe HG composite	23.8	1.40	94.8	88.3	
TCC-17	High Mo HG composite	26.3	1.42	96.3	88.3	

The good overall copper recoveries were confirmed but with variable concentrate grades and low grade in tests 16 and 17, probably owing to the presence of pyrite.

Based on these results, an estimated overall copper recovery of 91% and copper concentrate grade of 30% are reasonable for the PEA.

The 30% concentrate grades in TCC-16 and TCC-17 composites are achievable at a lower overall recovery, especially in composites with recovery rates higher than those considered in the PEA.

13.5 Conclusions

It seems unlikely that heap leaching of Vizcachitas sulphide mineralization will be appropriate as the primary process route. The milling-flotation route for the sulphide material is a much better route, based on recoveries and likely economics.

A review of the mineralogical analysis of the samples submitted for the leach testing and a review of the location from where these samples were collected indicate that even the samples collected near to surface have a high percentage of chalcopyrite. Of all the leach samples the lowest percentage chalcopyrite is 46% in Composite 3, see Table 13.22 for further detail.

The mineralogical analysis indicates that there is not enough soluble copper in the domain that was identified in 2008 as Oxide and this zone should be renamed a Mixed mineralization zone. These findings were confirmed as part of the current inspection by the Qualified Persons.

Mineralogical analysis shows that the main copper mineral was chalcopyrite.

The amount of test work carried out is considered sufficient to support a PEA. A number of test work programmes were carried out at different times using different composites. However, much of the earlier work was with high-grade composites and thus this work is of limited relevance in relation to the current anticipated head grades.

Overall the flotation results are generally supportive of the copper and molybdenum recoveries selected for use in the economic evaluation. However, there is some variability in the results and in some tests the concentrate grades and recoveries were not satisfactory.

Although reasonable confidence can be given to the selected parameters some caution must be taken and further test work is considered to be essential for the PFS. It should be noted that the current report authors were not involved in any of the previous test work.

The results suggest that the rougher flotation is not very sensitive to the P_{80} and on this basis a primary grind P_{80} of 240 µm is proposed. This requires confirmation and further optimization in the next round of test work. The test work indicates a relatively fine regrind P_{80} is required in the range 30 - 50 µm. A value of 45 µm has been selected for this PEA. Again, further test work to confirm this important parameter will be necessary.

The test work indicates three stages of cleaning is required. In this PEA a first stage of cleaning using conventional cells is followed by secondary column flotation and this should achieve the desired concentrate grades. However, a third cleaner stage can be utilized by converting some of the scavenger flotation cells into cleaner cells. Once again, further flotation test work will be essential to confirm and optimise the cleaner circuit.

The results updated with the new metallurgical tests carried out in 2017 with 40 drill hole samples show that establishing a flotation grind with a P_{80} of 240 µm is reasonable.

The tests carried out with the 40 drill hole samples, the closed-circuit tests carried out under the new grain size conditions with a P_{80} of 240 µm in the feed to the Rougher circuit and a P_{80} of 45 µm in the regrinding circuit, confirms good recoveries. Based on these results it is a reasonable estimate to use the following process parameters:

- Optimization of the plant treatment capacity by operating at a coarser grain size corresponding to a P_{80} of 240 µm.
- An overall recovery of 91% Copper and 75% Molybdenum, with a copper concentrate grade estimated at 30%.
- A copper recovery of 95% in the rougher circuit. and 96% in the cleaner circuit
- A molybdenum recovery of 84% in the collective circuit and 89% in the selective circuit

14. MINERAL RESOURCE ESTIMATE

14.1 Geological 3-D Model, and Domains

The 3D geological model for Vizcachitas was constructed using Leapfrog software. From this model different lithological and mineral zone solids (overburden, leached, supergene, and hypogene) were developed. The different codes for each unit are shown in Table 14.1.

Table 14.1: Lithological Model Codes

Lithology Code	Description	Description
102	AND	Andesite
117	BFM	Phreatomagmatic Breccia
113	BXH	Hydrothermal Breccia
112	BXH-1	Hydrothermal Breccia One
118	BXI	Igneous Breccia
119	DEP	Early Diorite Porphyry
123	DEP-1	Early Diorite Porphyry One
104	DIO	Diorite
120	DIOF	Fine Diorite
121	DIOM	Medium Grain Diorite
122	DIOQ	Quartz Diorite
105	GRD	Granodiorite
101	GRV	Gravel
103	PDAC	Dacite Porphyry
107	PDIO	Diorite Porphyry
109	TON	Tonalite

Table 14.2: Mineral Zone Model Codes

Mineralized Zone Code	Description
301	Overburden
302	Leached
304	Supergene
305	Hypogene

14.1.1 Available Data

The database used for estimating resources covers the campaigns from 1993 to 2017 comprising a total of 165 drill holes. (Table 14.3).

Of these 165 drill holes, four drill holes were identified that did not contain information on grades, as shown in Table 14.4. Drill hole V2015-06b was drilled within a barren dacite dyke and was not sampled. The other drill holes did not successfully drill through the gravel cover. Finally, a total of 161 drill holes were used.

Table 14.3: Drill Hole Campaigns

Campaigns	No. of Drill Holes	Metres
1993	6	1,953
1995-98	61	15,815
2007	39	11,256
2008	40	11,360
2015	8	3,610
2017	11	8,262
Total	**165**	**52,256**

Table 14.4: Drill Holes without Copper Grade Information

Hole Number	East	North	Elev	Length	DH_Year
V-14A	365,967	6,413,327	1,980	20.43	1995-98
V2015-06b	366,041	6,413,855	2,104	67	2015
V2015-07	366,239	6,413,137	2,022	52	2015
V2017-01	365,778	6,413,544	2,003	100	2017

Table 14.5 details of the metres drilled for each lithology and Table 14.6 details the metres with mineralized zone information.

Table 14.5: Metres Drilled with Lithological Information

Lithology Code	Description	Description	Drilled Meters
102	AND	Andesite	16,136
117	BFM	Phreatomagmatic Breccia	1,269
113	BXH	Hydrothermal Breccia	2,259
112	BXH-1	Hydrothermal Breccia One	2,956
118	BXI	Igneous Breccia	1,340
119	DEP	Early Diorite Porphyry	4,724
123	DEP-1	Early Diorite Porphyry One	3,553
104	DIO	Diorite	22
120	DIOF	Fine Diorite	1,582
121	DIOM	Medium Grain Diorite	154
122	DIOQ	Quartz Diorite	319
105	GRD	Granodiorite	3,418
101	GRV	Gravel	5,561
103	PDAC	Dacite Porphyry	1,651
107	PDIO	Diorite Porphyry	370
109	TON	Tonalite	6,942
Grand Total			52,256

Table 14.6: Metres Drilled with Information by Mineral Zone

Mineralized Zone Code	Description	Drilled Meters
301	Overburden	5,683
302	Leached	2,935
304	Supergene	7,970
305	Hypogene	35,644
Grand Total		52,233

14.1.2 Geological 3-D Model Verification

To verify the information from the geological logging and from the 3D models (lithology and mineral zone), the drill holes were assigned codes from the solids of the 3D geological model. The correlation between the lithological logging and the 3D model was over 95% for all units, except for unit 103 Dacite Porphyry Dykes, which corresponds to 3% of the total drilled metres.

Table 14.7: Logging v/s Geological Model Matching Percent

Lithology Code		Solids													
		102	103	105	107	109	112	113	117	118	119	120	121	122	123
102	AND	98%													
103	PDAC		81%												
105	GRD			99%											
107	PDIO				95%										
109	TON					99%									
112	BXH-1						99%								
113	BXH							96%							
117	BFM								100%						
118	BXI									99%					
119	DEP										98%				
120	DIOF											99%			
121	DIOM												95%		
122	DIOQ													96%	
123	DEP-1														100%

The same procedure was used to review the mineralized zone model, obtaining the following results.

Table 14.8: Mineral Zone Matching Percentage

Mineralized Zone		Solids			
		301	302	304	305
Overburden	301	95%	0	0	0
Leached	302	0	98%	0	0
Supergene	304	0	0	99%	0
Hypogene	305	0	0	0	100%

14.2 Mineral Resource Estimate

14.2.1 Method for Estimate and Tools

The mineral resources of the Vizcachitas Project have been estimated by using the Ordinary Kriging method. The Minesight Version 11.50-1 software was used for this purpose, supported by the MSDA tool for statistical analysis of the database, variography, and copper, molybdenum, and silver model statistics.

14.2.2 Specific Gravity

The density included in the block model was assigned for each copper estimate unit, with values calculated as an average in each estimate unit (UE). Prior to this step a statistical analysis was performed for each unit to identify the outliers to avoid influencing in the values calculated.

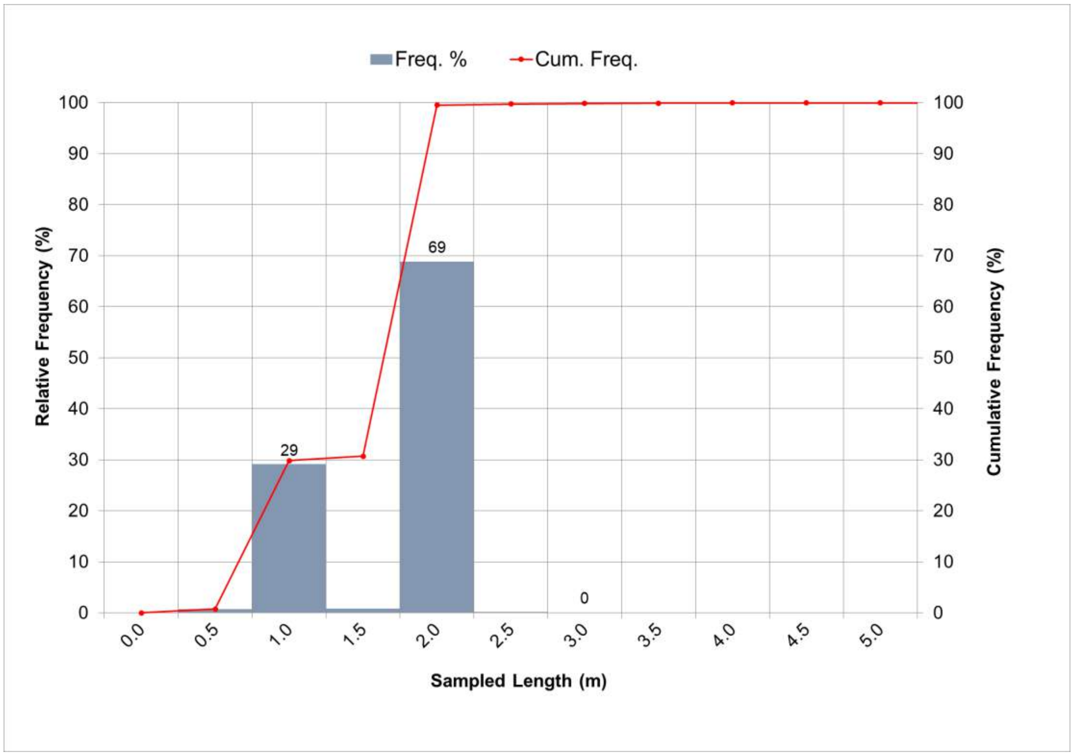
Table 14.9: Density by UE

Density	
UE	gr/cm³
1	2.5
2	2.54
3	2.58
4	2.62
5	2.56
6	2.62
7	2.73
8	2.68
9	2.69
10	2.73
11	2.56

14.2.3 Composites Study

Samples were composited at 2 m intervals, considering that 69% of samples had this length, and 29% of samples were 1 m long, as shown in Chart 14-1.

Chart 14-1: Sample Length



14.2.4 Assay Statistics

The following tables show the statistical analysis for each lithological unit and mineral zone for copper, molybdenum and silver.

14.2.4.1 Copper

Table 14.10: Copper Grade Statistics for Each Lithology

Lithology Code	Description	No. Samples	Length (m)	Minimum (%)	Maximum (%)	Mean (%)	Standard Deviation	Variance	Co. of Variation
101	GRV	111	166	0.004	0.678	0.085	0.119	0.014	1.398
102	AND	10,052	16,201	0.001	2.460	0.330	0.226	0.051	0.685
103	PDAC	1,045	1,460	0.000	0.953	0.091	0.136	0.019	1.491
104	DIO	11	22	0.140	0.603	0.312	0.127	0.016	0.409
105	GRD	2,372	3,413	0.002	1.167	0.178	0.148	0.022	0.832
107	PDIO	212	370	0.007	2.490	0.314	0.287	0.082	0.913
109	TON	4,246	6,927	0.005	1.684	0.281	0.173	0.030	0.616
112	BXH-1	1,823	2,991	0.003	2.530	0.533	0.274	0.075	0.515
113	BXH	1,408	2,255	0.002	2.261	0.308	0.226	0.051	0.734
117	BFM	798	928	0.001	3.000	0.103	0.157	0.025	1.529
118	BXI	680	1,328	0.014	1.742	0.389	0.208	0.043	0.535
119	DEP	2,730	4,083	0.003	1.315	0.259	0.171	0.029	0.661
120	DIOF	1,161	1,585	0.010	2.421	0.154	0.154	0.024	1.004
121	DIOM	83	155	0.025	2.080	0.461	0.271	0.074	0.589
122	DIOQ	285	320	0.086	1.710	0.415	0.253	0.064	0.610
123	DEP-1	2,375	3,496	0.005	3.080	0.544	0.222	0.049	0.408
Total		29,392	45,700	0.000	3.080	0.317	0.236	0.056	0.745

Table 14.11: Copper Grade Statistics for Each Mineralized Zone

Mineralized Zone Code	Description	N° Samples	Length (m)	Minimum (%)	Maximum (%)	Mean (%)	Standard Deviation	Variance	Co. of Variation
301	Overburden	61	115	0.004	0.364	0.060	0.071	0.005	1.187
302	Leached	2,019	2,801	0.002	2.017	0.128	0.129	0.017	1.011
304	Supergene	5,341	7,961	0.006	2.747	0.499	0.286	0.082	0.574
305	Hypogene	22,282	35,160	0.000	3.080	0.289	0.205	0.042	0.707
Total		29,703	46,037	0.000	3.080	0.315	0.236	0.056	0.750

14.2.4.2 Molybdenum

Table 14.12: Molybdenum Grade Statistics for Each Lithology

Lithology Code	Description	No. Samples	Length (m)	Minimum (ppm)	Maximum (ppm)	Mean (ppm)	Standard Deviation	Variance	Variation Code
101	GRV	111	166	1.00	749	67.32	112	12,455	1.66
102	AND	10,052	16,201	1.00	6,720	94.98	170	28,800	1.79
103	PDAC	1,045	1,460	0.52	943	39.89	92	8,452	2.30
104	DIO	11	22	50.60	800	263.45	203	41,350	0.77
105	GRD	2,372	3,413	1.00	1,860	50.80	100	9,912	1.96
107	PDIO	212	370	5.00	896	133.32	142	20,065	1.06
109	TON	4,246	6,927	1.00	5,850	129.29	215	46,174	1.66
112	BXH-1	1,823	2,991	2.00	5,550	202.76	329	108,221	1.62
113	BXH	1,408	2,255	0.60	2,640	169.67	225	50,626	1.33
117	BFM	798	928	1.00	2,580	38.92	177	31,370	4.55
118	BXI	680	1,328	5.00	1,410	129.14	144	20,710	1.11
119	DEP	2,730	4,083	0.96	3,580	131.70	227	51,708	1.73
120	DIOF	1,161	1,585	1.00	820	40.41	77	5,999	1.92
121	DIOM	83	155	10.00	1,160	205.53	229	52,224	1.11
122	DIOQ	285	320	5.00	1,740	92.35	139	19,311	1.50
123	DEP-1	2,375	3,496	0.88	3,080	137.57	192	36,750	1.39
Total		29,392	45,700	0.52	6,720	111.01	197	38,947	1.78

Table 14.13: Molybdenum Grade Statistics for Each Mineral Zone

Mineralization Zone Code	Description	No. Samples	Length (m)	Minimum (ppm)	Maximum (ppm)	Mean (ppm)	Standard Deviation	Variance	Variation Code
302	Leached	2,019	2,801	0.96	900	57.93	87	7,602	1.51
304	Supergene	5,341	7,961	0.60	5,850	106.65	162	26,277	1.52
305	Hypogene	22,282	35,160	0.52	6,720	115.35	209	43,867	1.82
Total		29,703	46,037	0.52	6,720	110.34	197	38,747	1.78

14.2.4.3 Silver

Table 14.14: Silver Grade Statistics for Each Lithology

Lithology Code	Description	No. Samples	Length (m)	Minimum (ppm)	Maximum (ppm)	Mean (ppm)	Standard Deviation	Variance	Variation Code
101	GRV	96	145	0.10	6.38	0.739	0.911	0.830	1.232
102	AND	6,352	9,691	0.02	53.90	1.025	1.115	1.243	1.088
103	PDAC	507	735	0.01	60.30	0.596	3.150	9.919	5.282
104	DIO	11	22	0.37	1.37	0.714	0.276	0.076	0.387
105	GRD	1,503	1,964	0.04	6.21	0.687	0.704	0.496	1.026
107	PDIO	149	251	0.13	7.05	1.312	0.956	0.914	0.729
109	TON	2,588	3,933	0.05	100.00	1.034	2.418	5.848	2.340
112	BXH-1	1,178	1,834	0.01	6.40	1.495	0.875	0.765	0.585
113	BXH	1,168	1,933	0.03	100.00	0.954	3.284	10.788	3.441
117	BFM	483	573	0.01	79.50	0.539	4.112	16.906	7.632
118	BXI	508	986	0.12	5.11	1.276	0.685	0.469	0.537
119	DEP	2,347	3,465	0.03	5.36	0.698	0.417	0.174	0.597
120	DIOF	616	823	0.08	8.45	0.770	0.778	0.606	1.010
121	DIOM	56	101	0.40	2.56	0.932	0.448	0.201	0.481
122	DIOQ	242	272	0.17	8.51	1.030	0.728	0.530	0.706
123	DEP-1	1,522	2,393	0.04	7.50	1.301	0.579	0.335	0.445
Total		19,326	29,122	0.01	100.00	0.994	1.655	2.738	1.665

Table 14.15: Silver Grade Statistics for Each Mineral Zone

Mineralization Zone Code	Description	No. Samples	Length (m)	Minimum (ppm)	Maximum (ppm)	Mean (ppm)	Standard Deviation	Variance	Variation Code
301	Overburden	42	83	0.10	6.38	0.909	1.055	1.114	1.161
302	Leached	1,497	2,003	0.02	100	0.946	3.732	13.927	3.946
304	Supergene	3,324	4,682	0.01	17.45	1.381	0.996	0.993	0.722
305	Hypogene	14,759	22,674	0.01	100	0.914	1.434	2.057	1.569
Total		19,622	29,441	0.01	100	0.991	1.650	2.722	1.665

14.2.5 Exploratory Data Analysis

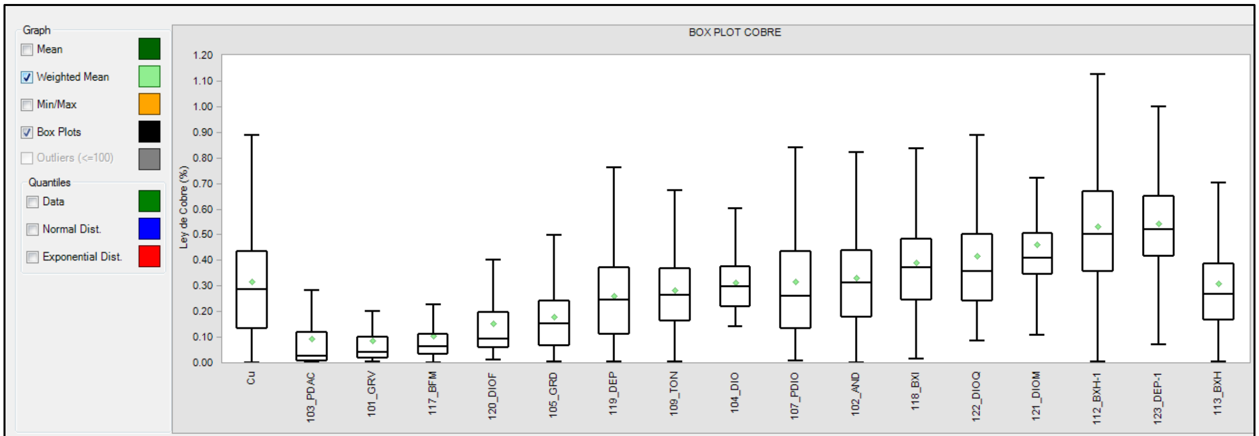
The exploratory data analysis was carried out by using different tools, such as box plots, log-probabilistic plots and histograms.

With this analysis, plus geological knowledge of the mineral deposit genesis, the estimate units (UE) were defined for each element estimated in the resource model.

14.2.5.1 Copper

The statistical analysis of copper was carried out for each lithology and mineral zone to identify the presence of similar statistical behavior between some populations.

Figure 14.1: Copper Grade Sample Box Plot by Lithology



	Cu	103_PD...	101_GRV	117_BFM	120_DIOF	105_GRD	119_DEP	109_TON	104_DIO	107_PDIO	102_AND	118_BXI	122_DIOQ	121_DI...	112_BX...	123_DE...	113_BXH
Num Samples	29703	1045	111	798	1161	2372	2730	4246	11	212	10052	680	285	83	1823	2375	1408
Num Missing Samples	390	10	334	34	0	0	2	1	0	0	3	1	0	0	1	0	0
Min	0.000	0.000	0.004	0.001	0.010	0.002	0.003	0.005	0.140	0.007	0.001	0.014	0.086	0.025	0.003	0.005	0.002
Max	3.080	0.953	0.678	3.000	2.421	1.167	1.315	1.684	0.603	2.490	2.460	1.742	1.710	2.080	2.530	3.080	2.261
Mean	0.3117	0.0860	0.0756	0.1021	0.1582	0.1697	0.2574	0.2733	0.3117	0.3102	0.3355	0.3900	0.4324	0.4343	0.5397	0.5587	0.3049
First quartile	0.1340	0.0080	0.0200	0.0320	0.0610	0.0670	0.1100	0.1630	0.2200	0.1340	0.1800	0.2460	0.2400	0.3470	0.3570	0.4160	0.1690
Median	0.2860	0.0250	0.0410	0.0620	0.0940	0.1520	0.2470	0.2640	0.2990	0.2600	0.3110	0.3700	0.3560	0.4100	0.5020	0.5210	0.2670
Third quartile	0.4360	0.1180	0.1000	0.1100	0.1970	0.2400	0.3710	0.3670	0.3760	0.4360	0.4370	0.4820	0.5010	0.5070	0.6680	0.6500	0.3850
SD	0.2459	0.1315	0.1025	0.1612	0.1619	0.1550	0.1742	0.1756	0.1336	0.3078	0.2374	0.2097	0.2585	0.2787	0.2843	0.2340	0.2362

Figure 14.2: Copper Grade Probability Curves by Lithology

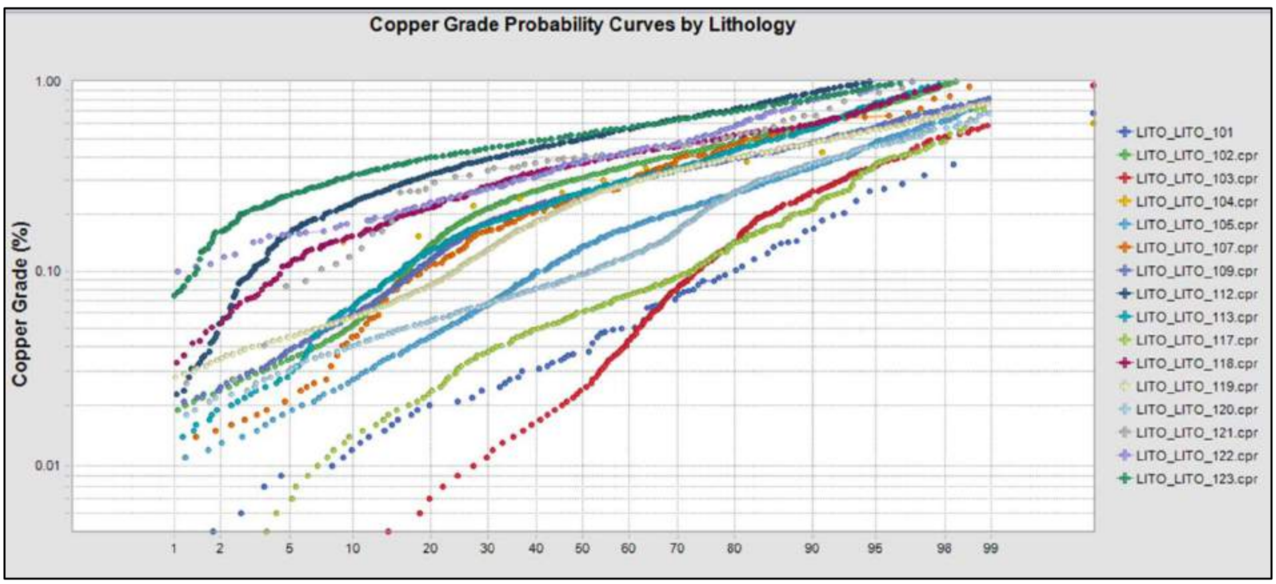
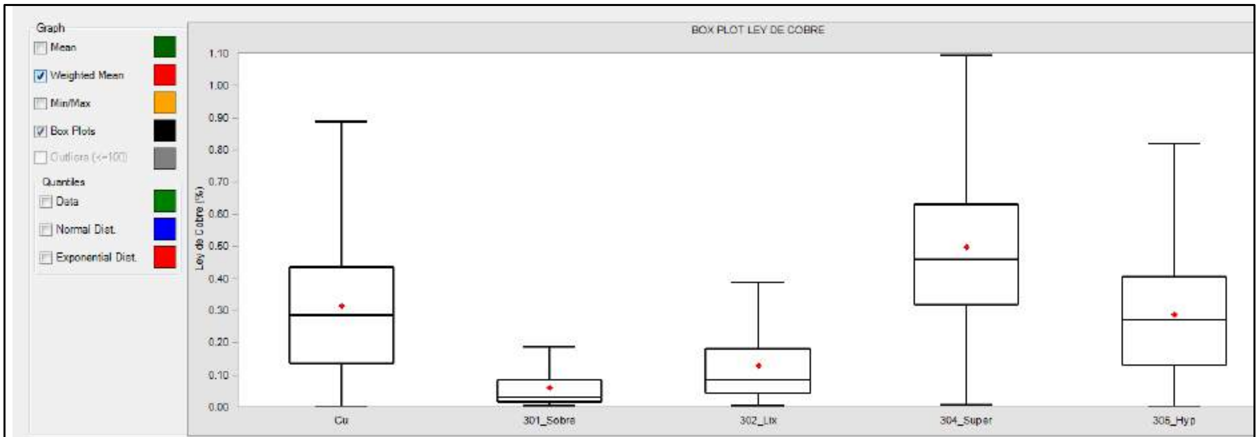


Figure 14.3: Copper Grade Box Plot by Mineral Zone



	Cu	301_Sobre	302_Lix	304_Super	305_Hyp
Num Samples	29703	61	2019	5341	22282
Num Missing Samples	390	335	9	3	43
Min	0.000	0.004	0.002	0.006	0.000
Max	3.080	0.364	2.017	2.747	3.080
Mean	0.3117	0.0625	0.1271	0.5055	0.2827
First quartile	0.1340	0.0150	0.0410	0.3170	0.1280
Median	0.2860	0.0310	0.0850	0.4600	0.2690
Third quartile	0.4360	0.0850	0.1810	0.6290	0.4040
SD	0.2459	0.0703	0.1311	0.3017	0.2101

Table 14.16 shows the estimate units defined for the copper grade estimate. The leached zone and the supergene zone include all lithologies, there are different combinations of lithologies for the hypogene zone.

Table 14.16: Copper Estimate Units

<table>
<tr><th colspan="6">UE</th></tr>
<tr><th rowspan="2">Lithology</th><th rowspan="2">Description</th><th colspan="4">Ore Zone</th></tr>
<tr><th>301
Overburden</th><th>302
Leached</th><th>304
Supergene</th><th>305
Hypogene</th></tr>
<tr><td>101</td><td>GRV</td><td rowspan="16"></td><td rowspan="16">1</td><td rowspan="16">2</td><td></td></tr>
<tr><td>120</td><td>DIOF</td><td rowspan="4">3</td></tr>
<tr><td>104</td><td>DIO</td></tr>
<tr><td>105</td><td>GRD</td></tr>
<tr><td>107</td><td>PDIO</td></tr>
<tr><td>102</td><td>AND</td><td>4</td></tr>
<tr><td>109</td><td>TON</td><td>5</td></tr>
<tr><td>118</td><td>BXI</td><td rowspan="3">6</td></tr>
<tr><td>121</td><td>DIOM</td></tr>
<tr><td>122</td><td>DIOQ</td></tr>
<tr><td>119</td><td>DEP</td><td>7</td></tr>
<tr><td>123</td><td>DEP-1</td><td>8</td></tr>
<tr><td>112</td><td>BXH-1</td><td>9</td></tr>
<tr><td>113</td><td>BXH</td><td>10</td></tr>
<tr><td>103</td><td>PDAC</td><td rowspan="2">11</td></tr>
<tr><td>117</td><td>BFM</td></tr>
</table>

Table 14.17: Composites Copper Statistics by UE

UE	N° Samples	Length (m)	Minimum (%)	Maximum (%)	Mean (%)	Standard Deviation	Variance	Co. of Variation
1	1,361	2,644	0.002	1.294	0.133	0.122	0.015	0.916
2	3,987	7,906	0.007	2.490	0.501	0.272	0.074	0.543
3	2,126	4,230	0.006	0.920	0.148	0.116	0.014	0.786
4	5,776	11,508	0.001	1.630	0.290	0.175	0.031	0.603
5	2,808	5,590	0.010	1.098	0.256	0.143	0.020	0.557
6	695	1,385	0.030	2.080	0.375	0.172	0.029	0.457
7	1,734	3,449	0.005	1.025	0.266	0.157	0.025	0.589
8	1,276	2,543	0.005	1.732	0.527	0.167	0.028	0.318
9	1,229	2,450	0.003	2.530	0.505	0.241	0.058	0.478
10	959	1,908	0.002	1.249	0.299	0.186	0.035	0.623
11	994	1,972	0.000	1.597	0.091	0.127	0.016	1.399
12	65	115	0.004	0.364	0.060	0.071	0.005	1.183
Total	23,010	45,700	0.000	2.530	0.317	0.230	0.053	0.725

Contact Analysis

A contact analysis was performed by estimate unit to verify the presence of hard or gradational contacts among UEs.

Chart 14-2: UE2 v/s UE1 Contact Analysis

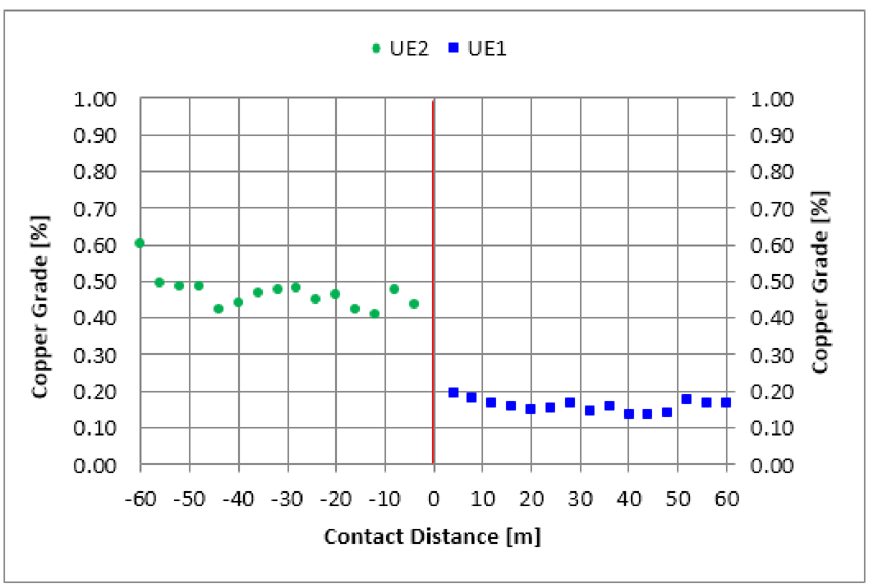


Chart 14-3: UE7 v/s UE2 Contact Analysis

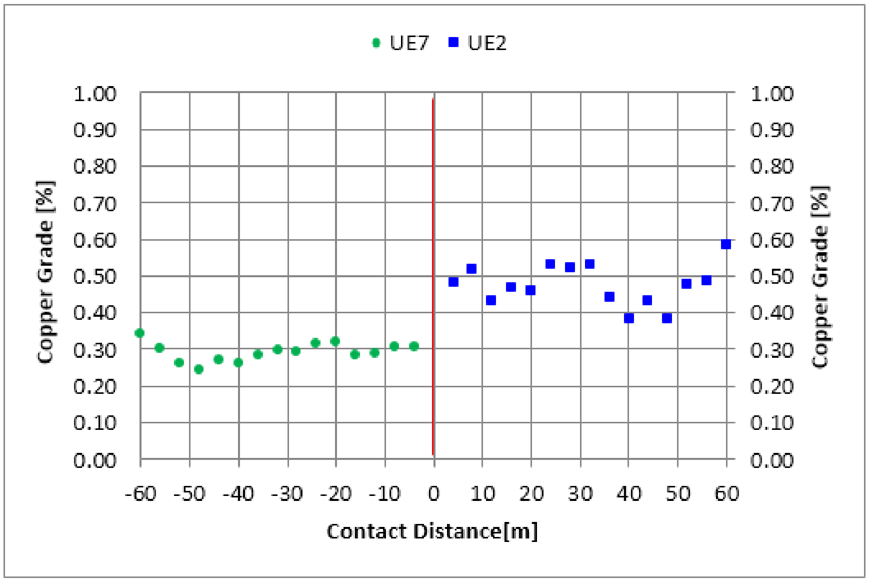


Chart 14-4: UE8 v/s UE2 Contact Analysis

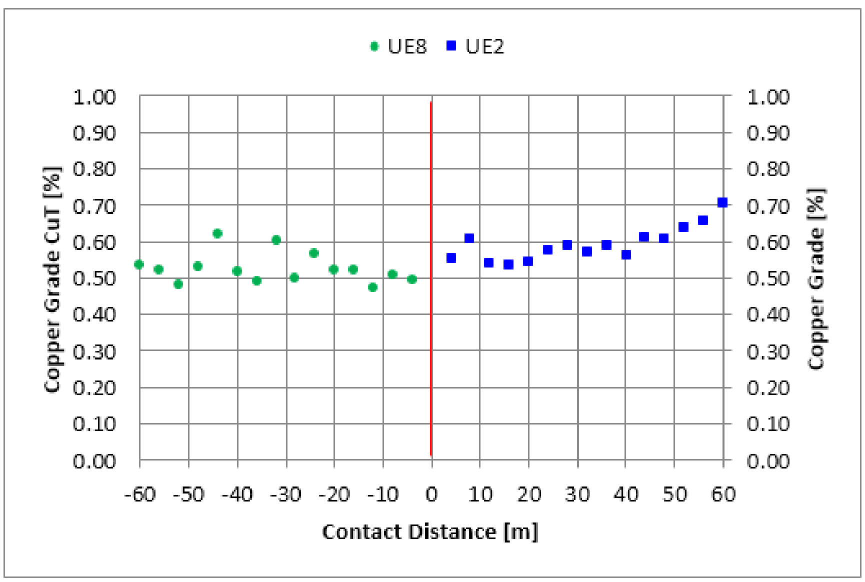


Chart 14-5: UE9 v/s UE2 Contact Analysis

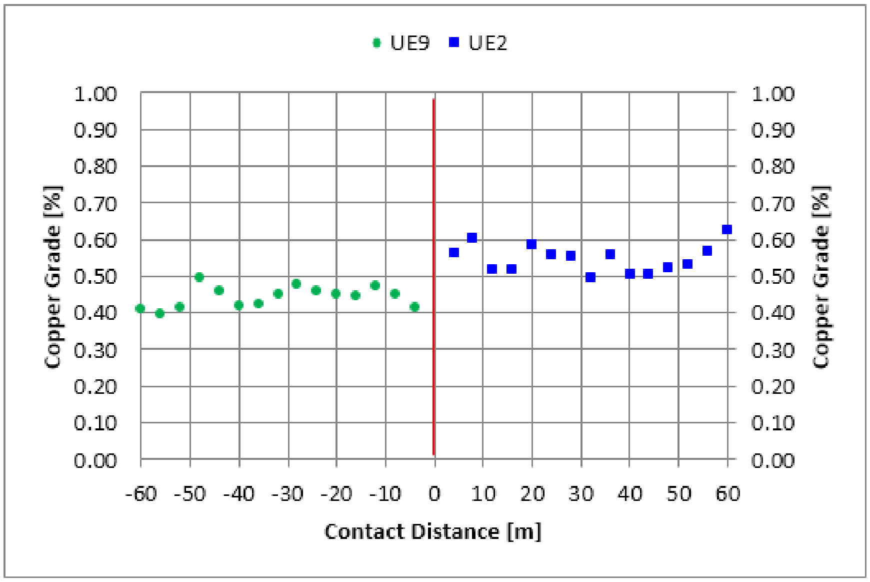


Chart 14-6: UE4 v/s UE2 Contact Analysis

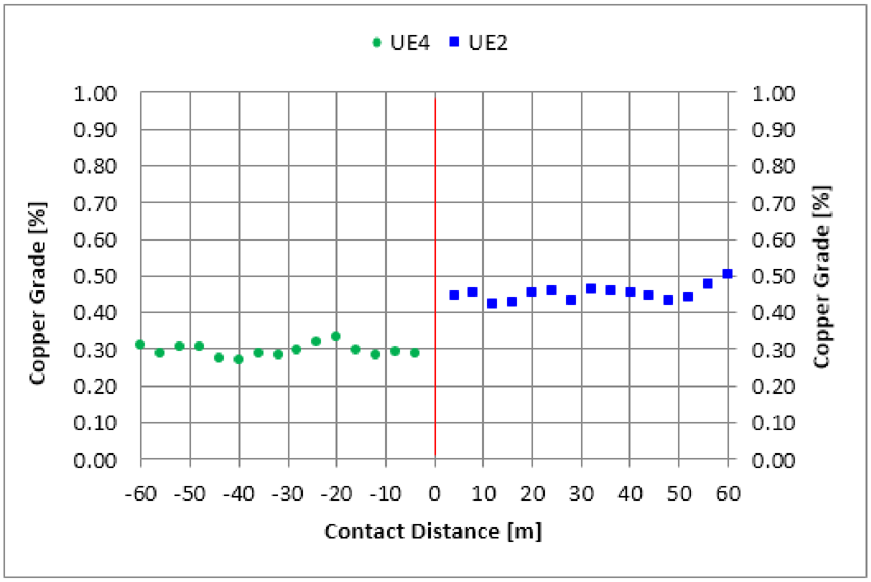


Chart 14-7: UE5 v/s UE2 Contact Analysis

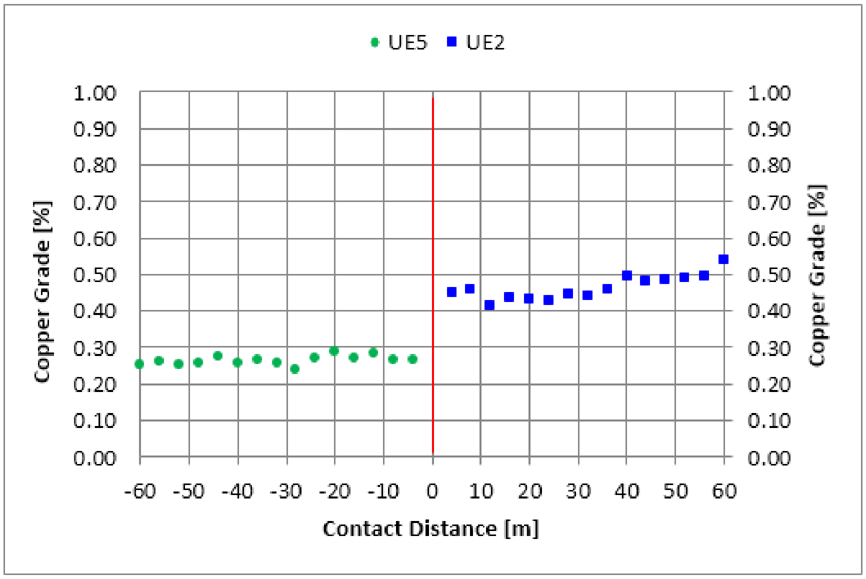


Chart 14-8: UE3 v/s UE2 Contact Analysis

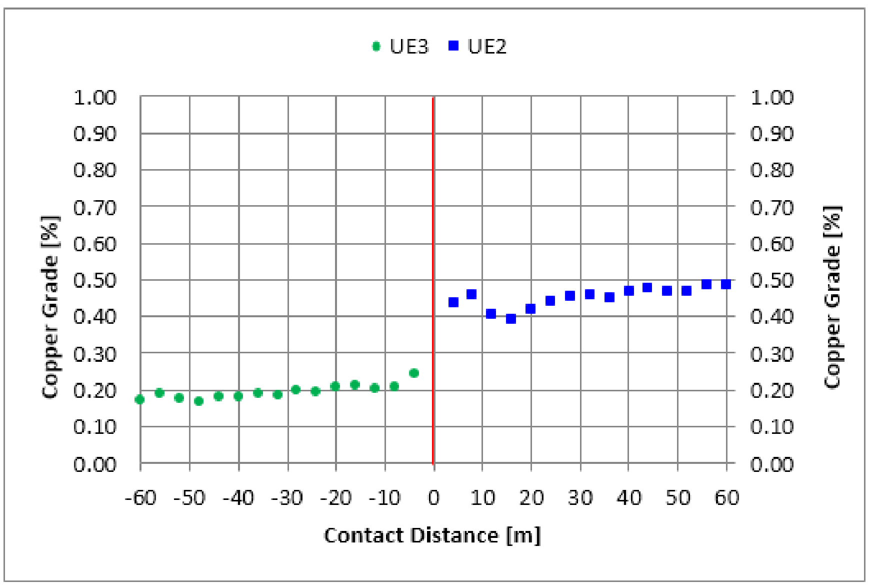


Chart 14-9: UE6 v/s UE2 Contact Analysis

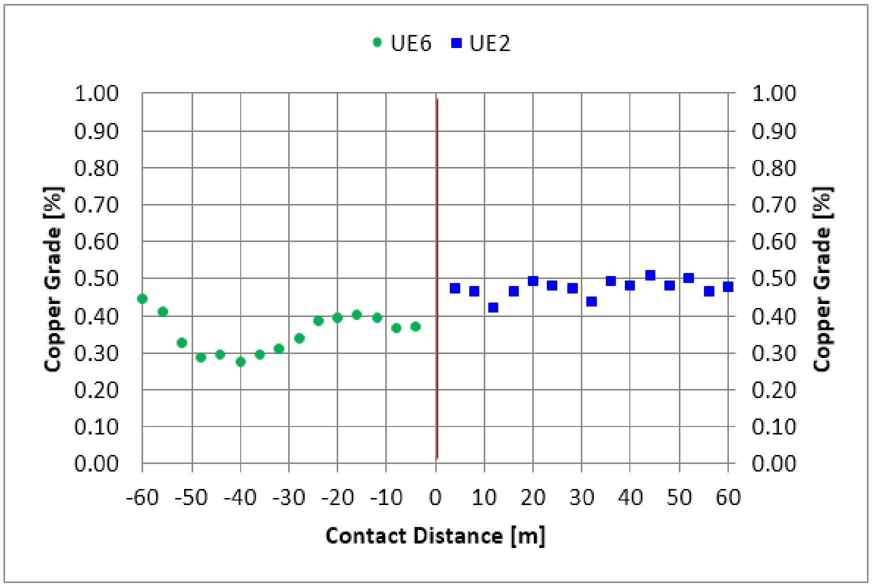


Chart 14-10: UE11 v/s UE10 Contact Analysis

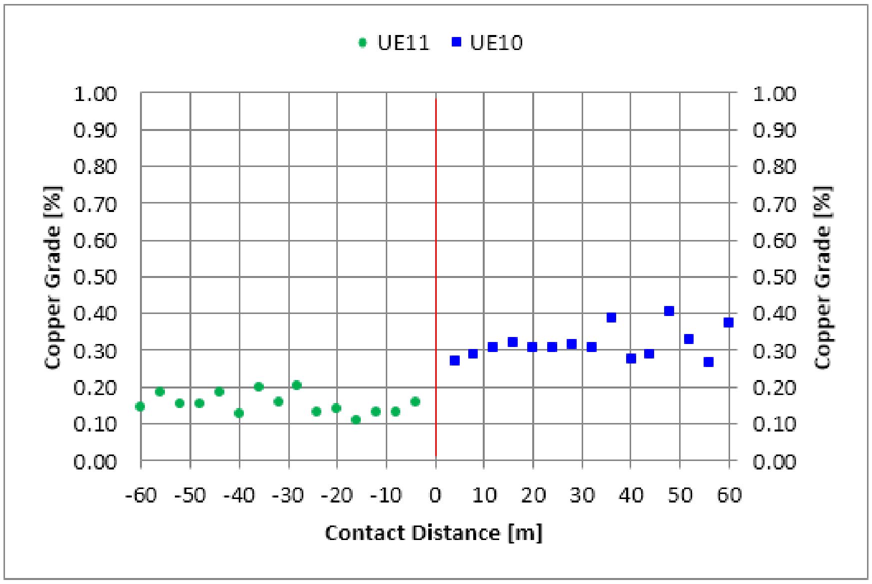
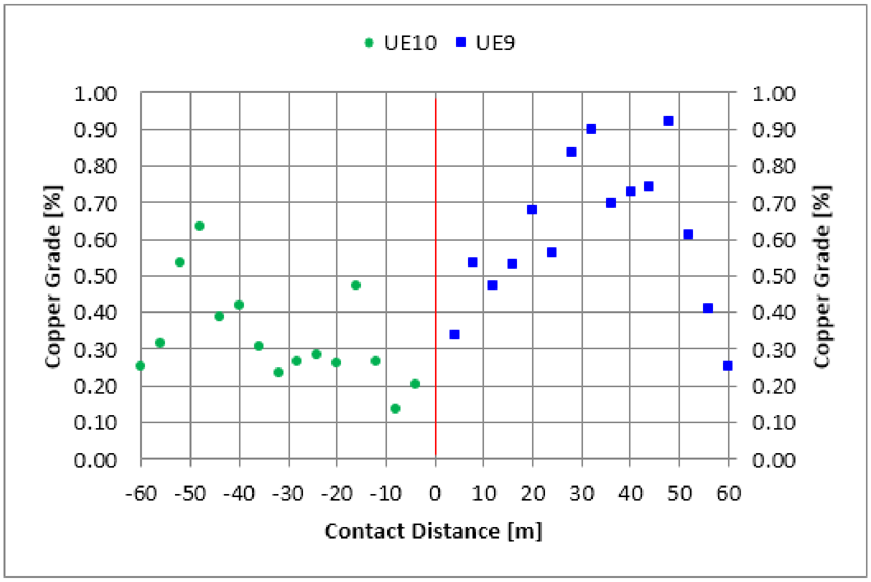


Chart 14-11: UE10 v/s UE9 Contact Analysis



14.2.5.2 Molybdenum

Based on a statistical and geological analysis, molybdenum estimate units were defined. Table 14.18 shows the UEs defined by lithology and mineral zone, and Table 14.19 illustrates the statistics by molybdenum estimate unit for each UE.

Table 14.18: Molybdenum Estimate Units

UE					
Lithology	Description	Mineral Zone			
		301 Overburden	302 Leached	304 Supergene	305 Hypogene
103	PDAC	2			
105	GRD				
107	PDIO				
102	AND	1			
104	DIO				
109	TON				
112	BXH-1				
113	BXH				
117	BFM				
118	BXI				
119	DEP				
120	DIOF				
121	DIOM				
122	DIOQ				
123	DEP-1				

Table 14.19: Composites Statistics for Molybdenum by UE

<table>
<tr><th rowspan="2">UE</th><th colspan="8">Total</th></tr>
<tr><th>N° Samples</th><th>Length (m)</th><th>Minimum (ppm)</th><th>Maximum (ppm)</th><th>Mean (ppm)</th><th>Standard Deviation</th><th>Variance</th><th>Co. of Variation</th></tr>
<tr><td>1</td><td>20,037</td><td>39,880</td><td>0.60</td><td>5,550</td><td>120.39</td><td>192.76</td><td>37,155</td><td>1.601</td></tr>
<tr><td>2</td><td>2,934</td><td>5,820</td><td>0.52</td><td>2,580</td><td>46.69</td><td>111.92</td><td>12,526</td><td>2.397</td></tr>
<tr><td>Total</td><td>22,971</td><td>45,700</td><td>0.52</td><td>5,550</td><td>111.01</td><td>186.07</td><td>34,622</td><td>1.676</td></tr>
</table>

14.2.5.3 Silver

Based on a statistical and geological analysis, silver estimate units were defined. Table 14.20 shows the estimate units defined for silver.

Table 14.20: Silver Estimate Unit

<table>
<tr><th colspan="6">UE</th></tr>
<tr><th rowspan="2">Lithology</th><th rowspan="2">Description</th><th colspan="4">Mineral Zone</th></tr>
<tr><th>301 Overburden</th><th>302 Leached</th><th>304 Supergene</th><th>305 Hypogene</th></tr>
<tr><td>101</td><td>GRV</td><td rowspan="16"></td><td rowspan="16">1</td><td rowspan="16">2</td><td></td></tr>
<tr><td>120</td><td>DIOF</td><td rowspan="6">4</td></tr>
<tr><td>104</td><td>DIO</td></tr>
<tr><td>105</td><td>GRD</td></tr>
<tr><td>107</td><td>PDIO</td></tr>
<tr><td>102</td><td>AND</td></tr>
<tr><td>109</td><td>TON</td></tr>
<tr><td>118</td><td>BXI</td><td rowspan="7">3</td></tr>
<tr><td>121</td><td>DIOM</td></tr>
<tr><td>122</td><td>DIOQ</td></tr>
<tr><td>119</td><td>DEP</td></tr>
<tr><td>123</td><td>DEP-1</td></tr>
<tr><td>112</td><td>BXH-1</td></tr>
<tr><td>113</td><td>BXH</td></tr>
<tr><td>103</td><td>PDAC</td><td rowspan="2">5</td></tr>
<tr><td>117</td><td>BFM</td></tr>
</table>

14.2.6 High-Grade Capping

From the review of log-probabilistic charts, box plots and histograms by estimate unit, outliers were identified both for high and low grades.

Chart 14.1 - Chart 14.9 show the probability curves used to identify the outliers for copper grades.

Chart 14.1: UE2

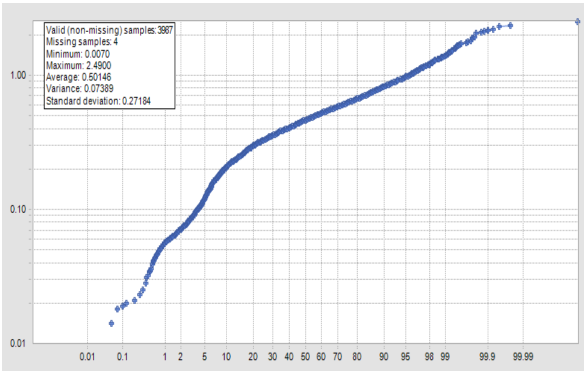


Chart 14.2: UE3

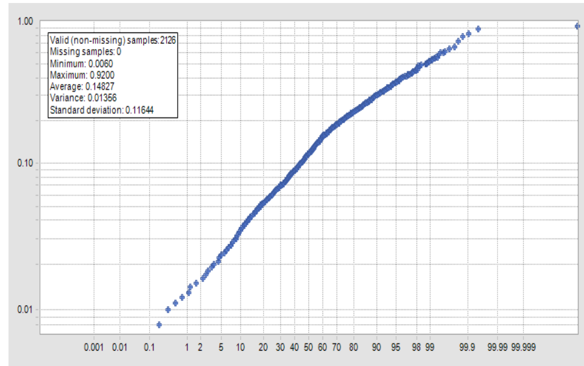


Chart 14.3: UE4

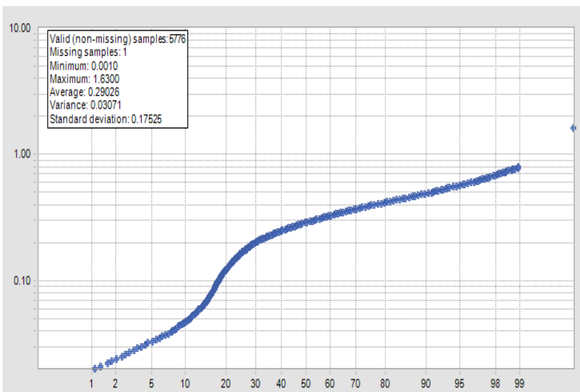


Chart 14.4: UE5

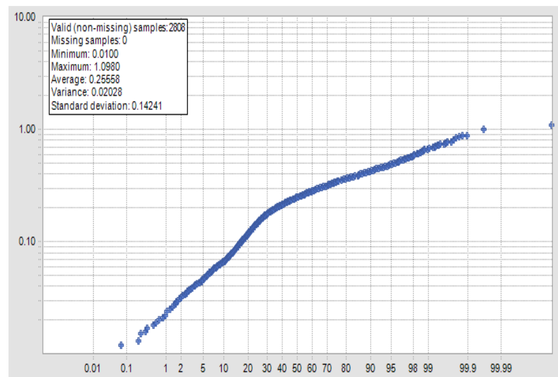


Chart 14.5: UE6

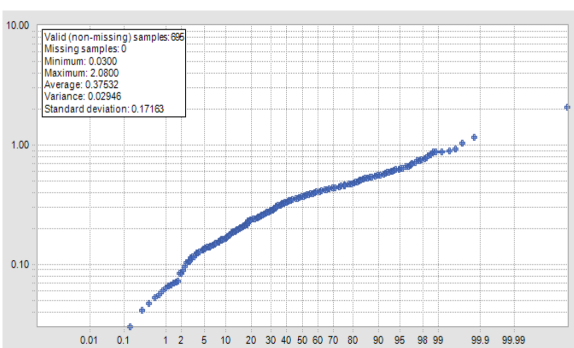


Chart 14.6: UE7

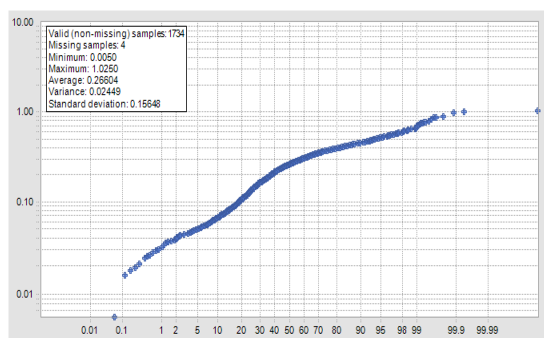


Chart 14.7: UE8	Chart 14.8: UE9

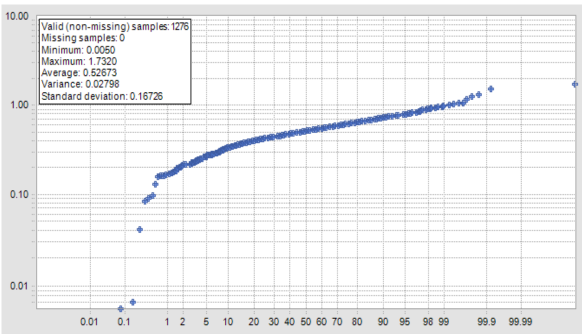
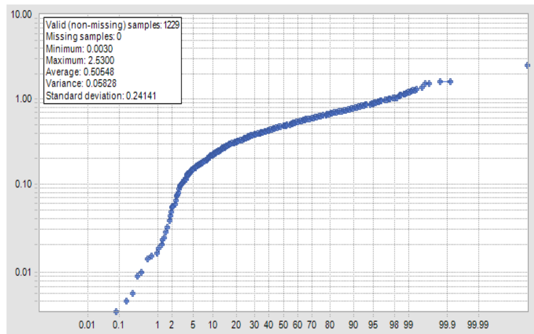
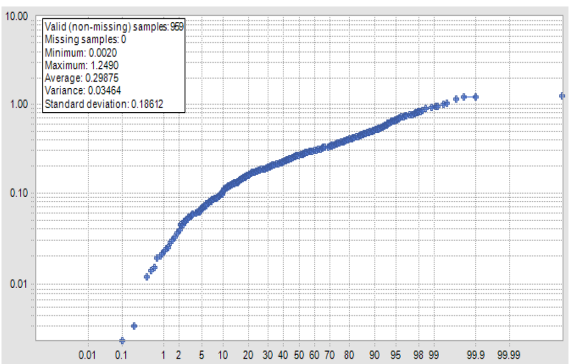

	

Chart 14.9: UE10	
	

As a result of this review, the following cut-off values (outliers) were obtained, which in most cases account for 1.1% of the population.

Table 14.21: Copper Grade Outliers by UE

EU	Minimum Cu (%)	Maximum Cu (%)
1	0.005	0.744
2	0.030	1.750
3	0.020	0.639
4	0.070	1.250
5	0.020	0.890
6	0.084	0.869
7	0.040	0.670
8	0.160	1.060
9	0.020	1.300
10	0.030	0.950

For molybdenum, UE1MO and UE2MO charts were used for the two estimate units.

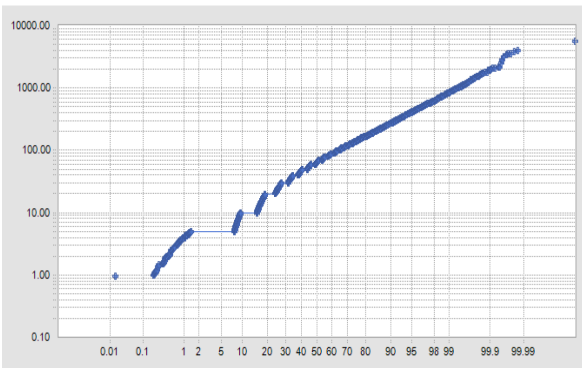
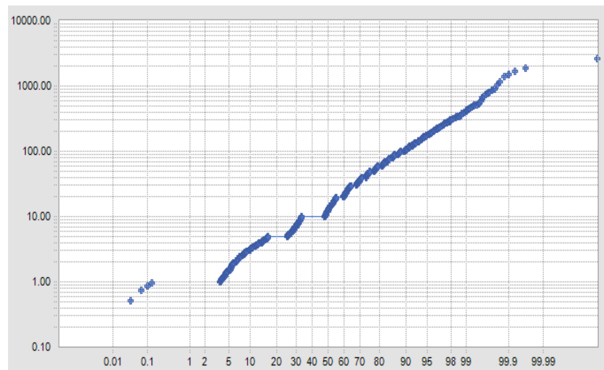
Chart 14.10: UE1MO	*Chart 14.11: UE2MO*
	

Table 14.22: Molybdenum Grade Outliers

EUMO	Minimum Mo (ppm)	Maximum Mo (ppm)
1	4.85	2,189
2	1.02	910

14.2.7 Spatial Analysis - Variography

For each estimate unit, the variographic analysis was conducted in different directions with different steps and allowances.

The variographic analysis for each of the elements studied (Cu, Mo and Ag) was conducted by UE, considering the different directions, paths and allowances. The following tables show the different variographic models for each of the mentioned elements.

14.2.7.1 Copper

Table 14.23: Variographic Models for Structures 1 and 2

Variographic Models												
UE	C0	Model	C1	1 Structure			Model	C2	2 Structure		
				X	Y	Z			X	Y	Z
UE1	0.2834	EXP	0.3513	79.412	26.471	102.100	SPH	0.356	101.890	79.328	185.290
UE2	0.4167	SPH	0.5796	260.920	211.760	221.220	SPH				
UE3	0.1293	SPH	0.2067	34.034	34.034	34.034	SPH	0.031	272.270	272.270	272.270
UE4	0.0079	EXP	0.0026	14.806	14.806	14.806	SPH	0.016	364.710	364.710	364.710
UE5	0.0038	SPH	0.0129	307.560	307.560	307.560	EXP	0.003	388.240	388.240	388.240
UE6	0.4512	SPH	0.4042	44.118	44.118	44.118	SPH	0.146	288.050	288.050	288.050
UE7	0.0037	SPH	0.0022	8.805	8.805	8.805	SPH	0.005	143.700	143.700	143.700
UE8	0.0077	SPH	0.0058	10.347	10.347	10.347	SPH	0.011	89.702	89.702	89.702
UE9	0.2571	SPH	0.7518	113.450	113.450	153.780					
UE10	0.0114	SPH	0.0095	33.170	33.170	33.170	SPH	0.006	188.380	188.380	188.380
UE11	0.4002	SPH	0.3291	30.928	30.928	30.928	SPH	0.272	136.080	136.080	136.080

Table 14.24: Variographic Models for Structure 3

Variographic Models										
UE	Model	C3	3 Structure			Rotation GSLIB-MS (zxy)				
			X	Y	Z	X	Y	Z	Outlier	Low grade
UE1				1.01		30	0	150	0.744	0.010
UE2						30	0	-30	0.030	1.750
UE3	SPH	0.631	600	600	600				0.639	0.020
UE4									1.250	0.070
UE5									0.890	0.020
UE6									0.869	0.084
UE7	SPH	0.009	700	700	700				0.670	0.040
UE8									1.060	0.160
UE9									1.300	0.020
UE10									0.950	0.030
UE11										

14.2.7.2 Molybdenum

Table 14.25: Molybdenum Variographic Model

Variographic Models											
UE	C0	Model	C1	1 Structure			Model	C2	2 Structure		
				X	Y	Z			X	Y	Z
UE1	5,927.80	SPH	6,722.20	54.8	64.5	129.3	SPH	16,683.30	383.300	321.300	161.600
UE2	2,632.70	SPH	897.91	155.74	155.74	155.74	SPH	2,479.00	331.910	331.910	331.910

Table 14.26: Molybdenum Variographic Model

Variographic Models							
UE	C0	Model	Rotation GSLIB-MS (zxy)				
			X	Y	Z	Outlier	Low grade
UE1	5,927.80	SPH	0	75	0	1.01	4.85
UE2	2,632.70	SPH	0	0	0	910	1.02

14.2.7.3 Silver

Table 14.27: Silver Variographic Model

Variographic Models																	
UE	C0	Model	C1	1 Structure			Model	C2	2 Structure			Rotation GSLIB-MS (zxy)					
				X	Y	Z			X	Y	Z	X	Y	Z	Outlier	Low grade
UE1	0.01987	SPH	0.0299	328.12	328.12	328.12									1.01	0.04
UE2	0.02209	SPH	0.0806	132.35	132.35	132.35	SPH	0.129	155.04	155.04	155.04				3.94	0.13
UE3	0.07685	SPH	0.0361	85.159	85.159	85.159	SPH	0.134	455.46	455.46	455.46				2.31	0.13
UE4	0.09020	SPH	0.0406	122.64	122.64	122.64	SPH	0.103	560.38	560.38	560.38				2.83	0.09
UE5	0.01987	SPH	0.0299	328.12	328.12	328.12									1.41	0.04

14.2.8 Resource Block Model

Vizcachitas Project block model has its origin at UTM WGS84 East 3,646,600, North 6,411,800 and with an elevation of 900 metres above sea level. The block dimensions have been maintained from the previous study at 20m x 20m x 10m.

Table 14.28: Block Model Dimensions

	East	North	Elev
Minimum	364,600	6,411,880	900
Maximum	367,900	6,415,300	3,200
Blocks (m)	20	20	10
Block No.	165	171	230

14.2.9 Interpolation Plan

14.2.9.1 Copper

Table 14.29 shows the Kriging interpolation plan considering R1 and R2 search radii. Table 14.35 shows the kriging plan for the R3 and R4 search radii. Finally, Table 14.31 illustrates Inverse Distance Weighting (IDW) estimates for R1 and R2 radio.

Table 14.29: Ordinary Kriging Estimate Plan, Copper Grade (R1 and R2)

UE	R1								R2							
	Search distance			Min No. Samples per Block	Max No. Samples per Block	Max No. Samples per Drill Hole	No. of Octants / Quadrants	No. Samples per Octant / Quadrant	Search distance			Min No. Samples per Block	Max No. Samples per Block	Max No. Samples per Drill Hole	No. of Octants / Quadrants	No. Samples per Octant / Quadrant
	X	Y	Z						X	Y	Z					
UE1	80	30	110	3	24	2			110	70	190	3	24	2		
UE2	100	100	100	4	12	3			270	220	220	4	12	3		
UE3	60	60	60	3	24	3	Quadrant	6	120	120	120	2	24	4	Quadrant	6
UE4	100	100	100	4	24	3			140	140	140	6	36	5		
UE5	120	120	120	3	12	2			200	200	200	4	12	3		
UE6	60	60	60	4	24	3			160	160	160	3	24	3		
UE7	130	130	130	4	24	3	Quadrant	6	180	180	180	4	24	3		
UE8	90	90	90	4	24	3			180	180	180	4	24	3		
UE9	60	60	80	4	24	3			115	115	160	4	24	3		
UE10	50	50	50	4	24	3			150	150	150	3	24	2		
UE11	70	70	70	4	24	3			150	150	150	4	24	3		

Table 14.30: Ordinary Kriging Estimate Plan, Copper Grade (R3 and R4)

UE	R3								R4							
	Search distance			Min No. Samples per Block	Max No. Samples per Block	Max No. Samples per Drill Hole	No. of Octants / Quadrants	No. Samples per Octant / Quadrant	Search distance			Min No. Samples per Block	Max No. Samples per Block	Max No. Samples per Drill Hole	No. of Octants / Quadrants	No. Samples per Octant / Quadrant
	X	Y	Z						X	Y	Z					
UE1	300	300	300	2	15	2										
UE2	400	400	400	1	15	2										
UE3	300	300	350	3	15	3										
UE4	250	250	250	2	15	2										
UE5	400	400	400	3	15	2										
UE6	300	300	300	1	15	3										
UE7	400	400	400	3	15	2										
UE8	300	300	300	2	15	2			500	500	500	1	6	1		
UE9	220	220	350	3	18	3										
UE10	340	340	340	2	15	2										
UE11	400	400	400	3	24	3										

Table 14.31: IDW2 Estimate Plan, Copper Grade

UE	R1							
	Search distance			Min No. Samples per Block	Max No. Samples per Block	Max No. Samples per Drill Hole	No. of Octants / Quadrants	No. Samples per Octant / Quadrant
	X	Y	Z					
UE1								
UE2								
UE3	500	500	500	1	6	3		
UE4	500	500	500	1	6	1		
UE5	500	500	500	1	6	1		
UE6								
UE7	500	500	500	1	6	1		
UE8								
UE9	500	500	500	1	6	1		
UE10	500	500	500	1	6	1		
UE11	500	500	500	1	6	1		
UE12	600	600	600	1	24	1		

14.2.9.2 Molybdenum

Table 14.32: Ordinary Kriging Estimate Plan, Molybdenum Grade (R1 and R2)

UE	R1								R2							
	Search distance			Min No. Samples	Max No. Samples per Block	Max No. Samples per Drill Hole	No. of Octants / Quadrants	No. Samples per Octant / Quadrant	Search distance			Min No. Samples	Max No. Samples per Block	Max No. Samples per Drill Hole	No. of Octants / Quadrants	No. Samples per Octant / Quadrant
	X	Y	Z						X	Y	Z					
UE1	70	70	120	6	16	5		4	180	180	150	7	15	6		
UE2	150	150	150	7	15	6			200	200	200	7	15	6		

Table 14.33: Ordinary Kriging Estimate Plan, Molybdenum Grade (R3 and R4)

UE	R3								R4							
	Search distance			Min No. Samples per Block	Max No. Samples per Block	Max No. Samples per Drill Hole	No. of Octants / Quadrants	No. Samples per Octant / Quadrant	Search distance			Min No. Samples per Block	Max No. Samples per Block	Max No. Samples per Drill Hole	No. of Octants / Quadrants	No. Samples per Octant / Quadrant
	X	Y	Z						X	Y	Z					
UE1	380	380	160	8	16	6			400	400	400	6	15	6		
UE2	320	320	320	7	15	6			400	400	400	6	15	6		

14.2.9.3 Silver

Table 14.34: Ordinary Kriging Estimate Plan, Silver Grade (R1 and R2)

UE	R1								R2							
	Search distance			Min No. Samples per Block	Max No. Samples per Block	Max No. Samples per Drill Hole	No. of Octants / Quadrants	No. Samples per Octant / Quadrant	Search distance			Min No. Samples per Block	Max No. Samples per Block	Max No. Samples per Drill Hole	No. of Octants / Quadrants	No. Samples per Octant / Quadrant
	X	Y	Z						X	Y	Z					
UE1	80	80	80	3	15	2			120	120	120	2	15	1		
UE2	120	120	120	3	15	2			132	132	132	3	15	2		
UE3	85	85	85	7	15	6			240	240	240	7	15	6		
UE4	150	150	150	7	15	6			210	210	210	3	15	2		
UE5	120	120	120	4	15	3			170	170	170	3	15	2		

Table 14.35: Ordinary Kriging Estimate Plan, Silver Grade (R3)

UE	R3							
	Search distance			Min No. Samples per Block	Max No. Samples per Block	Max No. Samples per Drill Hole	No. of Octants / Quadrants	No. Samples per Octant / Quadrant
	X	Y	Z					
UE1	500	500	500	2	15	2		
UE2	160	160	160	2	15	2		
UE3	460	460	460	2	15	2		
UE4	560	560	560	2	15	2		
UE5	500	500	500	3	15	3		

14.2.10 Block Model Validation

The validation of the block model estimate was performed by Ordinary Kriging using the Nearest Neighbor (NN) method, for radiuses 1 and 2 of the Kriging estimate.

A visual inspection was carried out section by section, between the grades estimated with Ordinary Kriging, and the grades of samples.

14.2.11 Global Bias Comparison

14.2.11.1 Copper

The estimate was validated for Radii 1 and 2 estimated by Ordinary Kriging. For this validation a comparison of the grades estimated by the Nearest Neighbor method was used, which were in turn compared with the disaggregated grades of the samples. When reviewing the results obtained between both methods the maximum deviations were about 5%, deviations that are acceptable by industry standards. See Table 14.36

Table 14.36: Copper Grade Estimate Validation

UE	Mean (%)	Mean (%)	Decluster (%)	Rb1 OK (%)		Rb1 NN		(OK-NN)/OK	Rb2 OK		Rb2 NN		(OK-NN)/OK
				Mean %	Diff. (%)	Mean %	Diff.		Mean %	Diff.	Mean %	Diff.	
1	0.1332	0.1332	0.141	0.130	7.7%	0.135	3.9%	-4.1%	0.142	-1.2%	0.143	-1.4%	-0.2%
2	0.5014	0.5014	0.478	0.477	0.3%	0.483	-0.9%	-1.2%	0.475	0.7%	0.481	-0.5%	-1.2%
3	0.1481	0.1481	0.147	0.139	5.6%	0.139	5.6%	0.1%	0.132	10.1%	0.132	10.5%	0.4%
4	0.2902	0.2902	0.283	0.274	3.4%	0.271	4.3%	1.0%	0.262	7.5%	0.249	12.0%	4.8%
5	0.2559	0.2559	0.252	0.272	-8.0%	0.273	-8.1%	-0.1%	0.245	3.0%	0.242	4.0%	1.1%
6	0.3752	0.3752	0.367	0.372	-1.4%	0.361	1.6%	2.9%	0.367	0.0%	0.380	-3.6%	-3.6%
7	0.2658	0.2658	0.261	0.278	-6.7%	0.277	-6.1%	0.5%	0.276	-6.1%	0.279	-7.1%	-1.0%
8	0.5265	0.5266	0.522	0.532	-1.8%	0.534	-2.3%	-0.4%	0.543	-4.0%	0.550	-5.3%	-1.3%
9	0.5049	0.5049	0.492	0.497	-0.8%	0.497	-0.8%	0.0%	0.479	2.7%	0.484	1.8%	-0.9%
10	0.2991	0.2991	0.295	0.314	-6.7%	0.330	-11.8%	-4.8%	0.317	-7.5%	0.316	-7.3%	0.2%
11	0.0908	0.0908	0.095	0.101	-6.8%	0.104	-10.1%	-3.2%	0.086	9.5%	0.086	9.2%	-0.4%

14.2.11.2 Molybdenum

For molybdenum a deviation of less than 2% for 87% of composites was obtained for the estimate unit 1 for both radii. The remaining 13% obtained a deviation between 5.5% and 7.6%, deviations that are acceptable by industry standards. See Table 14.37

Table 14.37: Molybdenum Grade Estimate Validation

UE	Mean (ppm)	Mean (ppm)	Decluster (ppm)	Rb1 OK		Rb1 NN		(OK-NN)/OK	Rb2 OK		Rb2 NN		(OK-NN)/OK
				Mean (ppm)	Diff. (%)	Mean (ppm)	Diff. (%)		Mean (ppm)	Diff. (%)	Mean (ppm)	Diff. (%)	
UE1	120.435	120.392	115.69	122.60	-6%	122.80	-6%	-0.2%	114.78	1%	1.01	3%	1.8%
UE2	46.153	46.695	44.380	44.46	0%	42.03	5%	5.5%	46.23	-4%	42.72	4%	7.6%

14.2.12 Visual Comparison

The following figures show the matching between the grades of the drill hole samples and the grades of the estimated blocks for copper and molybdenum.

14.2.12.1 Copper

Figure 14.4: North Section 6413500

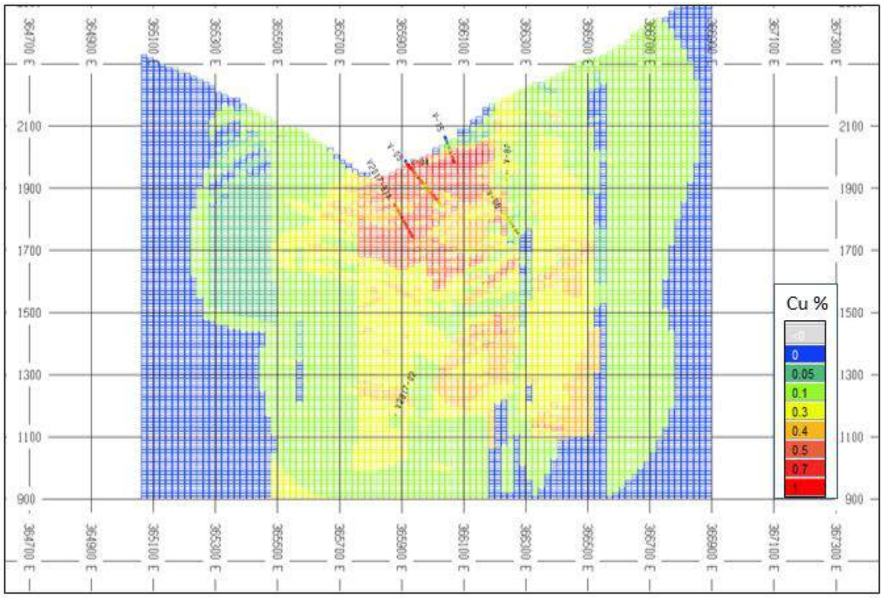


Figure 14.5: North Section 6413400

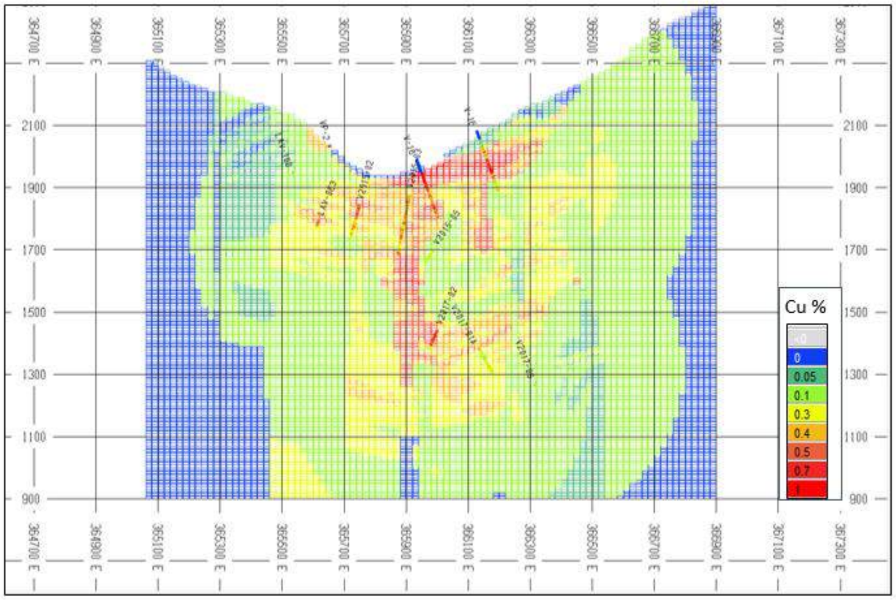
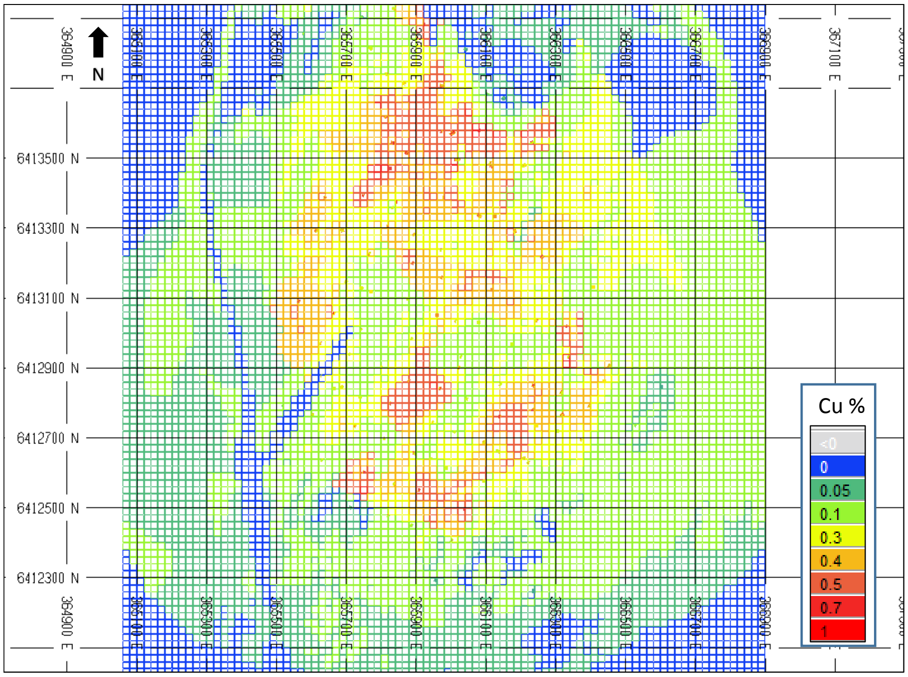


Figure 14.6: Plan 1850



14.2.12.2 Molybdenum

Figure 14.7: North Section 6413460

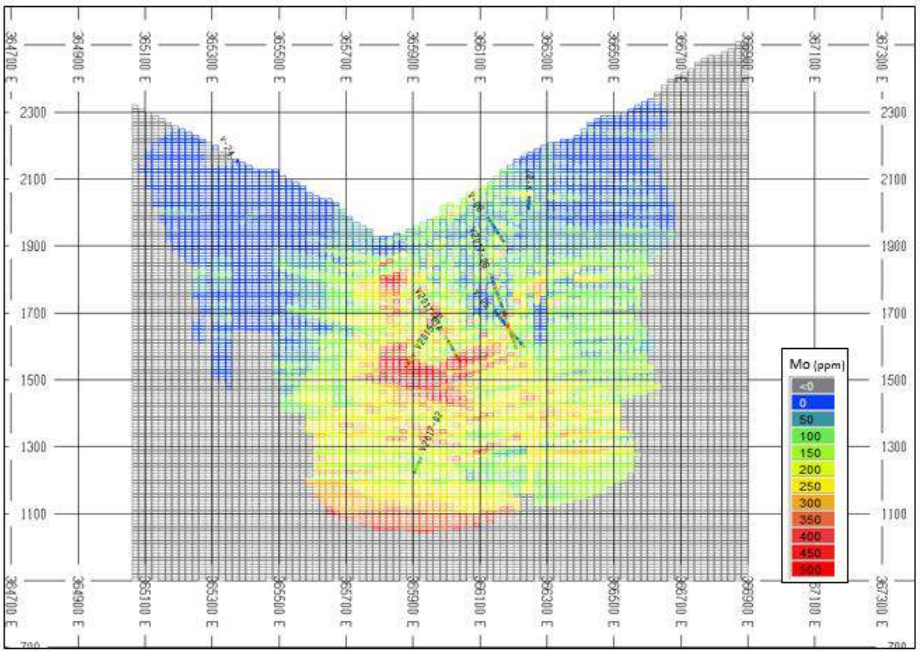
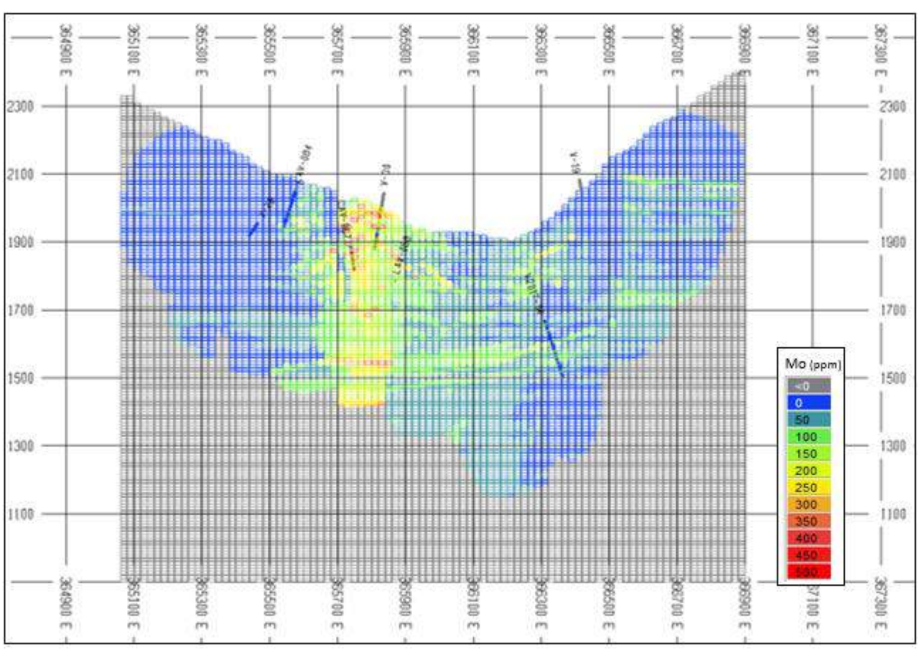


Figure 14.8: North Section 6413000



14.2.13 Mineral Resource Categorization

The categorization criterion used in this technical report has been based on the criteria used in the previous 2014 PEA by Coffey, linking the geological information with the geostatistics in which a tonnage associated with a certain production period is considered.

A monthly production block was used, the dimensions of which are = 320 x 320x 20 m = 2.048.000 m^3 x 2.4 t/m^3 = 4.915.200 t.

For this evaluation an indicator Kriging was developed with a cut-off grade of 0.25% Cu for four estimate units, which were defined as shown in Table 14.38.

CL (90% confidence limits) = +- 1.645 x σ 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 Coefficients Variation)

Results showed that with a 180 x 180 m mesh, a 9-10% range with a 90% confidence limit is obtained.

Table 14.38: Estimate Unit for Indicator Kriging used for Categorization

<table>
<tr><th colspan="5">UE IND</th></tr>
<tr><th rowspan="2">Lithology</th><th rowspan="2">Description</th><th colspan="3"></th></tr>
<tr><th>302
Leached</th><th>304
Supergene</th><th>305
Hypogene</th></tr>
<tr><td>101</td><td>GRV</td><td colspan="2" rowspan="16">1</td><td rowspan="11">4</td></tr>
<tr><td>120</td><td>DIOF</td></tr>
<tr><td>104</td><td>DIO</td></tr>
<tr><td>105</td><td>GRD</td></tr>
<tr><td>107</td><td>PDIO</td></tr>
<tr><td>102</td><td>AND</td></tr>
<tr><td>109</td><td>TON</td></tr>
<tr><td>118</td><td>BXI</td></tr>
<tr><td>121</td><td>DIOM</td></tr>
<tr><td>122</td><td>DIOQ</td></tr>
<tr><td>119</td><td>DEP</td></tr>
<tr><td>123</td><td>DEP-1</td><td>2</td></tr>
<tr><td>112</td><td>BXH-1</td><td>3</td></tr>
<tr><td>113</td><td>BXH</td><td rowspan="3">4</td></tr>
<tr><td>103</td><td>PDAC</td></tr>
<tr><td>117</td><td>BFM</td></tr>
</table>

Table 14.39: Drill Hole Spacing to Define the Categorization Search Scopes

UEIND1

Mesh	**220**	**210**	**200**	**180**	**160**	**140**
OK	0.157	0.069	0.069	0.064	0.036	0.021
CV	0.784	0.784	0.784	0.784	0.784	0.784
Monthly Relative Variance	0.097	0.042	0.042	0.039	0.022	0.013
Annual Relative Variance	0.008	0.004	0.004	0.003	0.002	0.001
V.A. Root	0.090	0.059	0.059	0.057	0.043	0.033
90% Confidence Limit (± 1.645)	14.76%	9.784%	9.777%	9.4%	7.0%	5.4%

UEIND2

Mesh	**220**	**210**	**200**	**180**	**160**	**140**
OK	0.861	0.418	0.412	0.406	0.211	0.138
CV	0.357	0.357	0.357	0.357	0.357	0.357
Monthly Relative Variance	0.110	0.053	0.053	0.052	0.027	0.018
Annual Relative Variance	0.009	0.004	0.004	0.004	0.002	0.001
V.A. Root	0.096	0.067	0.066	0.066	0.047	0.038
90% Confidence Limit (± 1.645)	15.75%	10.973%	10.894%	10.8%	7.8%	6.3%

UEIND3

Mesh	**220**	**210**	**200**	**180**	**160**	**140**
OK	0.101	0.049	0.048	0.047	0.047	0.022
CV	0.488	0.488	0.488	0.488	0.488	0.488
Monthly Relative Variance	0.024	0.012	0.011	0.011	0.011	0.005
Annual Relative Variance	0.002	0.001	0.001	0.001	0.001	0.000
V.A. Root	0.045	0.031	0.031	0.031	0.030	0.021
90% Confidence Limit (± 1.645)	7.36%	5.127%	5.079%	5.0%	5.0%	3.5%

UEIND4

Mesh	**220**	**210**	**200**	**180**	**160**	**140**
OK	0.194	0.094	0.093	0.089	0.046	0.028
CV	0.740	0.740	0.740	0.740	0.740	0.740
Monthly Relative Variance	0.106	0.051	0.051	0.048	0.025	0.015
Annual Relative Variance	0.009	0.004	0.004	0.004	0.002	0.001
V.A. Root	0.094	0.065	0.065	0.064	0.046	0.036
90% Confidence Limit (± 1.645)	15.46%	10.769%	10.683%	10.4%	7.6%	5.9%

The following criteria was considered for resource classification:

- Inferred Resources: One drill hole as a minimum and a distance < 400 m
- Indicated Resources: Two drill holes as a minimum and an average distance < 180 m

The previous study did not classify measured resources. For this study, measured resources have been classified according to the following criteria:

- Measured Resources: Three drill holes as a minimum and an average distance < 80 m.

To avoid having inferred blocks among measured blocks (salt and pepper effect) a smoothing using an IDW estimation was made. This smoothing was not applied to the already categorized measured and indicated blocks which were estimated by Ordinary Kriging. Based on this, the following criteria were used for the smoothing:

- Inferred Resources: One drill hole as a minimum and a distance < 400 m

- Indicated Resources: Two drill holes as a minimum and an average distance < 140 m
- Measured Resources: Three drill holes as a minimum and an average distance < 80 m.

Figure 14.9: Cu Resource Categorization, North Section 6412720

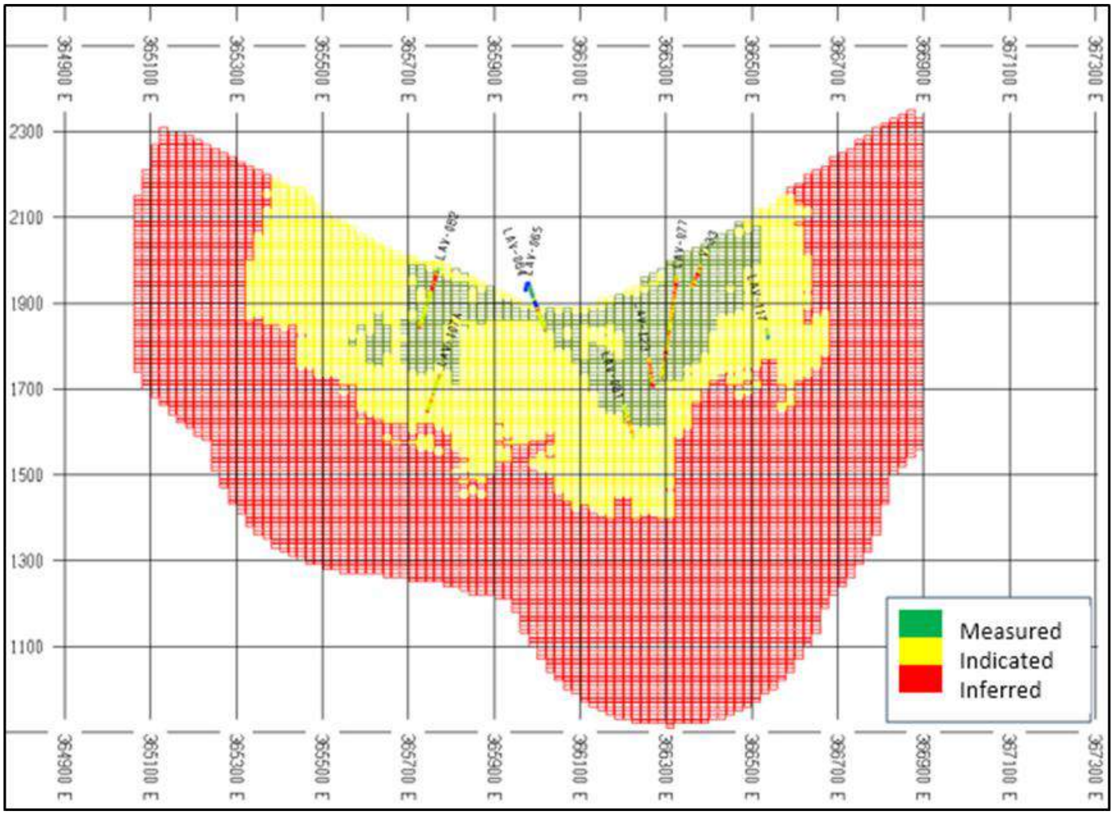
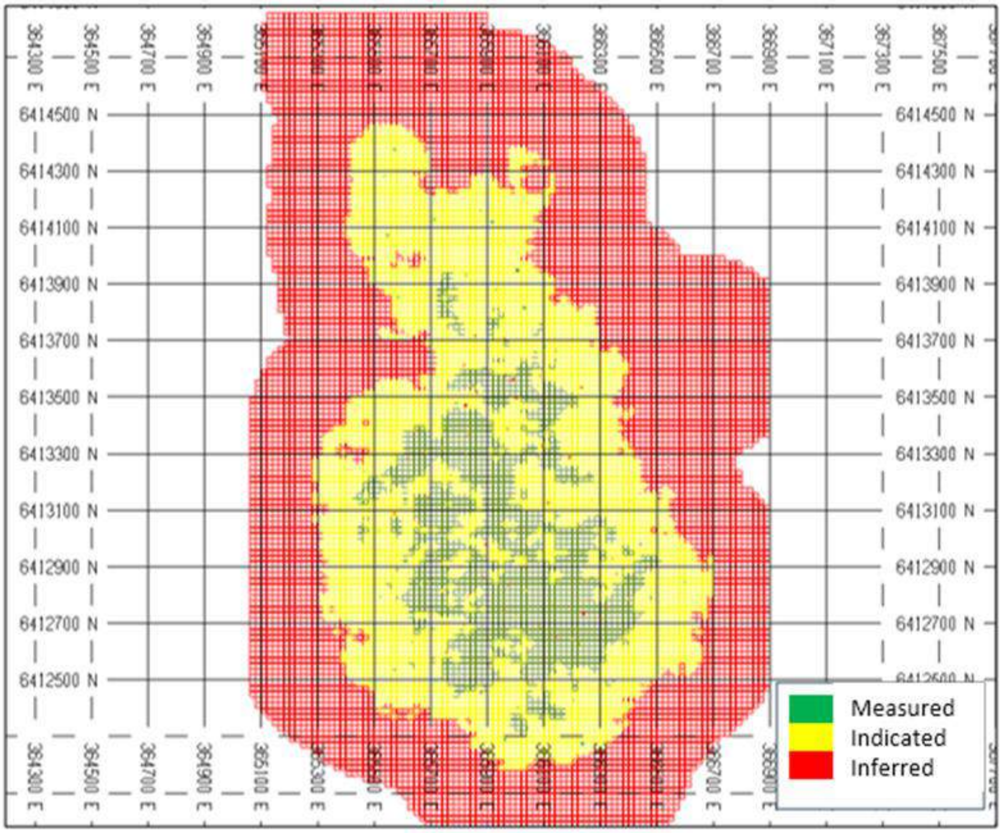


Figure 14.10: Cu Resource Categorization, Plan 1870



14.2.14 Mineral Resource

To assess reasonable prospects for eventual economic extraction, a Whittle pit shell was prepared using general technical and economic assumptions listed below to constrain the estimated resource blocks.
Plant cost : 4.9 USD/t

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

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.

The in-pit mineral resources are reported using a 0.25% copper cut-off.

- Measured mineral resources are 254.4 million tonnes grading 0.439% copper, 119.2 ppm molybdenum and 1.26 g/t silver giving a 0.489% copper equivalent.
- Indicated mineral resources are 1,029.67 million tonnes grading 0.385% copper, 146.9 ppm molybdenum and 1.00 g/t silver giving a 0.442% copper equivalent.
- Measured and Indicated mineral resources are 1,284.06 million tonnes grading 0.396% copper, 141.4 ppm molybdenum and 1.05 g/t silver giving a 0.451% copper equivalent.
- The Inferred mineral resources are 788.82 million tonnes grading 0.337% copper, 127.0 ppm molybdenum and 0.88 g/t silver giving a 0.386% copper equivalent.

The tables Table 14.40, Table 14.41, Table 14.42 and Table 14.43 present a sensitivity analysis for the mineral resources under different cut-off grades. The base case for the estimation of resources is 0.25% Cu.

Table 14.40: Measured Resources In-Pit, Cut-off Cu

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 14.41: Indicated Resources In-Pit, Cut Off Cu

Indicated Resources									
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 14.42: Measured and Indicated Resources In-Pit, Cut-off Cu

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 14.43: Inferred Resources In-Pit, Cut-off Cu

Inferred Resources									
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: 3.00 USD/lb Cu, 10.00 USD/lb Mo and 17.00 USD/oz Ag. No allowance for metallurgical recoveries has been considered*
- *Small discrepancies may exist due to rounding errors.*
- *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.*
- *Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability*

14.3 Conclusions

- The updated geological model better defines how the mineralization is controlled and therefore has produced a more robust resource estimate that the previous resource estimates.
- The database and geological models were found to be in good order and auditable.
- The database (assay and geological data) has been validated and is of sufficient quality to support this Mineral Resource estimate.
- The block model has been validated and is of sufficient quality to support the Mineral Resource estimate.
- The Mineral Resource estimates are in line with good industry practice.

15. MINERAL RESERVES ESTIMATE

The Project has no mineral reserves; all mineralization is considered as resources.

16. MINING METHODS

An open pit mining method has been selected for the Vizcachitas deposit mainly owing to the copper and molybdenum grades and the continuous mineralization occurring near the surface, which is a considerable benefit in relation to the waste/mineral ratio for open pit mining.

Previous engineering studies have discarded an underground mining option owing to the lack of grades to support such an option.

The mine has been scheduled to operate 360 days per year. The plan contemplates two 12-hour shifts per day. Mining operations include drilling, blasting, loading, hauling and support services.

The annual operation period defined for the mine considers minimal snow-related downtime. The deposit is located at an average elevation of 2000 masl, with potentially minor snow events. The roads and working areas may be promptly cleared by using support equipment considered in the operation.

This report shows the results obtained for three mine plans developed for 55 ktpd, 110 ktpd and 200 ktpd mill throughput capacities.

An optimization was made for the three mine plants scenarios, the 55 ktpd (scenario with minimum Capex) was updated from the previous 44 ktpd proposed by Coffey (Coffey et al., 2014) and the 200 ktpd (scenario with maximum starter NPV) updated from the previous 176 ktpd also proposed by Coffey (Coffey et al., 2014). The 110 ktpd case was developed from a Tetra Tech optimization exercise, as part of the PEA update, which delivers not only better economic results but also improves the mining and technical aspects for operating the Vizcachitas mineral deposit.

It should be noted that the pit optimization process is preceded by an economic valuation of the resource model (block models), whereby all blocks have an associated economic value that may be negative for waste or positive for mineral resource. This was accomplished by employing the block model described in Chapter 14. The pit optimization process was performed using Whittle Four X, which uses the Lerchs–Grossman optimization algorithm.

Based on the pit optimization results, 2 sets were designed. Set 1 considered average phase widths between 90 and 110 m, sufficient to support mining for the 55 ktpd and 110 ktpd scenarios. Set 2 had a greater operating width, reaching average widths between 110 and 150 m, consistent with a 200 ktpd scenario.

Mine plans were prepared using the COMET strategic planning software to maximize the total project value, obtain the best cut-off grade strategy, and comply with all relevant technical-operating restrictions, such as maximum development of phases by period, vertical distance, and inter-phase interferences, among others.

The strategic mine plans considered the following conditions:

- Bench height: 20 m (doubling the resource model block height set at 10 m).
- Maximum vertical head by phase: 11 benches per year, equivalent to a maximum vertical development of 220 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 third year.
- For mine plan purposes, 70% of mine and plant costs are considered variable and 30% are fixed.

All mine plan options were assessed with the same level of detail, determining the hauling distances for mineralized material and waste, and then estimating the equipment fleet and the purchasing requirements over time.

16.1 Bench Height

The bench height defined for the three scenarios under analysis was 20 m, which resulted from doubling the geological resource model block height (20 m x 20 m x 10 m). A bench height of 20 m instead of 10 m has the following major advantages:

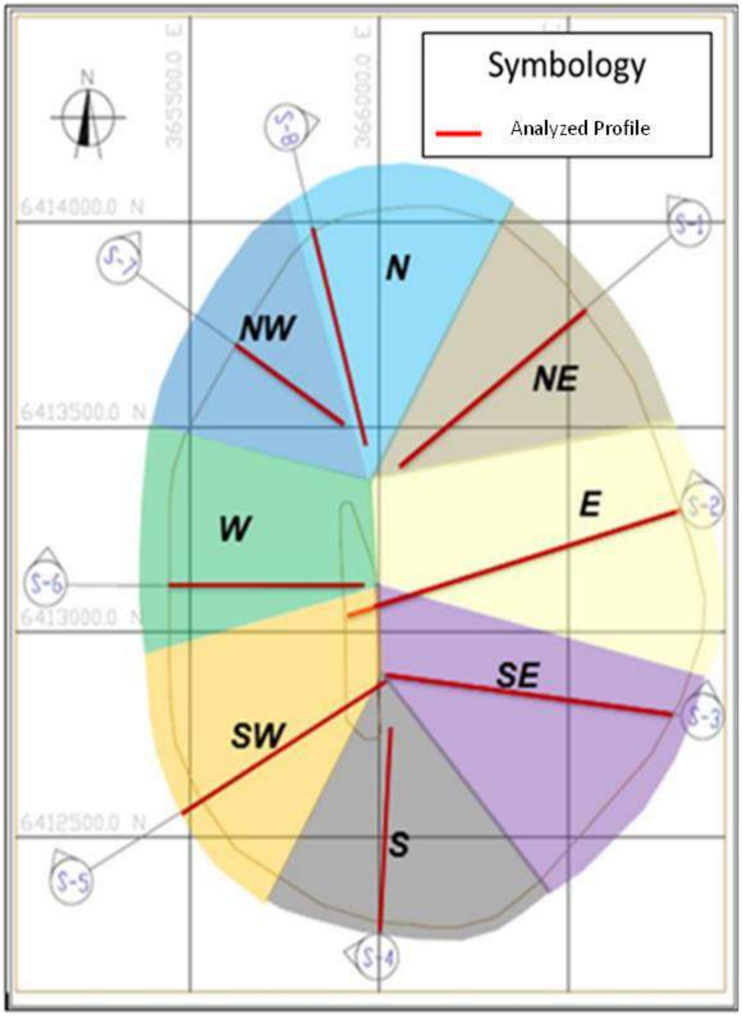
- Drilling and blasting cost savings (reduced metres and operating time)
- Earthworks cost savings (removing an intermediate level)
- Introduction of larger, heavier-duty equipment
- Reduction in number of equipment and operators

16.2 Geotechnical Parameters

The study "Preliminary Geotechnical Modeling Report and Global Geomechanical Stability Analysis for a Pre-Economic Assessment (PEA) –Vizcachitas Pit" prepared by FF GeoMechanics defined 24 geotechnical areas for the mine design.

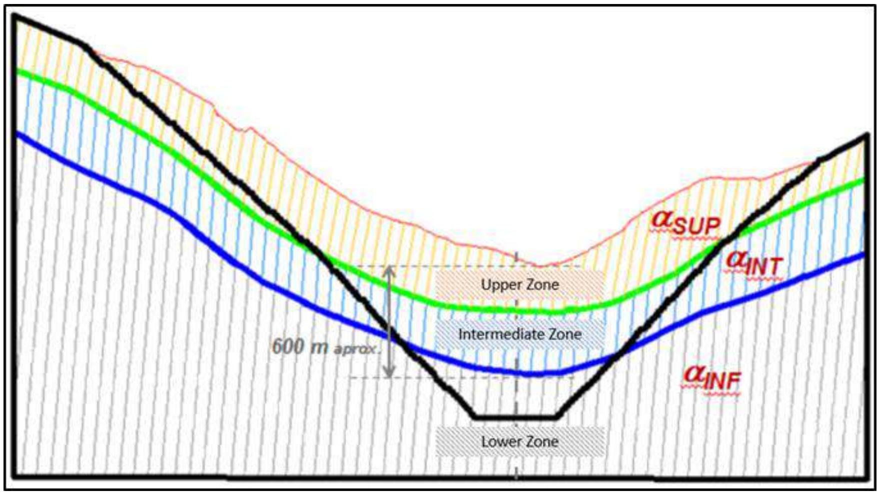
The identification of the geotechnical areas began with the definition of a design rosette based on the study of eight profiles covering the entire mine area. Figure 16.1 shows the position of the profiles under study (FF GeoMechanics 2018).

Figure 16.1: Mine Sectors Considered by Geotechnical Behavior



Subsequently the rock mass was zoned in depth, identifying a fractured surface layer and two deeper zones with better stability conditions labeled Upper, Intermediate and Lower Zones, as shown in Figure 16.2.

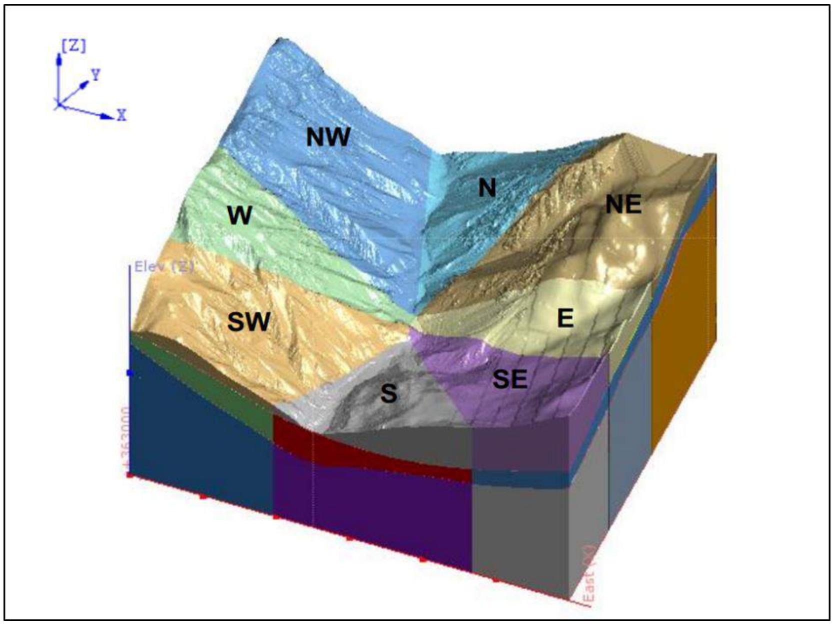
Figure 16.2: Rock Mass Zones



As a result of the intersection of the information presented in Figure 16.1 and Figure 16.2, 24 geotechnical zones were obtained (see Figure 16.3) that allow the mineralization to be modeled three-dimensionally, defining a global slope angle for each of them. The angles in the mine design satisfy the stability requirements of the rock mass, fulfilling the standard parameters for safety factors and probability of failure.

Conceptually, the global angles thus defined allow the design of bench by bench phases (face-slope angle, inter-angle angle and berm-width), incorporating access ramps and decoupling berms according to the required inter-ramp height. This study however did not consider the operational design of phases but generated a smoother design adjusting them to the Whittle-optimized pit shells.

Figure 16.3: Geotechnical Zones



The main geotechnical parameters for each zone are shown in Table 16.1

Table 16.1: Geometrical Parameters by Sector

Slope/Wall	Geotechnical Zone	Global Angle (°)	Width Berm (m)	High Bench (m)	Decouple Berm (m)
NE (section 1)	Upper	42	15	20	each 160 m, width 30 m
	Intermediate	49	12	20	each 160 m, width 24 m
	Lower	49	12	20	each 160 m, width 24 m
E (section 2)	Upper	43	15	20	each 160 m, width 30 m
	Intermediate	43	12	20	each 160 m, width 24 m
	Lower	43	12	20	each 160 m, width 24 m
SE (section 3)	Upper	42	15	20	each 160 m, width 30 m
	Intermediate	48	12	20	each 160 m, width 24 m
	Lower	46	12	20	each 160 m, width 24 m
S (section 4)	Upper	49	15	20	each 160 m, width 30 m
	Intermediate	51	12	20	each 160 m, width 24 m
	Lower	51	12	20	each 160 m, width 24 m
SW (section 5)	Upper	43	15	20	each 160 m, width 30 m
	Intermediate	48	12	20	each 160 m, width 24 m
	Lower	46	12	20	each 160 m, width 24 m
W (section 6)	Upper	41	15	20	each 160 m, width 30 m
	Intermediate	48	12	20	each 160 m, width 24 m
	Lower	48	12	20	each 160 m, width 24 m
NW (section 7)	Upper	44	15	20	each 160 m, width 30 m
	Intermediate	52	12	20	each 160 m, width 24 m
	Lower	49	12	20	each 160 m, width 24 m
N (section 8)	Upper	45	15	20	each 160 m, width 30 m
	Intermediate	50	12	20	each 160 m, width 24 m
	Lower	50	12	20	each 160 m, width 24 m

16.3 Waste Dump Design Parameters

The waste dump was designed to the north of the pit, with the following technical parameters (see Table 16.2).

Table 16.2: Geometrical Parameters for Waste Dumps

Parameters Dump Design		
Parameters	Value	Unit
Module Height	50.0	m
Face Slope	37.0	°
Ramp Width	35.0	m
Ramp Slope	10.0	%
South Berm Width	60.0	m
North Berm Width	100.0	m
South Global Slope	27.8	°
North Global Slope	30.9	°

Figure 16.4 shows the waste dump design for the Vizcachitas Project, located north of the pit. For the three cases under analysis (55 ktpd, 110 ktpd and 200 ktpd) the same waste dump design was used to measure the hauling distances and estimate the extraction truck fleet.

Figure 16.4: Waste Dump Design

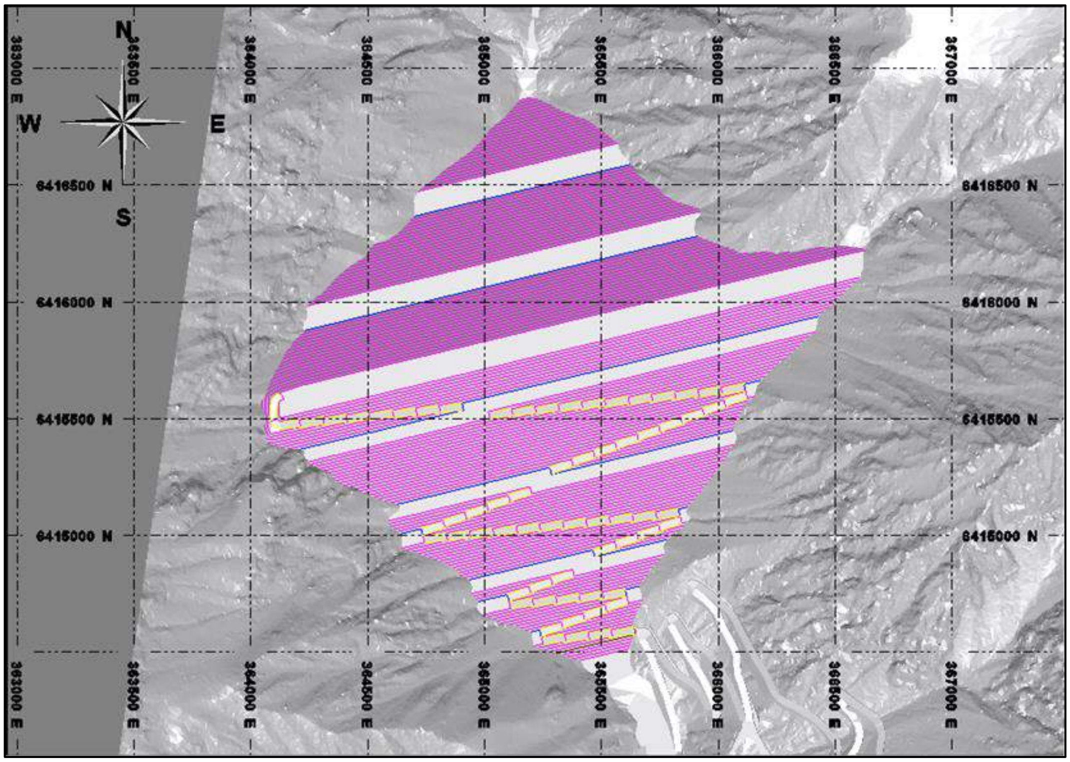


Table 16.3 summarizes the dump take-off for each level.

Table 16.3: Waste Dump Volume

Floor Elevation	Dumping Elevation	Height (m)	Volume	
			Partial m^3 x 1000	Cumulated m^3 x 1000
2030	2080	50	6,544	6,544
2080	2130	50	22,349	28,894
2130	2180	50	39,146	68,040
2180	2230	50	53,059	121,099
2230	2280	50	61,590	182,689
2280	2330	50	70,869	253,558
2330	2380	50	77,685	331,243
2380	2430	50	80,976	412,219
2430	2480	50	77,528	489,748
2480	2530	50	75,350	565,097
2530	2580	50	64,867	629,964
2580	2630	50	53,123	683,088
2630	2680	50	35,071	718,158
2650	2700	20	17,289	735,447
Total		670	735,447	

For the 200 ktpd case, the design capacity is insufficient. An expansion of the dump is possible generating the capacity for this scenario. In this case, hauling distances were escalated, simulating that the dump size is expanded to the north.

16.4 Hydrology and Hydrogeology

No studies have been made at this stage. The pit sits in a “V” shape valley through which the Rocin River flows. For this PEA, a river diversion has been considered allowing for open pit mining and the location of waste dumps.

A provision under item Others in the Operating Costs allows for water management in the pit and dumps, among other activities.

16.5 Treatment Capacity

Three process capacity options were assessed, using a mine planning process based on the optimization of the final pit, non-operating phase designs, strategic mine plans, equipment fleet estimates, and associated capital costs, concluding with an economic evaluation. The case studies were:

- Case 1: 55 ktpd
- Case 2: 110 ktpd

- Case 3: 200 ktpd

16.6 Optimum Pit Shells

To define the final boundary and the extraction sequence for the subsequent design of the operating phases in each case under analysis, the final pit was optimized with the existing resource model and following the defined economic and geotechnical parameters.

This exercise was performed by using the Lerchs & Grossman algorithm introduced into the Whittle Four-X software.

16.6.1 Final Pit Optimization Results

16.6.1.1 55 ktpd and 110 ktpd Cases

Figure 16.5 shows the final economic pit limit obtained in the optimization process for the 55 ktpd and 110 ktpd cases. With this limit, mineral extraction amounts to 3,634 Mt with a 0.28% Cu grade and 9,355 Mt of rock.

Figure 16.5: Final Whittle Pit - Case 55 and 110 ktpd

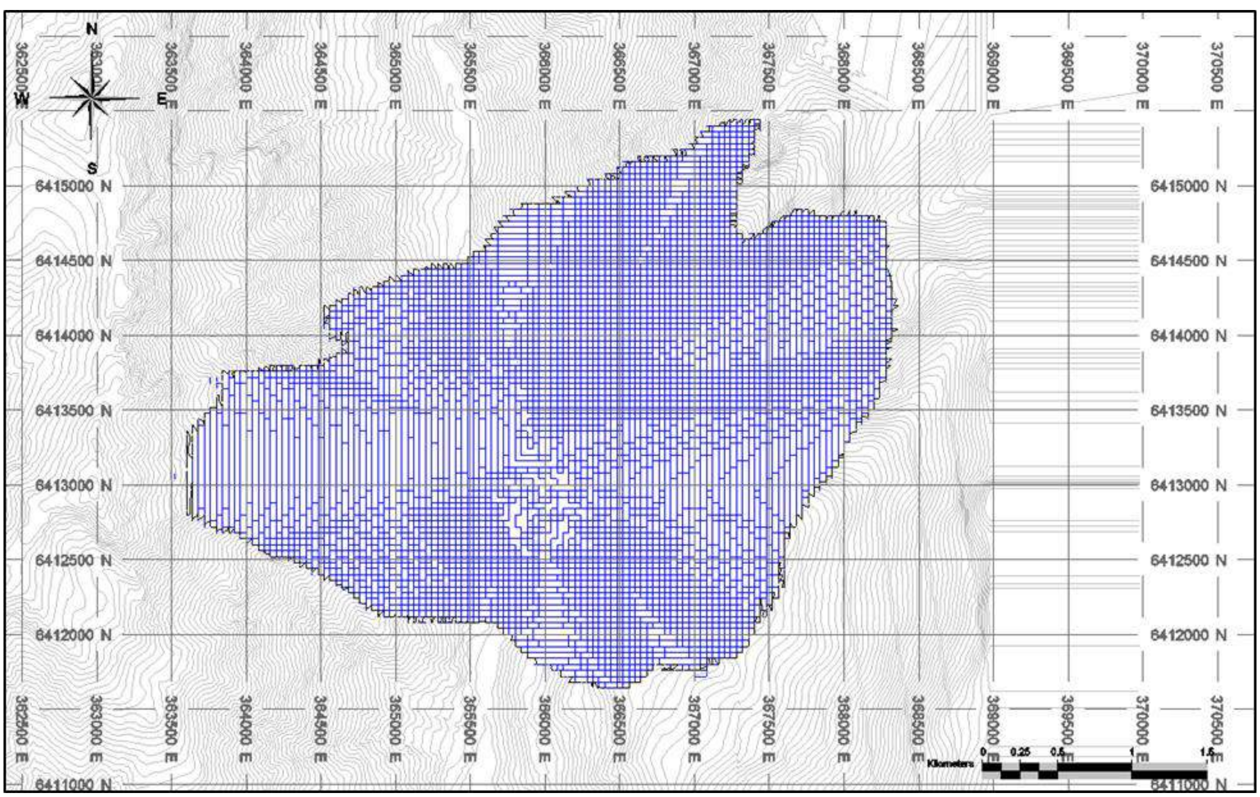
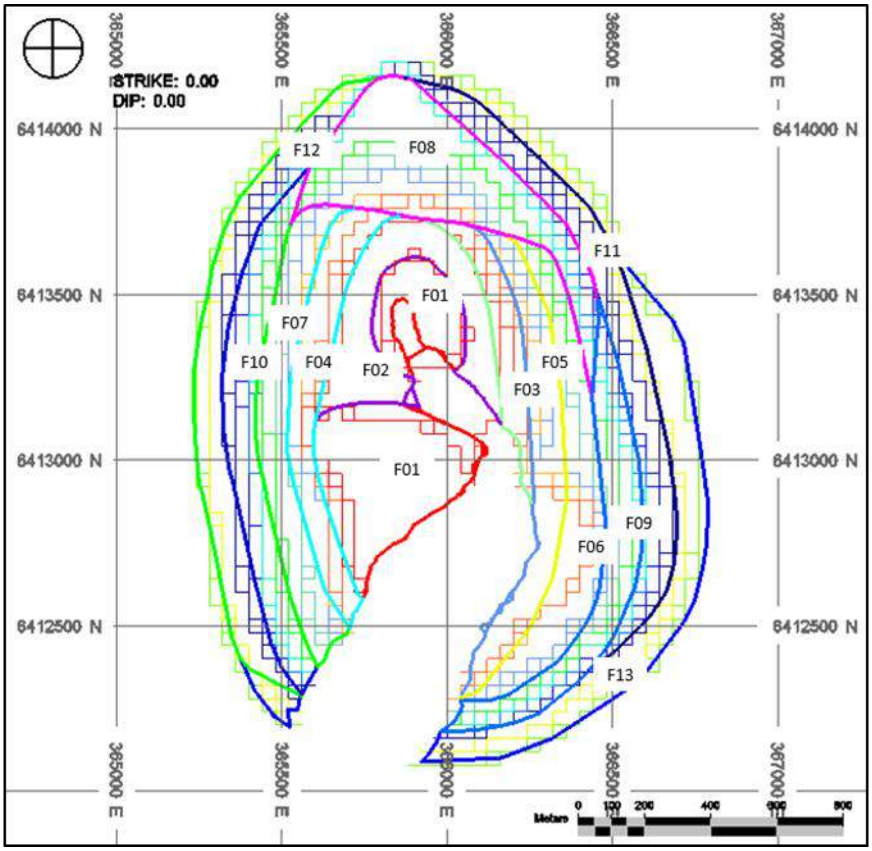


Table 16.4 summarizes the final pit optimization process results based on incremental copper price values, with variations from a baseline price of 1.20 USD/lb to 3.00 USD/lb (Revenue Factor (RF) 0.40 - 1.00). The table highlights the pits used in the further design phase (green shaded). These pits are shown in Figure 16.6 with the relevant design phases.

Table 16.4: Summary Final Pit Optimization for 55 ktpd and 110 ktpd Cases

Pit	Revenue Factor	Copper Price	Total Rock (kt)	Ore (kt)	Strip Ratio	CU (%)	MO (ppm)
1	0.40	1.20	229	157	0.46	0.56	31.55
2	0.41	1.23	11,229	3,387	2.31	0.76	114.42
3	0.42	1.26	14,382	4,705	2.06	0.71	118.06
4	0.43	1.29	76,486	33,329	1.29	0.57	129.09
5	0.44	1.32	134,669	57,682	1.33	0.56	122.06
6	0.45	1.35	206,571	95,636	1.16	0.53	127.67
7	0.46	1.38	227,059	106,752	1.13	0.52	126.28
8	0.47	1.41	271,300	131,564	1.06	0.51	123.28
9	0.48	1.44	304,071	148,802	1.04	0.50	121.94
10	0.49	1.47	384,117	190,772	1.01	0.48	118.00
11	0.50	1.50	440,130	226,098	0.95	0.47	117.38
12	0.51	1.53	501,556	263,343	0.90	0.45	117.19
13	0.52	1.56	623,469	335,004	0.86	0.43	115.63
14	0.53	1.59	701,892	378,274	0.86	0.43	115.48
15	0.54	1.62	769,083	418,738	0.84	0.42	113.98
16	0.55	1.65	859,574	475,576	0.81	0.41	111.07
17	0.56	1.68	928,677	517,708	0.79	0.40	110.69
18	0.57	1.71	1,008,614	565,039	0.79	0.39	110.62
19	0.58	1.74	1,118,382	625,035	0.79	0.39	109.61
20	0.59	1.77	1,193,229	661,548	0.80	0.38	110.00
21	0.60	1.80	1,316,847	731,192	0.80	0.37	109.24
22	0.61	1.83	1,551,709	838,726	0.85	0.37	109.49
23	0.62	1.86	1,694,302	910,975	0.86	0.36	109.03
24	0.63	1.89	1,845,561	985,905	0.87	0.36	108.88
25	0.64	1.92	1,960,063	1,040,755	0.88	0.35	109.08
26	0.65	1.95	2,145,262	1,122,713	0.91	0.35	109.46
27	0.66	1.98	2,292,806	1,186,470	0.93	0.35	109.28
28	0.67	2.01	2,624,990	1,327,282	0.98	0.34	110.06
29	0.68	2.04	2,754,020	1,381,553	0.99	0.34	109.86
30	0.69	2.07	3,052,068	1,492,794	1.04	0.33	110.57
31	0.70	2.10	3,202,654	1,556,521	1.06	0.33	110.33
32	0.71	2.13	3,391,690	1,630,021	1.08	0.33	110.58
33	0.72	2.16	3,573,797	1,707,096	1.09	0.33	109.22
34	0.73	2.19	3,914,734	1,829,750	1.14	0.32	109.55
35	0.74	2.22	4,105,115	1,902,512	1.16	0.32	109.13
36	0.75	2.25	4,325,005	1,978,591	1.19	0.32	108.69
37	0.76	2.28	4,702,375	2,094,267	1.25	0.32	109.58
38	0.77	2.31	4,881,835	2,171,861	1.25	0.31	108.38
39	0.78	2.34	4,989,569	2,211,222	1.26	0.31	108.38
40	0.79	2.37	5,157,541	2,272,924	1.27	0.31	107.88
41	0.80	2.40	5,410,176	2,356,381	1.30	0.31	107.38
42	0.81	2.43	5,657,006	2,439,086	1.32	0.31	106.95
43	0.82	2.46	5,952,005	2,542,809	1.34	0.30	106.64
44	0.83	2.49	6,049,298	2,575,435	1.35	0.30	106.32
45	0.84	2.52	6,283,559	2,642,507	1.38	0.30	106.85
46	0.85	2.55	6,556,040	2,739,398	1.39	0.30	105.56
47	0.86	2.58	7,481,660	2,964,465	1.52	0.30	110.58
48	0.87	2.61	7,606,200	3,026,670	1.51	0.29	109.73
49	0.88	2.64	7,724,189	3,068,817	1.52	0.29	109.92
50	0.89	2.67	7,900,710	3,118,259	1.53	0.29	110.43
51	0.90	2.70	7,976,640	3,150,391	1.53	0.29	109.91
52	0.91	2.73	8,176,129	3,210,953	1.55	0.29	109.99
53	0.92	2.76	8,195,788	3,234,335	1.53	0.29	109.51
54	0.93	2.79	8,413,949	3,306,586	1.54	0.29	109.39
55	0.94	2.82	8,645,216	3,380,031	1.56	0.29	109.53
56	0.95	2.85	8,695,116	3,403,575	1.55	0.28	109.19
57	0.96	2.88	8,795,972	3,438,763	1.56	0.28	109.13
58	0.97	2.91	8,864,997	3,465,052	1.56	0.28	109.42
59	0.98	2.94	9,069,006	3,520,124	1.58	0.28	109.81
60	0.99	2.97	9,091,105	3,543,421	1.57	0.28	109.51
61	1.00	3.00	9,355,266	3,633,852	1.57	0.28	108.96

Figure 16.6: Nested Pits and Phase Design for 55 ktpd and 110 ktpd Cases



16.6.1.2 200 ktpd Case

Figure 16.7 shows the final economic pit limit, which corresponds to the final pit optimization process for the 200 ktpd case obtained using the parameters described above.

With this final limit, mineral extraction amounts to 3,810 Mt at a 0.27% Cu grade and 9,780 Mt of rock.

Figure 16.7: Final Whittle Pit for 200 ktpd Case

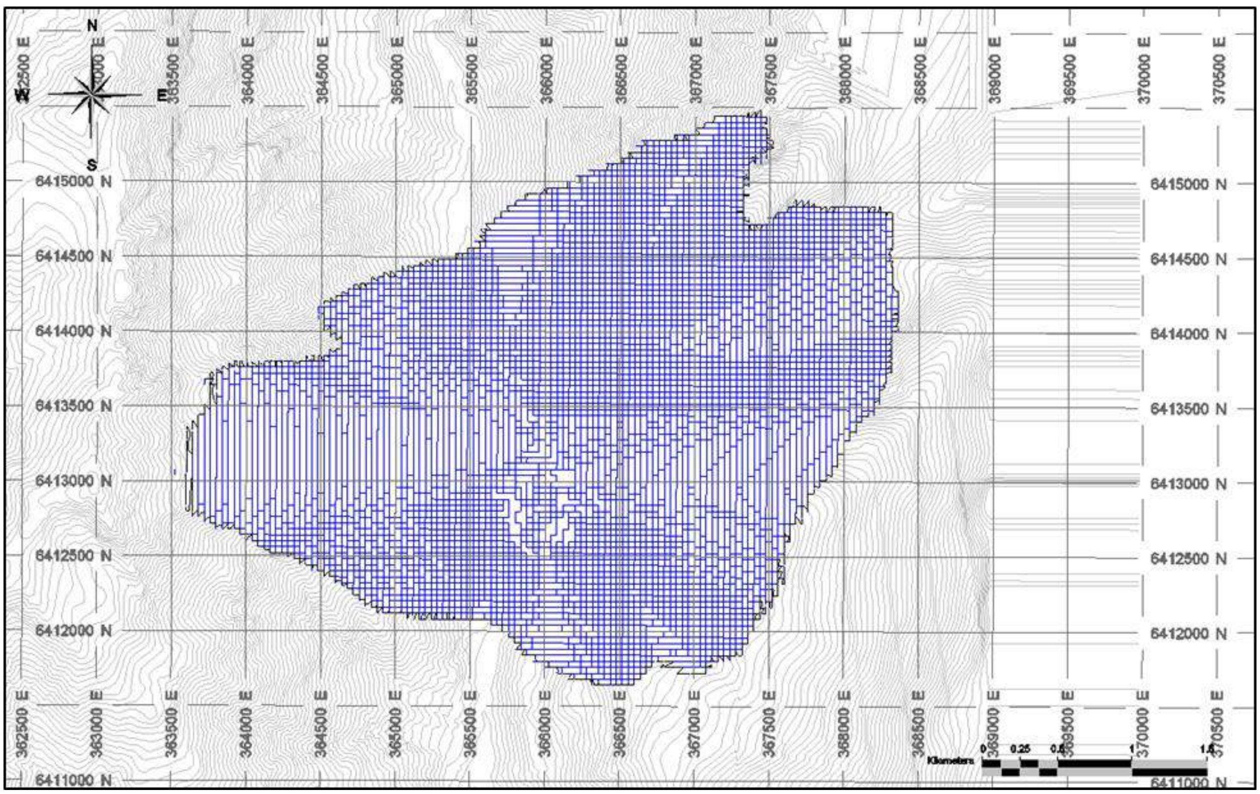
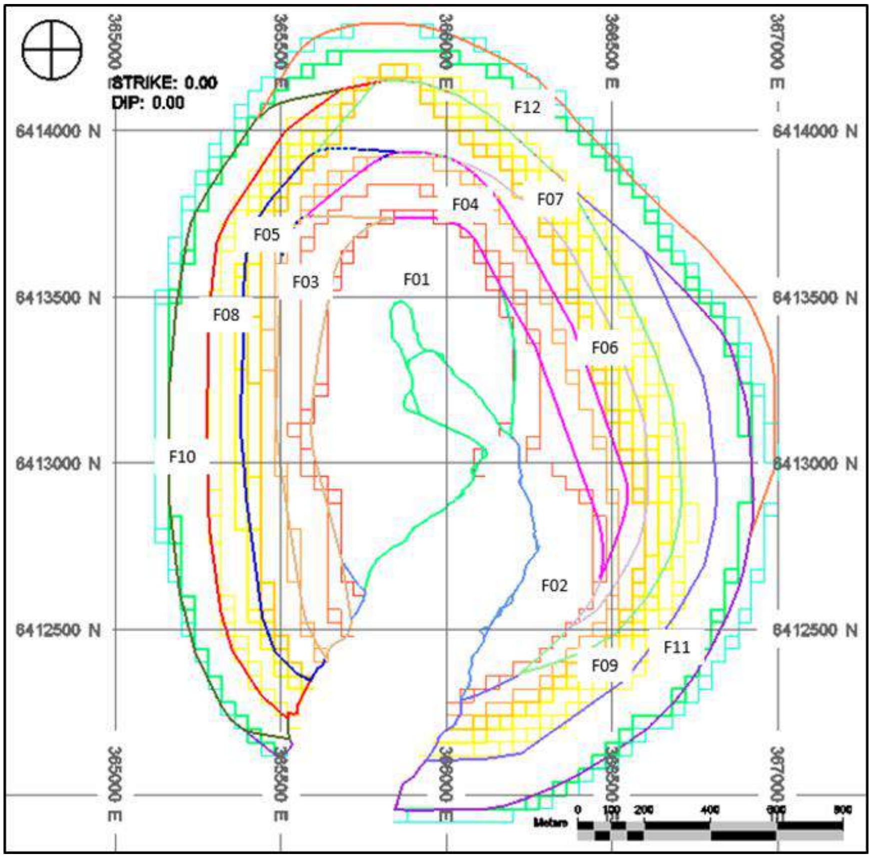


Table 16.5 summarizes the final pit optimization process results obtained for each nested pit by applying incremental factors to copper prices and highlighting the pits used in the further phase design stage. The highlighted pits are also shown in Figure 16.8 with the relevant designed phases.

Table 16.5: Summary Final Pit Optimization for 200 ktpd Case

Pit	Revenue Factor	Copper Price	Total Rock (kt)	Ore (kt)	Strip Ratio	CU (%)	MO (ppm)
1	0.39	1.17	229	157	0.46	0.56	31.55
2	0.40	1.20	11,578	3,529	2.28	0.75	113.30
3	0.41	1.23	17,486	6,017	1.91	0.69	115.08
4	0.42	1.26	77,575	33,917	1.29	0.57	129.12
5	0.43	1.29	141,159	61,065	1.31	0.56	122.57
6	0.44	1.32	210,395	97,909	1.15	0.53	127.85
7	0.45	1.35	252,047	119,954	1.10	0.51	125.14
8	0.46	1.38	284,504	136,964	1.08	0.51	122.52
9	0.47	1.41	361,854	176,515	1.05	0.49	119.98
10	0.48	1.44	419,367	211,491	0.98	0.47	117.55
11	0.49	1.47	477,872	248,618	0.92	0.46	117.47
12	0.50	1.50	595,718	317,877	0.87	0.44	115.24
13	0.51	1.53	652,992	352,430	0.85	0.43	115.63
14	0.52	1.56	754,292	407,367	0.85	0.42	114.91
15	0.53	1.59	832,623	455,551	0.83	0.41	112.28
16	0.54	1.62	915,891	506,018	0.81	0.40	111.27
17	0.55	1.65	994,745	555,113	0.79	0.39	110.55
18	0.56	1.68	1,104,343	614,458	0.80	0.39	109.79
19	0.57	1.71	1,193,106	659,719	0.81	0.38	110.18
20	0.58	1.74	1,316,806	729,330	0.81	0.38	109.41
21	0.59	1.77	1,552,856	837,724	0.85	0.37	109.61
22	0.60	1.80	1,694,302	908,748	0.86	0.36	109.20
23	0.61	1.83	1,856,493	989,239	0.88	0.36	109.26
24	0.62	1.86	1,962,109	1,040,807	0.89	0.35	109.09
25	0.63	1.89	2,145,595	1,122,039	0.91	0.35	109.50
26	0.64	1.92	2,434,178	1,239,873	0.96	0.34	109.80
27	0.65	1.95	2,625,248	1,327,343	0.98	0.34	110.06
28	0.66	1.98	2,834,801	1,410,349	1.01	0.34	110.20
29	0.67	2.01	3,108,151	1,514,954	1.05	0.33	110.27
30	0.68	2.04	3,390,772	1,621,494	1.09	0.33	110.93
31	0.69	2.07	3,444,633	1,650,817	1.09	0.33	110.17
32	0.70	2.10	3,914,486	1,817,284	1.15	0.32	110.14
33	0.71	2.13	4,101,172	1,895,486	1.16	0.32	109.32
34	0.72	2.16	4,169,807	1,924,449	1.17	0.32	108.92
35	0.73	2.19	4,559,868	2,049,344	1.23	0.32	109.15
36	0.74	2.22	4,871,612	2,156,158	1.26	0.31	108.96
37	0.75	2.25	4,989,569	2,208,774	1.26	0.31	108.44
38	0.76	2.28	5,157,541	2,270,006	1.27	0.31	107.96
39	0.77	2.31	5,410,176	2,351,375	1.30	0.31	107.54
40	0.78	2.34	5,657,006	2,432,368	1.33	0.31	107.19
41	0.79	2.37	5,952,005	2,532,716	1.35	0.31	107.01
42	0.80	2.40	6,153,030	2,597,147	1.37	0.30	106.83
43	0.81	2.43	6,283,559	2,639,991	1.38	0.30	106.91
44	0.82	2.46	6,814,796	2,795,140	1.44	0.30	107.23
45	0.83	2.49	7,500,398	2,963,317	1.53	0.30	110.69
46	0.84	2.52	7,719,482	3,042,688	1.54	0.30	110.62
47	0.85	2.55	7,886,182	3,102,266	1.54	0.29	110.68
48	0.86	2.58	7,917,857	3,120,518	1.54	0.29	110.45
49	0.87	2.61	8,115,954	3,182,391	1.55	0.29	109.91
50	0.88	2.64	8,177,161	3,210,078	1.55	0.29	110.03
51	0.89	2.67	8,388,779	3,271,166	1.56	0.29	110.22
52	0.90	2.70	8,459,690	3,316,545	1.55	0.29	109.82
53	0.91	2.73	8,694,709	3,395,998	1.56	0.29	109.31
54	0.92	2.76	8,747,012	3,416,074	1.56	0.28	109.39
55	0.93	2.79	8,864,997	3,456,433	1.56	0.28	109.59
56	0.94	2.82	9,069,006	3,508,989	1.58	0.28	110.05
57	0.95	2.85	9,091,105	3,529,118	1.58	0.28	109.84
58	0.96	2.88	9,294,512	3,593,271	1.59	0.28	109.79
59	0.97	2.91	9,522,091	3,679,519	1.59	0.28	109.36
60	0.98	2.94	9,529,017	3,736,787	1.55	0.27	107.91
61	0.99	2.97	9,735,969	3,790,265	1.57	0.27	108.66
62	1.00	3.00	9,780,487	3,809,581	1.57	0.27	108.62

Figure 16.8: Nested Pits and Phase Design for 200-ktpd Case



16.7 Mine Design

The study considered two sets of mine phase designs. The first set is suitable for 55 ktpd and 110 ktpd mining cases (Set 1), while the second set is for the 200 ktpd mining design (Set 2).

16.7.1 Operating Criteria

The minimum operating widths of the phase design in the different loading cases during bench mining are:

- In production blasting with control strip
  - 73 yd³ rope shovel with 330 st trucks loading from both sides: 90 m
  - 73 yd³ rope shovel with 330 st trucks loading from one side: 75 m
- In bench closing blasting
  - PC hydraulic shovel with 330 st trucks loading from one side: 40 m

The next three figures (Figure 16.9, Figure 16.10 and Figure 16.11) show the material loading scheme for the operating widths identified for engineering development.

Figure 16.9: 73 yd³ Shovel Loading from Both Sides + Wall Control Strip

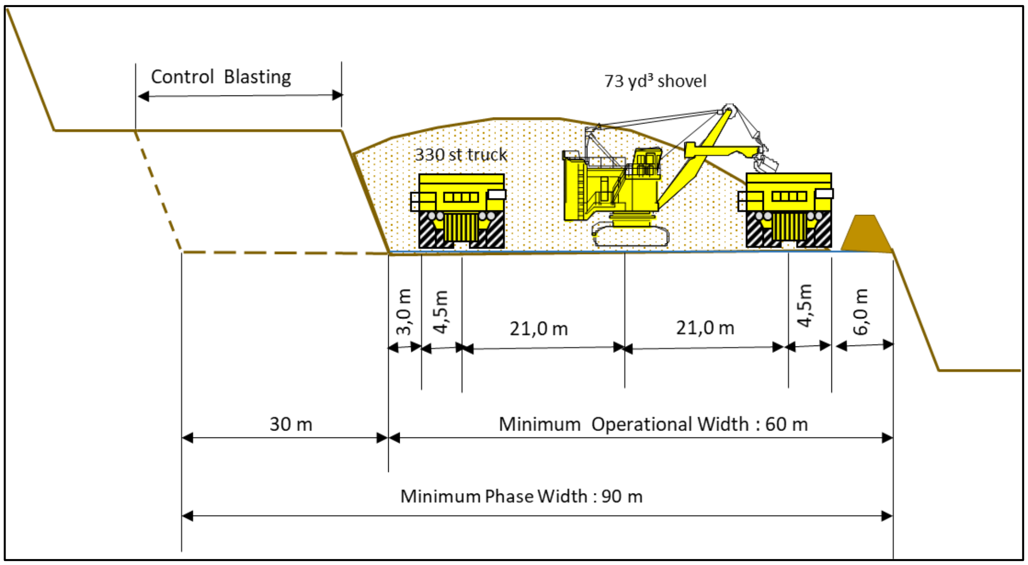


Figure 16.10: 73 yd³ Shovel Loading from One Side + Wall Control Strip

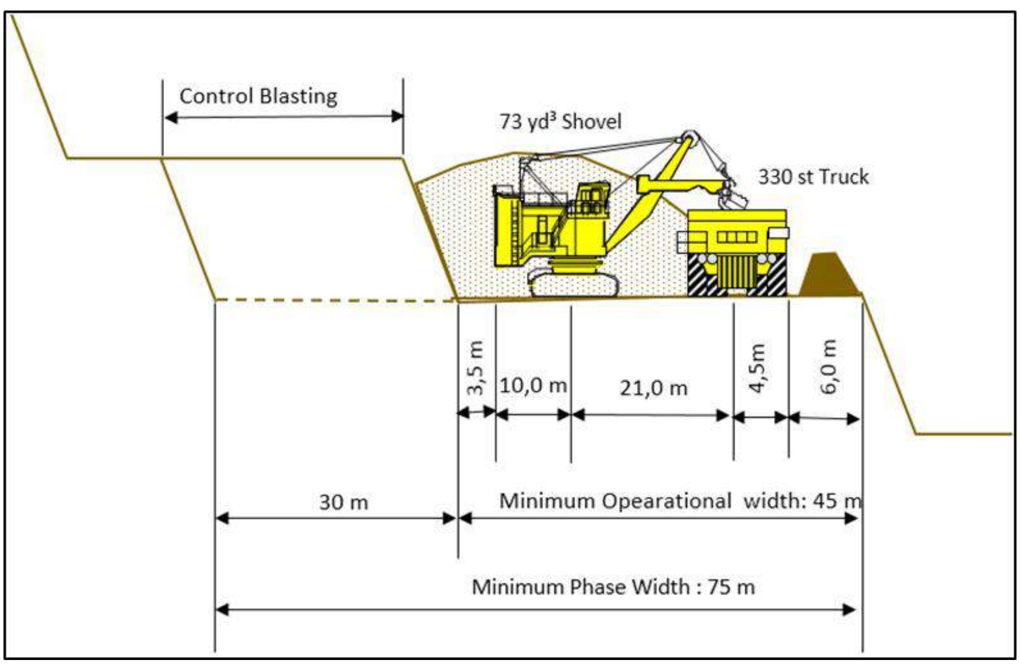
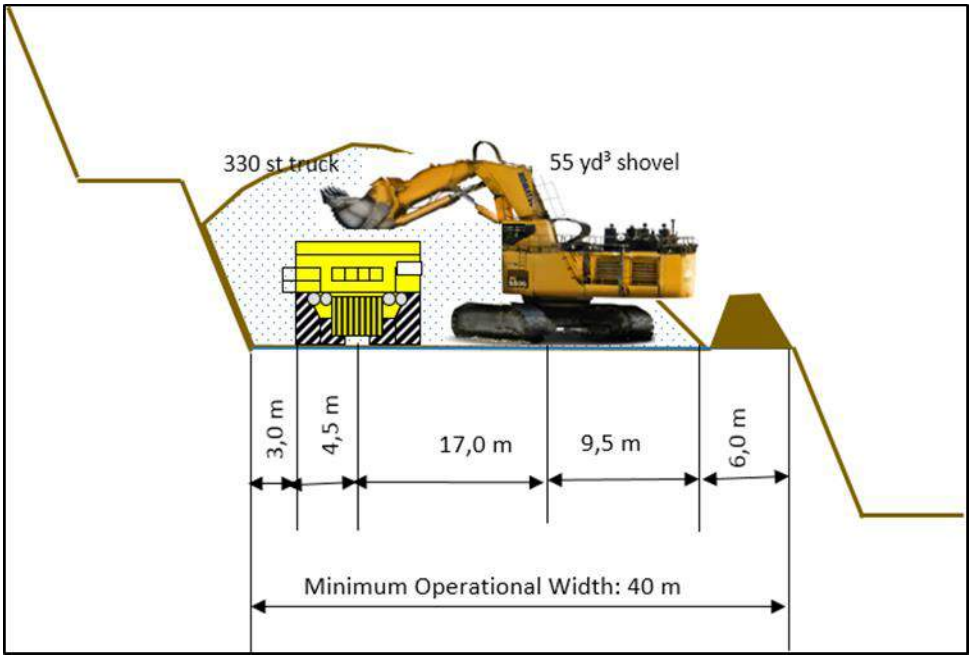


Figure 16.11: Hydraulic Shovel Loading from One Side (Bench End)



16.7.2 General Mine Design Criteria

16.7.2.1 Set 1

Set 1 is supported by the following design criteria:

Non-operating phase designs are considered (smoothed geometries) using the global angles defined in Figure 16.1.

Minimum operating widths under production are 75 m (Figure 16.10).

Minimum operating widths under bench closures are 40 m (Figure 16.11).

According to the widths defined for the phases of this set and considering the depth at which each phase is developed from the upper to the lower part of pit, the amount of mineralization contained (with a cut-off grade ≥ 0.18% Cu) in each phase is between 3 years and 5 years of mineral sent to mill.

Benches located in high summits may have lower widths, involving the use of smaller capacity equipment to facilitate their development (Caterpillar tractors, front-end loaders, and motor graders, among others).

Each of the mine plans for the 55 ktpd and 110 ktpd developed from Set 1 assigned the mineral to either the mill feed or to the different stockpiles.

16.7.2.2 Set 2

Set 2 is supported by the following design criteria:

Operating phase designs are considered (non-smoothed geometries), using the global angles defined in Table 16.1.

Minimum operating widths under production are 90 m (Figure 16.9).

Minimum operating widths under bench closures are 40 m (Figure 16.11).

The amount of mineralization contained (with a cut-off grade ≥ 0.15% Cu) in each phase is between 2 years and 4 years of mineral sent to mill.

Benches located in high summits may have lower widths, involving the use of lower capacity equipment to facilitate their development (Caterpillar tractors, front-end loaders and motor graders, among others).

The mine plan for the 200 ktpd developed from Set 2 assigned the mineral to either the mill feed or to the different stockpiles.

16.7.3 Set 1 Phase Design

The phase design consists in developing smoothed envelopes (excluding access ramps), following the extraction sequence defined in chapter 16.10.1.1 and considering the global angles defined in Table 16.1.

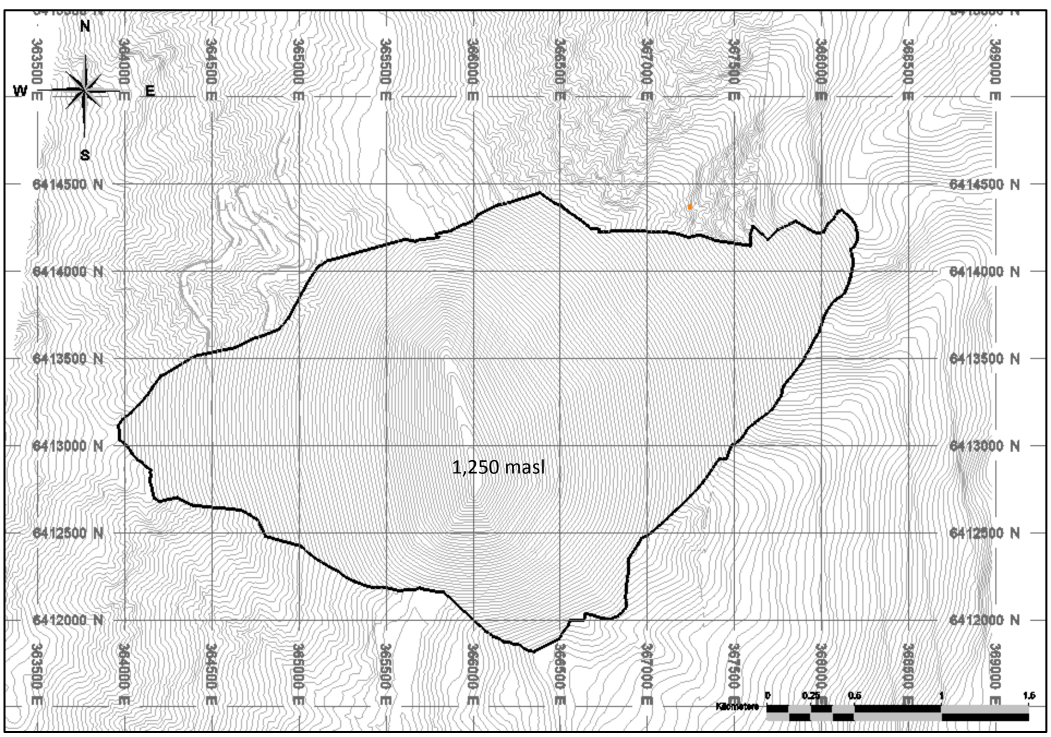
Set 1 contemplates 13 phases totaling 1,654 Mt of mineral with an average Cu grade of 0.33% and 2,170 Mt of waste, using a cut-off grade of ≥ 0.18% of Cu. The results are shown in Table 16.6.

Table 16.6: Phase Take-Off – Set 1

Phases	Mineral (Cu ≥ 0.18 %)				Waste	Total	W/O	IG	Mill at 55 ktpd	Mill at 110 ktpd
	Tonnage kt	Cu Grade %	Mo Grade ppm	CuEq %	kt	kt			Years	Years
Ph1	62,895	0.48	121.09	0.52	67,007	129,902	1.07	0.23	3.13	1.57
Ph2	64,574	0.44	123.97	0.48	43,991	108,565	0.68	0.26	3.22	1.61
Ph3	57,855	0.40	93.23	0.44	41,721	99,576	0.72	0.23	2.88	1.44
Ph4	79,240	0.35	135.10	0.40	54,390	133,629	0.69	0.21	3.95	1.97
Ph5	117,510	0.34	90.23	0.37	87,986	205,496	0.75	0.20	5.85	2.93
Ph6	168,493	0.33	77.37	0.35	119,666	288,159	0.71	0.19	8.39	4.20
Ph7	116,003	0.32	135.02	0.36	115,688	231,691	1.00	0.16	5.78	2.89
Ph8	136,680	0.37	132.55	0.42	250,142	386,822	1.83	0.13	6.81	3.40
Ph9	170,815	0.30	79.65	0.33	117,148	287,963	0.69	0.18	8.51	4.25
Ph10	135,187	0.31	148.56	0.36	204,764	339,952	1.51	0.12	6.73	3.37
Ph11	205,402	0.30	112.16	0.34	361,695	567,098	1.76	0.11	10.23	5.12
Ph12	146,816	0.31	138.18	0.36	321,710	468,526	2.19	0.10	7.31	3.66
Ph13	193,441	0.29	91.16	0.32	384,575	578,016	1.99	0.10	9.64	4.82
Total	**1,654,911**	**0.33**	**111.06**	**0.37**	**2,170,484**	**3,825,395**	**1.31**	**0.14**	**82.44**	**41.22**

The final condition of the mine is shown in Figure 16.12 at an elevation of 1,250 masl.

Figure 16.12: Final Pit – Set 1



16.7.4 Set 2 Phase Design

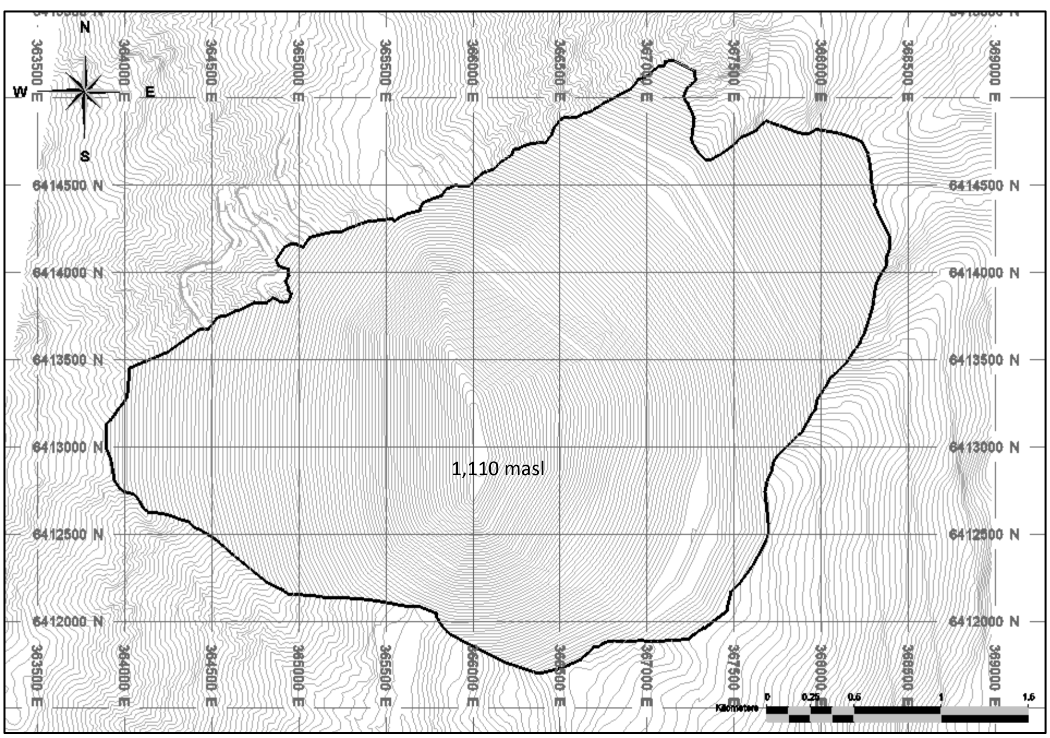
This set contemplates 10 phases, totaling 1,939 Mt of mineral with an average Cu grade of 0.32% and 2,655 Mt of waste, using a cut-off grade ≥ 0.15% of Cu. The results are shown in Table 16.7.

Table 16.7: Phase Take-Off – Set 2

Phases	Mineral (Cu ≥ 0.15%) Tonnage kt	Cu Grade %	Mo Grade ppm	CuEq %	Waste kt	Total kt	W/O	IG	Mill at 200 ktpd Years
Ph1	159,197	0.45	117.83	0.49	125,718	284,915	0.79	0.25	3.97
Ph2	175,805	0.35	81.17	0.37	138,266	314,071	0.79	0.19	4.38
Ph3	140,012	0.32	123.73	0.36	89,991	230,003	0.64	0.19	3.49
Ph4	225,933	0.33	89.60	0.36	217,126	443,060	0.96	0.17	5.63
Ph5	142,981	0.30	121.51	0.34	174,596	317,577	1.22	0.13	3.56
Ph6	189,788	0.30	100.61	0.33	220,556	410,345	1.16	0.14	4.73
Ph7	268,830	0.30	110.37	0.33	370,575	639,405	1.38	0.13	6.70
Ph8	192,330	0.29	129.81	0.34	332,138	524,467	1.73	0.11	4.79
Ph9	272,265	0.28	88.58	0.31	461,227	733,492	1.69	0.11	6.78
Ph10	171,497	0.29	127.95	0.34	524,401	695,899	3.06	0.07	4.27
Total	**1,938,639**	**0.32**	**107.17**	**0.35**	**2,654,593**	**4,593,232**	**1.37**	**0.13**	**48.28**

The final geometry of the pit as designed is shown in Figure 16.12 at an elevation of 1,110 masl.

Figure 16.13: Final Pit – Set 2



Both phase sets were submitted to a marginal economic analysis to validate the economic contribution of the latest phases. This study comprised the creation of strategic mine plans adding one phase in each plan until reaching the final pit. Finally, the phase set with the maximum economic performance (reference NPV) was selected for economic evaluation.

16.8 Mine Operations

The key aspects of the mining operation are described below.

For each case discussed and using a simplified model, the loading and hauling equipment fleet was estimated based on the mine plans and distances to mill, waste dump and stocks.

Subsequently, the drilling and support equipment fleets were estimated by using empirical ratios based on material movements, number of shovels, number of trucks, number of dumps and stocks, among others.

Once the equipment (fleet) requirements were identified, an equipment increase, replacement and removal schedule were determined, establishing the capital costs and making the relevant economic evaluations.

Equipment performance, speed and prices were taken from Tetra Tech’s database for similar operations, the values of which are shown in Chapters 16.8.2 and 16.8.3.

16.8.1 Drilling

Two types of equipment were identified for drilling activities: Pit Viper 351, 10-5/8” and 12¼” diameter production drillers and ROC L-8, 6½” diameter control (precut) driller.

The driller fleet estimate criteria in terms of the loading equipment required to meet the mine plan is detailed below:

- Production drillers (12¼’’): (No. of electric shovels + No. of hydraulic shovels) x 1.5
- Pre-cut drillers (No. of production drillers) / 2

16.8.2 Load and Haul

For loading and hauling under both options studied, electric shovels, hydraulic shovels, front-end loaders, and extraction trucks suitable for each loading equipment were considered.

The performance of these equipment units corresponds to low to medium altitude operations, so at this stage no equipment de-rating factors for altitude conditions were applied.

Material loading and hauling equipment are detailed below:

- Loading:
  - 73 yd³ electric shovel
  - 56 yd³ hydraulic shovel
  - 31 yd³ front-end loader
- Hauling
  - Truck (330 st)

Table 16.8: Electric Shovel Parameters

Electric Shovel	Unit	Value
Bucket Capacity	yd³	73
Fill Factor	%	95
Physical Availability	%	90
Utilization	%	60
Cycle time/pass	min	0.66
N° Pass	pass	3

Table 16.9: Hydraulic Shovel Parameters

Hydraulic Shovel	Unit	Value
Bucket Capacity	yd³	56
Fill Factor	%	95
Physical Availability	%	90
Utilization	%	60
Cycle time/pass	min	0.71
N° Pass	pass	4

Table 16.10: Front-End Loader Parameters

Frontal Loader	Unit	Value
Bucket Capacity	yd³	31
Fill Factor	%	95
Physical Availability	%	85
Utilization	%	55
Cycle time/pass	min	0.81
N° Pass	pass	6

Table 16.11: Truck Parameters

Mining Truck	Unit	Value
Hopper Capacity	st	330
Fill Factor	%	95
Velocity		
First 150 m	km/h	20.00
Up Loaded	km/h	11.00
Down Loaded	km/h	24
Up Empty	km/h	29.00
Down Empty	km/h	34
Horizontal Loaded	km/h	40.00
Horizontal Empty	km/h	42
Operation Time		
Positioning	min	0.50
Load Time	min	2.30
Empty Time	min	0.50
Total Op. Time	min	3.30

16.8.3 Ancillary Equipment

Equipment units that directly support the drilling, loading and hauling operations and participate in the mining operations owing to the type of function they perform. These equipment units are Caterpillar tractors, wheel dozers, motor graders, water trucks, and fuel oil tanker trucks.

The criteria used to estimate the main support equipment are:

- Bulldozer: ((No. of electric shovels + No. of hydraulic shovels) / 2+1+ No. of active dumps) / 0.85
- Wheel dozer: No. of bulldozers + 1
- Motor grader: No. of trucks / 10
- Watering truck: No. of trucks / 10
- Fuel oil tanker truck: No. of trucks / 20

16.9 Manpower

In all cases under study, the staffing for management, mine operation, and mine equipment maintenance activities was estimated.

Shovels and drillers are considered critical equipment, so the staffing was estimated based on the number of nominal equipment. For extraction trucks and support equipment, the staffing was estimated based on the number of equipment available. The criteria to estimate the number of mine equipment operators and maintenance staff is detailed in Table 16.12

Table 16.12: 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

Factors of 3.5% and 4.0% were used for absenteeism and vacations respectively. Additionally, an average availability of 83% was used to estimate the extraction truck and support equipment requirements.

16.10 Mining Plan Options

A long-term mine plan study was conducted, which shows the production scenarios defined for the concentrator for each of the mill throughput cases.

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, a condition mitigated with stockpiles.

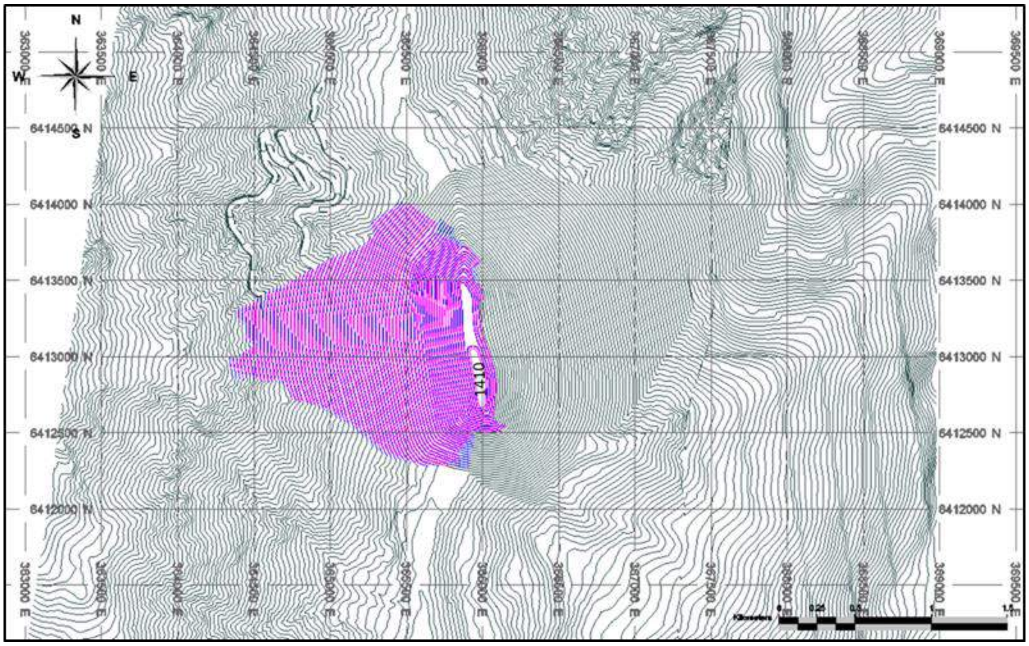
The analysis considered a wide array of mine capacity possibilities and the adjustment of stockpile cut-off grades. Finally, the plans developed in this report show the best economic value.

16.10.1 Case 1: 55 ktpd

16.10.1.1 Final Pit: 55 ktpd

Non-operating phase designs for Set 1 were prepared for the mine plan, based on the sequence defined in the final pit optimization stage. By performing a marginal phase analysis, the final pit was defined in phase 10, as shown in Figure 16.14

Figure 16.14: Final Pit: 55 ktpd (Phase 10)



16.10.1.2 Mine Production Schedule

Table 16.13 and Figure 16.15 summarize the movement of materials in the mine plan for a 55 ktpd plant. To comply with the production schedule the total movement reaches a maximum of 165 ktpd until year 40, when it begins to decline until the depletion of waste and low grade material. In the final periods of the mine plan, the minerals sent to the mill come from re-handled stocks.

The total movement in the 59 years of mine life for the option analyzed amounts to 2,626 Mt (including re-handling) with a total feed of 1,109 Mt to the concentrator.

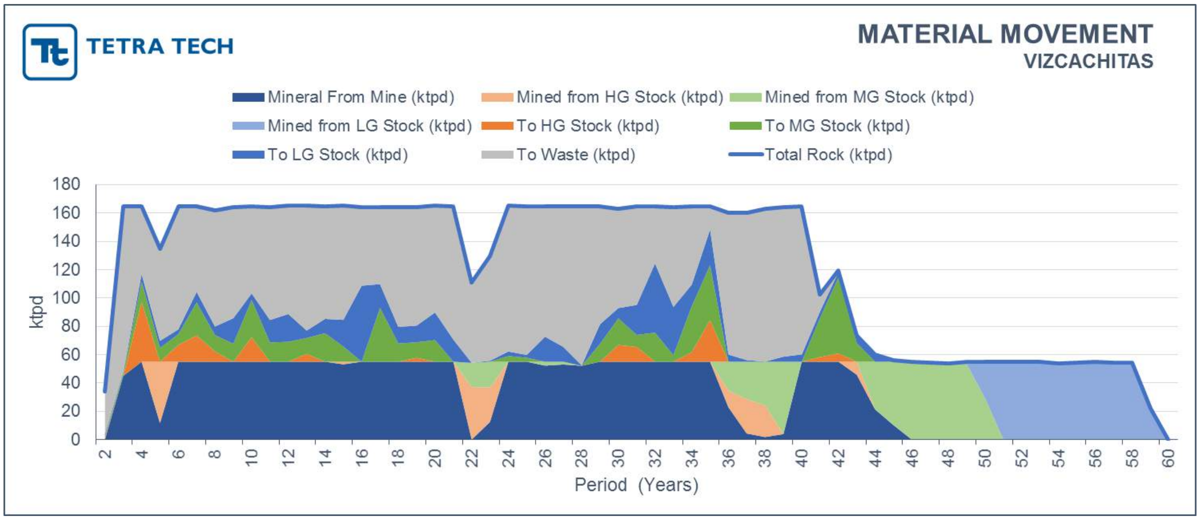
The mineral flow in the mine plan was optimized by separating the mined material into four categories namely:

- Mine to Mill: mineral shipped direct to the mill, with a variable cut-off grade
- HG Stockpile: mineral with grade below direct Mine to Mill and above a cut-off grade of 0.34% CuEq
- MG Stockpile: mineral with grade below HG Stockpile and above a cut-off grade of 0.25% CuEq
- LG Stockpile: mineral with grade below MG Stockpile and above a cut-off grade of 0.18% CuEq

Table 16.13: Production Schedule - 55 ktpd

Vizcachitas Project - 55ktpd													
Period (Years)	Days	COG	Mineral to Mill					Waste	W/O	Total			ktpd
			Total							On site	Rehandling	W/Rehandling	
			kt	CuEq (%)	Cu (%)	Mo (ppm)	Ag (g/t)	kt		kt	kt	kt	
2	360							10,887		12,275		12,275	34
3	360	0.31%	16,200	0.58%	0.54%	94.37	1.52	42,001	2.59	59,263		59,263	165
4	360	0.51%	19,800	0.68%	0.62%	148.97	1.80	17,259	0.87	59,329		59,329	165
5	360	0.30%	19,800	0.43%	0.38%	119.52	1.37	23,310	1.18	32,787	15,612	48,399	134
6	360	0.44%	19,800	0.62%	0.57%	107.48	1.37	31,180	1.57	59,177		59,177	164
7	360	0.41%	19,800	0.54%	0.48%	151.23	1.34	21,749	1.10	59,298.2		59,298	165
8	360	0.38%	19,800	0.61%	0.56%	114.41	1.26	29,468	1.49	58,220		58,220	162
9	360	0.32%	19,800	0.48%	0.43%	124.07	1.31	28,169	1.42	59,021		59,021	164
10	360	0.38%	19,800	0.50%	0.45%	125.32	1.28	22,139	1.12	59,286		59,286	165
11	360	0.30%	19,800	0.43%	0.35%	203.28	1.17	28,742	1.45	59,137		59,137	164
12	360	0.29%	19,800	0.40%	0.35%	119.83	1.00	27,459	1.39	59,377		59,377	165
13	360	0.37%	19,800	0.53%	0.49%	98.42	1.30	31,675	1.60	59,395		59,395	165
14	360	0.33%	19,800	0.47%	0.42%	120.83	1.19	28,488	1.44	59,174		59,174	164
15	360	0.27%	19,800	0.40%	0.34%	137.41	1.10	28,962	1.46	58,737	660	59,397	165
16	360	0.25%	19,800	0.33%	0.30%	67.68	0.79	19,968	1.01	59,086		59,086	164
17	360	0.34%	19,800	0.48%	0.44%	83.98	1.26	19,579	0.99	59,088		59,088	164
18	360	0.33%	19,800	0.45%	0.41%	98.47	1.16	30,474	1.54	59,098		59,098	164
19	360	0.36%	19,800	0.48%	0.44%	109.87	1.16	30,070	1.52	59,060		59,060	164
20	360	0.28%	19,800	0.43%	0.38%	136.57	1.09	27,075	1.37	59,390		59,390	165
21	360	0.23%	19,800	0.36%	0.34%	44.34	0.90	33,824	1.71	59,244		59,244	165
22	360	0.28%	19,507	0.36%	0.32%	92.68	0.99	20,482	1.05	20,482	19,507	39,988	111
23	360	0.21%	19,800	0.36%	0.32%	83.80	1.00	26,633	1.35	31,384	15,297	46,680	130
24	360	0.29%	19,800	0.43%	0.38%	125.72	1.19	36,997	1.87	59,384		59,384	165
25	360	0.26%	19,800	0.38%	0.31%	164.14	1.02	37,710	1.90	59,223		59,223	165
26	360	0.25%	19,800	0.38%	0.30%	208.61	1.03	33,203	1.68	58,330	1,012	59,342	165
27	360	0.24%	19,800	0.36%	0.30%	151.98	0.98	35,633	1.80	58,483	717	59,200	164
28	360	0.19%	18,750	0.34%	0.30%	113.93	0.85	40,417	2.16	59,292		59,292	165
29	360	0.31%	19,800	0.45%	0.41%	105.73	0.85	29,995	1.51	59,333		59,333	165
30	360	0.39%	19,800	0.54%	0.48%	145.30	0.99	25,283	1.28	58,709		58,709	163
31	360	0.42%	19,800	0.58%	0.51%	184.62	1.07	25,022	1.26	59,236		59,236	165
32	360	0.30%	19,800	0.49%	0.41%	231.65	0.96	14,470	0.73	59,260		59,260	165
33	360	0.26%	19,800	0.35%	0.32%	85.36	0.92	25,424	1.28	59,134		59,134	164
34	360	0.35%	19,800	0.45%	0.41%	92.86	1.14	19,897	1.00	59,272		59,272	165
35	360	0.43%	19,800	0.56%	0.50%	125.99	1.26	5,809	0.29	59,230		59,230	165
36	360	0.24%	19,800	0.33%	0.29%	109.67	0.90	35,967	1.82	46,011	11,574	57,585	160
37	360	0.24%	19,800	0.34%	0.30%	97.94	0.90	37,409	1.89	39,370	18,250	57,619	160
38	360	0.25%	19,800	0.33%	0.30%	94.37	0.89	38,862	1.96	39,552	19,144	58,696	163
39	360	0.21%	19,800	0.30%	0.26%	72.91	0.85	37,967	1.92	40,620	18,414	59,034	164
40	360	0.20%	19,800	0.35%	0.32%	51.08	0.77	37,628	1.90	59,266		59,266	165
41	360	0.35%	19,800	0.44%	0.40%	107.80	1.05	4,327	0.22	36,920		36,920	103
42	360	0.35%	19,800	0.46%	0.38%	207.75	1.11	456	0.02	43,035		43,035	120
43	360	0.29%	19,800	0.42%	0.33%	240.59	0.98			23,182	3,314	26,496	74
44	360	0.24%	19,800	0.34%	0.29%	123.54	0.91	194	0.01	9,547	12,134	21,681	60
45	360	0.24%	19,800	0.31%	0.27%	99.14	0.88	157	0.01	3,969	16,116	20,085	56
46	360	0.21%	19,745	0.30%	0.26%	97.51	0.86				19,745	19,745	55
47	360	0.21%	19,549	0.30%	0.26%	97.50	0.86				19,549	19,549	54
48	360	0.28%	19,412	0.30%	0.26%	97.65	0.86				19,412	19,412	54
49	360	0.21%	19,702	0.30%	0.26%	96.38	0.86				19,702	19,702	55
50	360	0.21%	19,749	0.26%	0.23%	76.80	0.80				19,749	19,749	55
51	360	0.21%	19,722	0.22%	0.20%	53.28	0.73				19,722	19,722	55
52	360	0.21%	19,656	0.22%	0.20%	53.28	0.73				19,656	19,656	55
53	360	0.21%	19,693	0.22%	0.20%	53.28	0.73				19,693	19,693	55
54	360	0.21%	19,405	0.22%	0.20%	53.28	0.73				19,405	19,405	54
55	360	0.21%	19,561	0.22%	0.20%	53.28	0.73				19,561	19,561	54
56	360	0.21%	19,694	0.22%	0.20%	53.28	0.73				19,694	19,694	55
57	360	0.21%	19,503	0.22%	0.20%	53.28	0.73				19,503	19,503	54
58	360	0.21%	19,550	0.22%	0.20%	53.28	0.73				19,550	19,550	54
59	360	0.21%	7,854	0.22%	0.20%	53.28	0.73				7,854	7,854	22
Total			1,109,251	0.39%	0.35%	110.73	1.01	1,102,416	0.99	2,211,668	414,543	2,626,210	

Figure 16.15: Production Schedule - 55 ktpd



As shown in the previous figure, the material profile shows a mine capacity decrease in years 5 and 22, which is explained by the economic convenience of introducing higher grade stockpile mineral to replace the mineral coming directly from the mine. This condition is more cost-effective than increasing the waste extraction to meet the need to feed the mill with mineral coming directly from the mine.

The mineral sent to mill is detailed in Figure 16.16

Mine production starts in year three, reaching the design capacity at the beginning of year four. From then on, the concentrator operates at capacity until year 58, decreasing in the last year.

At the end of mine life the mineral is sent to the mill from the remaining stockpiles generated in the mine plan according to the variable cut-off grade strategy applied.

Figure 16.16: Mineral to Mill in 55 ktpd Mine Schedule

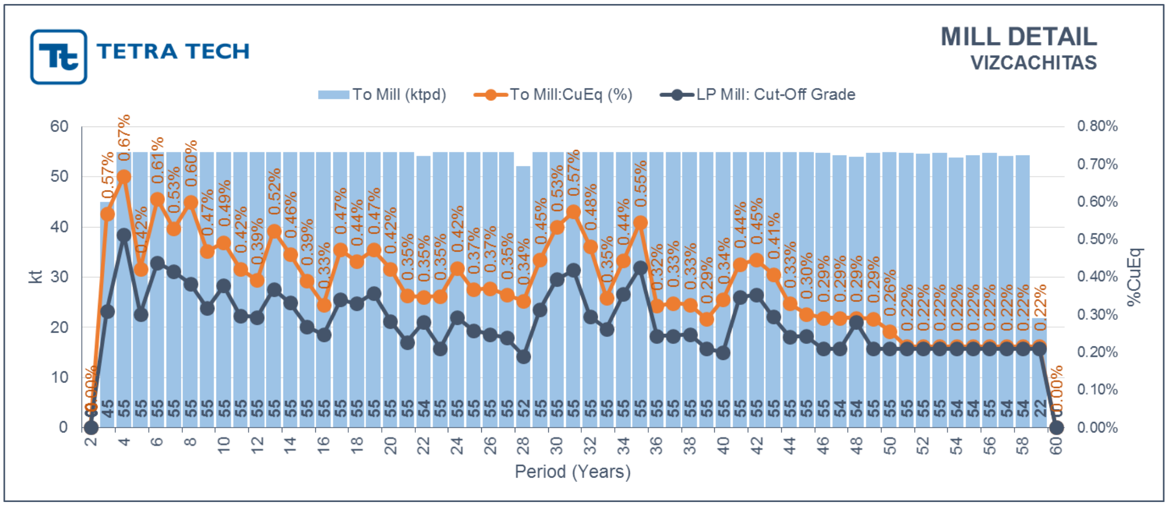
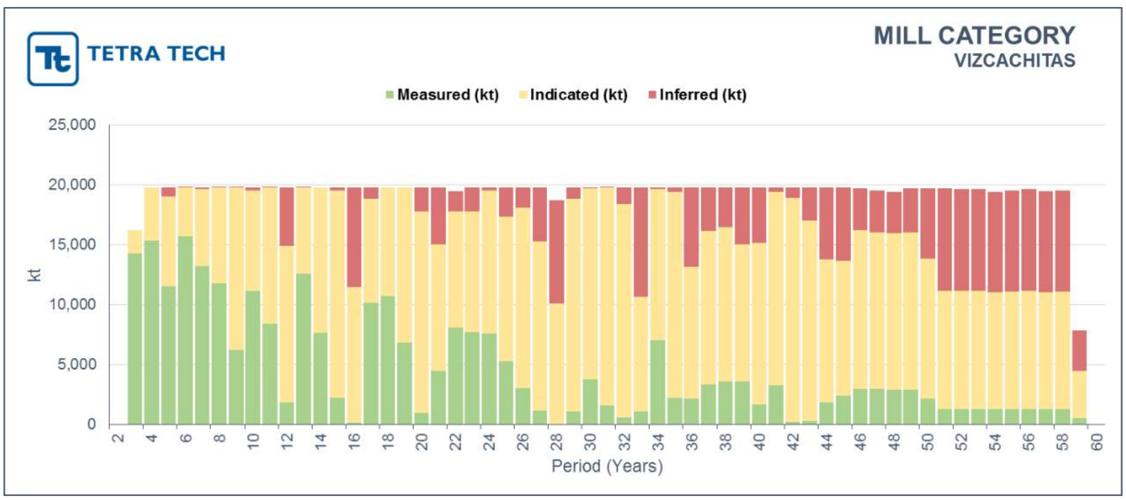


Figure 16.16 shows the detail of mineral fed to the mill according to resource category. Of the total resources, 23.7% are measured, 59.5% are indicated and the remaining 16.8% are inferred.

Figure 16.17: Measured, Indicated and Inferred Resources in 55 ktpd Mine Schedule



16.10.1.3 Equipment Fleet

The main mine equipment considered to meet the production schedule was as follows:

- Production driller (10-5/8” - 12¼’’)
- Electric shovel (73 yd^3)
- Hydraulic shovel (56 yd^3)
- Front-end loader (31 yd^3)

Support equipment is specified below:

- Bulldozer
- Wheel dozer
- Motor grader
- Water truck

The mine equipment units required for the 55 ktpd plan are summarized in Table 16.14.

Table 16.14: Mine Equipment Required in 55 ktpd Mine Plan (1 of 2)

Equipment Required		**Revenue Factor**	**Fleet**																		
			2	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**	**15**	**16**	**17**	**18**	**19**	**20**
Cable Shovel - 73 yd³	un	2		1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Hydraulic Shovel - 56 yd³	un	4		1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Front Loader - 31 yd³	un	4	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Truck - 330 st	un	62	3	12	15	15	15	15	15	15	21	21	21	21	21	21	21	21	21	21	23
Driller - Diesel 12 1/4"	un	18	2	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5
Bulldozer - 890 HP	un	24	3	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5
Wheel Dozer - 853 HP	un	29	4	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6
Motor Grader - 533 HP	un	12	1	2	2	2	2	2	2	2	3	3	3	3	3	3	3	3	3	3	3
Sprinkler Truck - 75.7 m³	un	10	1	2	2	2	2	2	2	2	3	3	3	3	3	3	3	3	3	3	3
Diesel Truck	un	7	1	1	1	1	1	1	1	1	2	2	2	2	2	2	2	2	2	2	2
Front Loader - 5 yd³	un	5	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Service Truck	un	10	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Precut Driller	un	11	1	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3

Equipment Required		**Fleet**																			
		21	**22**	**23**	**24**	**25**	**26**	**27**	**28**	**29**	**30**	**31**	**32**	**33**	**34**	**35**	**36**	**37**	**38**	**39**	**40**
Cable Shovel - 73 yd³	un	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Hydraulic Shovel - 56 yd³	un	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Front loader - 31 yd³	un	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Truck - 330 st	un	23	23	23	23	23	23	23	23	23	23	23	23	23	23	24	24	24	24	24	24
Driller - Diesel 12 1/4"	un	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5
Bulldozer - 890 HP	un	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5
Wheel Dozer - 853 HP	un	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6
Motor Grader - 533 HP	un	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3
Sprinkler Truck - 75.7 m³	un	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3
Diesel Truck	un	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Front Loader - 5 yd³	un	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Service Truck	un	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Precut Driller	un	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3

Table 16.15: Mine Equipment Required in 55 ktpd Mine Plan (2 of 2)

Equipment Required		Fleet																		
		41	42	43	44	45	46	47	48	49	50	51	52	53	54	55	56	57	58	59
Cable Shovel - 73 yd³	un	1	1	1																
Hydraulic Shovel - 56 yd³	un	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Front Loader - 31 yd³	un	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Truck - 330 st	un	18	18	18	18	18	18	6	6	6	2	2	2	2	2	2	2	2	2	2
Driller - Diesel 12 1/4"	un	5	5	5	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3
Bulldozer - 890 HP	un	5	5	5	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Wheel Dozer - 853 HP	un	6	6	6	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5
Motor Grader - 533 HP	un	2	2	2	2	2	2	1	1	1	1	1	1	1	1	1	1	1	1	1
Sprinkler Truck - 75.7 m³	un	2	2	2	2	2	2	1	1	1	1	1	1	1	1	1	1	1	1	1
Diesel Truck	un	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Front Loader - 5 yd³	un	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Service Truck	un	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Precut Driller	un	3	3	3	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2

16.10.1.4 Manpower

The staffing estimated for the mine plan with a 55 ktpd plant capacity is summarized in Table 16.16 through Table 16.18.

Table 16.16: Supervisory Staffing in 55 ktpd Mine Plan

Supervision	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**	**15**	**16**	**17**	**18**	**19**	**20**	**21**
Administration	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6
Mine Superintendent	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Maintenance Superintendent	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Management	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Geology	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5
Geologist	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Assistant	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Topography	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3
Topographer	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Assistant	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Mine Supervisor	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Maintenance Supervision	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Total	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**

Supervision	**22**	**23**	**24**	**25**	**26**	**27**	**28**	**29**	**30**	**31**	**32**	**33**	**34**	**35**	**36**	**37**	**38**	**39**	**40**	**41**
Administration	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6
Mine Superintendent	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Maintenance Superintendent	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Management	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Geology	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5
Geologist	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Assistant	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Topography	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3
Topographer	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Assistant	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Mine Supervisor	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Maintenance Supervision	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Total	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**

Supervision	**42**	**43**	**44**	**45**	**46**	**47**	**48**	**49**	**50**	**51**	**52**	**53**	**54**	**55**	**56**	**57**	**58**	**59**
Administration	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6
Mine Superintendent	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Maintenance Superintendent	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Management	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Geology	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5
Geologist	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Assistant	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Topography	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3
Topographer	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Assistant	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Mine Supervisor	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Maintenance Supervisor	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Total	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**

Table 16.17: Mine Operator Staffing in 55 ktpd Mine Plan

Equipment	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21
Cable Shovel - 73 yd³	0	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5
Hydraulic Shovel - 56 yd³	0	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5
Front Loader - 31 yd³	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5
Truck - 330 st	11	43	54	54	54	54	54	54	75	75	75	75	75	75	75	75	75	75	83	83
Driller - Diesel 12 1/4"	9	22	22	22	22	22	22	22	22	22	22	22	22	22	22	22	22	22	22	22
Bulldozer - 890 HP	11	18	18	18	18	18	18	18	18	18	18	18	18	18	18	18	18	18	18	18
Wheel Dozer - 853 HP	15	22	22	22	22	22	22	22	22	22	22	22	22	22	22	22	22	22	22	22
Motor Grader - 533 HP	4	8	8	8	8	8	8	8	11	11	11	11	11	11	11	11	11	11	11	11
Sprinkler Truck - 75.7 m³	4	8	8	8	8	8	8	8	11	11	11	11	11	11	11	11	11	11	11	11
Diesel Truck	4	4	4	4	4	4	4	4	8	8	8	8	8	8	8	8	8	8	8	8
Front Loader - 5 yd³	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Service Truck	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8
Precut Driller	5	13	13	13	13	13	13	13	13	13	13	13	13	13	13	13	13	13	13	13
Total	**80**	**165**	**176**	**176**	**176**	**176**	**176**	**176**	**207**	**207**	**207**	**207**	**207**	**207**	**207**	**207**	**207**	**207**	**215**	**215**

Equipment	22	23	24	25	26	27	28	29	30	31	32	33	34	35	36	37	38	39	40	41
Cable Shovel - 73 yd³	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5
Hydraulic Shovel - 56 yd³	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5
Front Loader - 31 yd³	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5
Truck - 330 st	83	83	83	83	83	83	83	83	83	83	83	83	83	86	86	86	86	86	86	65
Driller - Diesel 12 1/4"	22	22	22	22	22	22	22	22	22	22	22	22	22	22	22	22	22	22	22	22
Bulldozer - 890 HP	18	18	18	18	18	18	18	18	18	18	18	18	18	18	18	18	18	18	18	18
Wheel Dozer - 853 HP	22	22	22	22	22	22	22	22	22	22	22	22	22	22	22	22	22	22	22	22
Motor Grader - 533 HP	11	11	11	11	11	11	11	11	11	11	11	11	11	11	11	11	11	11	11	8
Sprinkler Truck - 75.7 m³	11	11	11	11	11	11	11	11	11	11	11	11	11	11	11	11	11	11	11	8
Diesel Truck	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	4
Front Loader - 5 yd³	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Service Truck	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8
Precut Driller	13	13	13	13	13	13	13	13	13	13	13	13	13	13	13	13	13	13	13	13
Total	**215**	**215**	**215**	**215**	**215**	**215**	**215**	**215**	**215**	**215**	**215**	**215**	**215**	**218**	**218**	**218**	**218**	**218**	**218**	**187**

Equipment	42	43	44	45	46	47	48	49	50	51	52	53	54	55	56	57	58	59
Cable Shovel - 73 yd³	5	5	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Hydraulic Shovel - 56 yd³	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5
Front Loader - 31 yd³	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5
Truck - 330 st	65	65	65	65	65	22	22	22	8	8	8	8	8	8	8	8	8	8
Driller - Diesel 12 1/4"	22	22	13	13	13	13	13	13	13	13	13	13	13	13	13	13	13	13
Bulldozer - 890 HP	18	18	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15
Wheel Dozer - 853 HP	22	22	18	18	18	18	18	18	18	18	18	18	18	18	18	18	18	18
Motor Grader - 533 HP	8	8	8	8	8	4	4	4	4	4	4	4	4	4	4	4	4	4
Sprinkler Truck - 75.7 m³	8	8	8	8	8	4	4	4	4	4	4	4	4	4	4	4	4	4
Diesel Truck	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Front Loader - 5 yd³	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Service Truck	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8
Precut Driller	13	13	9	9	9	9	9	9	9	9	9	9	9	9	9	9	9	9
Total	**187**	**187**	**162**	**162**	**162**	**111**	**111**	**111**	**97**	**97**	**97**	**97**	**97**	**97**	**97**	**97**	**97**	**97**

Table 16.18: Mine Equipment Maintenance Staffing in 55 ktpd Mine Plan

Maintenance	**Period**																			
Mine Equipment Maintenance	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**	**15**	**16**	**17**	**18**	**19**	**20**	**21**
Total	104	215	229	229	229	229	229	229	270	270	270	270	270	270	270	270	270	270	280	280

Mine Equipment Maintenance	**22**	**23**	**24**	**25**	**26**	**27**	**28**	**29**	**30**	**31**	**32**	**33**	**34**	**35**	**36**	**37**	**38**	**39**	**40**	**41**
Total	280	280	280	280	280	280	280	280	280	280	280	280	280	284	284	284	284	284	284	244

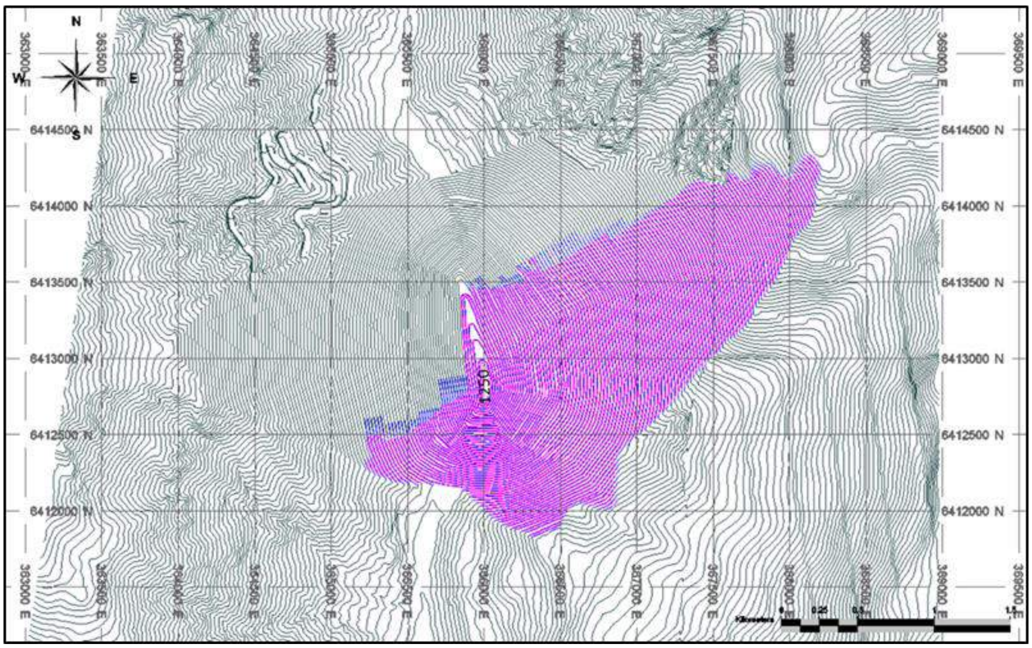
Mine Equipment Maintenance	**42**	**43**	**44**	**45**	**46**	**47**	**48**	**49**	**50**	**51**	**52**	**53**	**54**	**55**	**56**	**57**	**58**	**59**
Total	244	244	211	211	211	145	145	145	127	127	127	127	127	127	127	127	127	127

16.10.2 Case 2: 110 ktpd

16.10.2.1 Final Pit: 110 ktpd

Non-operating phase designs for Set 1 were used for this mine plan. Thus, in conducting a marginal phase analysis the final pit scope was defined in phase 13 as shown in Figure 16.18.

Figure 16.18: Final Pit: 110 ktpd (Phase 13)



16.10.2.2 Mine Production Schedule

Table 16.19 and Figure 16.19 summarize the movement of materials of the mine plan for a 110 ktpd plant. To comply with the production schedule, the total movement reaches a maximum of 310 ktpd until year 31, when it begins to decline. In the final periods of the mine plan, the minerals sent to the mill come from re-handled stockpiles.

The total movement in the 45 years of mine life for the option analyzed amounts to 4,163 Mt (including re-handling), with a total feed of 1,655 Mt to the concentrator.

The mineral flow in the mine plan was optimized by separating the mined material into four categories namely:

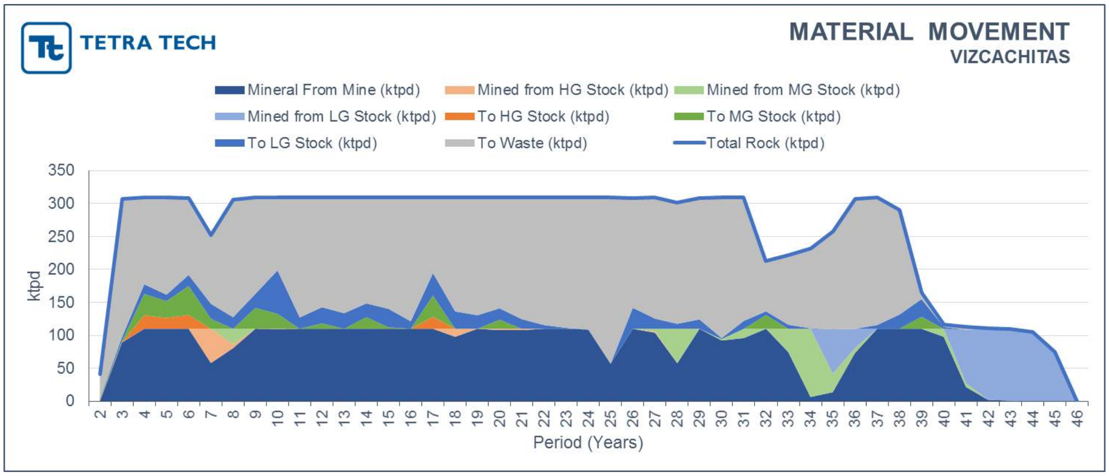
- Mine to Mill: mineral shipped direct to the mill, with a variable cut-off grade
- HG Stockpile: mineral with grade below direct Mine to Mill and above a cut-off grade of 0.34% CuEq

- MG Stockpile: mineral with grade below HG Stockpile and above a cut-off grade of 0.25% CuEq
- LG Stockpile: mineral with grade below MG Stockpile and above a cut-off grade of 0.18% CuEq

Table 16.19: Production Schedule - 110 ktpd

Vizcachitas Project - 110ktpd													
Period (Years)	Days	COG	Mineral to Mill					Waste	W/O	Total			ktpd
			Total							Onsite	Rehandling	W/Rehandling	
			kt	CuEq (%)	Cu (%)	Mo (ppm)	Ag (g/t)	kt		kt	kt	kt	
2	360							13,109		14,629		14,629	41
3	360	0.25%	32,400	0.56%	0.52%	103.45	1.42	75,270	2.32	110,174	204	110,378	307
4	360	0.40%	39,600	0.59%	0.54%	120.90	1.52	47,637	1.20	111,564		111,564	310
5	360	0.38%	39,600	0.57%	0.52%	133.85	1.33	53,146	1.34	111,389		111,389	309
6	360	0.37%	39,600	0.51%	0.45%	124.68	1.28	42,015	1.06	111,040		111,040	308
7	360	0.30%	39,600	0.41%	0.35%	155.64	1.16	37,456	0.95	71,985	18,713	90,698	252
8	360	0.25%	39,600	0.37%	0.33%	103.11	0.95	64,153	1.62	99,513	10,567	110,080	306
9	360	0.32%	39,600	0.49%	0.45%	106.89	1.23	52,791	1.33	111,462		111,462	310
10	360	0.29%	39,600	0.38%	0.34%	102.59	0.98	40,027	1.01	111,336	204	111,540	310
11	360	0.24%	39,600	0.42%	0.39%	79.90	1.12	65,557	1.66	111,374		111,374	309
12	360	0.28%	39,600	0.45%	0.41%	107.24	1.14	59,925	1.51	111,310		111,310	309
13	360	0.24%	39,600	0.40%	0.37%	73.80	1.04	63,232	1.60	111,374		111,374	309
14	360	0.27%	39,600	0.40%	0.33%	155.81	1.08	57,901	1.46	111,380		111,380	309
15	360	0.27%	39,600	0.38%	0.31%	189.45	1.04	60,867	1.54	111,384		111,384	309
16	360	0.23%	39,600	0.40%	0.36%	96.57	0.79	67,701	1.71	111,408		111,408	310
17	360	0.39%	39,600	0.55%	0.49%	166.33	1.03	41,141	1.04	111,211		111,211	309
18	360	0.22%	39,600	0.40%	0.34%	155.36	0.85	62,218	1.57	106,925	4,384	111,309	309
19	360	0.21%	39,600	0.33%	0.30%	73.17	0.91	64,278	1.62	111,274		111,274	309
20	360	0.29%	39,600	0.44%	0.40%	99.59	1.11	60,752	1.53	110,804	731	111,535	310
21	360	0.22%	39,600	0.38%	0.33%	127.77	1.00	66,572	1.68	110,691	731	111,422	310
22	360	0.20%	39,600	0.38%	0.35%	73.66	0.90	69,753	1.76	111,401		111,401	309
23	360	0.22%	39,600	0.37%	0.31%	152.88	1.00	71,181	1.80	111,287		111,287	309
24	360	0.18%	39,145	0.38%	0.30%	237.15	0.92	72,410	1.85	111,555		111,555	310
25	360	0.18%	20,544	0.31%	0.27%	98.11	0.69	90,955	4.43	111,499		111,499	310
26	360	0.21%	39,600	0.28%	0.25%	47.30	0.75	60,054	1.52	111,129		111,129	309
27	360	0.23%	39,600	0.35%	0.31%	91.61	0.90	66,342	1.68	109,365	2,143	111,508	310
28	360	0.24%	39,600	0.35%	0.31%	97.92	0.89	66,322	1.67	89,856	18,807	108,663	302
29	360	0.25%	39,600	0.45%	0.38%	177.79	0.95	66,449	1.68	111,176		111,176	309
30	360	0.19%	34,009	0.39%	0.32%	185.82	0.89	76,968	2.26	110,735	786	111,521	310
31	360	0.20%	39,600	0.35%	0.33%	65.71	0.82	67,326	1.70	106,220	5,068	111,288	309
32	360	0.29%	39,600	0.40%	0.36%	116.33	0.99	27,439	0.69	76,603		76,603	213
33	360	0.25%	39,600	0.35%	0.29%	183.32	0.88	38,027	0.96	67,090	12,737	79,826	222
34	360	0.22%	39,600	0.29%	0.26%	85.65	0.85	43,529	1.10	46,130	37,255	83,385	232
35	360	0.18%	39,320	0.25%	0.22%	76.66	0.77	53,495	1.36	58,439	34,375	92,815	258
36	360	0.18%	39,574	0.30%	0.25%	129.87	0.79	71,098	1.80	97,415	13,257	110,672	307
37	360	0.19%	39,600	0.26%	0.23%	58.05	0.78	69,584	1.76	111,244		111,244	309
38	360	0.21%	39,600	0.32%	0.29%	75.19	0.87	57,054	1.44	104,386		104,386	290
39	360	0.28%	39,600	0.41%	0.36%	133.73	0.98	3,430	0.09	59,239		59,239	165
40	360	0.24%	39,600	0.38%	0.34%	116.42	0.99	1,299	0.03	37,519	4,391	41,910	116
41	360	0.18%	39,102	0.24%	0.21%	68.07	0.75	1,439	0.04	9,234	31,307	40,541	113
42	360	0.19%	39,582	0.22%	0.20%	53.28	0.72	247	0.01	980	38,849	39,829	111
43	360	0.19%	39,359	0.22%	0.19%	52.55	0.72	148	0.00	454	39,053	39,507	110
44	360	0.20%	37,698	0.22%	0.19%	51.85	0.71				37,698	37,698	105
45	360	0.20%	26,957	0.22%	0.19%	51.85	0.71				26,957	26,957	75
Total			**1,654,890**	**0.38%**	**0.33%**	**111.06**	**0.96**	**2,170,295**	**1.31**	**3,825,185**	**338,217**	**4,163,402**	

Figure 16.19: Production Schedule - 110 ktpd



As with the 55 ktpd case, periods of feed to mill from stocks are observed in some years of the plan, specifically in years 7, 28 and between years 32 and 36. On the other hand, in years 25 and 30 the plan does not completely use the mill capacity, which is more cost-effective than increasing the mine movement to meet such goal.

The mineral fed to mill is detailed in Figure 16.20.

Mine production starts in year three, reaching the design capacity at the beginning of year four. From then on, the process capacity is met until year 44, decreasing in the last year of mine life, which is consistent with the depletion of phase 13 and low-grade stock (LG).

Figure 16.20: Mineral to Mill in 110 ktpd Mine Schedule

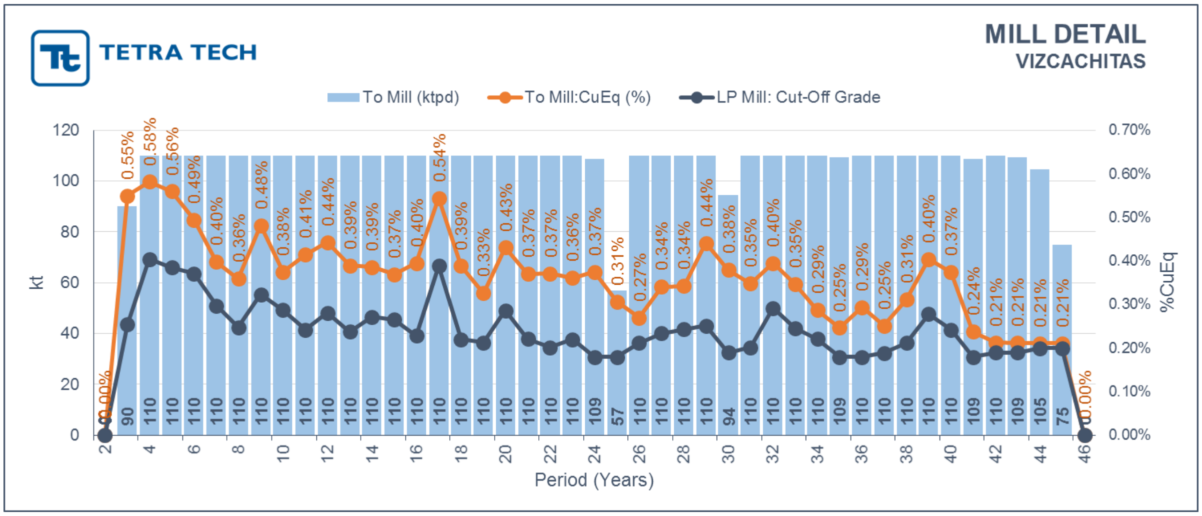
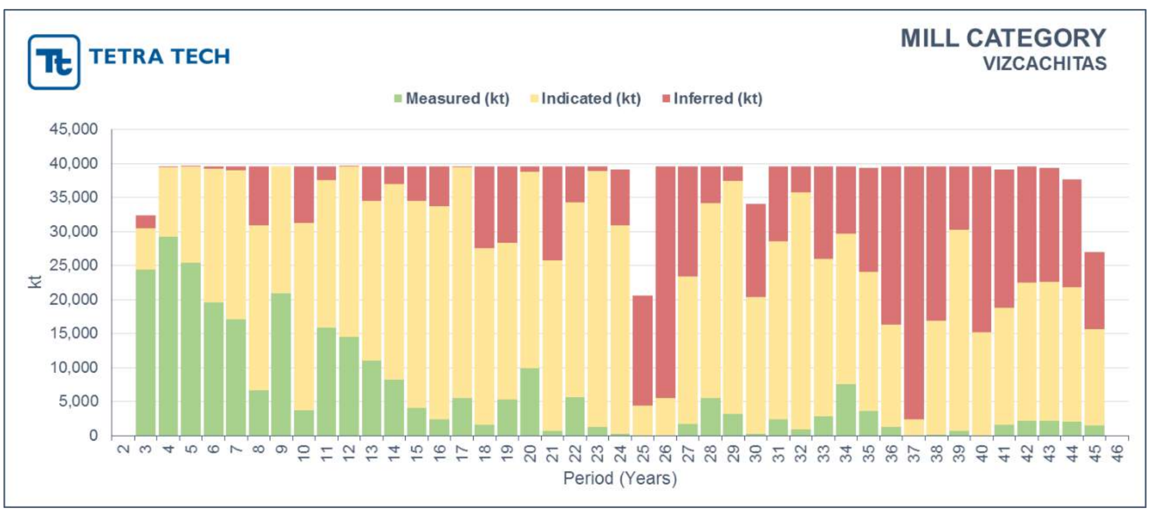


Figure 16.21 details the mineral fed to the mill by resource category. Of these resources 16.5% are measured, 57.3% are indicated and 26.2% are inferred. The largest concentration of inferred resources occurs from year 25 to the end of the mining plan.

Figure 16.21: Measured, Indicated and Inferred Resources in 110 ktpd Mine Schedule



16.10.2.3 Equipment Fleet

The main mine equipment considered to meet the production schedule are as follows:

- Production Drills (10-5/8" – 12¼'')
- Electric Shovel (73 yd^3)
- Hydraulic Shovel (56 yd^3)
- Front-End Loader (31 yd^3)

The support equipment units are listed below:

- Bulldozer
- Wheel dozer
- Motor grader
- Water truck

The mine equipment units required for the 110 ktpd plan are summarized in Table 16.20.

Table 16.20: Mine Equipment Required in 110 ktpd Mine Plan

Equipment Required		Purchases	Fleet																					
			2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23
Cable Shovel - 73 yd^3	un	4		2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Hydraulic Shovel - 56yd^3	un	4		1	1	1	1	1	1	1	1	1	1	1	2	2	2	2	2	2	2	2	2	2
Front Loader - 31 yd^3	un	5	1	1	1	1	1	1	1	2	2	2	2	2	1	1	1	1	1	1	1	1	1	1
Truck - 330 st	un	131	4	22	26	26	26	26	26	26	34	40	40	40	40	40	40	40	48	44	44	55	55	55
Driller - Diesel 12 1/4"	un	16	2	6	6	6	6	6	6	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8
Bulldozer - 890 HP	un	22	3	5	5	5	5	5	5	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6
Wheel Dozer - 853 HP	un	27	4	6	6	6	6	6	6	7	7	7	7	7	7	7	7	7	7	7	7	7	7	7
Motor Grader - 533 HP	un	19	1	3	3	3	3	3	3	3	4	4	4	4	4	4	4	4	5	5	5	6	6	6
Sprinkler Truck - 75.7 m^3	un	14	1	3	3	3	3	3	3	3	4	4	4	4	4	4	4	4	5	5	5	6	6	6
Diesel Truck	un	8	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	3	3	3	3	3	3
Front Loader - 5 yd^3	un	4	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Service Truck	un	8	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Precut Driller	un	14	1	3	3	3	3	3	3	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4

Equipment Required		Fleet																					
		24	25	26	27	28	29	30	31	32	33	34	35	36	37	38	39	40	41	42	43	44	45
Cable Shovel - 73 yd^3	un	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	
Hydraulic Shovel - 56yd^3	un	2	2	2	2	2	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Front Loader - 31 yd^3	un	1	1	1	1	1	2	2	2	2	1	1	1	2	2	2	2	2	1	1	1	1	1
Truck - 330 st	un	55	47	41	50	50	50	50	55	55	55	50	50	50	50	50	50	50	50	41	41	41	41
Driller - Diesel 12 1/4"	un	8	8	8	8	8	8	8	8	8	6	6	6	8	8	8	8	8	6	6	6	6	3
Bulldozer - 890 HP	un	6	6	6	6	6	6	6	6	6	5	5	5	6	6	6	6	6	5	5	5	5	4
Wheel Dozer - 853 HP	un	7	7	7	7	7	7	7	7	7	6	6	6	7	7	7	7	7	6	6	6	6	5
Motor Grader - 533 HP	un	6	5	5	5	5	5	5	6	6	6	5	5	5	5	5	5	5	5	5	5	5	5
Sprinkler Truck - 75.7 m^3	un	6	5	5	5	5	5	5	6	6	6	5	5	5	5	5	5	5	5	5	5	5	5
Diesel Truck	un	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3
Front Loader - 5 yd^3	un	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Service Truck	un	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Precut Driller	un	4	4	4	4	4	4	4	4	4	3	3	3	4	4	4	4	4	3	3	3	3	2

16.10.2.4 Manpower

The staffing estimated for the mine plan of a 110 ktpd plant capacity is summarized Table 16.21 through Table 16.22.

Table 16.21: Supervisory Staffing in 110 ktpd Mine Plan

Supervision	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**	**15**	**16**
Administration	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6
Mine Superintendent	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Maintenance Superintendent	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Management	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Geology	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5
Geologist	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Assistant	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Topography	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3
Topographer	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Assistant	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Mine Supervisor	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Maintenance Supervisor	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Total	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16

Supervision	**17**	**18**	**19**	**20**	**21**	**22**	**23**	**24**	**25**	**26**	**27**	**28**	**29**	**30**	**31**
Administration	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6
Mine Superintendent	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Maintenance Superintendent	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Management	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Geology	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5
Geologist	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Assistant	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Topography	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3
Topographer	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Assistant	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Mine Supervisor	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Maintenance Supervisor	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Total	16	16	16	16	16	16	16	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**

Supervision	**32**	**33**	**34**	**35**	**36**	**37**	**38**	**39**	**40**	**41**	**42**	**43**	**44**	**45**
Administration	6	6	6	6	6	6	6	6	6	6	6	6	6	6
Mine Superintendent	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Maintenance Superintendent	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Management	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Geology	5	5	5	5	5	5	5	5	5	5	5	5	5	5
Geologist	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Assistant	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Topography	3	3	3	3	3	3	3	3	3	3	3	3	3	3
Topographer	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Assistant	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Mine Supervisor	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Maintenance Supervisor	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Total	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**

Table 16.22: Mine Operator Staffing in 110 ktpd Mine Plan

Equipment	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
Cable Shovel - 73 yd3	0	9	9	9	9	9	9	9	9	9	9	9	9	9	9	9
Hydraulic Shovel - 56 yd3	0	5	5	5	5	5	5	5	5	5	5	5	9	9	9	9
Front Loader - 31 yd3	5	5	5	5	5	5	5	9	9	9	9	9	5	5	5	5
Truck - 330 st	15	79	93	93	93	93	93	93	122	143	143	143	143	143	143	143
Driller - Diesel 12 1/4"	9	26	26	26	26	26	26	35	35	35	35	35	35	35	35	35
Bulldozer - 890 HP	11	18	18	18	18	18	18	22	22	22	22	22	22	22	22	22
Wheel Dozer - 853 HP	15	22	22	22	22	22	22	25	25	25	25	25	25	25	25	25
Motor Grader - 533 HP	4	11	11	11	11	11	11	11	15	15	15	15	15	15	15	15
Sprinkler Truck - 75.7 m³	4	11	11	11	11	11	11	11	15	15	15	15	15	15	15	15
Diesel Truck	4	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8
Front Loader - 5 yd3	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Service Truck	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8
Precut Driller	5	13	13	13	13	13	13	18	18	18	18	18	18	18	18	18
Total	**84**	**219**	**233**	**233**	**233**	**233**	**233**	**258**	**295**	**316**	**316**	**316**	**316**	**316**	**316**	**316**

Equipment	18	19	20	21	22	23	24	25	26	27	28	29	30	31	32	33
Cable Shovel - 73 yd3	9	9	9	9	9	9	9	9	9	9	9	9	9	9	9	9
Hydraulic Shovel - 56 yd3	9	9	9	9	9	9	9	9	9	9	9	5	5	5	5	5
Front Loader - 31 yd3	5	5	5	5	5	5	5	5	5	5	5	9	9	9	9	5
Truck - 330 st	172	158	158	197	197	197	197	168	147	179	179	179	179	197	197	197
Driller - Diesel 12 1/4"	35	35	35	35	35	35	35	35	35	35	35	35	35	35	35	26
Bulldozer - 890 HP	22	22	22	22	22	22	22	22	22	22	22	22	22	22	22	18
Wheel Dozer - 853 HP	25	25	25	25	25	25	25	25	25	25	25	25	25	25	25	22
Motor Grader - 533 HP	18	18	18	22	22	22	22	18	18	18	18	18	18	22	22	22
Sprinkler Truck - 75.7 m³	18	18	18	22	22	22	22	18	18	18	18	18	18	22	22	22
Diesel Truck	11	11	11	11	11	11	11	11	11	11	11	11	11	11	11	11
Front Loader - 5 yd3	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Service Truck	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8
Precut Driller	18	18	18	18	18	18	18	18	18	18	18	18	18	18	18	13
Total	**354**	**340**	**340**	**387**	**387**	**387**	**387**	**350**	**329**	**361**	**361**	**361**	**361**	**387**	**387**	**362**

Equipment	34	35	36	37	38	39	40	41	42	43	44	45
Cable Shovel - 73 yd3	9	9	9	9	9	9	9	9	9	9	9	0
Hydraulic Shovel - 56 yd3	5	5	5	5	5	5	5	5	5	5	5	5
Front Loader - 31 yd3	5	5	9	9	9	9	9	5	5	5	5	5
Truck - 330 st	179	179	179	179	179	179	179	179	147	147	147	147
Driller - Diesel 12 1/4"	26	26	35	35	35	35	35	26	26	26	26	13
Bulldozer - 890 HP	18	18	22	22	22	22	22	18	18	18	18	15
Wheel Dozer - 853 HP	22	22	25	25	25	25	25	22	22	22	22	18
Motor Grader - 533 HP	18	18	18	18	18	18	18	18	18	18	18	18
Sprinkler Truck - 75.7 m³	18	18	18	18	18	18	18	18	18	18	18	18
Diesel Truck	11	11	11	11	11	11	11	11	11	11	11	11
Front Loader - 5 yd3	4	4	4	4	4	4	4	4	4	4	4	4
Service Truck	8	8	8	8	8	8	8	8	8	8	8	8
Precut Driller	13	13	18	18	18	18	18	13	13	13	13	9
Total	**336**	**336**	**361**	**361**	**361**	**361**	**361**	**336**	**304**	**304**	**304**	**271**

Table 16.23: Mine Equipment Maintenance Staffing in 110 ktpd Mine Plan

Mine Equipment Maintenance	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16
Total	110	285	303	303	303	303	303	336	384	411	411	411	411	411	411

Mine Equipment Maintenance	17	18	19	20	21	22	23	24	25	26	27	28	29	30	31
Total	411	461	442	442	504	504	504	504	455	428	470	470	470	470	504

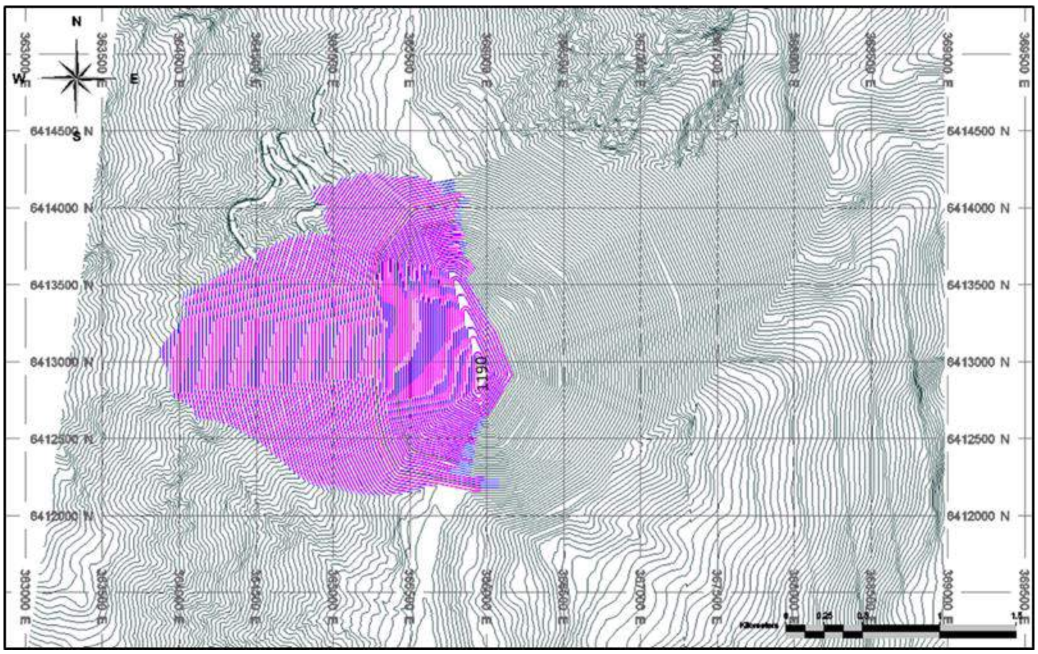
Mine Equipment Maintenance	32	33	34	35	36	37	38	39	40	41	42	43	44	45
Total	504	471	437	437	470	470	470	470	470	437	396	396	396	353

16.10.3 Case 3: 200 ktpd

16.10.3.1 Final Pit: 200 ktpd

The Set 2 phases were used for this mining plan. Based on the sequence defined in the final pit optimization stage, a marginal phase analysis was conducted. The final pit of this scenario corresponds to the topography of phase 10 as shown in Figure 16.22.

Figure 16.22: Final Pit: 200 ktpd (Phase 10)



16.10.3.2 Mine Production Schedule

Table 16.24 and Figure 16.23 summarize the movement of materials in the mine plan for a 200 ktpd plant. To comply with the production schedule, the total movement reaches a maximum of 680 ktpd.

The total movement in the 30 years of mine life for the option analyzed amounts to 5,065 Mt (including re-handling) with a total feed of 1,939 Mt to the concentrator.

The mineral flow in the mine plan was optimized by separating the mined material into four categories namely:

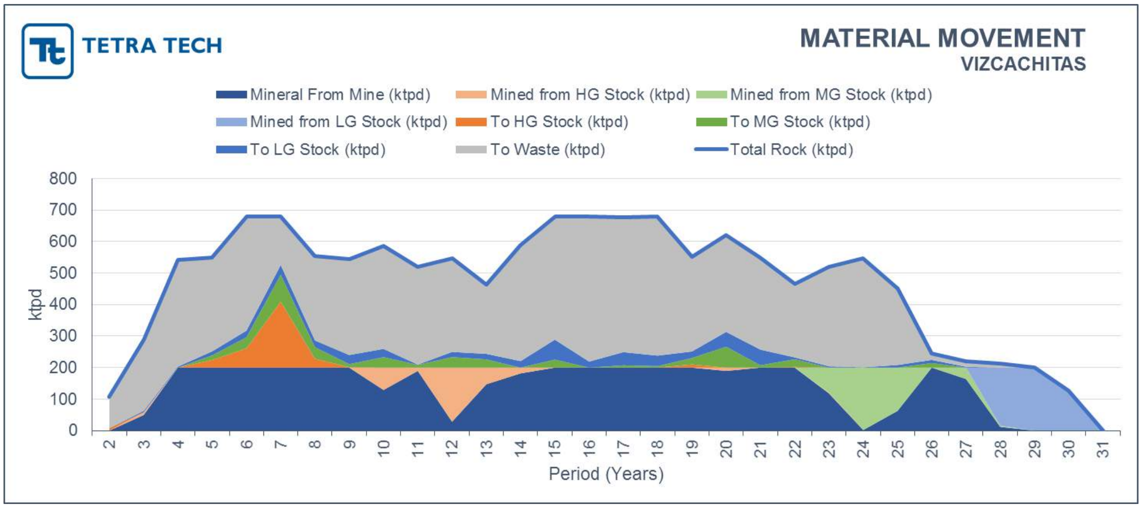
- Mine to Mill: mineral shipped direct to the mill, with a variable cut-off grade
- HG Stockpile: mineral with grade below direct Mine to Mill and above a cut-off grade of 0.25% CuEq

- MG Stockpile: mineral with grade below HG Stockpile and above a cut-off grade of 0.20% CuEq
- LG Stockpile: mineral with grade below MG Stockpile and above a cut-off grade of 0.15% CuEq

Table 16.24: Production Schedule - 200 ktpd

Vizcachitas Project - 200ktpd													
Period (Years)	Days	COG	Mineral to Mill					Waste	W/O	Total			ktpd
			Total							Onsite	Rehandling	W/Rehandling	
			kt	CuEq (%)	Cu (%)	Mo (ppm)	Ag (g/t)	kt		kt	kt	kt	
2	360							35,119		38,243		38,243	106
3	360	0.16%	20,937	0.30%	0.28%	40.73	0.72	81,320	3.88	100,794	3,052	103,847	289
4	360	0.18%	72,000	0.51%	0.47%	90.51	1.21	122,120	1.70	195,352		195,352	543
5	360	0.32%	72,000	0.51%	0.46%	124.09	1.33	107,237	1.49	197,869		197,869	550
6	360	0.32%	72,000	0.49%	0.44%	104.01	1.30	130,259	1.81	244,526		244,526	679
7	360	0.38%	72,000	0.49%	0.43%	153.57	1.25	55,174	0.77	244,520		244,520	679
8	360	0.29%	72,000	0.40%	0.36%	99.48	1.01	96,317	1.34	199,330		199,330	554
9	360	0.26%	72,000	0.48%	0.43%	130.94	1.08	109,421	1.52	195,837		195,837	544
10	360	0.25%	72,000	0.37%	0.33%	103.55	0.95	117,770	1.64	185,619	25,549	211,169	587
11	360	0.25%	72,000	0.38%	0.32%	156.89	1.02	111,601	1.55	183,386	3,729	187,115	520
12	360	0.24%	72,000	0.31%	0.27%	88.22	0.91	106,691	1.48	134,820	61,808	196,628	546
13	360	0.23%	72,000	0.32%	0.29%	86.21	0.89	78,710	1.09	147,456	19,042	166,498	463
14	360	0.19%	72,000	0.38%	0.34%	106.85	0.94	132,426	1.84	205,367	6,550	211,917	589
15	360	0.22%	72,000	0.35%	0.30%	134.50	0.83	140,713	1.95	244,672		244,672	680
16	360	0.18%	72,000	0.33%	0.29%	83.95	0.82	165,395	2.30	244,324		244,324	679
17	360	0.22%	72,000	0.43%	0.37%	133.06	0.93	153,853	2.14	243,562		243,562	677
18	360	0.21%	72,000	0.41%	0.35%	167.46	0.85	158,599	2.20	244,303		244,303	679
19	360	0.26%	72,000	0.39%	0.34%	118.66	0.96	107,848	1.50	198,339		198,339	551
20	360	0.24%	72,000	0.36%	0.29%	168.06	0.86	110,919	1.54	219,992	3,734	223,726	622
21	360	0.21%	72,000	0.33%	0.29%	95.32	0.83	104,960	1.46	197,525		197,525	549
22	360	0.25%	72,000	0.42%	0.37%	130.97	1.04	83,522	1.16	167,272		167,272	465
23	360	0.17%	72,000	0.31%	0.27%	94.45	0.88	113,838	1.58	157,846	29,483	187,330	520
24	360	0.17%	72,000	0.23%	0.20%	58.03	0.76	124,496	1.73	125,493	71,328	196,821	547
25	360	0.16%	72,000	0.26%	0.24%	52.14	0.77	87,298	1.21	112,803	49,490	162,292	451
26	360	0.24%	72,000	0.38%	0.34%	110.91	0.99	7,393	0.10	88,197		88,197	245
27	360	0.17%	72,000	0.32%	0.26%	171.14	0.84	6,702	0.09	66,678	12,961	79,639	221
28	360	0.15%	71,982	0.19%	0.17%	48.04	0.62	4,724	0.07	8,925	67,781	76,706	213
29	360	0.16%	71,972	0.18%	0.16%	38.33	0.61				71,972	71,972	200
30	360	0.16%	45,733	0.18%	0.16%	38.33	0.61				45,733	45,733	127
Total			**1,938,625**	**0.36%**	**0.32%**	**107.17**	**0.93**	**2,654,427**	**1.37**	**4,593,052**	**472,212**	**5,065,264**	

Figure 16.23: Production Schedule - 200 ktpd



The mineral fed to mill is detailed in Figure 16.24.

Mine production starts in year three, reaching the design capacity during year four. From then on, the process capacity is met until year 29, decreasing in the last year of mine life, which is consistent with the depletion of phase 10 and low-grade stock (LG).

Figure 16.24: Mineral to Mill in 200 ktpd Mine Schedule

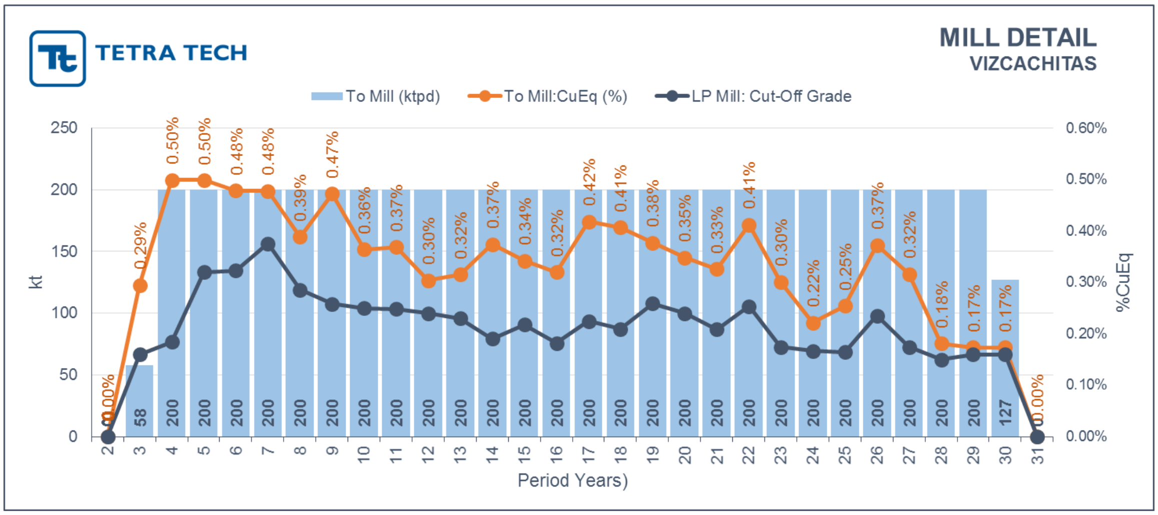
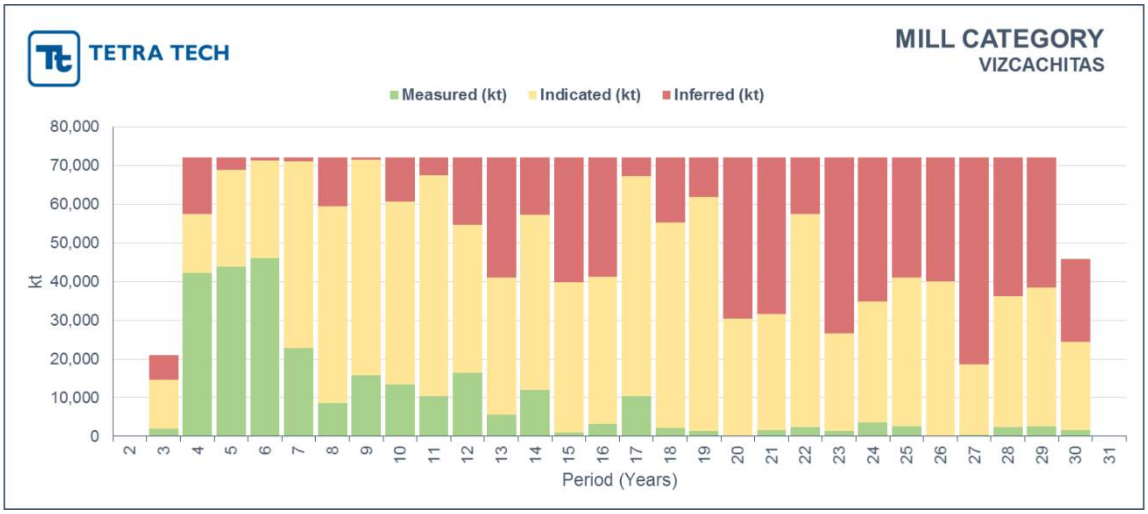


Figure 16.25 details the mineral sent to mill by resource category. Of these resources 14.3% are measured, 54.9% are indicated and 30.8% are inferred. The largest concentration of inferred resources occurs between periods 20 and 29.

Figure 16.25: Measured, Indicated and Inferred Resources in 200-ktpd Mine Schedule



16.10.3.3 Equipment Fleet

The main mine equipment units considered to meet the production schedule are as follows:

- Production Drills (10-5/8" - 12¼'')
- Electric Shovel (73 yd^3)
- Hydraulic Shovel (56 yd^3)
- Front-End Loader (31 yd^3)

The support equipment units are listed below:

- Bulldozer
- Wheel dozer
- Motor grader
- Water truck

The additional mine equipment units required for the 200 ktpd plan are summarized in Table 16.25.

Table 16.25: Mine Equipment Requirements in 200 ktpd Mine Plan

Equipment Required		Purchase	Fleet													
			2	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**	**15**
Cable Shovel - 73 yd3	un	4		2	4	4	4	4	4	4	4	4	4	4	4	4
Hydraulic Shovel - 56 yd3	un	8	1	4	4	4	4	4	4	4	4	4	4	4	4	4
Front Loader - 31 yd3	un	5	1	1	1	1	1	2	2	2	2	2	2	2	2	2
Truck - 330 st	un	160	13	26	58	58	76	91	91	91	91	91	91	91	110	110
Driller - Diesel 12 1/4"	un	26	3	11	14	14	14	15	15	15	15	15	15	15	15	15
Bulldozer - 890 HP	un	18	4	7	8	8	8	9	9	9	9	9	9	9	9	9
Wheel dozer - 853 HP	un	25	5	8	9	9	9	10	10	10	10	10	10	10	10	10
Motor Grader - 533 HP	un	21	2	3	6	6	8	10	10	10	10	10	10	10	11	11
Sprinkler Truck - 75.7 m³	un	17	2	3	6	6	8	10	10	10	10	10	10	10	11	11
Diesel Truck	un	9	1	2	3	3	4	5	5	5	5	5	5	5	6	6
Front Loader - 5 yd3	un	3	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Service Truck	un	6	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Precut Driller	un	16	2	6	7	7	7	8	8	8	8	8	8	8	8	8

Equipment Required		Fleet														
		16	**17**	**18**	**19**	**20**	**21**	**22**	**23**	**24**	**25**	**26**	**27**	**28**	**29**	**30**
Cable Shovel - 73 yd3	un	4	4	4	4	4	4	4	4	3	2					
Hydraulic Shovel - 56 yd3	un	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Front Loader - 31 yd3	un	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Truck - 330 st	un	117	117	117	102	102	102	84	84	69	69	69	69	69	50	50
Driller - Diesel 12 1/4"	un	15	15	15	15	15	15	15	15	14	12	9	9	9	9	9
Bulldozer - 890 HP	un	9	9	9	9	9	9	9	9	8	8	6	6	6	6	6
Wheel Dozer - 853 HP	un	10	10	10	10	10	10	10	10	9	9	7	7	7	7	7
Motor Grader - 533 HP	un	12	12	12	11	11	11	9	9	7	7	7	7	7	5	5
Sprinkler Truck - 75.7 m³	un	12	12	12	11	11	11	9	9	7	7	7	7	7	5	5
Diesel Truck	un	6	6	6	6	6	6	5	5	4	4	4	4	4	3	3
Front Loader - 5 yd3	un	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Service Truck	un	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Precut Driller	un	8	8	8	8	8	8	8	8	7	6	5	5	5	5	5

16.10.3.4 Manpower

The staffing estimated for the 200 ktpd mine plan is summarized in Table 16.26 through Table 16.27.

Table 16.26: Supervisory Staffing in 200 ktpd Mine Plan

Supervision	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16
Administration	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6
Mine Superintendent	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Maintenance Superintendent	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Management	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Geology	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5
Geologist	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Assistant	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Topography	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3
Topographer	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Assistant	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Mine Supervisor	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Maintenance Supervisor	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Total	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**

Supervision	17	18	19	20	21	22	23	24	25	26	27	28	29	30
Administration	6	6	6	6	6	6	6	6	6	6	6	6	6	6
Mine Superintendent	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Maintenance Superintendent	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Management	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Geology	5	5	5	5	5	5	5	5	5	5	5	5	5	5
Geologist	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Assistant	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Topography	3	3	3	3	3	3	3	3	3	3	3	3	3	3
Topographer	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Assistant	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Mine Supervisor	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Maintenance Supervisor	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Total	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**

Table 16.27: Mine Operator Staffing in 200 ktpd Mine Plan

Equipment	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16
Cable Shovel - 73 yd3	0	9	18	18	18	18	18	18	18	18	18	18	18	18	18
Hydraulic Shovel - 56 yd3	5	18	18	18	18	18	18	18	18	18	18	18	18	18	18
Front Loader - 31 yd3	5	5	5	5	5	9	9	9	9	9	9	9	9	9	9
Truck - 330 st	47	93	208	208	272	325	325	325	325	325	325	325	393	393	418
Driller - Diesel 12 1/4"	13	48	61	61	61	65	65	65	65	65	65	65	65	65	65
Bulldozer - 890 HP	15	25	29	29	29	33	33	33	33	33	33	33	33	33	33
Wheel Dozer - 853 HP	18	29	33	33	33	36	36	36	36	36	36	36	36	36	36
Motor Grader - 533 HP	8	11	22	22	29	36	36	36	36	36	36	36	40	40	43
Sprinkler Truck - 75.7 m^3	8	11	22	22	29	36	36	36	36	36	36	36	40	40	43
Diesel Truck	4	8	11	11	15	18	18	18	18	18	18	18	22	22	22
Front Loader - 5 yd3	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Service Truck	8	8	8	8	8	8	8	8	8	8	8	8	8	8	29
Precut Driller	9	26	31	31	31	35	35	35	35	35	35	35	35	35	0
Total	**144**	**295**	**470**	**470**	**552**	**641**	**641**	**641**	**641**	**641**	**641**	**641**	**721**	**721**	**738**

Equipment	17	18	19	20	21	22	23	24	25	26	27	28	29	30
Cable Shovel - 73 yd3	18	18	18	18	18	18	18	13	9	0	0	0	0	0
Hydraulic Shovel - 56 yd3	18	18	18	18	18	18	18	18	18	18	18	18	18	18
Front Loader - 31 yd3	9	9	9	9	9	9	9	9	9	9	9	9	9	9
Truck - 330 st	418	418	365	365	365	300	300	247	247	247	247	247	179	179
Driller - Diesel 12 1/4"	65	65	65	65	65	65	65	61	52	39	39	39	39	39
Bulldozer - 890 HP	33	33	33	33	33	33	33	29	29	22	22	22	22	22
Wheel Dozer - 853 HP	36	36	36	36	36	36	36	33	33	25	25	25	25	25
Motor Grader - 533 HP	43	43	40	40	40	33	33	25	25	25	25	25	18	18
Sprinkler Truck - 75.7 m³	43	43	40	40	40	33	33	25	25	25	25	25	18	18
Diesel Truck	22	22	22	22	22	18	18	15	15	15	15	15	11	11
Front Loader - 5 yd3	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Service Truck	8	8	8	8	8	8	8	8	8	8	8	8	8	8
Precut Driller	35	35	35	35	35	35	35	31	26	22	22	22	22	22
Total	**752**	**752**	**693**	**693**	**693**	**610**	**610**	**518**	**500**	**459**	**459**	**459**	**373**	**373**

Table 16.28: Equipment Maintenance Staffing in 200 ktpd Mine Plan

Mine Equipment Maintenance	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16
Total	188	384	611	611	718	834	834	834	834	834	834	834	938	938	960

Mine Equipment Maintenance	17	18	19	20	21	22	23	24	25	26	27	28	29	30
Total	978	978	901	901	901	793	793	674	650	597	597	597	485	485

17. RECOVERY METHODS

17.1 Introduction

Based on the results obtained in the series of metallurgical tests conducted to date, the recommended process for copper and molybdenum recovery includes the following main circuits:

- Comminution circuit made up of:
  - Primary crushing
  - SAG mill grinding
  - Ball mill grinding
- Flotation circuit made up of:
  - Rougher circuit
  - Regrinding
  - Copper and molybdenum cleaner circuit
  - Copper and molybdenum separation by selective flotation
  - Molybdenum cleaner flotation
  - Dewatering of copper and molybdenum concentrates.

The recommended comminution circuit is a standard and widely used copper concentration process. The mineral resource from the mine will be hauled by trucks to the primary crusher where its size will be reduced to an estimated P_{80} between 200 mm and 170 mm. The crushed product will be stored in a stockpile that feeds the comminution circuit, where it will be further ground to a P_{80} of 240 µm and sent to the flotation circuit.

17.2 Process Design Basis and Design Criteria Summary

Table 17.1 shows the general criteria considered in the design of the copper-molybdenum concentrator.

Table 17.1: Summary of Key Process Design Criteria

Item	Unit	Value	Source
General Parameters			
Operating days	d/y	365	Parameters defined by Alquimia, PEA 2014
Operating hours	h/d	24	Parameters defined by Alquimia, PEA 2014
Utilization			
Primary crushing	%	70	Parameters defined by Alquimia, PEA 2014
Grinding	%	92	Parameters defined by Alquimia, PEA 2014
Cu - Mo flotation	%	92	Parameters defined by Alquimia, PEA 2014
Mo flotation	%	92	Parameters defined by Alquimia, PEA 2014
Thickening	%	98	Parameters defined by Alquimia, PEA 2014
Filtering	%	85	Parameters defined by Alquimia, PEA 2014
Drying	%	90	Parameters defined by Alquimia, PEA 2014
Mineral Parameters			
Cu head grade	%	0.45	Parameters obtained from analysis of Vizcachitas test work 2018
Mo head grade	ppm	130	Parameters obtained from analysis of Vizcachitas test work 2018
Work index (Wi)	kWh/tm	13.1	Parameters obtained from analysis of Vizcachitas test work 2018
Moisture	%	3	Parameters defined by Alquimia, PEA 2014
Specific solid gravity	-	2.6	Parameters obtained from analysis of Vizcachitas test work
Cu-Mo Flotation			
Cu recovery	%	91	Parameters obtained from analysis of Vizcachitas test work
Mo recovery	%	84	Parameters obtained from analysis of Vizcachitas test work
Cu concentrate grade	%	30	Parameters obtained from analysis of Vizcachitas test work
Mo concentrate grade	%	0.72	Parameters obtained from analysis of Vizcachitas test work
Mo Flotation			
Cu recovery	%	0.04	Calculated
Mo recovery	%	89	Calculated
Cu concentrate grade	%	2	Benchmarking Parameters and industrial practice
Mo concentrate grade	%	48	Benchmarking Parameters and industrial practice
Global Process			
Cu recovery	%	91	Calculated
Mo recovery	%	75	Calculated
Cu concentrate grade	%	30	Benchmarking Parameters and industrial practice
Mo concentrate grade	%	48	Benchmarking Parameters and industrial practice

17.3 Comminution Circuit

17.3.1 Crushing

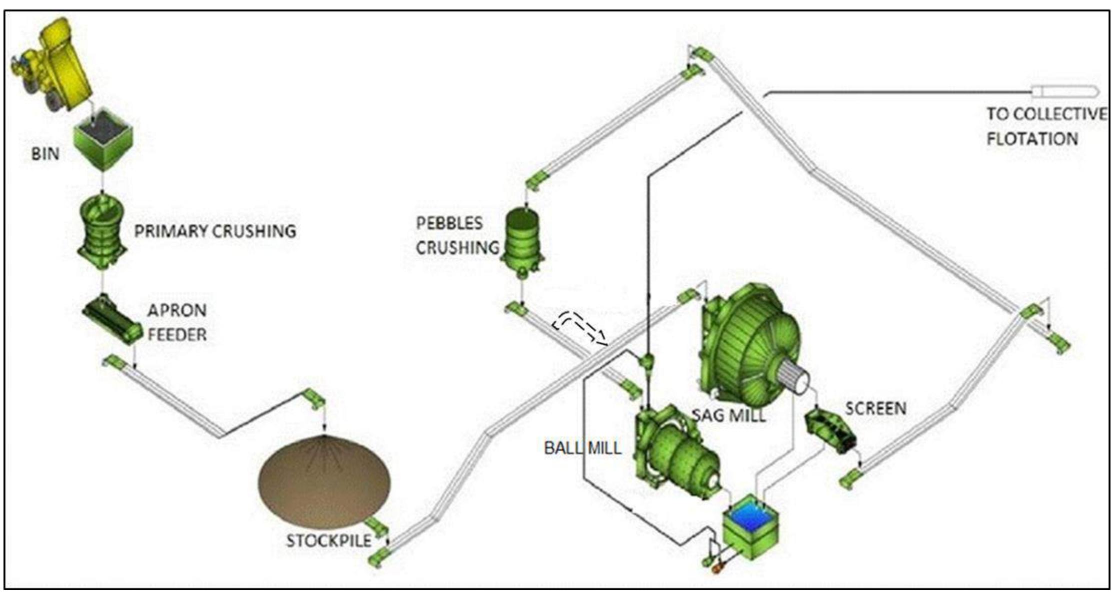
The mineralized material from the mine will be transported by trucks to the primary crushing facility, where it will be discharged into the bin which feeds a primary gyratory crusher. The crusher will discharge onto an apron feeder and the crushed material will then be transported by a belt conveyor to the covered coarse material stockpile.

17.3.2 Grinding

Semi-Autogenous Grinding (SAG) was selected for the milling stage, as it is efficient for this scale of operation and adds flexibility for future expansions. A SAG circuit has the capacity to be easily doubled by adding one or two ball mills whilst maintaining only one grinding line.

The SAG mill will be fed by a conveyor belt from the coarse material stockpile. The SAG mill discharge slurry will pass through a double deck screen with 13 mm sieve size to remove pebbles. The pebbles will be transported by conveyor belt to the pebble crusher plant and then returned to either the primary SAG mill circuit or the secondary ball mill circuit. The double screen undersize and the ball mill will discharge into a sump pump. The combined product will be pumped to the cyclone cluster. The cyclone underflow will return to the ball mills circuit and the overflow will report by gravity to rougher flotation. (See Figure 17.1)

Figure 17.1: Comminution Circuit



The design of the SAG mill circuit considered:

- AxB test results and curves that relate the AxB factor and the specific power consumption.
- T_{80} derived from existing SAG mill operations.
- Load factor to size the nameplate power for the motor.
- Specific power consumption obtained from the Starkey time curves.

The design of the pebble crusher circuit considered a pebble circulating load of 30% and 20% of the new feed rate for SAG circuit. This is based on industry experience with mineralized material of similar competency.

The ball mill secondary grinding circuit was designed to produce a final product size of P_{80} at 240 µm. The key design criteria considered for the primary and secondary grinding circuits for the proposed treatment options are shown in Table 17.2 and Table 17.3. The loading factor includes drive losses.

Table 17.2: Primary Circuit

Primary Grinding (SAG)	ktpd		
Treatment Options	**55**	**110**	**200**
General Parameters			
AxB	36.8	36.8	36.8
CEE kwh / tm	5,658	5,050	5,658
OT %	92%	92%	92%
tph Nominal tm /hr	2,491	4,982	9,058
Power required by mill shaft (kw)	14,094	25,159	51,250
Power required by mill shaft (hp)	18,900	33,738	68,726
Loading factor	90%	90%	90%
Motor power (hp)	21,000	37,487	76,363
Recommended motor power	**21,000**	**38,000**	**38,000**
T_{80} microns	3,800	5,500	3,800
Number of SAG mills required	1	**1**	2

Table 17.3: Secondary Circuit

Ball Grinding Circuit			
Treatment Options	ktpd		
	55	110	200
General Parameters			
Wib kwh /tm	13.1	13.1	13.1
OT %	92%	92%	92%
Nominal tph	2,491	4,876	9,058
Design tph	2,865	5,607	10,417
Power required by mill shaft (kw)	17,936	20,732	16,295
Power required by mill shaft (hp)	24,052	27,802	21,852
Loading factor	95%	95%	95%
Motor Power (hp)	25,318	29,265	23,002
Recommended motor power	**25,000**	**29,500**	**23,000**
T_{80} microns	240	240	240
Number of ball mills required	1	2	4

17.4 Flotation Circuit

17.4.1 Copper-Molybdenum Flotation Circuit Design

The flow sheet selected shown in Figure 17.2 consists of rougher flotation, rougher concentrate regrind, three stages of cleaner flotation and three stages of scavenger flotation. The mineralized material will require a fine regrind of the rougher concentrate to a P_{80} of 45 µm to achieve adequate recovery. After regrinding, conventional flotation will be used to enable the production of commercial grade concentrates.

Rougher concentrate, as well as first and second scavenger concentrate, will report to the regrind cyclone pump box. The regrind cyclones will target an overflow size of P_{80} at 45 µm, which will be achieved with a cluster of cyclones operating in reverse closed circuit. The cyclones overflow will report to the first cleaner flotation while the underflow will feed the vertical mill. The vertical mill discharge will recirculate back into the cyclone pump box.

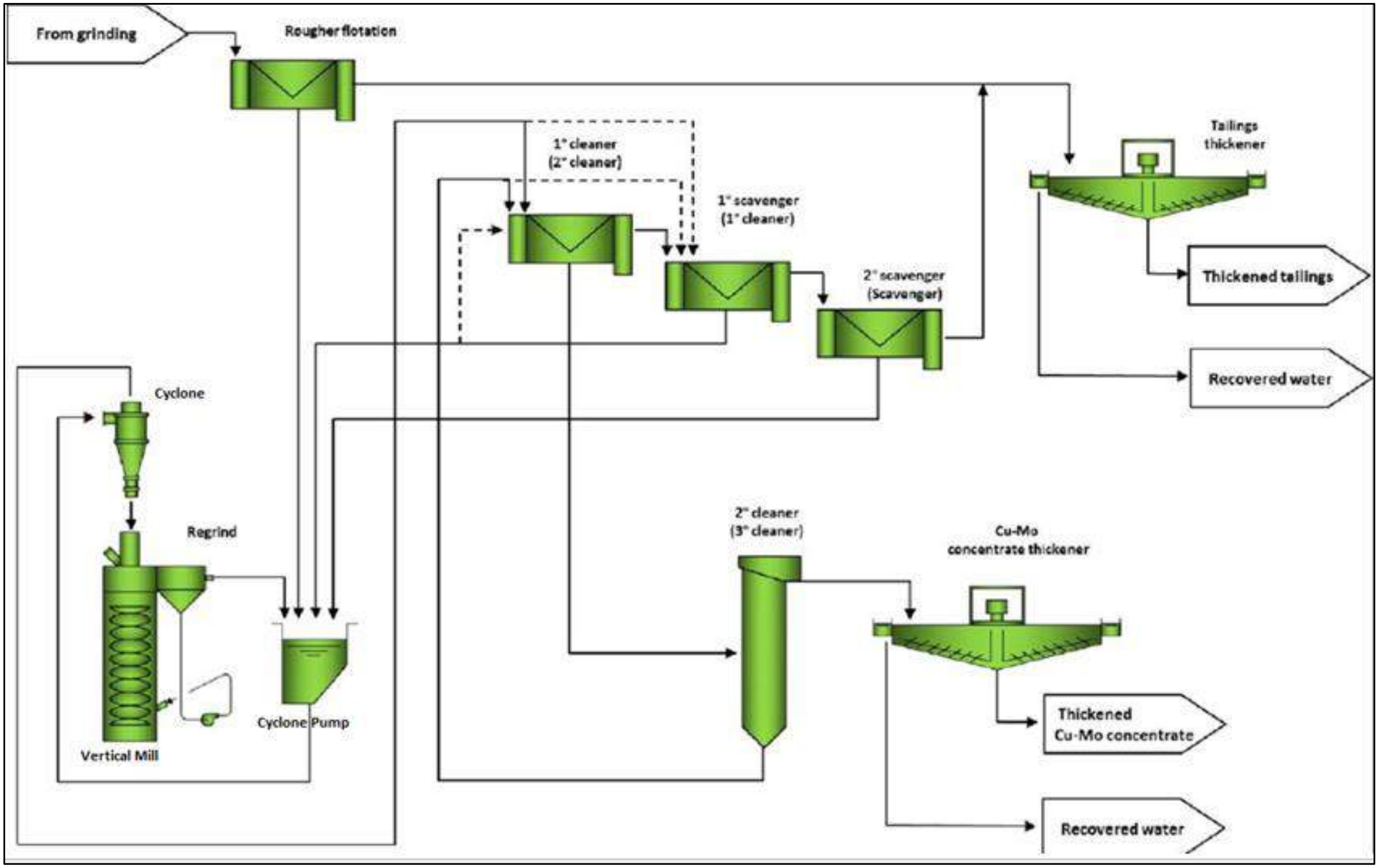
The first cleaner stage will be composed of conventional flotation cells and will be fed by the overflow of the cyclone cluster and the tail of the second cleaner stage. The product will feed the second cleaner while the tail will feed the first scavenger stage.

Second cleaner flotation (column cells) concentrate will report to tertiary cleaners and then to the Cu-Mo concentrate thickener while the tail will report back to the first cleaner stage.

First scavenger stage (conventional cells) concentrate will report to the regrind cyclone feed pump box, while the tail will feed the second scavenger stage.

Second scavenger stage (conventional cells) concentrate will report to the regrind cyclone feed pump box, while the tail will report to the tailings thickener.

Figure 17.2: Cu - Mo Flotation



The flotation process has been designed so that the first scavenger may be converted to a cleaner stage if additional cleaning is required. If this modification is adopted the circuit consists of three stages of cleaners:

- The first cleaner in the normal circuit becomes second cleaner
- The first scavenger becomes the first cleaner
- The second cleaner becomes the third cleaner
- The second scavenger becomes the only scavenger

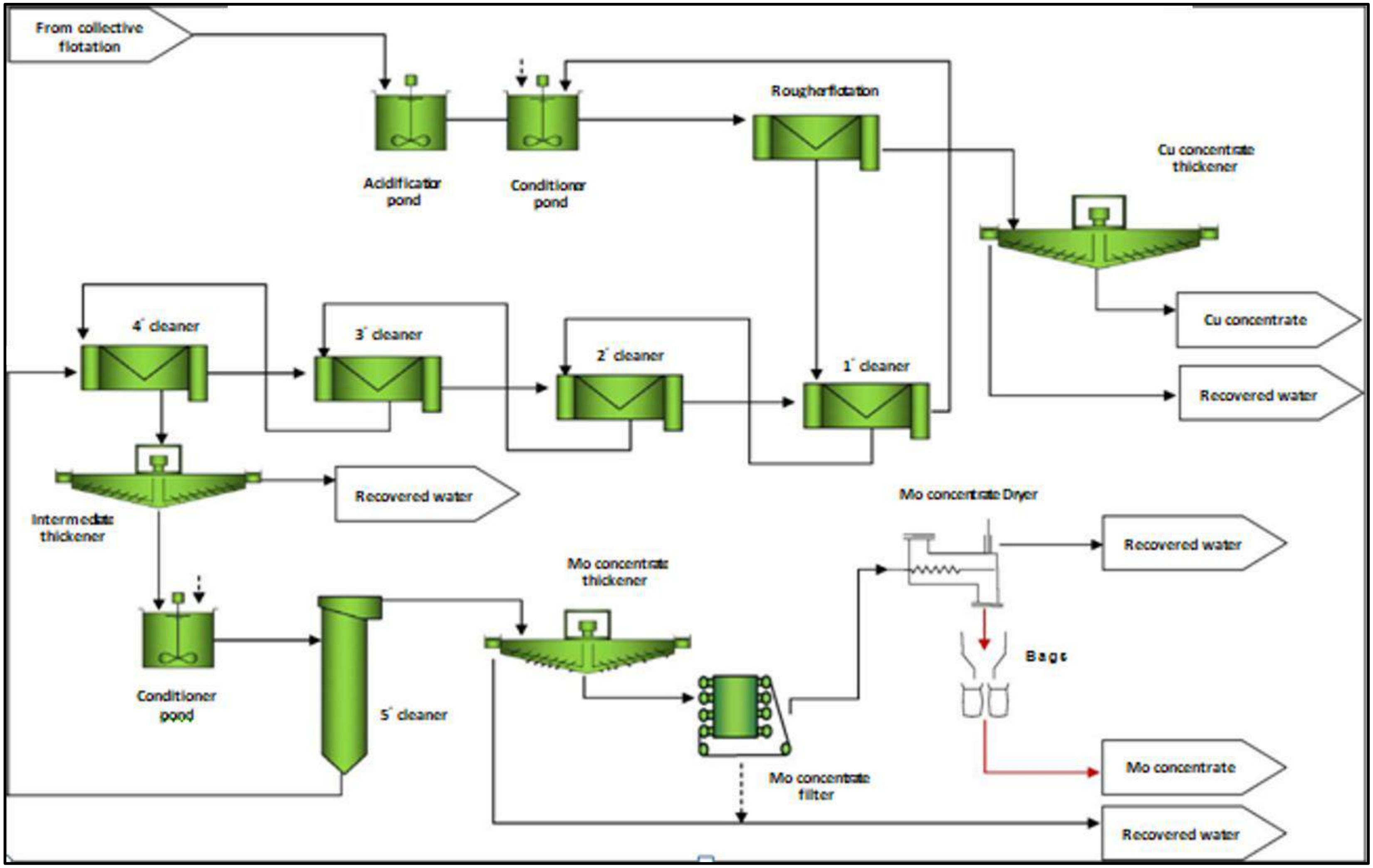
In this modified circuit, the concentrate of the first cleaner will feed the second cleaner and the concentrate of the second cleaner will feed the third cleaner. The tail of the third cleaner will feed the first cleaner and the concentrate will report to the Cu-Mo concentrate thickener (see segmented lines in Figure 17.2).

17.4.2 Copper-Molybdenum Separation and Molybdenum Cleaner

The Cu-Mo concentrate thickener discharge will report to an acidification pond and then will be pumped to a conditioning pond, where it mixes with dilution water and reagents as necessary (collector, frother, etc.).

The flow sheet selected to separate copper and molybdenum consists of a rougher flotation, four stages of cleaner flotation in conventional cells, intermediate thickening and a fifth cleaner flotation in column cells. The tailings from the rougher flotation will report to the copper concentrate thickener. Concentrate from the fifth cleaner flotation cells will report to the molybdenum concentrate thickener. This circuit has been designed based on benchmark data only as no selective molybdenum flotation test work has been performed to date.

Figure 17.3: Molybdenum Flotation Circuit



The product of the conditioning pond will feed the rougher flotation stage. This stage is composed of conventional sealed and self-aspirated cells. The concentrate from this stage will report to the first cleaner stage while the tailings will report to the copper concentrate thickener. The discharge will be thickened to approximately 58% w/w solids, which constitute the final copper concentrate.

Thickened copper concentrate slurry will be pumped to the copper concentrate filters. The filter presses are designed to reduce the moisture to 9% prior to transport. Filter cake will be discharged through the floor of the filter building and stored in the covered concentrate storage shed. Concentrate will then be loaded into trucks for transport to the rail load-out facility.

In the molybdenum flotation circuit, the concentrate from the fifth cleaner cells will be thickened to approximately 58% w/w solids in the molybdenum concentrate thickener. The discharge of the thickener will be filtered, dried and stored in a hopper. The dried concentrate will then be packed in 1 t maxi bags for transport to final sale.

17.5 PEA Options and Cases

Based on a rougher grind size of P_{80} of 240 µm, three cases were analyzed, namely mill throughputs of 55 ktpd, 110 ktpd and 200 ktpd. The main plant equipment for these cases are shown from Table 17.4 to Table 17.7

The following tables list the main equipment for each of the options evaluated in the crushing, grinding, copper flotation, and molybdenum flotation processes.

Table 17.4: Process Plant Equipment – Crushing

Equipment	Case 55 ktpd		Case 110 ktpd		200 ktpd	
	Quantity	Characteristics	Quantity	Characteristics	Quantity	Characteristics
Primary Gyratory Crusher	1	54´ x 75´	1	60' x 110´	2	60' x 110´
	1	650 hp	1	1,600 hp	2	1,400 hp

Table 17.5: Process Plant Equipment – Grinding

Equipment	Case 55 ktpd		Case 110 ktpd		Case 200 ktpd	
	Quantity	Characteristics	Quantity	Characteristics	Quantity	Characteristics
SAG Mill	1	34' x 17'	1	42' x 27'	2	42'x 27'
	1	21,000 hp	1	38,000 hp	2	38,000 hp
Pebbles Crusher	2	800 hp	3	1,000 hp	6	1,000 hp
Ball Mill	1	27' x 45'	2	27' x 45'	4	27' x 45'
	1	25,000 hp	2	29,500 hp	4	23,000 hp

Table 17.6: Process Plant Equipment – Flotation

Equipment	Case 55 ktpd		Case 110 ktpd		Case 200 ktpd	
	Quantity	Characteristics	Quantity	Characteristics	Quantity	Characteristics
Rougher Flotation Cell	12	300 m³	24	300 m³	42	300 m³
First Cleaner Flotation Cell	4	100 m³	6	160 m³	10	160 m³
First Scavenger Flotation Cell	4	100 m³	4	200 m³	8	160 m³
Second Scavenger Flotation Cell	5	130 m³	4	250 m³	8	250 m³
Second Cleaner Flotation Cell	2	7 m²	2	18 m²	3	18 m²
Regrinding Mill	2	600 hp	2	1,500 hp	3	1,500 hp

Table 17.7: Process Plant Equipment – Molybdenum Flotation

Equipment	Case 55 ktpd		Case 110 ktpd		Case 200 ktpd	
	Quantity	Characteristics	Quantity	Characteristics	Quantity	Characteristics
Rougher Flotation Cell	9	300 ft³	5	1,000 ft³	9	1,000 ft³
First Cleaner Cell	8	150 ft³	3	1,000 ft³	5	1,000 ft³
Second Cleaner Cell	5	150 ft³	4	500 ft³	6	500 ft³
Third Cleaner Cell	4	100 ft³	3	300 ft³	5	300 ft³
Fourth Cleaner Cell	5	40 ft³	3	150 ft³	5	150 ft³
Fifth Cleaner Cell	1	1.0 m	1	2 m	2	1.5 m

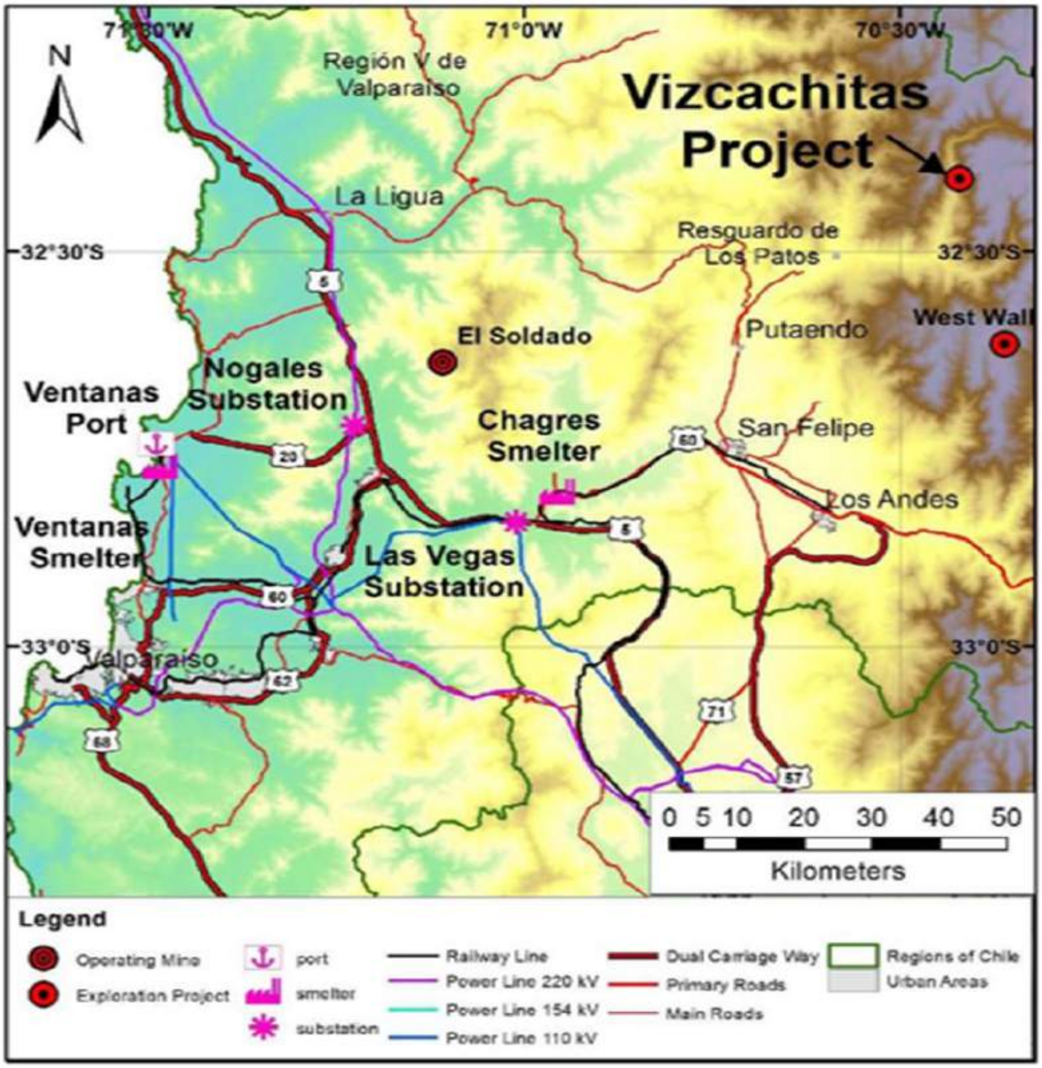
18. PROJECT INFRASTRUCTURE

18.1 Vizcachitas Site

The Vizcachitas Project is located at approximately 32.41°S and 70.42°W in the western foothills of the Andes at an average elevation of 2,100 masl. The central UTM coordinates are 6,413,600N and 366,200E (Datum WGS84, Zone 19H). The Property is approximately 150 km northeast of Santiago and 46 km northeast of the town of Putaendo in the Province of San Felipe, Valparaiso Region, Chile.

Figure 18.1 shows the Vizcachitas site location and regional reference points.

Figure 18.1: Vizcachitas Property Location



The Vizcachitas Project is located close to extensive infrastructure such as roads, rail and port access. A project of this size will require substantial additional infrastructure to maintain operations.

The PEA reviewed three throughput capacity cases with the corresponding infrastructure:

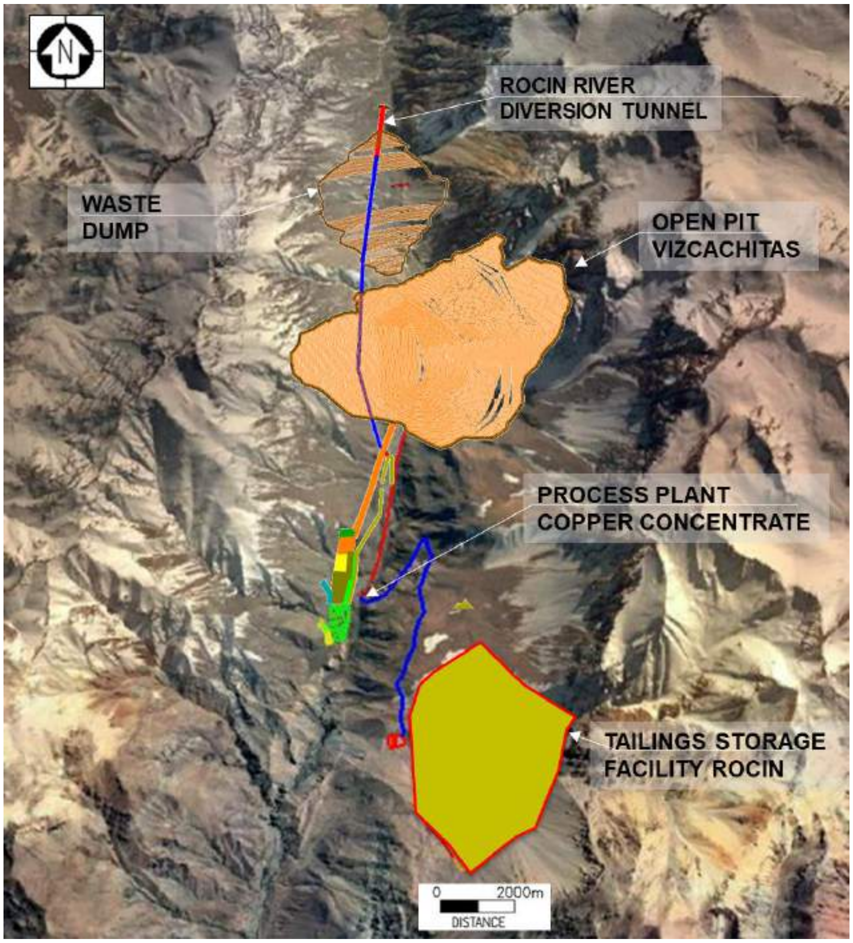
- 55 ktpd process plant throughput capacity
- 110 ktpd process plant throughput capacity
- 200 ktpd process plant throughput capacity

To assure optimum performance, the project will require the following infrastructure:

- Process Water Supply
- Rocin River Diversion Tunnel
- Power Supply
- Tailings Storage Facilities (TSF)
- Access and Site Roads
- Concentrate Storage, Loading and Transport System
- Operations Platforms
- Mine and Plant Infrastructure
- Ground Material Transport

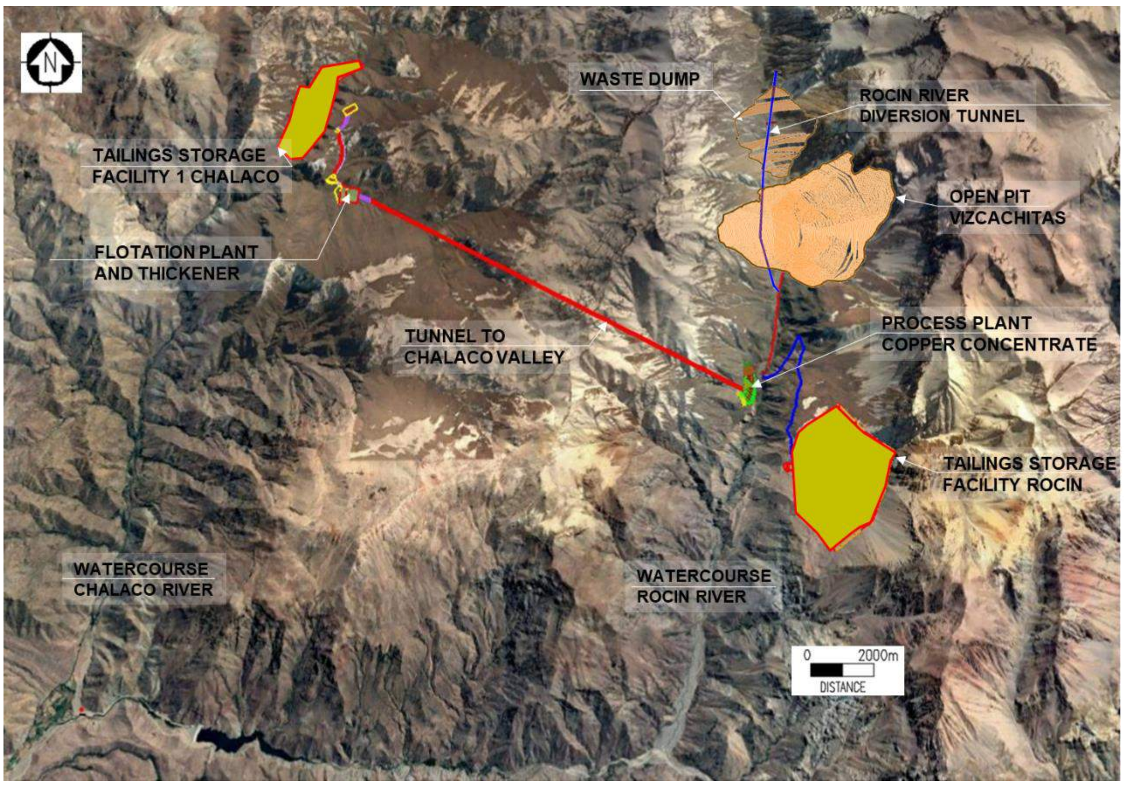
The 55 ktpd case considers that all project infrastructure facilities (open pit mine, concentrator plant, High-Density Thickening Plant and TSF) are located in the Rocin valley. See (Figure 18.2)

Figure 18.2: General Layout for 55 ktpd Case



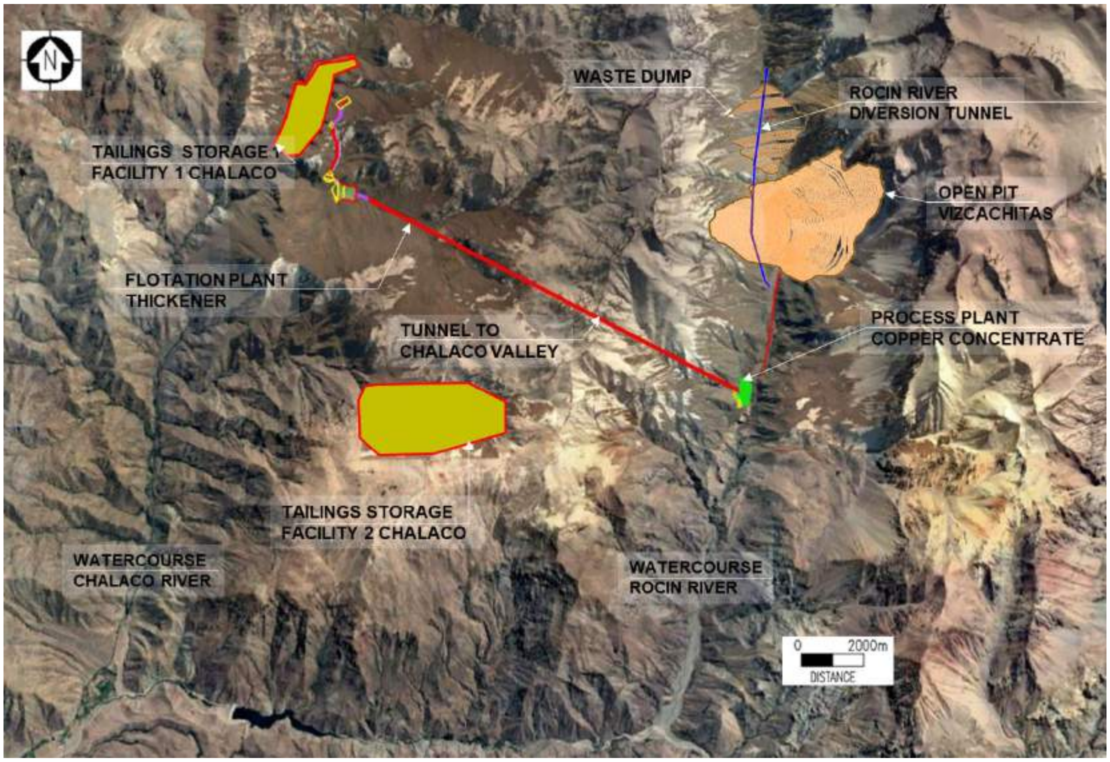
The 110 ktpd case starts the operation with a similar configuration as the 55 ktpd case (with all the infrastructure in Rocin valley), but once the Rocin TSF is filled (year 18 of production approximately) conventional tailings (55% solids) will be transported to Chalaco valley to the High-Density Thickening Plant (72% solids) to be deposit on TSF1 Chalaco. See (Figure 18.3)

Figure 18.3: General Layout for 110 ktpd Case



For the 200 ktpd case the open pit mine, crushing and grinding will take place in the Rocin valley. However, the flotation and tailing plants will be located in the Chalaco valley. The 55% solids tailings produced in the flotation and conventional tailings plant will be transported to the High-Density Thickening Plant (72% solids) to be deposited on TSF1 Chalaco and then on TSF 2 Chalaco. See Figure 18.4

Figure 18.4: General Layout for 200 ktpd Case



18.2 Process Water Supply

The process fresh water make-up requirements by case are presented in Table 18.1. Fresh water is required for cleaner flotation, molybdenum flotation, cooling, reagent mixing and concentrate washing.

Table 18.1: Make-Up Water Requirement

Plant Size ktpd	Water Demand m^3/h (l/s)	Design m^3/h (l/s)
55	864 (240)	1000 (280)
110	1730 (480)	2000 (560)
200	3150 (875)	3620 (1000)

These estimates use the throughput as input and assume thickened tailings. A summary is shown on Table 18.2.

Table 18.2: Water Requirement Estimate by Case

55 ktpd Case								
Item	Solid %	Water %	Solid tph	Water tph	Total tph	Density t/m³	Volume m³/h	Water Make-Up l/s
Thickened Tailings	72	28	2,077	807	2,884	1.82	1,584	240

110 ktpd Case								
Item	Solid %	Water %	Solid tph	Water tph	Total tph	Density t/m³	Volume m³/h	Water Make-Up l/s
Thickened Tailings	72	28	4,153	1,615	5,768	1.82	3,169	480

200 ktpd Case								
Item	Solid %	Water %	Solid tph	Water tph	Total tph	Density t/m³	Volume m³/h	Water Make-Up l/s
Thickened Tailings	72	28	7,551	2,937	10,488	1.82	5,763	875

Subsidiaries of Los Andes currently own consumptive water rights for 500 l/s in the Aconcagua River (sufficient for the 55 ktpd and 110 ktpd cases) with an extraction point located on the Aconcagua River approximately 80 km from the project site. If the 200 ktpd case is selected, Los Andes Copper will need to secure additional consumptive water rights to meet requirements and make water available for the Project. Alternatives are being evaluated for the fresh water extraction and pipeline transportation to the plant sites for the different options.

18.3 Rocin River Diversion Tunnel

The open pit mine and waste areas are in the Rocin River watercourse. It is necessary to divert the watercourse using a tunnel and return it to the river downstream of the project installations, as shown in Figure 18.5

The river diversion civil works will include:

- Catchment upstream of the mine installations.
- A tunnel of approximately 7 km with a section of 5 m x 5 m and a slope of 1%. The tunnel design has considered a precipitation event associated with a return period of 1 in 100 years of 200 m^3/s, as shown in Table 18.3.
- Tunnel exit and civil works to return the water to its normal course.
- Rain water collecting channel surrounding lateral hills on both sides of the Rocin valley and delivering it to the water course.
- Return water to the river downstream is designed considering a series of retaining walls located at the tunnel exit to slow the water flow down and return it to the original water course.

Table 18.3: Average Daily Peak Water Flow at Rocin River by Month (m^3/s)

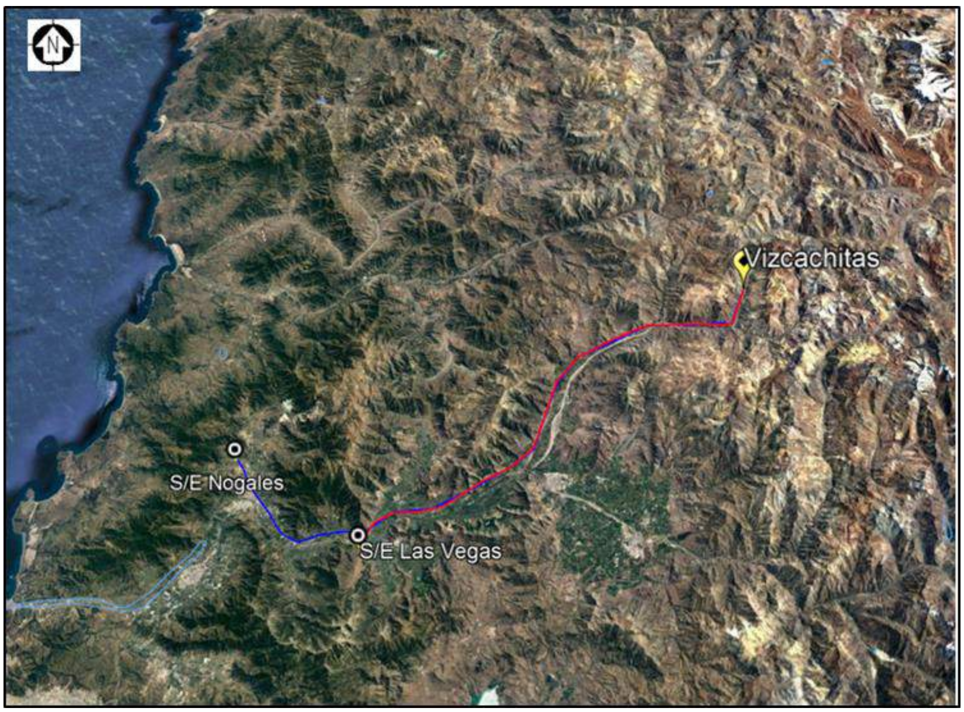
Period Return (Years)	APR	MAY	JUN	JUL	AUG	SEP	OCT	NOV	DEC	JAN	FEB	MAR
2	1.6	2	2.4	2.4	3.2	4.1	14.7	24.4	17.3	10	5.7	3.9
5	2.7	3.7	5.4	6.1	7.6	7.8	25	45	41.9	23	11	6.5
10	3.6	5.3	8.6	11.1	12.9	11	31.8	58.7	66.6	35.5	15.6	8.6
25	4.9	8.1	15	23.3	24	15.7	40.4	75.9	109	56.5	22.6	11.4
50	6	10.9	22	39.8	37.1	19.9	46.7	88.7	149.9	76.2	28.8	13.8
100	7.3	14.3	31.7	67.2	56.3	24.5	53.1	101.4	199.6	99.8	35.7	16.3
Distribution	LP	LP	LP	LP	LP	LN	Gumbel	Gumbel	LN	LN	LN	LN

Source: Compiled from fluviometric statistics from DGA (Chile) – 1950 - 2007

18.4 Power Supply

The power requirements for the three cases exceed 100 Mva requiring a 220 kV supply line. The tap point would be at the Nogales substation located south-west of the project site. The prospective routing of the transmission line from Nogales to the project site is shown in Figure 18.5. The overall transmission line extends for approximately 105 km to the plant sites.

Figure 18.5: 220 kV Overhead Powerline Route (source: Google Earth, Nov 2018)



18.4.1 Site Electrical Power Distribution

Table 18.4 shows the power requirement estimated for each case evaluated.

Table 18.4: Electrical System Requirement by Plant Size

Item	Power Demand (MW)		
	55 ktpd	110 ktpd	200 ktpd
Mine	4.6	7.8	12.5
Crushing	2.5	2.5	2.5
Grinding	54.0	90.0	144.0
Bulk Flotation	9.0	14.5	24.0
Selective Flotation	1.0	1.2	1.5
Tailing Handling	28.2	41.9	68.7
Concentrate Handling	15.0	15.0	15.0
Water Supply	7.3	15.0	21.5
Power Supply	3.0	3.0	3.0
Grinded Mat. Transport	0.0	0.0	28.2
Infrastructure	1.5	1.5	1.5
TOTAL	126.1	192.4	322.4

The Vizcachitas main substation will be located directly adjacent to the process plant building. From there, medium voltage electrical power will be distributed to the concentrator and the rest of the site at either 23 kV or 12 kV.

The electrical power supply for the water pipeline and transport systems for tailings and ground material will be by parallel 23 kV powerlines.

18.5 Process Plant Earthworks

The cubic metres of excavation and fill vary according to each case and topography of the selected location. Estimated values are presented in Table 18.5.

Table 18.5: Excavation and Fill by Plant Size

Plant Size (ktpd)	Excavation (m^3)	Fill (m^3)
55	600,000	250,000
110	750,000	550,000
200	1,000,000	750,000

18.6 Tailings Storage Facilities (TSF)

Mineral processing produces copper and molybdenum concentrates and tailings. The flotation tailings should be chemically benign and are anticipated to be non-acid generating.

Final location and design for tailings storage facilities in Rocin and Chalaco valleys will be defined in more detail in future engineering studies. For this PEA, a referential position and volume was assumed.

The 55 ktpd and 110 ktpd cases will have infrastructure facilities on the Rocin River valley, including open pit mine, crushing, grinding, flotation, tailings thickening and TSF. However, when the capacity of Rocin TSF is reached (year 18 of production for 110 ktpd case) conventional tailings will be pumped through a tunnel to Chalaco valley, where they will be thickened and deposited in TSF1 Chalaco. The layouts for the 55 ktpd and 110 ktpd cases are shown in Figure 18.2 and Figure 18.3.

The 55 ktpd case fills the Rocin TSF initial capacity in year 36 of production. At that time, different alternatives will have to be analysed to increase the capacity for tailings disposal in the Rocin valley.

For the 200 ktpd case, the processing facility will be divided between the Rocin and Chalaco valleys. Crushing and grinding will be located in the Rocin valley. Flotation and tailings process and storage will be located in the Chalaco valley, immediately west of the Rocin valley. A tunnel and a pumping system will be utilized to transport the ground material from the Rocin valley to the process plant in the Chalaco valley. A gravitational connection will carry the tailings to the TSF. Figure 18.4 shows two potential areas for TSFs, but more areas are possible in Chalaco valley.

Table 18.6 shows an estimate of tailings generated for each plant size.

Table 18.6: Annual Tails & Lifetime 72% Solids Tailings Production by Case

Item	Unit	Plant Size		
		55 ktpd	100 ktpd	200 ktpd
Tailings produce per year	Mt/y	27.3	54.6	99.3
Mine life	years	57	43	28
TSF Rocin life	years	36	18	N.A.
TSF 1 and TSF 2 Chalaco life	years	N.A.	25	28
Total tailings produced	Mt	1,556	2,348	2,780

18.6.1 Tailings Storage Facility Design

The tailings from the flotation plant will be concentrated in the tails thickener to 55% solids and then pumped to the High-Density Thickening Plant adjacent to the TSF. Tailings will be then concentrated to 72% solids and will be pumped by positive displacement pumps and distributed along the TSF area.

18.6.2 Underdrain System

To ensure tailings storage facility stability and avoid embankment liquefaction, a drainage system is envisioned. Given the use of thickened tailings (up to 72% solid) a simpler system can be considered. Drainage water will be collected and returned to the process plant for reuse. As the size of the TSF increases, its footprint expands downstream requiring phased expansion of the underdrain system.

18.6.3 TSF Diversion Channel

A water diversion channel will be required to avoid rain and other fresh water coming into contact with the tailings. The design of the channel considers a precipitation level associated with a return period of 1 in 100 years.

18.6.4 Tailings Transport System

Tailings will be pumped from a conventional thickening plant at 55% solids to a High-Density plant next to each TSF, where they will be deposited at 72% solids.

For the 110 ktpd case, after TSF Rocin is filled, conventional tailings will be pumped through a tunnel from the Rocin valley to the High-Density Thickening plant in the Chalaco valley, where 72% solids tailings will be deposited.

The preliminary design includes the use of slurry pumps and building a tunnel that provides room for the pipes, pump stations, electrical substations for every pumping station, return water for the grinding process, tanks and electric rooms.

To pump from an elevation of 1,860 masl to 2,150 masl will require seven pumping stations in a 9 km tunnel with a 3.5 m width for a service road, two 36” diameter slurry pipes, one 36” water return pipe and tanks. Electric rooms and a 23 kV energy powerline must be included in the design area.

18.6.5 TSF Reclaim Water System

The TSF water reclaim system includes a water collecting system located next to the thickening system and a gravity pipeline to return the water to the reclaimed water tank located next to the process plant. Drainage embankment water will be also discharged to the reclaimed water tank. All reclaimed water will be pumped back to the process plant by a pipeline that follows the same route as the slurry pipeline. Centrifugal pumps will be used for pumping the reclaimed water back to the plant.

18.7 Roads

18.7.1 Access Road

Access road improvements from San Felipe to Resguardo Los Patos for the different plant options will be the same. Access from a Truck/Rail transfer station (to be located near San Felipe) to Resguardo Los Patos will be along approximately 40 km of existing roads. These roads will require expansions and improvements, as well as bypass alternatives in specific sections.

18.7.2 Site Roads

Currently the project site is accessible from the village of Resguardo Los Patos via a 24 km gravel road that is 3 m to 5 m wide.

The plant location in the Rocin Valley will be accessed from the Chacrillas Dam via the existing gravel road which will need to be expanded to a minimum 9 m width and will require the construction of three bridges. A new 5 km road will be required to connect the plant with the TSF.

For the 200 ktpd case, the processing facility will be divided between the Rocin and Chalaco valleys. Crushing and grinding will be installed in the Rocin valley. Flotation and tailings processing and storage will be located at the Chalaco valley. A tunnel and a pumping system will be utilized to transport the ground material from the Rocin valley to the process plant in the Chalaco valley. A gravitational connection will carry the tailings to the TSF. A new 18 km gravel road will be required to access the areas in the Chalaco valley.

18.8 Concentrate Storage, Loading and Transport

Copper concentrate will be stockpiled at an enclosed warehouse with 6 kt capacity. The storage warehouse will be located adjacent to the filter discharge conveyor and will have a tripper belt conveyor for concentrate distribution.

The concentrate will be loaded into 28 t sealed container trucks and be transported to a transfer station with 15 kt storage capacity near San Felipe (65 km to the south-west from the plant). The concentrate will then be loaded into trains to be delivered to the port of Ventanas.

18.9 Ventanas Port Facilities

Copper concentrate will be unloaded at Ventanas port which currently handles copper concentrate volumes from other mining operations. Concentrate storage facilities as well as the ship loading system maybe need to be upgraded or expanded. The cost for these facilities was not included in this PEA study.

18.10 Site Accommodation

No installation is provided for a permanent camp. Project staff will use daily transportation from San Felipe, Los Andes, Putaendo and/or other neighboring towns.

The following infrastructure is considered for operations staff:

- Dining hall
- Change house
- Warehouse and laboratories

18.11 Fuel Storage

Fuel storage will consist of a horizontal cylindrical metal tank made of carbon steel with a 75,000 litre capacity. The storage facility will be located in a bunded area with a spillage containment capacity of 110% of the tank volume. The fuel storage area will be clearly marked and will have a perimeter fence 1.8 m high and be equipped with fire extinguishers. This facility will be managed by a fuel supplier.

18.12 Potable Water Supply

Potable water will be provided in sealed containers by a company approved by the health authority. Potable water will have to meet all the requirements of Chilean standard NCh 409.

18.13 Ancillary Site Buildings and Facilities

Various ancillary facilities will be located near the process plant. The buildings and facilities will include the following:

- Administration building
- Assay and metallurgical laboratory facilities

- Change house for personnel
- First aid or clinic building
- Gatehouse at the entrance to site
- Concentrator warehouse and an attached workshop building
- Potable water system
- Sewage treatment plant

Mine ancillary facilities will be located near the primary crusher. These facilities will include:

- Warehouse with offices
- Heavy vehicle workshop
- Tire shop
- Maintenance and welding shop
- Truck wash bay
- Fuel storage depot
- Effluent treatment facility
- Mine explosives storage facility

Various ancillary facilities will be located near the TSF. The buildings and facilities will include the following:

- Administration building
- Change house for personnel
- Gatehouse at the entrance to site
- Potable water system
- Sewage treatment plant

Commercially available packaged sewage treatment plants will be installed at the main concentrator facilities, mine support facilities and TSF. Effluent will need to meet Chilean regulations for discharge or undergo additional treatment until it does. Technical details will need to be developed during the next phases of project development.

19. MARKETING STUDIES AND CONTRACTS

For this marketing assessment, assumptions are based on metallurgical data with respect to the copper and molybdenum characteristics of the Vizcachitas concentrates. The comment and outlook on concentrate marketability and related smelter charges, including treatment, refining, penalty details, payment timing, metal accountability, and other contract terms, are based on data from other projects, current market understanding and information available in the public domain. As the Project progresses through the next phases of development, it is recommended that further review be made of market conditions and conclusions drawn as required.

19.1 Commodity Supply and Demand

Most analysts believe that the long negative cycle for copper and other commodities has already ended. It is expected that for normal economic growth in China, India, Europe and United States medium term prices should rise again. In the short term, uncertainty due to the commercial trade dispute between USA and China is causing slower recovery than expected.

19.2 Smelter Capacities and Utilization.

For the most part, smelter capacity is fixed. The relationship between capacity and utilization dictates a smelter’s profitability, hence it’s setting of treatment charges (TC), refining charges (RC) and other costs.

In the long term, TCs and RCs are expected to increase to cover additional smelter costs, particularly as environmental legislation becomes more stringent regarding airborne discharge and other effluents.

The PEA has considered TCs of 102 USD/t and RCs of 10.2 cUSD/lb.

19.3 Ocean Freight

Currently, the availability of vessels significantly exceeds the demand. Opportunities for reasonable freight costs are available, particularly with negotiation of long-term freight contracts.

The PEA has assumed a cost of 10.09 cUSD/lb for freight and insurance.

19.4 Future Metals Pricing

The general industry consensus is for copper price to stabilise as developing countries such as India and China take up production once again.

Medium and long term consensus copper values are as shown in Table 19.1 and Table 19.2. The PEA has considered using metals values of 3.00 USD/lb copper, and 22 USD/kg molybdenum and 17.0 USD/oz silver as the base case.

Table 19.1: Copper (USD/lb) - Nominal[3]

Survey Date: July 16, 2018

Long Term 2023-2027	
High	3.7
Low	2.8
Consensus (mean)	3.3

Table 19.2: Molybdenum (USD/kg) - Nominal[4]

Survey Date: July 16, 2018

Long Term 2023-2027	
High	26.6
Low	18.3
Consensus (mean)	22.5

[3] *Source: Energy and Metals Consensus Forecast (July, 2018)*
[4] *Source: Energy and Metals Consensus Forecast (July, 2018)*

20. ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Introduction

The environmental studies and permits that are necessary to execute the Project can be divided into those required to complete the pre-feasibility and feasibility studies and those that will be needed to construct and operate the mine.

The Project will have to conduct further infill, metallurgical and condemnation drilling to complete the pre-feasibility and feasibility studies. For this drilling, an Environmental Impact Statement or *"Declaración de Impacto Ambiental"* ("DIA" for its acronym in Spanish) is required.

For the construction and operation of the mine, further environmental studies and permits would be necessary. These would include an Environmental Impact Study or *"Estudio de Impacto Ambiental"* ("EIA" for its acronym in Spanish), environmental sectorial permits and an approved mine closure plan.

20.2 Legal Framework for Environmental Studies

The key pieces of environmental legislation are:

1. **Chilean Constitution**.

Article 19 No. 8 of the Chilean Constitution provides that:

- Citizens have the right to live in a pollution-free environment and;
- The state, through legislation, must protect this right and ensure the conservation of nature.

2. **Environmental Legislation**.

The main regulatory framework is set out in Law 19,300 (Environmental Law). This law aims to:

- Bring together fragmented and sector-specific regulations into a single legislative framework.
- Create a single environmental liability system.
- Introduce several procedures for assessing environmental impact. The most important of these, the Environmental Impact Assessment System (SEIA)

The Environmental Law was amended by Law 20,417, which was published in the *Official Gazette* on 26 January 2010. Law 20,417 introduced:

- An Environmental Superintendence ("SMA" for its acronym in Spanish).
- Substantial changes to the Environmental Law, for example: enhancing the role of the local community and involving it more in environmental screenings; new strategic environmental assessment; alternative and faster proceedings; larger fines and innovative sanctions to encourage compliance; and the creation of an integrated system of conservation and protected areas.

Finally, the Law 20,600, which was published in the *Official Gazette* on 28 June 2012, established the new Environmental Courts.

3. **General environmental bases**, including International Labour Organization 169, participation of indigenous communities.

4. **Specific Regulations for:**

- Emissions to the atmosphere and air quality
- Liquid and solid waste
- Dangerous substances
- Cultural heritage
- Labour conditions
- Transport safety

20.2.1 Environmental Impact Assessment

According to article 8 of the Chilean Environmental Law No 19,300 enacted in 1994, and its Regulation, Decree Supreme No 40 / 2012 of the Ministry of the Environment (the "EA Regulation"), every project or activity likely to have an environmental impact and included in a list of projects or activities established in article 10 of the Environmental Law, must undergo a mandatory environmental impact assessment process. Said list includes a variety of projects, such as dams, ports and airports, industrial and mining projects, among others. Therefore, to determine if the project needs to be assessed, it is necessary to analyze whether it is included in the list established in the Environmental Law and if it meets the specific thresholds as described by the EIA Regulation.

Once an environmental impact assessment is mandatory, the Environmental Law sets forth two ways to comply with its requirements: (i) by filing an Environmental Impact Statement ("DIA") or (ii) by filing an Environmental Impact Study ("EIA"). Environmental Studies are designed for large-scale projects that may produce significant impacts on the environment, whereas Environmental Statements are thought for simpler and lower scale projects. Once an EIA has been filed, the Environmental Assessment Agency ("SEA") has a 120 business-day term to evaluate it. Such term may be extended in certain cases up to 60 additional business days. In case of a DIA, the original term is 60 business days extendable for another 30 business days if required. In practice, an Environmental Impact Study may take much longer to be approved than the period set forth in the Law. For an Environmental Impact Study, the average term can be approximately one year, whereas for an Environmental Statement the average term is approximately six months.

An important aspect to bear in mind is that EIAs contemplate a mandatory 60-day term for a public participation process, whereas DIAs would only be subject to public participation provided either 2 citizen organizations or 10 individuals request such procedure and the SEA determines its applicability. In this case, the term for public participation would last for 30 days. Within the aforementioned terms of the public participation process, any individual or legal entity is entitled to submit comments regarding the project, all of which must be considered in the final decision.

After the submission of the DIA or EIA, the environmental impact assessment procedure continues with one or more rounds of questions made by diverse administrative agencies with environmental jurisdiction, such as the National Forest Corporation, the Agricultural and Livestock Agency, the Health or Sanitary Authority, among others. Those rounds of questions are intended to clarify the environmental aspects of a project. After those questions have been answered by the project developer, the environmental impact assessment finalizes with an Environmental Qualification Resolution ("RCA") that approves (sometimes introducing conditions or restrictions) or rejects the project or activity that has been evaluated.

20.2.2 Environmental sectoral regulation and permits

Upon obtaining the RCA, the company has to obtain other specific environmental sectoral permits that could be applicable. In these cases, the administrative agencies cannot deny granting these permits on environmental grounds, as the environmental impact assessment procedure serves as a 'single tenet' mechanism to obtain all relevant environmental sectoral permits once the RCA is granted.

Sectoral environmental permits are issued by a State Administration entity, which because of their environmental content are listed in the environmental impact assessment regulation. These permits must be issued for all projects and activities submitted to the SEA.

Within the sectoral permits, the key permits for the Project are granted by the National Geology and Mining Service ("SERNAGEOMIN" for its acronym in Spanish), and the General Water

Directorate ("DGA" for its acronym in Spanish), which can take up to 12 months or more to be approved.

DGA approvals are important when scheduling the permitting process for the Project, at least two permits from the DGA are required: approval of the water works, such as a tailings deposit, and a water course intervention, such as the Rocin River diversion.

SERNAGEOMIN provides the main permits for the operation of the mine, including the construction of tailing deposits, the accumulation of minerals or waste dump and the mine closure plan.

Finally, other possible permits to be identified as applicable in the environmental impact assessment of the mine are permits for wastewater and sewage treatment plants, for the accumulation of hazardous waste, industrial qualification and land use change.

20.2.3 Legal Requirements Regarding Indigenous People

Environmental baseline studies include the observation of the International Labour Organization recommendation 169, which defines the rights and levels of participation required from any indigenous stakeholders within the direct or indirect impact area of a project. The Chilean government has a National Indigenous Development Corporation ("CONADI" for its acronym in Spanish) which works with both the property owners and the local indigenous population to ensure that their rights and participation in the environmental permitting processes are correctly considered.

In accordance with the environmental impact assessment regulation, if a project has the ability to directly and significantly affect indigenous people located in the nearby area, it is required to submit an EIA, and SEA shall undergo a special consultation process with those indigenous people, in accordance with article 6 of the Indigenous and Tribal People Convention (No. 169), in force in Chile since September 2009. This is an obligation for the State, not the developer of the project. Thus, it is the SEA–the entity that shall conduct the consultation process. It is important to bear in mind that direct and significant affectation to indigenous communities could be either recognized in the EIA or detected during any stage of the environmental impact assessment procedure, thereby triggering the duty to consult.

The criteria as to the direct and significant affectation required to trigger the duty to consult is a unique and discretionary decision of SEA, regardless of other sectoral agencies, for it is only SEA the administrative agency in charge of issuing the administrative measure (the RCA) which may affect the indigenous people directly.

20.3 Historical Environmental Information

As part of the "Initial Feasibility Study" completed by General Minerals, an environmental baseline was developed in 1998 by *Ingeniería y Geología Dos Limitada* (INGEDOS). The report studied the vegetation, vertebrate terrestrial animals, invertebrate aquatic animals and vertebrate aquatic animals. A total of 169 plant species were identified of which 69 % are indigenous. Out of a potential total of 243 vertebrate species, only 56 were identified in the area of the study. This low ratio is probably due to the pre-existing roads in the area and the fact that the study was completed at the end of winter. Two non-indigenous fish species were identified in the Rocin River (INGEDOS 1998).

In 2007, an environmental baseline study was carried by *Jaime Illanes y Asociados* to support a DIA submitted in 2008. The study identified 49 species of flora and 2 species of reptile in the area. Bibliographic review estimated the presence of 24 species of birds and 8 species of mammals, 6 of which have conservation issues, including Lama Guanicoe.

A DIA was submitted by Los Andes Copper on February 22, 2008 to the environmental authority at that time, Chile's National Environmental Commission or *"Corporación Nacional del Medio Ambiente*" in Spanish (CONAMA). CONAMA denied the permit on October 23, 2008 based on 4 main points:

- There was no adequate road between Casablanca and Resguardo de Los Patos
- The project contingency plan was inadequate
- The river crossings had been made without the correct DGA approvals
- The project may unduly impact the touristic value of the area

On May 28, 2018 Los Andes Copper submitted a DIA into the SEIA. Although it should be noticed that this time, the DIA was the result of a compliance programme in response to a notice of an environmental administrative infringement issued by the SMA, requesting that Los Andes Copper conduct an environmental approval process for the drilling campaigns conducted during 2015/2017 and that the SMA claimed had been conducted without environmental permission.

In a dispute of competencies between two state institutions, the SMA dismissed a letter sent by the SEA to the Company explicitly exempting the 2015/2017 drilling campaigns from requiring an environmental impact assessment process. It should be noted that the SEA is the entity responsible for the environmental impact assessment process, and as part of its responsibilities, is the entity that determines whether a project has to undergo such process. In a dismissal of such attributes, and based on a 2013 change in the environmental assessment regulations, the SMA sought to bundle the recent 2015/2017 drilling campaigns with the 2007/2008 drilling programme considering the recent campaigns a modification of the 2007/2008 programme.

Therefore, the SMA concluded that the campaigns must have been submitted to an environmental impact approval process. As a compromise and without admitting to any wrongdoing, the Company agreed to submit the 2015/2017 drilling to such assessment process.

On April 3, 2019, the all members of the Regional Environmental Committee (Comision de Evaluacion Ambiental), unanimously moved to grant environmental approval for the DIA related to the drilling carried out in during 2015-2017.

20.4 Future Environmental Studies

These studies can be subdivided into those studies required to complete the pre-feasibility and feasibility studies and those that will be required to construct and operate the mine.

20.4.1 Studies Required to Complete the Pre-feasibility and Feasibility Studies.

To complete a pre-feasibility or feasibility study further drilling will be needed. Under Chilean law, to carry out intensive drilling to define projects as opposed to exploration drilling it is necessary to submit a DIA to the SEA for approval.

20.4.2 Studies Required to Construct and Operate the Mine.

20.4.2.1 Baseline Studies

The formal baseline studies and environmental data compilation around the Property must be extended or initiated to support an EIA application. The baseline studies must be carried out over at least one full year to ensure that the fauna and flora can be studied in all seasons. For this reason, these studies would normally be completed during pre-feasibility and feasibility phases. The studies correspond mainly to existing information review, analysis and development of documents to comply with environmental regulations. In addition to natural environmental data collection, the engineering completed during the pre-feasibility and feasibility studies is used to support project robustness and detail specific engineering requirements needed by the environmental services.

20.4.2.2 Gap Analysis

Gap analysis gathers all environmental background information available for the Project. This identifies the studies needed to be completed, updated or revised according to the requirements of the regulations or requirements of the different governmental authorities.

20.4.2.3 Environmental Baseline Update/Completion

After carrying out the gap analysis, further studies may be identified for components of the Environmental Baseline (including: flora, fauna, archaeology and others). Potential project impacts will need to be evaluated to determine which studies must be completed and which require additional information.

20.4.2.4 Environmental Permitting Conceptual Requirements

The environmental permits associated with the different potential impacts for the whole lifecycle of the Project (design, construction, operation, closure) need to be identified. Some of the environmental sectoral permits' worth highlighting are:

- **Permit for the approval of closure plan of a mining site.**

The permit for the execution of the closure plan of a mining site is established in article 6 of Law No 20,551 that approves the Closure of Works and Mining Facilities Law. This statute requires all mining sites to have a closure plan approved by Sernageomin, prior to the start of mining operations and must contain all the facilities of the project.

There are two types of procedures for approval of the closure plans which depend on the production capacity of the mining site:

General Application Procedure, which must be submitted by those mining companies whose purpose is the extraction of one or more mining deposits, and whose mineral extraction capacity is greater than 10,000 t per month per mining site.

Simplified Procedure, which must be presented by those mining companies whose purpose is the extraction of one or more mining deposits, and whose capacity to extract is less than or equal to 10,000 t per month per mining site.

Additionally, exploration and mining survey must be subject to this procedure.

- **Permit for the construction of tailing deposits.**

This permit is set forth in article 9 of Supreme Decree No. 248 of 2006 of the Ministry of Mining, the Regulation for the approval of design, construction, operation and closure of the tailings deposits.

In order to be granted, the developer shall ensure the physical and chemical stability of the deposit and its environment, so as to protect the environment and avoid generation of risks for the people's health.

- **Permit for mineral waste dump or mineral accumulation.**

This permit is set forth in paragraph 1 of article 339 of Supreme Decree No. 132, of 2002, of the Ministry of Mining, which establishes Mining Safety Regulation.

The requirements for its granting consist in ensuring the physical and chemical stability of the dump or deposit and that it contains the maximum safety measures both in its construction and growth, in order to protect the environment and the life and physical integrity of people.

- **Industrial liquid waste and sewage treatment plants.**

Pursuant to article 71, letter b) of the Sanitary Code, the Health or Sanitary Authority is responsible for approving the projects related to the construction, repair, modification and extension of any public or private work intended for the evacuation, treatment or final disposal of drains, wastewater of any nature and industrial or mining waste.

Pursuant to article 71, paragraph 2, before putting into operation the mentioned works, they must be authorized by the Sanitary Authority. In the same sense, DFL Nº 1/89 of the Ministry of Health, indicates that they require express sanitary authorization, the operation of works destined to the provision or purification of potable water of a population or to the evacuation, treatment or final disposition of drains, wastewater of any nature and industrial or mining waste.

- **Hazardous waste accumulation permit.**

According to article 29 of Supreme Decree No. 148 of 2003, the Sanitary Regulation on Hazardous Wastes Management, any site dedicated to the accumulation of hazardous waste must have the relevant sanitary authorization, unless it is included in the sanitary authorization of the main activity.

The design, construction, expansion and / or modification of any site that implies accumulation of 2 or more incompatible hazardous waste or that contemplates the accumulation of 12 or more kg of acute toxic waste, or 12 t or more of hazardous waste that present any other characteristic of danger, must have a project previously approved by the Health Authority.

- **Land use change for facilities.**

Pursuant to article 55 of the General Urban and Constructions Law, every project located outside the urban limit set forth in the specific urban planning plan, shall obtain a favourable report from the Housing Regional Ministerial Secretary (Secretaría Regional Ministerial), together with the Agriculture and Livestock Agency ("SAG"), prior obtaining the applicable construction permit to be issued by the Municipality.

This permit is intended to protect rural areas deemed to be preserved by law as areas likely for agricultural, timber and/or livestock activities. Based on past experience with this permit, SAG usually requires compensation for the soil losing its rural conditions, thereby classifying those rural soils in accordance with their agricultural potential and requiring different compensation ratios depending on its quality.

21. CAPITAL AND OPERATING COSTS

The capital and operating cost estimates were developed in Chilean Pesos (CLP) and United States dollars (USD) according to the source currency of costs. The exchange rate (CLP/USD) used was 620. All costs are estimated as of the Effective Date of this Technical Report. All cost projections are presented on a nominal dollar basis.

21.1 Capital Cost Estimate

Capital cost estimates are composed of the following:

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

Table 21.1 shows the initial, sustaining and deferred capital cost estimated for the alternatives evaluated.

Table 21.1: Capital Cost Estimate (Nominal Values (kUSD))

Description	Initial			Sustaining and Deferred		
	55 ktpd	110 ktpd	200 ktpd	55 ktpd	110 ktpd	200 ktpd
Direct Costs						
Diversion Rocin River	52,912	52,912	52,912	2,500	2,500	2,500
Access	29,731	29,731	23,557			
Concentrate Transport	29,932	29,932	29,932			
Pipeline Rocin-Chalaco			73,749			
General Facilities	32,746	35,000	40,754			
Operations Platform	29,820	35,145	76,680			
Mine	184,363	277,465	359,328	624,333	1,025,196	1,218,390
Plant	228,440	439,016	687,999			
Tailing Management Facilities	152,290	173,057	230,639		98,250	
Water Reclaim System	2,926	3,653	4,430			
Water Supply System	35,844	47,382	62,132			
Power Supply System	88,125	124,539	168,597			
Total Direct Costs	**867,129**	**1,247,831**	**1,810,708**	**626,833**	**1,125,946**	**1,220,890**
Total Indirect Costs	**164,299**	**242,672**	**361,191**			
Contingency	**268,605**	**384,294**	**651,570**			
Total Capital Cost	**1,300,034**	**1,874,797**	**2,823,469**			

Direct costs were estimated using:

- Material take-offs (MTOs) based on preliminary layouts, process flow diagrams, and topographic information
- Historical data
- Allowances for similar projects

Table 21.2 shows the main estimation basis by area.

Table 21.2: Estimation Basis by Area

Area	Cost Estimation	
	MTOs	**Price**
Mine		
Mine Works	Calculated	Calculated/Factorized
Mine Equipment	Calculated	Data Base
Workshops	Benchmark	Data Base
Infrastructure	Benchmark	Calculated/Factorized
Building	Benchmark	Data Base
Power Supply	Factorized	Data Base
Process Plant		
Process Equipment	Calculated	Data Base
Tailings Management Facilities	Factorized	Data Base
Building	Factorized	Calculated/Factorized
Civil Works	Factorized	Data Base
Power Supply	Calculated	Calculated/Factorized
Tailings Storage Facility		
Civil Works	Factorized	Factorized
Piping	Factorized	Factorized
Infrastructure	Factorized	Factorized
Contour Channels	Factorized	Factorized
Building	Factorized	Factorized
Support Equipment	Factorized	Factorized
Power Supply	Factorized	Factorized
Infrastructure		
Power Supply	Calculated	Data Base
Mine/Plant Water Supply	Calculated	Calculated/Factorized
Train Loading Station	Factorized	Calculated/Factorized
Contour Channels	Factorized	Calculated/Factorized
Building	Factorized	Data Base
Support Equipment	Factorized	Data Base
Infrastructure	Benchmark	Benchmark

Table 21.3 shows the mine equipment purchase value and Table 21.4 shows the unit construction costs considered.

Table 21.3: Mine Equipment Purchase Value

Equipment	Market Price kUSD
Electric Shovel - 73 yd^3	20,129
Hydraulic Shovel - 56 yd^3	8,064
Frontal Load - 31 yd^3	5,630
Truck - 330 st	4,590
Drill - Diesel 12 1/4"	2,880
Bulldozer - 890 HP	1,782
Whelldozer - 853 HP	1,800
Motor Grader - 533 HP	2,097
Watering Truck - 75.7 m^3	1,719
Diesel Charger Truck	720
Frontal Load - 5 yd^3	573
Service Truck	180
Contour Drill	1,260

Table 21.4: Unit Construction Values

Area	Price	Unit
Concrete	650	USD/m^3
Metallic Structure	7,000	USD/t
Piping	6	USD/kg
Rock Excavation	15	USD/m^3
Soil Excavation	5	USD/m^3
Filling	10	USD/m^3

21.1.1 Indirect Costs

Lump sum allowances or factors have been used to calculate indirect costs as is applicable for a PEA. At this level many of the sourcing and contract strategies are not defined, so reasonable and customary assumptions have been made based on experience with similar projects. Table 21.5 shows detailed indirect cost.

Table 21.5: Indirect Costs (Nominal Values (kUSD))

Description	55 ktpd	110 ktpd	200 ktpd
Freight & Insurance	20,811	29,948	43,457
Import Dutes	10,406	14,974	21,728
Spare Parts	13,007	18,717	27,161
Vendor Representatives	4,552	6,551	9,506
PFS-FS-EPCM	89,509	128,807	186,910
Start Up	13,007	24,957	45,268
Owner Cost	13,007	18,717	27,161
Total Indirect Costs	**164,299**	**242,671**	**361,191**

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 project 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 contingency of 30% has been included in the initial capital cost for all items, excluding mining equipment in certain scenarios. The mining equipment for the 55 ktpd and 110 ktpd cases was excluded from the contingency estimate, as they are based on recent actual quotes and a detailed estimation of quantities. The mining equipment for the 200 ktpd case, is included in the basis for contingency estimation.

21.1.3 Accuracy

This estimate has been developed to a level sufficient to assess/evaluate the project concept, various 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 +/-35%. This is based on the level of contingency applied, the confidence levels of the authors in their respective estimates and an assessment comparing this estimate to standard accuracy levels on PEA estimates.

21.1.4 Estimate Exclusions

The following items are not included in the capital estimate:

- All Owner's taxes, including any financial transaction tax, withholding tax, or value-added tax (VAT)
- Future foreign currency exchange rate fluctuations
- Interest and financing costs
- Escalation beyond first quarter 2019
- Risk due to political upheaval, government policy changes, labour disputes, permitting delays, weather delays, or any other force majeure occurrences.

21.2 Operating Cost Estimates

Operating costs have been estimated for the operating areas of Mining, Process Plant, Infrastructure and Administration. Costs were reported under subheadings related to the function of each of the areas identified.

The operating cost estimates are based on energy prices of 45 USD/MWh for electricity and 1.00 USD/l for diesel fuel. Table 21.6 summarizes the average unit operating cost for the first 8 years of operation by area. Labour costs for mine and process plant consider only up to Superintendent level and all superior positions are considered as administration costs.

The operating costs are considered to have accuracy of ± 35%, based on the assumptions listed in this section of the Report. All unitary operating costs are expressed in processed tonnes.

Table 21.6: Unit Operating Costs (USD/t plant feed; Nominal values, average first 8 years)

Description	Case 55 ktpd	Case 110 ktpd	Case 200 ktpd
Mine (*)	4.75	4.27	4.90
Plant	5.11	4.92	4.70
Infraestructure	0.18	0.18	0.18
Administration	0.20	0.19	0.20
Total (USD/t)	**10.24**	**9.57**	**9.98**

(*) Mine costs include the waste/mineral ratio for the first 8 year of operation

Table 21.7: Unit Operating Costs (USD/t plant feed; Nominal values, average LOM)

Description	Case 55 ktpd	Case 110 ktpd	Case 200 ktpd
Mina	3.59	4.40	4.72
Plant	5.11	4.92	4.70
Infraestruture	0.18	0.18	0.18
Administration	0.18	0.19	0.19
Total (USD/t)	**9.06**	**9.70**	**9.79**

21.2.1 Mining 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 and blast-hole stemming and other earthworks as may be 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, energy, diesel, lubricant consumption, tyres, materials, spare parts, third party services and others. These operating costs were adjusted for local labour rates and supply costs, while tracking recent experience for projects with similar fleets.

Table 21.8 and Table 21.9 show the mine unit operating costs with nominal values for material moved and by expense item.

Table 21.8: Mine Unit Operating Costs (USD/t) (Nominal values, for material moved)

Description	Case 55 ktpd	Case 110 ktpd	Case 200 ktpd
Drilling	0.18	0.15	0.15
Blasting	0.15	0.13	0.13
Loading	0.32	0.28	0.26
Earth movement	0.10	0.08	0.08
Support	0.02	0.02	0.02
Administration	0.10	0.08	0.08
Total [USD/t]	**1.85**	**1.72**	**1.70**

Table 21.9: Mine Unit Operating Costs (USD/t) by Expense Item (Nominal values, for material moved)

Description	Case 55 ktpd	Case 110 ktpd	Case 200 ktpd
Materials	0.28	0.26	0.26
Fuel	0.43	0.40	0.39
Third party services	0.11	0.10	0.10
Labour	0.26	0.24	0.24
Supplies	0.04	0.03	0.03
Maintenance	0.72	0.67	0.66
Others	0.02	0.02	0.02
Total [USD/t]	**1.85**	**1.72**	**1.70**

21.2.2 Process Plant Operating Cost

Process plant operating costs were developed covering the following unitary operations:

- Primary crushing and stockpiling
- Grinding
- Copper-molybdenum bulk flotation
- Molybdenum flotation
- Thickening of concentrates and tails
- Filtering
- Concentrates handling

Unit operating costs incorporate labour, energy, materials, spare parts, third party services and others. These operating costs were adjusted for local labour rates and supply costs, while tracking recent experience for projects with similar equipment. Table 21.10 shows operating costs for the process plant by expense item.

Table 21.10: Process Plant Operating Cost (USD/t plant feed) by Expense Item (Nominal values)

Description	Case 55 ktpd	Case 110 ktpd	Case 200 ktpd
Labor	0.75	0.72	0.48
Energy	0.97	0.94	0.91
Materials	2.29	2.20	2.29
Parts	0.34	0.33	0.32
Third party services	0.52	0.50	0.48
Other	0.24	0.23	0.22
Total [USD/t]	**5.11**	**4.92**	**4.70**

21.2.3 Infrastructure and Administration

Infrastructure operating costs were developed considering the following areas:

- Tailings storage facility
- Tailings transport
- Water supply
- Power supply
- Administration buildings
- Others

Administration costs include general administration of the company.

For all cases an Infrastructure cost of 0.2 USD/t was assumed, which includes labour, energy, materials, spare parts and third-party services. Administration cost is estimated at 0.2 USD/t.

21.2.4 C-1 Cash Costs

The C-1 cash costs were calculated using the economic model cash flow forecast values:

- Total operating costs
- Royalty costs including Mining Royalty and third party NSR
- Treatment costs, refining costs and transportation costs (i.e. third party rail fee, port handling, and ocean freight)
- Revenue from molybdenum and silver

To calculate the cash cost per pound of copper, total expenses (operating cost, NSR / royalty, and TCs, RCs, and transportation) less total revenue from molybdenum and silver were divided by the number of pounds of copper to be sold over the life of mine. The average life of mine cash cost is shown in Table 21.11 and Table 21.12 shows the average first 8 years (of operation) cash cost.

Table 21.11: Average Life of Mine Cash Costs

Description	Unit	55 ktpd	110 ktpd	200 ktpd
Operating Costs	kUSD	10,097,016	15,268,600	17,469,645
NSR	kUSD	434,760	620,835	696,754
Royalty	kUSD	641,679	790,004	819,253
TC/RC	kUSD	1,958,779	2,785,373	3,123,334
Transportation	kUSD	727,768	1,034,882	1,160,448
Total Cash Cost w/o Credits	**kUSD**	**13,860,001**	**20,499,695**	**23,269,434**
Molybdenum and Silver Credit	kUSD	2,068,426	3,071,756	3,473,998
Total Cash Cost w/ Credits	**kUSD**	**11,791,575**	**17,427,938**	**19,795,437**
Total Copper to be Sold	Mlb	7,742,210	11,009,381	12,345,195
Life of Mine Cash Cost				
Average Cu Cash Cost w/o Mo-Ag Credit	**USD/lb**	**1.79**	**1.86**	**1.88**
Average Cu Cash Cost w/ Mo-Ag Credit	**USD/lb**	**1.52**	**1.58**	**1.60**

Table 21.12: Average First 8 Years Cash Costs

Description	Unit	55 ktpd	110 ktpd	200 ktpd
Operating Costs	kUSD	1,599,569	2,975,458	5,148,375
NSR	kUSD	86,010	149,521	239,750
Royalty	kUSD	129,807	231,674	328,655
TC/RC	kUSD	395,032	681,472	1,093,403
Transportation	kUSD	146,771	253,195	406,245
Total Cash Cost w/o Credits	**kUSD**	**2,357,188**	**4,291,322**	**7,216,429**
Molybdenum and Silver Credit	kUSD	333,678	632,903	1,007,854
Total Cash Cost w/ Credits	**kUSD**	**2,023,511**	**3,658,418**	**6,208,574**
Total Copper to be Sold	Mlb	1,561,392	2,693,566	4,321,751
First 8 Years Cash Cost				
Average Cu Cash Cost w/o Mo-Ag Credit	**USD/lb**	**1.51**	**1.59**	**1.67**
Average Cu Cash Cost w/ Mo-Ag Credit	**USD/lb**	**1.30**	**1.36**	**1.44**

22. ECONOMIC ANALYSIS

This PEA assumes an economic evaluation based on a production plan that includes mineral resources in all categories (measured, indicated and inferred).

The results of the economic analyses discussed in this section represent forward-looking information as defined under Canadian securities law. The results depend on inputs that are subject to several known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here. Some of the information that is forward-looking includes, but is not limited to:

- Mineral resource estimates
- Assumed copper price
- The proposed mine and process production plan
- Projected recovery rates
- Energy price
- Infrastructure costs
- Ability to obtain sufficient process water to support planned project activities
- Ability to obtain sufficient electrical power to support planned project activities
- Ability to permit the Project.

22.1 Methodology Used

Tetra Tech has estimated the Project's net present value based on a discounted cash flow model. Using the mine plan as input, the model calculates annual quantities of metal production, the associated revenues, and the capital, operating and other costs to sustain the production.

The model considers closure costs as required by the Chilean legislative authorities (Law 20.551, see http://www.leychile.cl/Navegar?idNorma=1032158) and includes a valuation for remaining copper in the ground.

Chilean legislation contemplates three alternatives to guarantee the estimated cost for closure:

- Cash lodged in a bank account
- Bank guarantee
- Standby Letter of Credit issued by a bank with a minimum of A credit rating

The model has assumed that one of the latter two mechanisms will be used and has applied the estimated closure costs for three years starting the following year after operation finalizes.

22.2 Financial Model Parameters

The base-case discount rate is 8%. Chile is a politically stable country and the Vizcachitas Project has the same technical features as many other projects or operations in Chile.

The annual period when capital expenditures are initiated is defined as Period 1. NPV has been calculated to the year prior to initial capital expenditures.

The exchange rate is not a direct input in the financial model since all the input costs are converted to United States Dollars. However, a significant part of the cost will actually be in Chilean pesos (CLP) and Tetra Tech applied an exchange rate of 620 pesos per dollar in the cost estimation.

Since the analysis is based on a cash flow estimate, it should be expected that actual economic results might vary from these results. The PEA has been completed to a level of accuracy of ± 35%. The PEA is not a Preliminary Feasibility Study or Feasibility Study as defined by the NI 43-101 guidelines.

Economic parameters used for the evaluation are shown in Table 22.1.

Table 22.1: Main Economic Parameters

Description	Value	Unit
Cu Price	3.0	USD/lb
Mo Price	22.0	USD/kg
Ag Price	17.0	USD/oz
Energy Costs	45.0	USD/MWh
Inflation	None	---
Currency Fluctuation	None	---

22.3 Taxes and Royalties

The following is a summarized description of the Chilean income tax code applicable to mining companies.

22.3.1 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 would be applicable to the Project.

First Category Tax

First Category Tax is due on income derived from commercial, industrial and agricultural activities, mining, fishing and other extractive activities, investment and real estate. The tax rate applicable is 27%.

Additional Tax (Impuesto Adicional)

This tax operates as a withholding tax and affects, among others, 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 the shareholders not domiciled or resident in Chile are subject to an additional withholding tax on distribution at a rate of 35%. If the distributed amounts had been subject to First Category tax, a 27% credit is given against the additional tax. The additional tax must be withheld by the corporation.

The after-tax economic evaluation shown in this PEA assumes no dividends will be payable outside of Chile, i.e. only the 27% First Category Tax would be applicable.

Depreciation

Depreciation on fixed assets, except for land, is tax deductible by the straight-line method based on the asset's useful life in accordance with the guidelines of the SII (Servicio de Impuestos Internos de Chile) computed on the restated 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 useful lives of over five years. Accelerated depreciation considers useful lives equivalent to one-third of the normal useful lives, 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. A difference between accelerated and straight-line method is that the latter will 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, trademarks, etc. Depletion is not tax deductible.

Stock/Inventory

The costing of goods sold of production materials 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 so determined is however, adjusted for the manner stipulated for the annual monetary correction procedures.

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 as such does not exist. Dividends are necessarily 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.

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.

Losses

Losses incurred in the fiscal year are deductible. Furthermore, 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.

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 affected by income tax, except those 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%.

22.3.2 Mining Royalty Tax

All mining properties are subject to statutory obligations to the Chilean Government in the form of a Mining Royalty Tax or "Impuesto Específico a la Minería" in Spanish (IEM). 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 to 12,000 t copper equivalent: No tax applied

- 12,001 to 50,000 t copper equivalent: 0.5% to 4.5% of the Mining Operating Income according to the scale showed in Table 22.2.

More than 50,000 t copper equivalent: A different scale applies that starts at 5% of the Mining Operating Income for Mining Operating Margins less than 35%, and up to 34.5% for Mining Operating Margins in excess of 85%. This scale is shown in Table 22.2 and Table 22.3.

Table 22.2: Mining Royalty Tax Scale for Mining Exploitation under 50,000 t of Equivalent Copper

Cu Eq (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 22.3: Mining Royalty Tax Scale for Mining Exploitation 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 certain specific rules. Certain expenses such as losses from past periods, accelerated depreciation of fixed assets, etc. 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.

22.4 Production Summary

Data from the mine production schedule was used as the basis for the process production as presented in Chapter 16 (Mine Plan, Table 16.13, Table 16.19 and Table 16.24).

The life of mine revenue profile is shown in Table 22.4.

Table 22.4: Life of Mine Revenue Profile

Period	Case 55 ktpd				Case 110 ktpd				Case 200 ktpd			
	Total (kUSD)	Cu (kUSD)	Ag (kUSD)	Mo (kUSD)	Total (kUSD)	Cu (kUSD)	Ag (kUSD)	Mo (kUSD)	Total (kUSD)	Cu (kUSD)	Ag (kUSD)	Mo (kUSD)
1												
2												
3	**472,135**	443,647	6,375	22,113	**911,051**	850,632	11,939	48,480	**316,550**	300,299	3,916	12,335
4	**676,317**	624,382	9,273	42,662	**1,178,061**	1,093,148	15,662	69,251	**1,838,648**	1,721,852	22,542	94,254
5	**427,294**	386,013	7,053	34,228	**1,130,155**	1,039,863	13,622	76,669	**1,832,565**	1,678,549	24,790	129,227
6	**614,473**	576,648	7,044	30,781	**999,109**	914,523	13,175	71,411	**1,760,245**	1,627,539	24,384	108,322
7	**532,485**	482,289	6,886	43,309	**800,668**	699,566	11,955	89,146	**1,747,807**	1,564,419	23,458	159,930
8	**605,981**	566,758	6,457	32,767	**725,037**	656,172	9,809	59,057	**1,427,454**	1,304,967	18,882	103,604
9	**473,985**	431,729	6,724	35,532	**974,904**	901,000	12,682	61,223	**1,729,491**	1,572,859	20,265	136,367
10	**497,817**	455,344	6,583	35,890	**757,087**	688,264	10,062	58,761	**1,334,765**	1,209,186	17,738	107,840
11	**421,895**	357,655	6,023	58,217	**841,020**	783,732	11,520	45,767	**1,344,349**	1,161,846	19,119	163,383
12	**395,216**	355,737	5,161	34,318	**895,671**	822,485	11,761	61,425	**1,116,229**	1,007,349	17,005	91,874
13	**530,255**	495,409	6,662	28,185	**790,093**	737,108	10,714	42,270	**1,158,554**	1,052,120	16,650	89,784
14	**465,664**	424,967	6,093	34,604	**775,077**	674,741	11,094	89,242	**1,370,974**	1,242,214	17,487	111,272
15	**392,485**	347,462	5,670	39,353	**737,790**	618,630	10,647	108,513	**1,246,318**	1,090,653	15,591	140,074
16	**330,520**	307,095	4,043	19,383	**795,932**	732,466	8,154	55,313	**1,173,505**	1,070,836	15,238	87,430
17	**479,196**	448,683	6,463	24,050	**1,092,888**	986,982	10,636	95,270	**1,527,733**	1,371,757	17,408	138,569
18	**447,423**	413,261	5,962	28,200	**777,142**	679,383	8,771	88,988	**1,478,710**	1,288,453	15,866	174,391
19	**477,839**	440,408	5,966	31,465	**660,205**	608,942	9,351	41,912	**1,381,244**	1,239,645	18,025	123,573
20	**424,455**	379,756	5,588	39,112	**870,909**	802,503	11,366	57,040	**1,262,393**	1,071,203	16,166	175,023
21	**355,830**	338,480	4,651	12,698	**744,934**	661,419	10,329	73,185	**1,193,727**	1,078,910	15,554	99,264
22	**345,623**	314,478	4,996	26,149	**751,617**	700,172	9,251	42,193	**1,508,832**	1,353,041	19,398	136,394
23	**353,884**	324,755	5,128	24,000	**724,387**	626,502	10,321	87,564	**1,104,836**	990,101	16,370	98,364
24	**427,930**	385,811	6,114	36,005	**733,375**	589,776	9,324	134,275	**816,673**	742,026	14,215	60,433
25	**367,532**	315,269	5,254	47,009	**319,657**	286,799	3,704	29,154	**938,821**	870,100	14,420	54,300
26	**368,619**	303,563	5,313	59,743	**548,114**	513,271	7,752	27,091	**1,365,984**	1,231,939	18,541	115,504
27	**353,558**	304,975	5,057	43,525	**687,811**	626,098	9,240	52,473	**1,143,616**	949,734	15,654	178,229
28	**321,051**	286,037	4,114	30,900	**691,937**	626,703	9,150	56,084	**668,367**	606,829	11,518	50,019
29	**449,628**	414,977	4,370	30,281	**883,590**	772,026	9,733	101,831	**641,624**	590,238	11,479	39,906
30	**535,763**	489,051	5,100	41,613	**650,138**	550,869	7,866	91,403	**407,700**	375,049	7,294	25,357
31	**576,385**	518,014	5,496	52,874	**704,243**	658,190	8,416	37,637				
32	**480,482**	409,224	4,915	66,343	**796,800**	719,952	10,219	66,629				
33	**348,597**	319,407	4,746	24,445	**692,493**	578,464	9,029	105,000				
34	**449,053**	416,591	5,868	26,594	**581,183**	523,363	8,762	49,057				
35	**550,266**	507,721	6,463	36,082	**494,209**	442,703	7,907	43,599				
36	**325,685**	289,636	4,642	31,407	**588,312**	505,877	8,095	74,340				
37	**333,720**	301,040	4,633	28,047	**508,887**	467,656	7,984	33,248				
38	**329,420**	297,810	4,584	27,026	**630,032**	578,061	8,907	43,064				
39	**291,752**	266,526	4,345	20,881	**812,103**	725,402	10,104	76,598				
40	**345,461**	326,875	3,957	14,630	**753,226**	676,355	10,189	66,683				
41	**439,012**	402,765	5,374	30,873	**474,404**	428,294	7,614	38,496				
42	**445,453**	380,261	5,697	59,496	**431,690**	393,808	7,378	30,505				
43	**403,969**	330,003	5,063	68,902	**427,017**	389,782	7,319	29,916				
44	**331,604**	291,553	4,671	35,379	**407,443**	372,179	6,990	28,274				
45	**304,091**	271,168	4,532	28,391	**291,356**	266,139	4,998	20,219				
46	**292,903**	260,647	4,409	27,846								
47	**289,994**	258,059	4,365	27,569								
48	**288,204**	256,450	4,337	27,417								
49	**290,432**	258,582	4,383	27,467								
50	**257,954**	231,923	4,095	21,937								
51	**217,858**	198,928	3,731	15,200								
52	**217,130**	198,263	3,718	15,149								
53	**217,543**	198,640	3,725	15,178								
54	**214,355**	195,729	3,671	14,955								
55	**216,085**	197,309	3,700	15,076								
56	**217,547**	198,644	3,725	15,178								
57	**215,441**	196,721	3,689	15,031								
58	**215,964**	197,199	3,698	15,067								
59	**86,763**	79,224	1,486	6,053								

22.5 Residual Value In-Situ

Residual values of minerals were considered in the cases when in-situ copper remained after the life of mine plan. In-situ copper was valued at 6% of the copper price considered (18 cUSD/lb. = 6%*300 cUSD/lb.). Table 22.5 shows the copper residual value for each case.

Table 22.5: Copper Resource Residual Values

Description	Unit	Case	Case	Case
		55 ktpd	110 ktpd	200 ktpd
Residual Cu	Mlb	13,259	9,993	8,658
Period applied		60	46	31
Residual Value	kUSD	2,386,796	1,798,871	1,558,492
Present Value (8%)	kUSD	87,216	52,180	143,406

22.6 After-Tax Analysis

Tetra Tech is not a financial adviser and these economic models are indicative only. Tetra Tech recommends that the Company and other readers of this report seek their own financial and tax advice before acting in relation to the financial matters described herein.

The preparation of a comprehensive after-tax model results from proper tax planning modelled with the advice of taxation specialists, which rely on a number of material assumptions that cannot be defined at this point, but can be generally grouped into:

- The optimal capital structure (leverage).
- The financial and commercial terms and conditions available in the markets at the time of preparing the actual funding for the Project.
- The tax regimes of the jurisdictions affecting the Project.

These assumptions in turn depend on multiple variables including, but not limited to:

- The financial, operational and commercial strength and the country of origin of strategic partners, joint venture partners and/or other sponsors that would be involved in the development and operation of the Project.
- The conditions prevailing in the debt, equity and other financial markets relevant to the Project.
- The country of origin of the Project's main equipment suppliers.
- The conditions prevailing in the main commercial markets relevant to the Project (off-take, EPCM, power supply, etc.).

22.7 Economic Evaluation Results

Based on the projections resulting from the financial model, the pre-tax and after-tax NPV, IRR and payback periods are shown in Table 22.6.

Table 22.6: Summary Economic Results

Financial Indicators - Pre-Tax				
Description	Unit	55 ktpd	110 ktpd	200 ktpd
Net Present Value - 8%	kUSD	1,370,914	2,595,839	3,201,879
IRR	%	19.73%	24.73%	20.07%
Payback Period from operation (*)	Years	4.1	3.0	4.2
Payback Period from construction (**)	Years	6.1	5.0	6.2
Financial Indicators - After-Tax				
Description	Unit	55 ktpd	110 ktpd	200 ktpd
Net Present Value - 8%	kUSD	931,120	1,797,425	2,198,359
IRR	%	16.90%	20.77%	17.37%
Payback Period from operation (*)	Years	4.3	3.4	4.4
Payback Period from construction (**)	Years	6.3	5.4	6.4

(*) Referred to the first year of mill production

(**) Referred to the beginning of construction

Payback period calculated with nominal cash flows

22.8 Sensitivity Analysis

An NPV sensitivity analysis has been performed for changes in market price for copper and molybdenum, changes in capital costs, operating costs and discount rate.

22.8.1 Copper Price Variation

Table 22.7 and Figure 22.1 show the pre-tax NPV sensitivity to copper price variations. Table 22.8 and Figure 22.2 show the after-tax NPV sensitivity to copper price variations.

Table 22.7: NPV (kUSD) Sensitivity Analysis – Copper Price Variation (Pre-Tax)

Cu Price Variation cUSD/lb	55 ktpd	110 ktpd	200 ktpd
250	606,410	1,260,270	1,246,939
270	914,531	1,798,183	2,033,529
285	1,143,671	2,198,439	2,619,395
300	1,370,914	2,595,839	3,201,879
315	1,596,515	2,990,556	3,780,478
330	1,820,473	3,381,953	4,354,549
350	2,116,905	3,899,771	5,114,696

Table 22.8: NPV (kUSD) Sensitivity Analysis – Copper Price Variation (After-Tax)

Cu Price Variation cUSD/lb	55 ktpd	110 ktpd	200 ktpd
250	367,496	808,359	771,111
270	597,583	1,213,473	1,345,465
285	765,232	1,507,323	1,773,147
300	931,120	1,797,425	2,198,359
315	1,095,809	2,085,568	2,620,737
330	1,259,298	2,371,288	3,039,809
350	1,475,694	2,749,296	3,594,716

Figure 22.1: Copper Price Sensitivity Analysis (Pre-Tax)

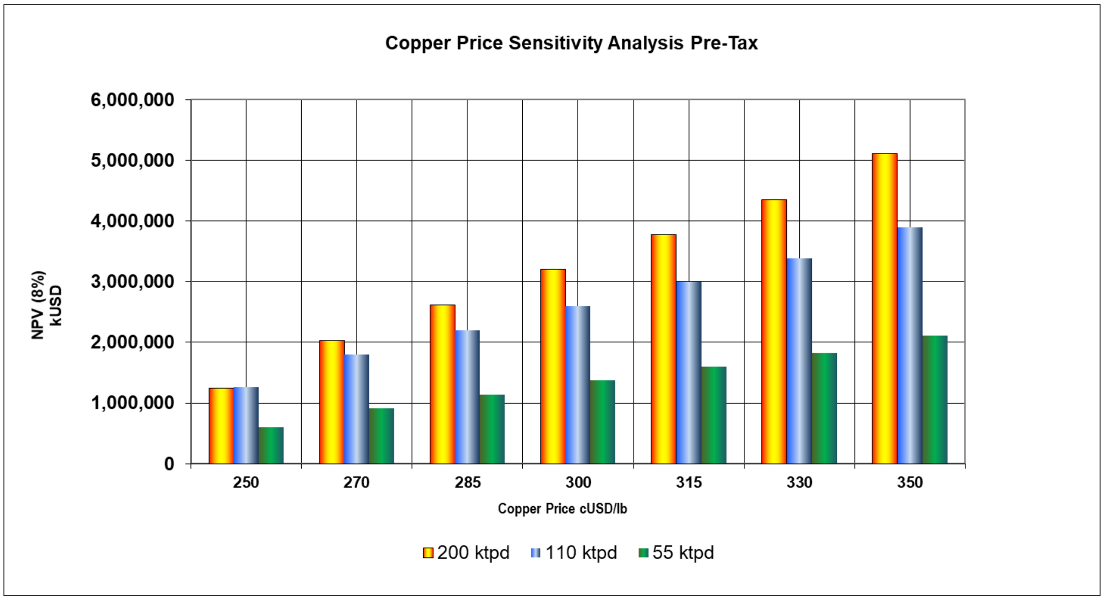
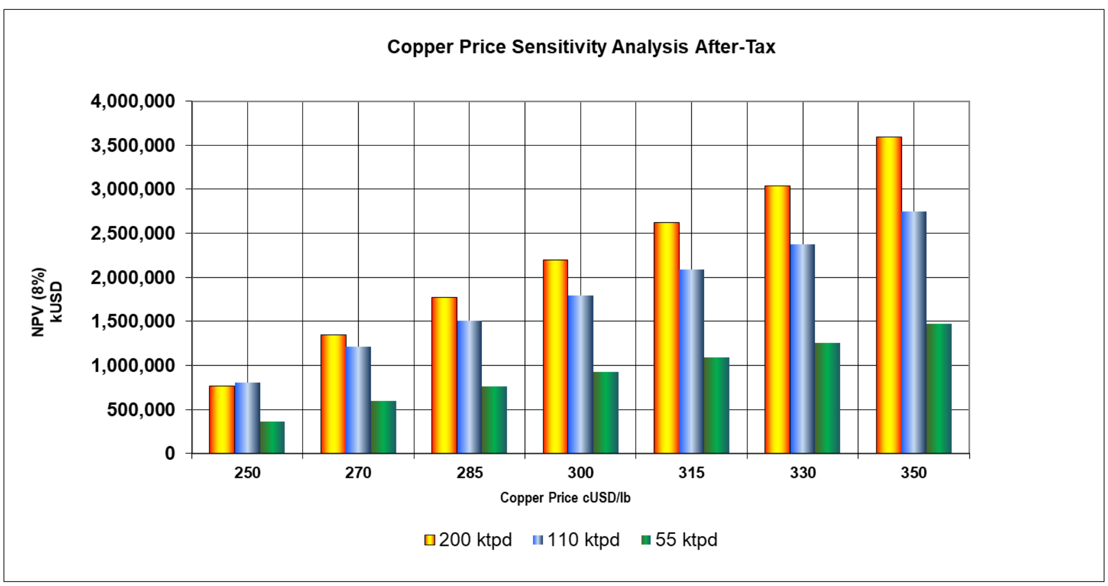


Figure 22.2: Copper Price Sensitivity Analysis (After-Tax)



22.8.2 Molybdenum Price Variation

Table 22.9 and Figure 22.3 show the pre-tax NPV sensitivity to molybdenum price variations. Table 22.10 and Figure 22.4 show the after-tax NPV sensitivity to molybdenum price variations.

Table 22.9: NPV (kUSD) Sensitivity Analysis – Molybdenum Price Variation (Pre-Tax)

Mo Price Variation USD/kg	200 ktpd	110 ktpd	55 ktpd
17.0	2,969,065	2,441,650	1,289,223
19.5	3,085,567	2,518,775	1,330,116
22.0	3,201,879	2,595,839	1,370,914
24.5	3,318,076	2,672,749	1,411,654
27.0	3,434,194	2,749,515	1,452,302

Table 22.10: NPV (kUSD) Sensitivity Analysis – Molybdenum Price Variation (After-Tax)

Mo Price Variation USD/kg	200 ktpd	110 ktpd	55 ktpd
17.0	2,028,405	1,684,867	871,486
19.5	2,113,452	1,741,168	901,337
22.0	2,198,359	1,797,425	931,120
24.5	2,283,184	1,853,569	960,860
27.0	2,367,949	1,909,609	990,533

Figure 22.3: Molybdenum Price Sensitivity Analysis (Pre-Tax)

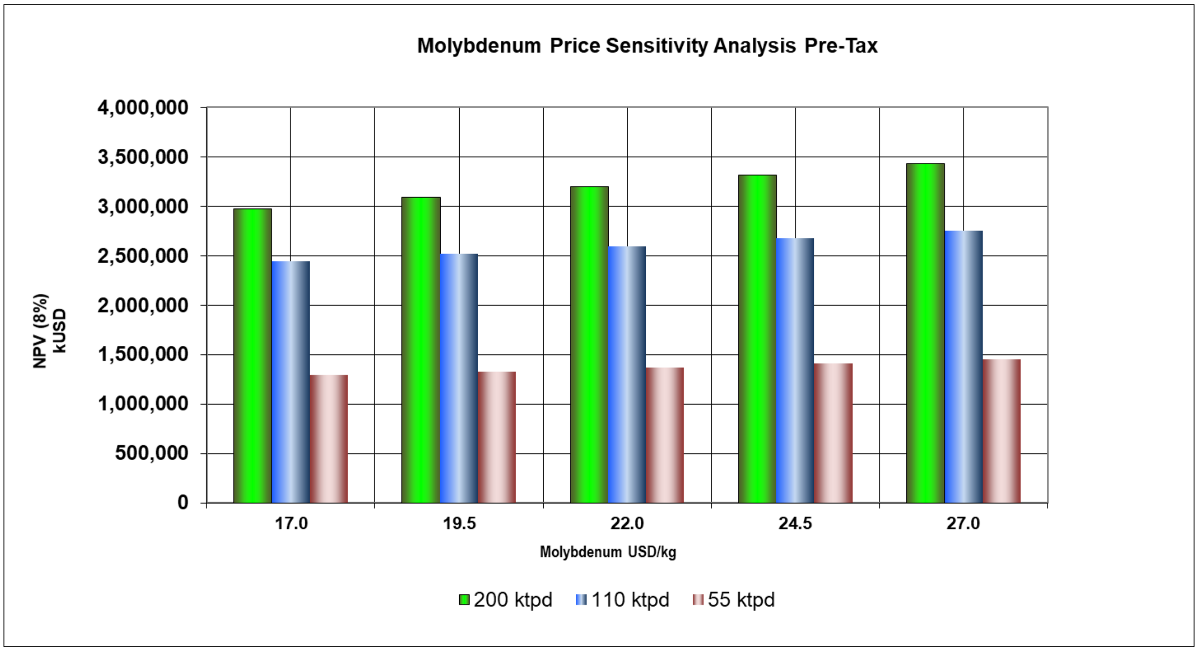
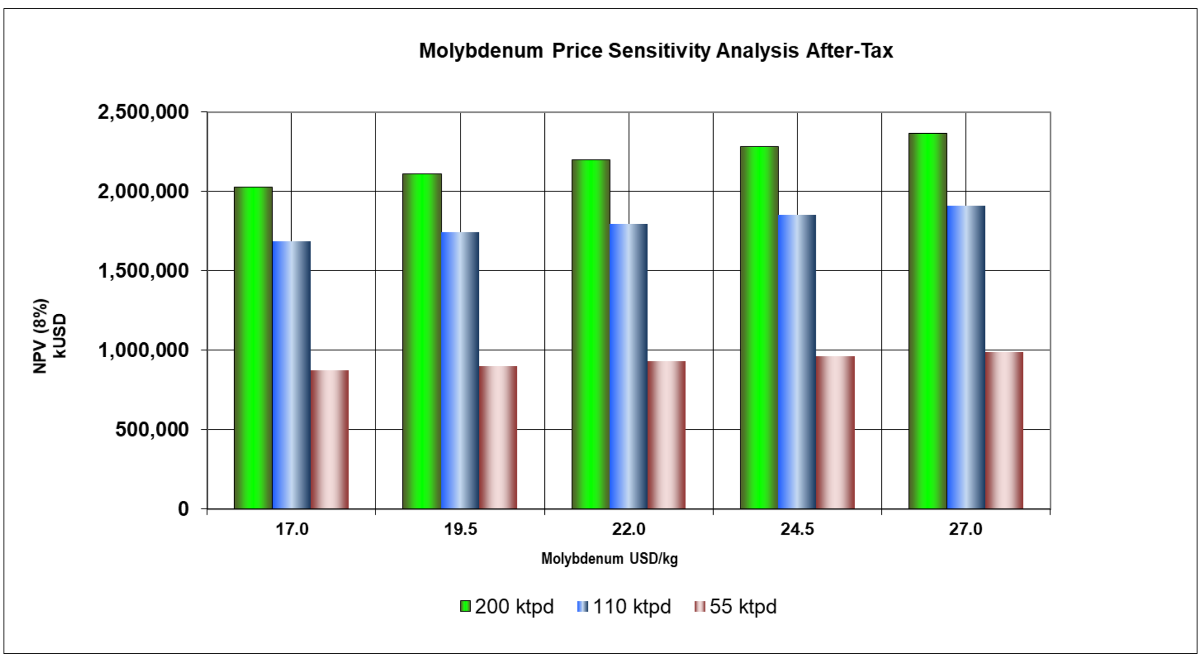


Figure 22.4: Molybdenum Price Sensitivity Analysis (After-Tax)



22.8.3 Capex and Opex Variation

On a pre-tax analysis, Table 22.11 presents NPV sensitivity to Capex variations and Table 22.13 presents NPV sensitivity to Opex variations. Figure 22.5 to Figure 22.7 present the data graphically.

On an after-tax analysis, Table 22.12 presents NPV sensitivity to Capex variations and Table 22.14 presents NPV sensitivity to Opex variation. Figure 22.8 to Figure 22.10 present the data graphically.

Table 22.11: NPV (kUSD) Sensitivity Analysis – Capex Variation (Pre-Tax)

Capex Variation	55 ktpd	110 ktpd	200 ktpd
-35%	1,790,598	3,220,765	4,208,631
-30%	1,730,643	3,131,490	4,064,809
-25%	1,670,688	3,042,215	3,920,987
-20%	1,610,734	2,952,940	3,777,166
-15%	1,550,779	2,863,664	3,633,344
-10%	1,490,824	2,774,389	3,489,522
-5%	1,430,869	2,685,114	3,345,700
0%	**1,370,914**	**2,595,839**	**3,201,879**
5%	1,310,959	2,506,564	3,058,057
10%	1,251,004	2,417,289	2,914,235
15%	1,191,050	2,328,013	2,770,413
20%	1,131,095	2,238,738	2,626,592
25%	1,071,140	2,149,463	2,482,770
30%	1,011,185	2,060,188	2,338,948
35%	951,230	1,970,913	2,195,126

Table 22.12: NPV (kUSD) Sensitivity Analysis – Capex Variation (After-Tax)

Capex Variation	55 ktpd	110 ktpd	200 ktpd
-35%	1,263,928	2,292,350	2,994,153
-30%	1,216,384	2,221,647	2,880,468
-25%	1,168,840	2,150,943	2,766,784
-20%	1,121,296	2,080,239	2,653,099
-15%	1,073,752	2,009,536	2,539,414
-10%	1,026,208	1,938,832	2,425,729
-5%	978,664	1,868,129	2,312,044
0%	**931,120**	**1,797,425**	**2,198,359**
5%	883,576	1,726,721	2,084,675
10%	836,032	1,656,018	1,970,990
15%	788,488	1,585,314	1,857,305
20%	740,944	1,514,610	1,743,620
25%	693,400	1,443,907	1,629,935
30%	645,856	1,373,203	1,516,250
35%	598,313	1,302,499	1,402,566

Table 22.13: NPV (USD) Sensitivity Analysis – Opex Variation (Pre-Tax)

Opex Variation	55 ktpd	110 ktpd	200 ktpd
-35%	2,028,322	3,806,704	5,126,132
-30%	1,934,406	3,633,723	4,851,239
-25%	1,840,491	3,460,743	4,576,345
-20%	1,746,576	3,287,762	4,301,452
-15%	1,652,660	3,114,781	4,026,559
-10%	1,558,745	2,941,800	3,751,665
-5%	1,464,830	2,768,820	3,476,772
0%	**1,370,914**	**2,595,839**	**3,201,879**
5%	1,276,999	2,422,858	2,926,985
10%	1,183,083	2,249,877	2,652,092
15%	1,089,168	2,076,897	2,377,198
20%	995,253	1,903,916	2,102,305
25%	901,337	1,730,935	1,827,412
30%	807,422	1,557,954	1,552,518
35%	713,507	1,384,974	1,277,625

Table 22.14: NPV (USD) Sensitivity Analysis – Opex Variation (After-Tax)

Opex Variation	55 ktpd	110 ktpd	200 ktpd
-35%	1,411,142	2,681,489	3,603,413
-30%	1,342,568	2,555,194	3,402,691
-25%	1,273,993	2,428,899	3,201,969
-20%	1,205,419	2,302,604	3,001,247
-15%	1,136,844	2,176,309	2,800,525
-10%	1,068,269	2,050,015	2,599,803
-5%	999,695	1,923,720	2,399,081
0%	**931,120**	**1,797,425**	**2,198,359**
5%	862,546	1,671,130	1,997,638
10%	793,971	1,544,835	1,796,916
15%	725,397	1,418,540	1,596,194
20%	656,822	1,292,245	1,395,472
25%	588,247	1,165,951	1,194,750
30%	519,673	1,039,656	994,028
35%	451,098	913,361	793,306

Figure 22.5: Capex and Opex Variation — Case 55 ktpd (Pre-Tax)

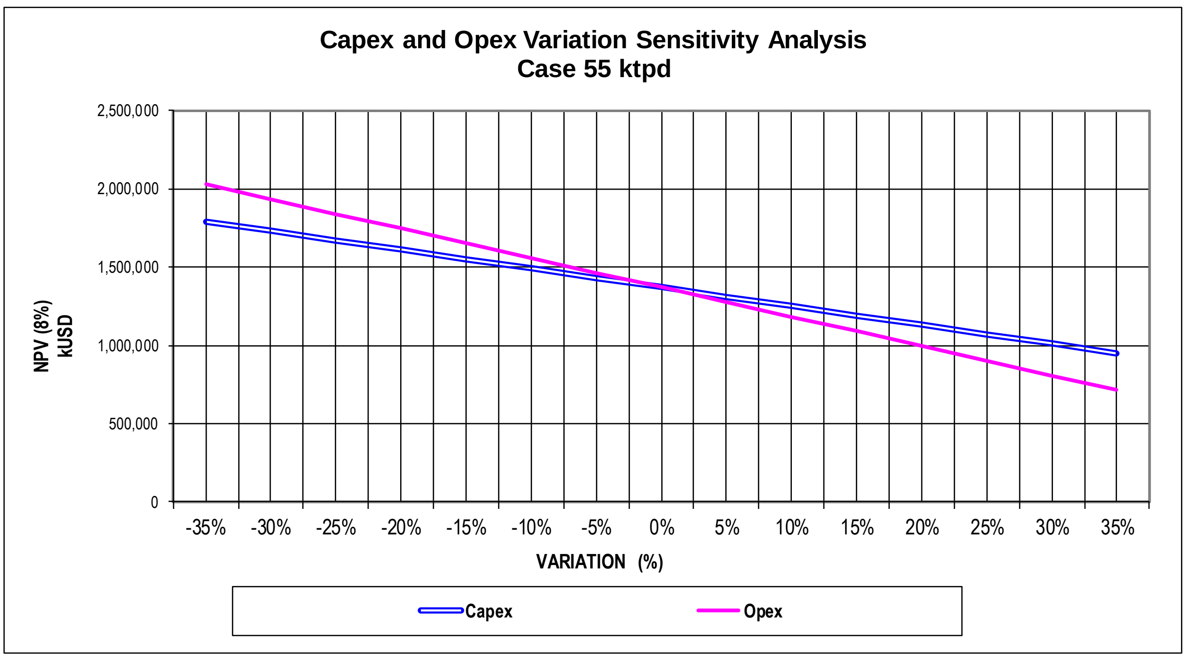


Figure 22.6: Capex and Opex Variation — Case 110 ktpd (Pre-Tax)

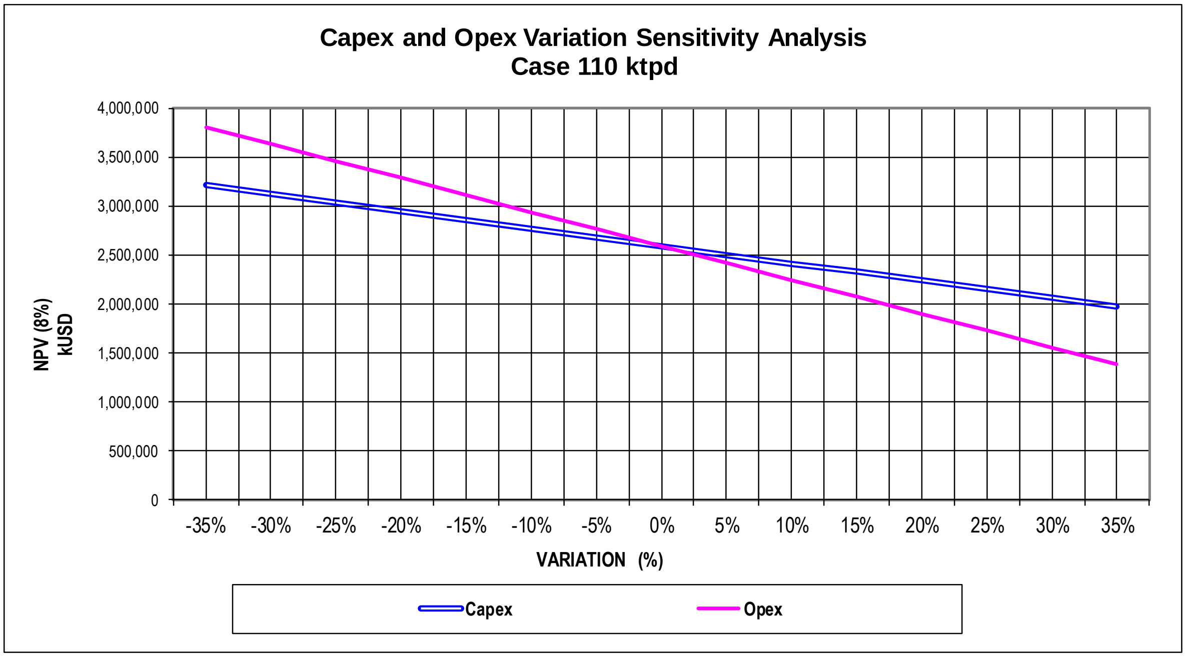


Figure 22.7: Capex and Opex Variation — Case 200 ktpd (Pre-Tax)

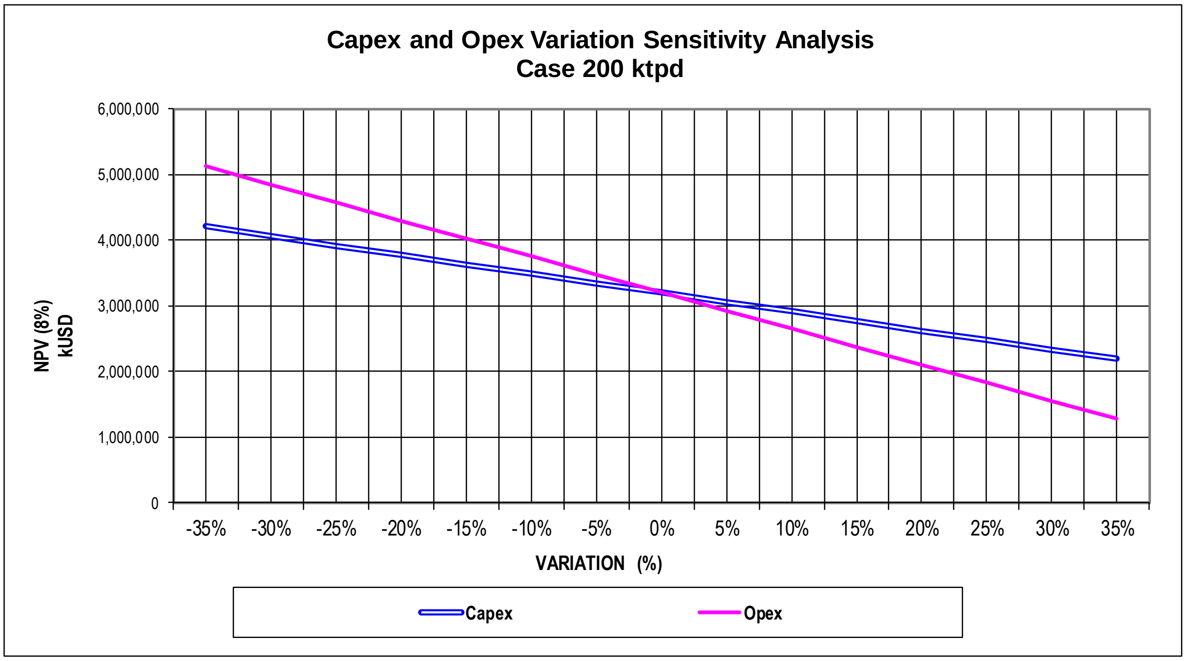


Figure 22.8: Capex and Opex Variation — Case 55 ktpd (After-Tax)

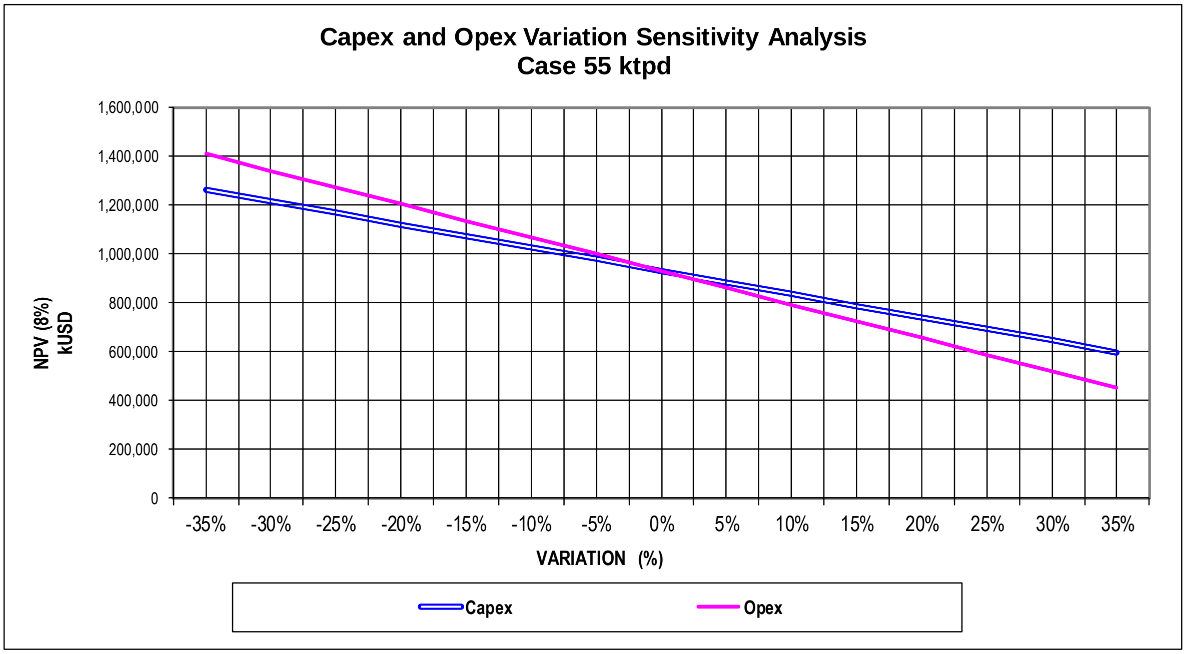


Figure 22.9: Capex and Opex Variation — Case 110 ktpd (After-Tax)

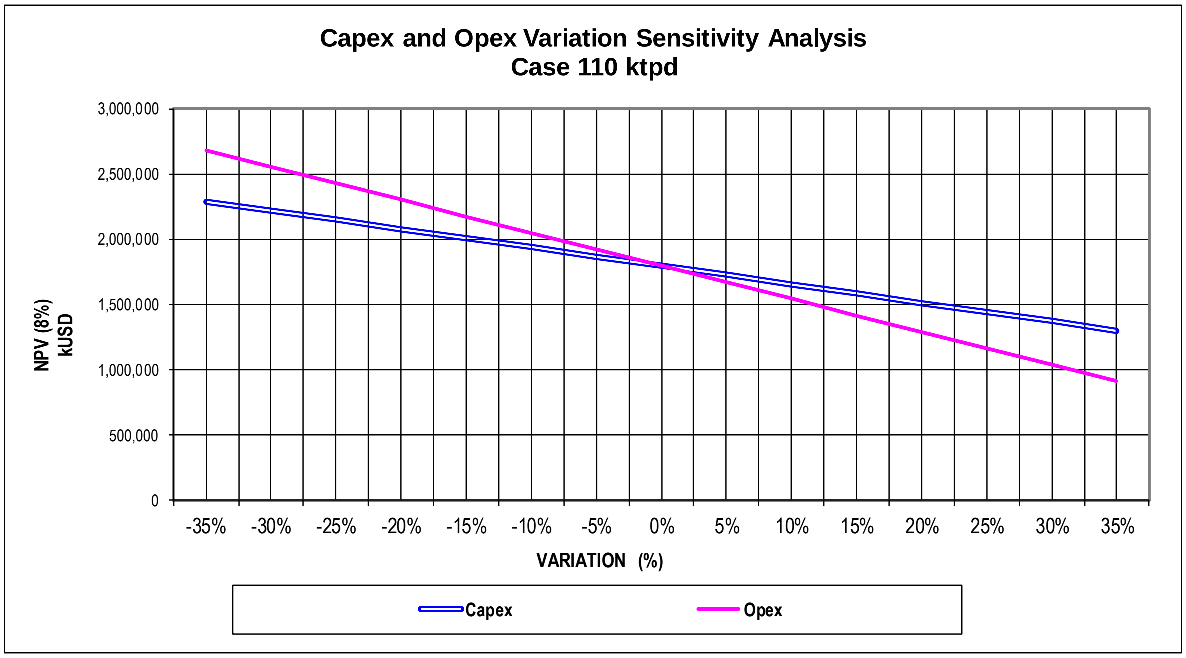
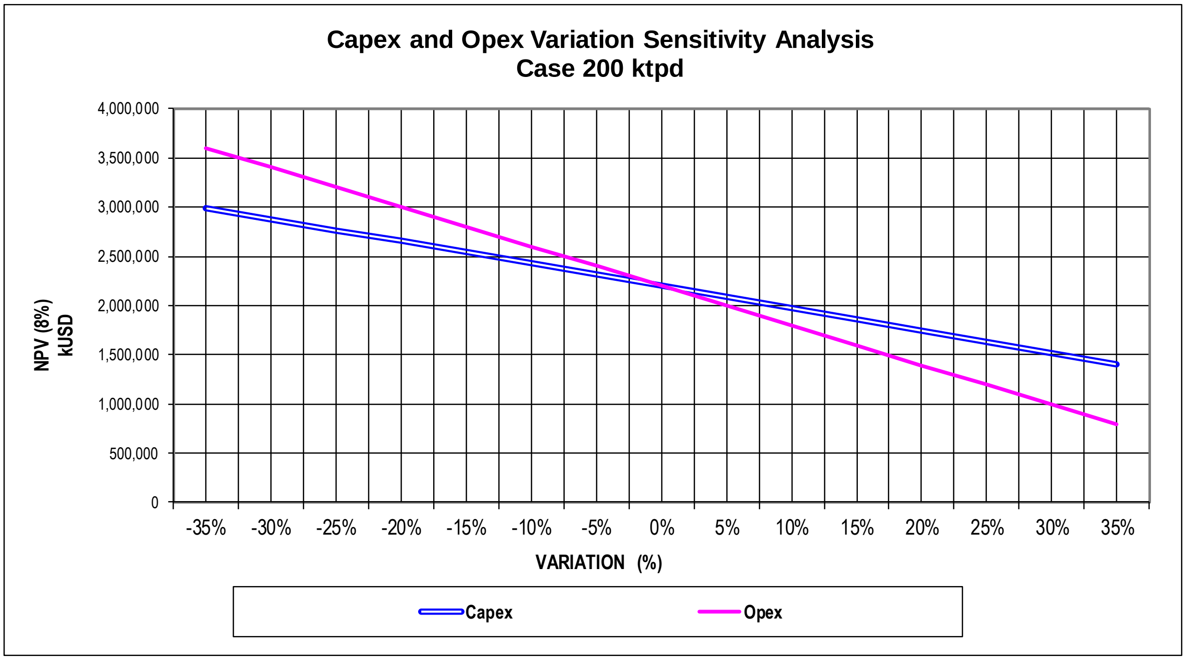


Figure 22.10: Capex and Opex Variation — Case 200 ktpd (After-Tax)



23. ADJACENT PROPERTIES

West Wall

West Wall is a copper porphyry exploration project located approximately 20 km south-east from the Vizcachitas Property. Glencore plc and Anglo American plc each own a 50% interest in the project. The estimated resources (according to the 2004 Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves - JORC Code) as published by the owners in their respective 2018 reports for Resources & Reserves, are the following:

- Indicated resources of 861 Mt @ 0.51% Cu, 0.05 g/t Au and 0.008% Mo and
- Inferred resources of 1,072 Mt @ 0.42% Cu, 0.05 g/t Au and 0.006% Mo
- Reporting cut-off grade of 0.2% Cu.

The porphyry copper style hydrothermal alteration identified in the West Wall property covers a large area of approximately 7 km by 3 km. The primary control is structural with primary NS structures, secondary NNE structures and strong WWN shearing. The mineralization is associated with quartz diorite porphyry intruding into an Oligocene volcano-sedimentary sequence. The mineralization is mainly chalcopyrite and bornite associated with potassic alteration.

Exploration has focused in the south of the prospect at Lagunillas and West Wall North. During the 2011-2012 drilling programme, a total of 24,000 m of infill were completed and incorporated into the geological models and Mineral Resource estimate.

24. OTHER RELEVANT DATA

The Qualified Persons are unaware 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. INTERPRETATION AND CONCLUSIONS

This section presents the conclusions and recommendations of the Qualified Persons for the Project and this Technical Report.

25.1 Interpretations and Conclusions

- The results of the PEA indicate that the Vizcachitas Project is robust at this stage of development demonstrating favourable economic potential that warrants further work toward the development of pre-feasibility studies.
- The exploration programme continues to demonstrate the potential for future growth of the resource.
- The sample preparation, security, and procedures followed by Los Andes Copper are adequate to support a Mineral Resource estimate.
- Assay data provided by Los Andes Copper was represented accurately and is suitable for use in resource estimation.
- There are no known environmental issues existing or anticipated that could materially impact the ability to develop the Vizcachitas Project.
- 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.
- The metallurgical test work undertaken is reasonably extensive and suitable for this level of study. The comminution data are considered adequate for a conceptual milling circuit design. The design of the processing circuits is based on this test work data in conjunction with assumptions based on typical industry values.
- The Vizcachitas mineralized material is of moderate competency and hardness, and amenable to grinding in a conventional SABC circuit. The mineralogy is fine grained and test work indicates a requirement to re-grind to a fine particle size to achieve adequate liberation for flotation as is common within the industry.
- Overall recoveries are estimated at 91% for copper and 75% for molybdenum, which are contained in metals concentrates.
- The Project has been 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 once mining has been completed.

25.2 Risks and Opportunities

The following risks and opportunities associated with development of the Project have been identified by the Qualified Persons.

During the pre-feasibility study phase, several risks will need to be investigated further and possibly reduced or eliminated. Similarly, further investigation and evaluation of opportunities may allow their incorporation in the Project.

25.2.1 Risks

- Political risks and uncertainties affecting legislation, regulatory requirements or general business climate, including for example: (i) potential changes in existing laws and approval of more onerous laws in the future or (ii) 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 necessary permits or approvals from government authorities.
- Diesel fuel is a significant component of the mine operating costs. Higher fuel prices could impact project returns given the stripping ratio, pit depth, and corresponding long haulage profiles.
- Electrical power is a significant component of the plant operating costs. Higher power prices and overall power availability could impact project returns.

25.2.2 Opportunities

- Higher metals pricing, particularly for copper, than those used as a long-term forecast in the financial model.
- Potential for expansion of mineral resources.
- Capital and operating cost improvements as alternative designs, equipment and processes become available or can be properly evaluated with more detailed information.

26. RECOMMENDATIONS

Based on the results of the PEA, the Qualified Persons recommend that Los Andes Copper complete a PFS to further define the Project, to more accurately assess its technical and economic viability and to support permitting activities.

The tasks and estimate of the costs to complete the PFS are summarized below in Table 26.1.

Table 26.1: Task and Budget Estimate of the Pre-feasibility Study

Task	Estimated Cost (kUSD)
Exploration and Drilling For PFS it is estimated 36,600 m for exploration, condemnation, metallurgical, infill, geotechnical and hydrogeologic studies. Including assays.	**7,700**
Metallurgical testing for PFS Includes 200 samples	**1,600**
Engineering Prefeasibility Study	
Resources, Mining, Processing and Infrastructure	**6,840**
Mining Engineering	700
Processing Engineering	1,100
Tailings Engineering	500
Water and energy Supply	360
Roads and Access	80
Geology/Resources	800
Geotechnical Studies (including Hydrogeology)	800
Topography	300
Engineering Counterpart	400
Environmental Studies	1,800
Owner Costs	**2,300**
Total	**18,440**

The exploration, drilling and metallurgical testing costs were based on budgets presented by Los Andes Copper.

The PFS engineering costs were estimated assuming 76,000 MH at 90 USD/MH. This estimate is taken from benchmarking similar studies. The distribution in main disciplines is also based on the experience of similar studies. The owner costs were estimated as 14.25% of the other costs for the PFS.

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28. CERTIFICATES AND SIGNATURES

The undersigned prepared this technical report titled “Preliminary Economic Assessment on the Vizcachitas Project” National Instrument 43-101 Technical Report, Effective date May 10, 2019.

CERTIFICATE OF QUALIFIED PERSON

Evaristo Lillo 78, Office 51, Las Condes,
Santiago, Chile
Telephone: +56 (2) 2906911
Central Telephone: +56 (2) 2906910
Severino.ModenaSe@tetratech.com

I, Severino Modena do hereby certify that:

1. My position at Tetra Tech Chile is Chief Executive Officer.
2. I hold the professional degree of Civil Mining Engineer.
3. I am a Registered Member of the Chilean Mining Commission since 2011, and hold a valid Certificate of Qualified Competency from the Chilean Mining Commission with number 0134.
4. I am a member of the Australian Institute of Mines and Metallurgy (AusiMM) since 2009 and my registration number is 301987.
5. I state that I have read the definition of "Qualified Competent Person" of the Qualifying Commission for Competencies in Mineral Resources and Reserves, set forth in Code CH 20235, Chapter IV, Number 11. My qualifications to subscribe to the report identified above are as follows: Civil Mining Engineering from Universidad de Santiago de Chile, practicing the profession since 1979. I have a double certification of Competent Person (CP): i) Register No. 301987 issued in year 2009 by the Australian Institute of Mines and Metallurgy (AusiMM) and ii) Register No. 0134, issued in 2011 by the Qualifying Commission for Competencies in Mineral Resources and Reserves of Chile. Additionally, I am a member of the Engineering Society of Chile and the Mining Institute of Chile.
6. My expertise is in the following areas:
   a. Reserve Estimates, Classification and Audits
   b. Open Pit Mining Planning (Profile, Conceptual, Basic and Detail Engineering):
      i. Pit Optimization
      ii. Mining Design (Pit and Dumps)
      iii. Mining Plans (Short- and Long-Term)
      iv. Estimation of Transportation Distances and Mining Equipment
      v. CAPEX and OPEX Estimates and Economic Assessment

c. Analysis of Operational Productivity
d. Due Diligence and Expert Revisions
e. Preparation of Reports for Banks and Financial Institutions (CPs)
f. Support in the Analysis and Definition of Mine Infrastructure and Processes

7. I visited the Vizcachitas Project on October 13, 2017.
8. I am responsible for the revision and preparation chapters 1.1,1.2,1.3,1.4,19.9,1.10, 1.12, 1.13, 1.14, 1.15, 1.16, 1.17, 2, 3, 4, 15, 16, 18, 19, 20, 21, 22, 23, 24 25, 26, 27 and those portions of the Summary, Interpretations and Conclusions, and Recommendations that pertain to those Sections of the Technical Report (NI 43-101) of the Preliminary Economic Assessment for Vizcachitas Copper Project, Region V, Chile, for Los Andes Copper Ltd., effective date May 10, 2019.
9. I state that I have no previous relationship or partnership with Los Andes Copper Ltd. mining business.
10. I state that, to my best knowledge, no biased data have been used that could make this report misleading.
11. The author of this report, Severino Modena, is a professional who works for Tetra Tech Chile.
12. I state to have read the rules that govern the Evaluation of Mineral Resources and Reserves of the Mining Commission, which provide reassurance that the technical report has been prepared in accordance with NI 43 101.
13. I hereby authorize Tetra Tech to submit the Report "PEA of The Vizcachitas Project, under NI 43 101 Reporting Standard", prepared for Los Andes Copper Ltd.

As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this June 05, 2019.

"Signed and sealed"

____

Severino Modena
Mining Engineer
General Manager Tetra Tech Chile
Member of the Chilean Mining Commission

CERTIFICATE OF QUALIFIED PERSON

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Santiago, Chile
Telephone: +56 (2) 2906951
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Mario.Riveros@tetratech.com

I, Mario Riveros do hereby certify that:

1. My position at Tetra Tech Chile is Technical Chief Executive Officer, Process Area, and Project Manager.
2. I hold the professional degree of Chemical and Industrial Engineer.
3. I am a Registered Member of the Chilean Mining Commission and hold a valid Certificate of Qualified Competency from the Chilean Mining Commission with number 0129.
4. My registration in the Public Register dates form September 2011.
5. I state that I have read the definition of "Qualified Competent Person" of the Qualifying Commission for Competencies in Mineral Resources and Reserves, set forth in Code CH 20235, Chapter IV, Number 11. My qualifications to subscribe to the report identified above are as follows: Industrial and Chemical Engineer from Universidad de Santiago de Chile, practicing the profession since 1971. Competent Person in Metallurgical Mining Processes issued by the Qualifying Commission of Competences in Mining Resources and Reserves of Chile (Law of the Republic of Chile No. 20235) (Registration No. 0129), specialized in the following areas:
    1. • Design and Optimization of Metallurgical Processes - Sulphides Area.
    2. • Conceptual, Basic and Detailed Engineering Studies of Ore Processing Plants.
    3. • Design, Management, Monitoring, Control and Evaluation of Metallurgical Tests in Laboratory and Pilot Plants.
    4. • Development of testing programs, mass balance and water in metallurgical processes.
6. I visited the Vizcachitas Project on Friday, October 13, 2017.

   I am responsible for the revision for the chapters 1.7, 1.11, 13, 13.1,1 3.2, 13.3, 13.4, 13.5, 13.6, 17, 17.1, 17.2, 17.3, 17.4, 17.5, 17.6, 17.7 and those portions of the Summary, Interpretations and Conclusions, and Recommendations that pertain to those Sections of the Technical Report (NI 43-101) of the Preliminary Economic Assessment for Vizcachitas Copper Project, Region V, Chile, for Los Andes Copper Ltd., effective date May 10, 2019.

7. I state that I have no previous relationship or partnership with Los Andes Copper Ltd. mining business.
8. I state that, to my best knowledge, no biased data have been used that could make this report misleading.

9. I state that the author of this report, Mario Riveros, is a professional who works for Tetra Tech Chile.
10. I state to have read the rules that govern the Evaluation of Mineral Resources and Reserves of the Mining Commission, which provide reassurance that the technical report has been prepared in accordance with NI 43 101.

11. I hereby authorize Tetra Tech to submit the Report "PEA of The Vizcachitas Project, under NI 43 101 Reporting Standard", prepared for Los Andes Copper Ltd.

As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this June 05, 2019.

"Signed and sealed"

____

Mario Riveros
Chemical and Industrial Engineer
Member of Chilean Mining Commission

CERTIFICATE OF QUALIFIED PERSON

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Jose.Fuenzalida@tetratech.com

I, José Fuenzalida do hereby certify that:

1. My position at Tetra Tech is consultant.
2. I hold the professional degree of Geologist.
3. I am a Registered Member of the Chilean Mining Commission and hold a valid Certificate of Qualified Competency from the Chilean Mining Commission with number 171.
4. I have been registered in the Public Register since June 2012.
5. I state that I have read the definition of "Qualified Competent Person" of the Qualifying Commission for Competencies in Mineral Resources and Reserves, set forth in Code CH 20235, Chapter IV, Number 11. My qualifications to subscribe to the report identified above are as follows: Geologist from Universidad de Chile, practicing the profession since 1979 with more than 30 years in the Resource Estimates area and international and national experience.
    - Consultant: took part in the preparation and revision of reports for several mining operations and projects; mining project assessments; analysis of mining processes; support in geometallurgical modeling; statistic and geostatistics analysis;
    - Geologist in charge of different mining operations in mineral resource estimates, working in the estimation of copper, molybdenum, arsenic, zinc, iron, gold and phosphate models.
6. I visited the Vizcachitas Project on May 10, 2017.
7. I state that I have no previous relationship or partnership with Los Andes Copper Ltd. mining business.

8. I am responsible for the revision and preparation chapters 1.5, 1.6, 1.8, 4.1, 5, 6, 7, 8, 9, 10, 11, 12, 14 and those portions of the Summary, Interpretations and Conclusions, and Recommendations that pertain to those Sections of the Technical Report (NI 43-101) of the Preliminary Economic Assessment for Vizcachitas Copper Project, Region V, Chile, for Los Andes Copper Ltd., effective date May 10, 2019.
9. I state that, to my best knowledge, no biased data have been used that could make this report misleading.
10. I state that José Luis Fuenzalida is a professional who works for Tetra Tech Chile.
11. I state to have read the rules that govern the Evaluation of Mineral Resources and Reserves of the Mining Commission, which provide reassurance that the technical report has been prepared in accordance with NI 43 101.
12. I hereby authorize Tetra Tech to submit the Report “PEA of The Vizcachitas Project, under NI 43 101 Reporting Standard”, prepared for Los Andes Copper Ltd.

As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this June 05, 2019.

“Signed and sealed”

____

José Luis Fuenzalida
Geologist
Geostatistics Specialist
Member of Chilean Mining Commission

29. APPENDIX I – QA/QC CHARTS

The diagrams for correlation coefficient, relative error and dispersion for the 1996-1997 ACME results and 2007-2008 SGS results are shown in Figure 29.1.

Figure 29.1: Statistical Diagrams for Duplicate Samples

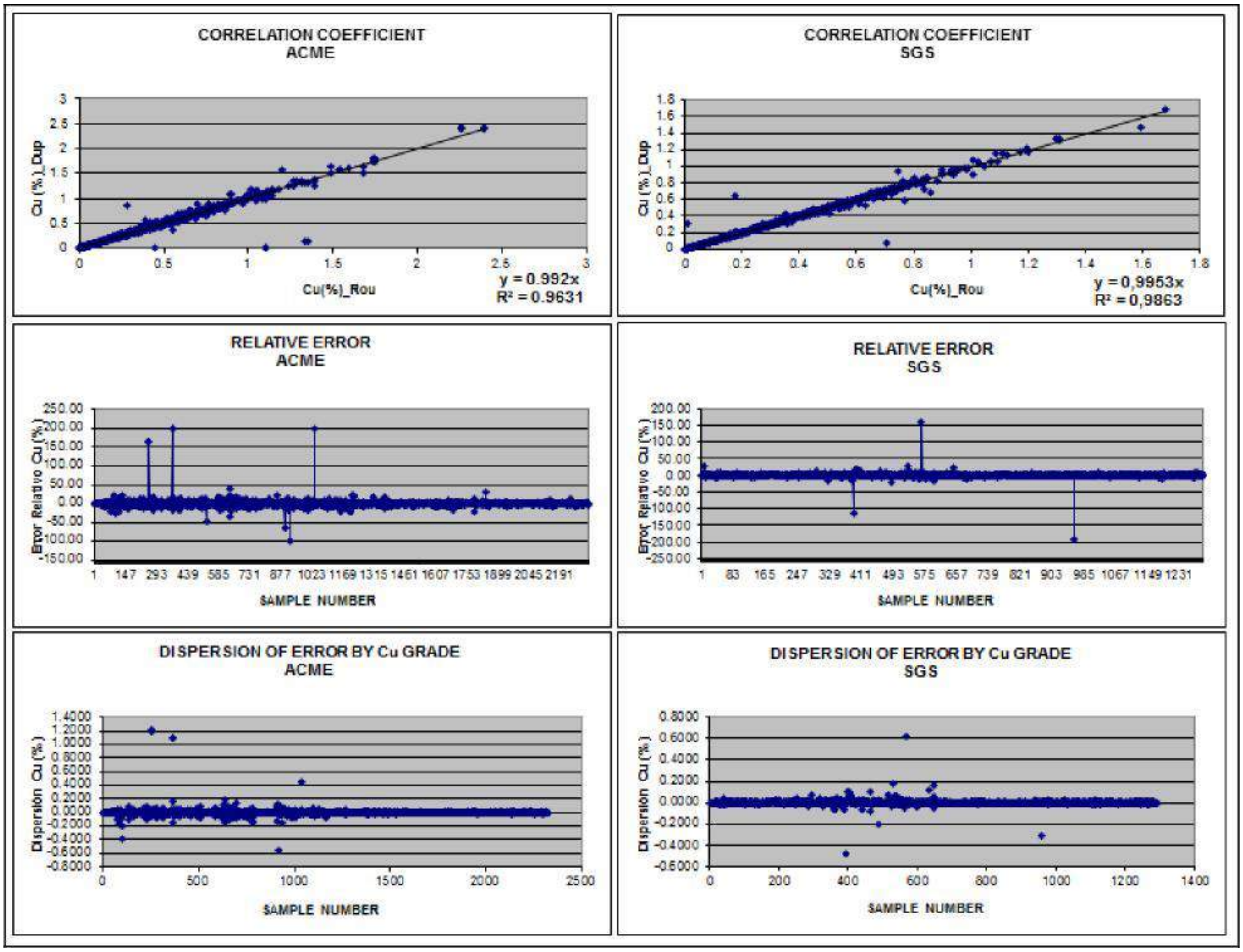
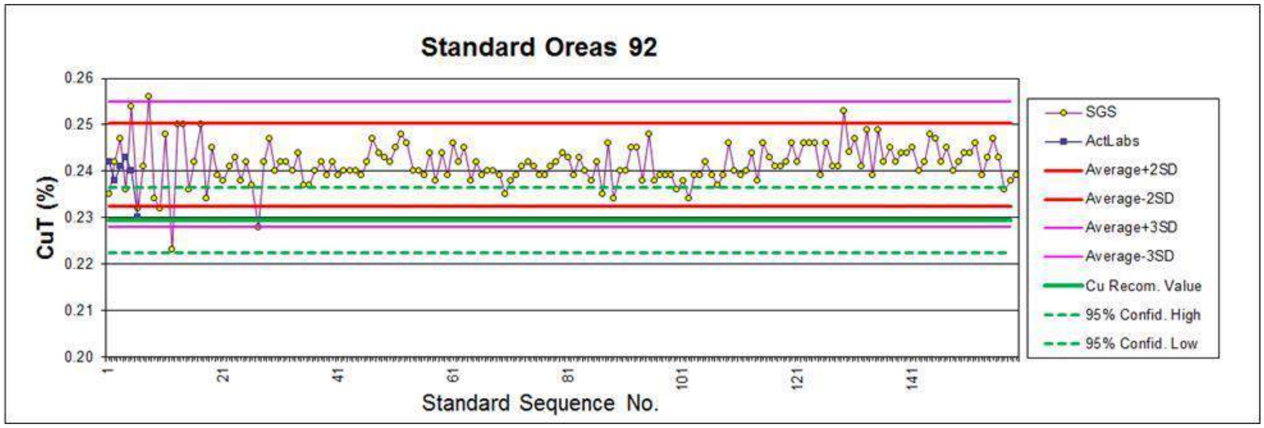


Figure 29.2 graphically presents the assay results and statistical parameters for Oreas 92.

Figure 29.2: 2007-2008 Oreas 92 Assay Results



The graph shows that most of the values of Oreas 92 are above the recommended value but below the upper recommended value. Only a few samples are outside the ± 2 SD limits, mainly at the beginning of the process. The laboratory has adjusted precision with time and the values in general are acceptable. It is clear that the position of the assay results of this CRM above the recommended value with a 5.46% bias indicates that the low-grade copper values could be overestimated by this amount.

Figure 29.3 illustrates the distribution of Oreas 93 the medium copper grade standard.

Figure 29.3: 2007-2008 Oreas 93 Assay Results

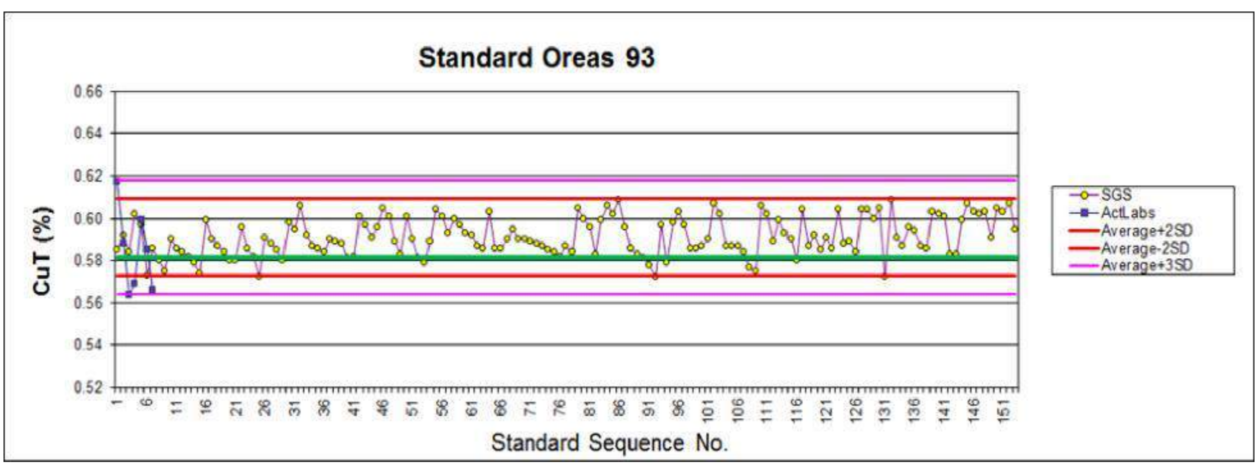
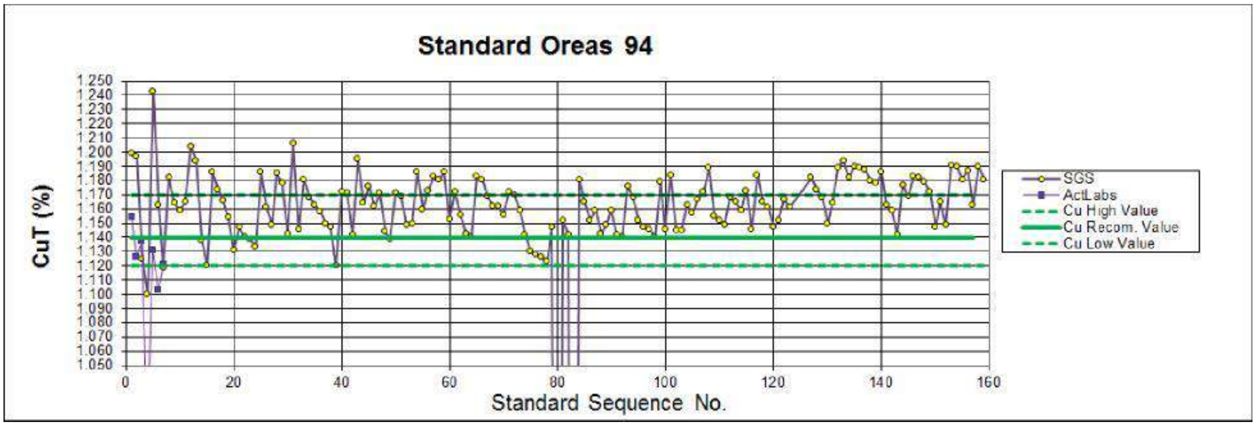


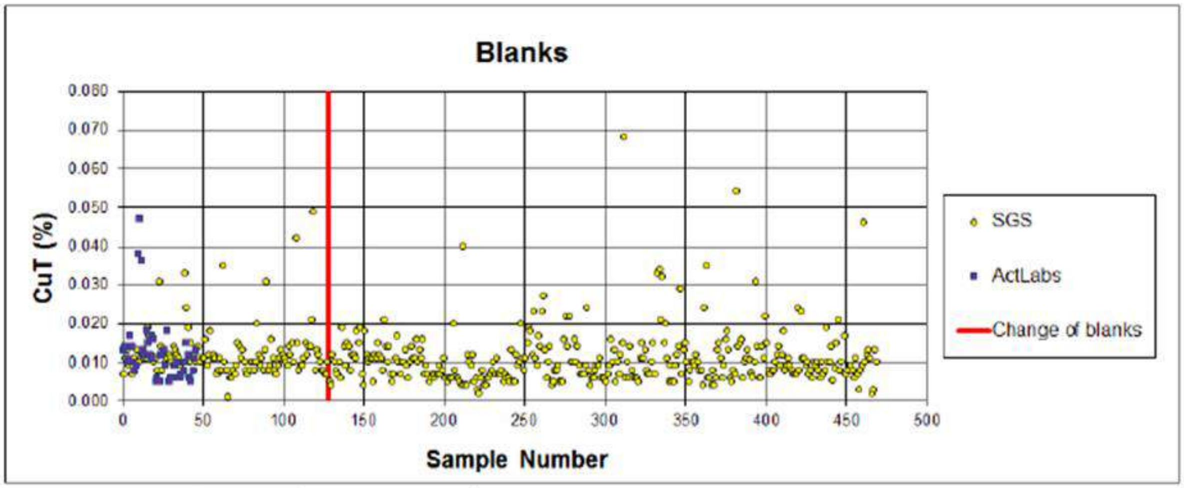
Figure 29.4 presents the distribution of standard Oreas 94, the high copper grade CRM.

Figure 29.4: 2007-2008 Oreas 94 Assay Results

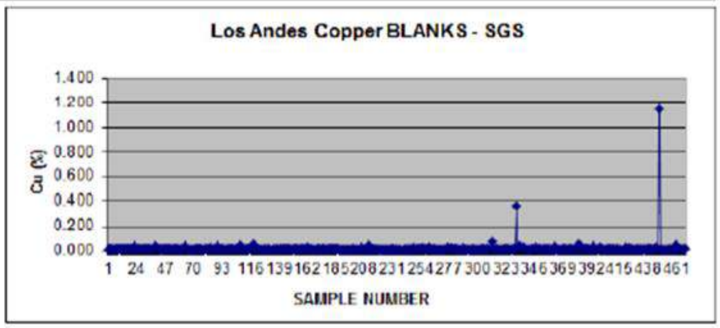


As in the other two standards, this figure shows that most of the values are located above the Recommended Value limit, with many of them even above the +2 SD limit. This situation means that it is probable that the higher-grade copper values may be slightly over estimated.

Figure 29.5: 2007-2008 Blanks Assay Values in the Vizcachitas Project

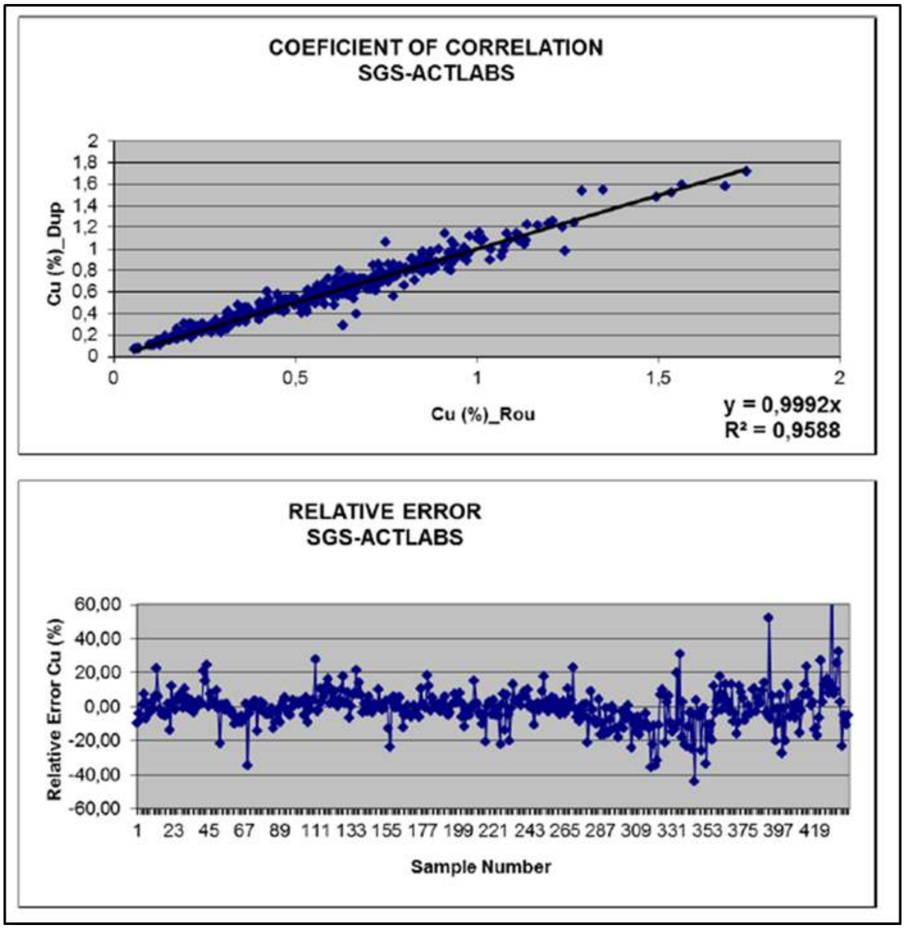


Parameters	Cu (%)
Mean	0.014387846
Standard Error	0.002569295
Median	0.01
Mode	0.01
Standard Deviation	0.055641695
Sample Variance	0.003095998
Kurtosis	381.2421066
Asymmetry Coeficient	18.977059
Range	1.153
Minimum	0.001
Maximum	1.154
Sum	6.7479
Count	469
Higher	1.154
Lower	0.001
Confidence Level (95.0%)	0.00504878



The figure illustrates that the blank samples submitted during the Los Andes Copper drilling campaign show very low copper values varying mainly between 0.005% and 0.020% Cu. There are two values with higher copper content which may be due to the uncertified nature of these two samples, both were taken in the area of the Vizcachitas deposit. There are some clear outliers that are probably related to external reasons rather than to laboratory accuracy such as incorrect numbering of samples.

Figure 29.6: 2007-2008 Actlabs Check Samples



30. APPENDIX II – DRILL HOLE COORDINATES

List of all drill holes used in the Resource Estimate

Drill Hole Number	East	North	Elevation	Total Depth	Dip	Azimuth	Drilling Campaign	Company
VP-1	365,860	6,414,170	2,033	300.25	-65	135	1993	Placer Dome
VP-2	365,647	6,413,393	2,035	300.20	-60	160	1993	Placer Dome
VP-3	365,847	6,413,575	1,999	303.85	-70	315	1993	Placer Dome
VP-4	366,050	6,413,595	2,082	251.50	-65	30	1993	Placer Dome
VP-5	366,190	6,413,271	2,040	300.00	-65	130	1993	Placer Dome
VP-6	366,102	6,413,139	1,966	497.15	-80	135	1993	Placer Dome
V-01	365,838	6,414,161	2,022	307.13	-75	100	1996-1997	General Minerals
V-02	366,047	6,413,590	2,077	302.25	-60	50	1996-1997	General Minerals
V-03	366,439	6,412,863	2,074	578.58	-90	0	1996-1997	General Minerals
V-04	365,909	6,413,510	1,985	407.78	-50	55	1996-1997	General Minerals
V-05	365,907	6,413,510	1,982	508.43	-50	100	1996-1997	General Minerals
V-06	366,066	6,413,432	2,063	584.68	-55	70	1996-1997	General Minerals
V-07	366,249	6,413,411	2,134	532.83	-65	350	1996-1997	General Minerals
V-08	365,936	6,413,856	2,049	535.88	-60	115	1996-1997	General Minerals
V-09	366,474	6,413,641	2,366	368.13	-60	90	1996-1997	General Minerals
V-10	366,421	6,413,706	2,372	438.28	-65	300	1996-1997	General Minerals
V-11	366,476	6,413,641	2,366	452.62	-70	260	1996-1997	General Minerals
V-12	366,195	6,414,193	2,234	299.51	-60	305	1996-1997	General Minerals
V-13	365,922	6,413,580	1,990	203.43	-70	100	1996-1997	General Minerals
V-14A	365,967	6,413,327	1,980	20.43	-70	100	1996-1997	General Minerals
V-15	366,034	6,413,495	2,067	292.80	-70	100	1996-1997	General Minerals
V-16	366,122	6,413,397	2,082	209.53	-70	80	1996-1997	General Minerals
V-17	366,265	6,413,352	2,104	200.38	-70	80	1996-1997	General Minerals
V-18	365,928	6,413,393	1,992	206.48	-70	90	1996-1997	General Minerals
V-19	366,407	6,412,992	2,085	224.05	-80	110	1996-1997	General Minerals
V-20	365,909	6,413,525	1,987	252.23	-60	280	1996-1997	General Minerals
V-21	366,527	6,412,600	2,150	196.42	-80	20	1996-1997	General Minerals
V-22	366,367	6,413,147	2,095	253.00	-80	110	1996-1997	General Minerals
V-23	365,571	6,413,085	2,121	325.74	-70	110	1996-1997	General Minerals
V-24	365,367	6,413,452	2,155	218.88	-60	110	1996-1997	General Minerals
V-25	365,527	6,412,909	2,147	312.02	-70	110	1996-1997	General Minerals
V-26	365,573	6,413,087	2,120	264.43	-50	235	1996-1997	General Minerals
V-27	365,524	6,412,911	2,147	230.88	-60	300	1996-1997	General Minerals
V-28	365,861	6,413,283	1,981	252.65	-70	290	1996-1997	General Minerals
V-29	365,632	6,413,173	2,048	251.60	-70	115	1996-1997	General Minerals
V-30	365,832	6,412,997	2,048	174.30	-80	290	1996-1997	General Minerals
V-31	366,233	6,412,357	2,015	226.90	-60	180	1996-1997	General Minerals
V-32	366,276	6,412,605	2,015	358.45	-70	290	1996-1997	General Minerals
V-33	366,420	6,412,675	2,086	415.05	-60	310	1996-1997	General Minerals
V-34	366,267	6,412,747	1,978	357.90	-70	310	1996-1997	General Minerals

Drill Hole Number	East	North	Elevation	Total Depth	Dip	Azimuth	Drilling Campaign	Company
V-35	366,179	6,412,835	1,964	248.10	-70	290	1996-1997	General Minerals
V-36	365,635	6,414,035	2,033	250.15	-60	290	1996-1997	General Minerals
V-37	365,470	6,414,175	2,111	246.40	-70	110	1996-1997	General Minerals
V-38	365,645	6,413,827	2,049	258.05	-70	290	1996-1997	General Minerals
V-39	365,743	6,413,947	2,015	440.15	-60	105	1996-1997	General Minerals
V-40	365,983	6,414,206	2,111	149.55	-60	280	1996-1997	General Minerals
V-41	365,983	6,414,200	2,111	142.15	-60	180	1996-1997	General Minerals
V-42	365,465	6,415,252	2,100	114.40	-60	300	1996-1997	General Minerals
V-43	365,981	6,414,409	2,159	195.25	-70	280	1996-1997	General Minerals
V-45	365,596	6,414,707	2,103	213.75	-60	270	1996-1997	General Minerals
V-46	366,097	6,413,337	2,055	173.80	-60	260	1996-1997	General Minerals
V-47	366,200	6,412,633	1,976	173.85	-60	290	1996-1997	General Minerals
V-48	365,903	6,413,088	2,017	190.00	-80	290	1996-1997	General Minerals
V-49	365,789	6,413,380	1,993	175.85	-70	90	1996-1997	General Minerals
V-50	366,277	6,412,927	2,000	151.25	-70	290	1996-1997	General Minerals
V-52	366,002	6,413,227	1,975	150.70	-75	280	1996-1997	General Minerals
V-53	366,064	6,412,888	1,969	164.10	-70	280	1996-1997	General Minerals
V-54	366,102	6,413,027	1,985	162.20	-80	280	1996-1997	General Minerals
V-55	365,782	6,413,302	1,991	136.85	-70	290	1996-1997	General Minerals
V-56	366,333	6,412,874	2,014	161.25	-80	90	1996-1997	General Minerals
V-57	366,223	6,412,550	1,991	144.35	-75	270	1996-1997	General Minerals
V-58	365,800	6,413,470	1,995	202.25	-80	90	1996-1997	General Minerals
V-59	366,243	6,413,138	2,022	167.70	-80	90	1996-1997	General Minerals
V-60	365,887	6,413,717	2,000	216.15	-60	100	1996-1997	General Minerals
V-61	366,111	6,413,232	1,999	114.95	-80	90	1996-1997	General Minerals
V-62	365,812	6,413,205	1,990	155.55	-70	290	1996-1997	General Minerals
V-63	365,877	6,413,807	2,005	152.50	-60	100	1996-1997	General Minerals
LAV-064	365,973	6,412,729	1,948	424.00	-69	114	2007-2008	Los Andes Copper
LAV-065	365,972	6,412,727	1,948	280.00	-76	295	2007-2008	Los Andes Copper
LAV-066	365,858	6,412,779	1,989	256.00	-73	290	2007-2008	Los Andes Copper
LAV-067	365,705	6,413,029	2,047	240.00	-78	124	2007-2008	Los Andes Copper
LAV-068	365,942	6,412,951	2,017	250.00	-70	294	2007-2008	Los Andes Copper
LAV-069	366,012	6,413,136	1,969	200.00	-70	296	2007-2008	Los Andes Copper
LAV-070	366,424	6,412,676	2,084	210.00	-74	117	2007-2008	Los Andes Copper
LAV-071	365,931	6,412,638	1,950	250.00	-68	292	2007-2008	Los Andes Copper
LAV-072	365,922	6,412,838	1,987	250.00	-69	294	2007-2008	Los Andes Copper
LAV-073	366,474	6,412,754	2,078	250.00	-69	286	2007-2008	Los Andes Copper
LAV-074	365,934	6,413,255	1,976	369.30	-68	295	2007-2008	Los Andes Copper
LAV-075	365,680	6,413,258	2,019	358.00	-65	292	2007-2008	Los Andes Copper
LAV-076B	365,847	6,412,458	1,935	250.00	-67	297	2007-2008	Los Andes Copper
LAV-077	366,330	6,412,706	2,020	350.00	-80	292	2007-2008	Los Andes Copper
LAV-078	366,405	6,412,896	2,060	300.00	-69	292	2007-2008	Los Andes Copper
LAV-079	366,288	6,412,824	1,988	250.00	-68	294	2007-2008	Los Andes Copper
LAV-080	366,136	6,412,554	1,975	250.00	-70	289	2007-2008	Los Andes Copper
LAV-081	366,102	6,412,770	1,947	386.00	-70	111	2007-2008	Los Andes Copper
LAV-082	365,769	6,412,709	1,993	250.00	-71	293	2007-2008	Los Andes Copper

Drill Hole Number	East	North	Elevation	Total Depth	Dip	Azimuth	Drilling Campaign	Company
LAV-083	365,698	6,413,359	2,005	250.00	-69	289	2007-2008	Los Andes Copper
LAV-084	365,586	6,412,984	2,095	250.00	-68	298	2007-2008	Los Andes Copper
LAV-085	365,577	6,413,086	2,118	292.00	-64	287	2007-2008	Los Andes Copper
LAV-086	366,005	6,412,818	1,956	280.00	-69	107	2007-2008	Los Andes Copper
LAV-087	365,763	6,412,911	2,062	250.00	-73	294	2007-2008	Los Andes Copper
LAV-088	366,004	6,412,820	1,956	250.00	-69	293	2007-2008	Los Andes Copper
LAV-089	365,842	6,412,880	2,036	254.05	-75	289	2007-2008	Los Andes Copper
LAV-090	365,677	6,412,947	2,066	412.00	-76	287	2007-2008	Los Andes Copper
LAV-091	365,644	6,413,061	2,079	362.00	-70	286	2007-2008	Los Andes Copper
LAV-092	365,932	6,412,640	1,950	250.00	-83	91	2007-2008	Los Andes Copper
LAV-093	365,931	6,413,175	1,975	478.00	-69	294	2007-2008	Los Andes Copper
LAV-094	365,624	6,413,179	2,050	500.00	-74	289	2007-2008	Los Andes Copper
LAV-095	365,507	6,413,014	2,139	250.00	-75	286	2007-2008	Los Andes Copper
LAV-096	365,662	6,412,732	2,045	296.00	-69	293	2007-2008	Los Andes Copper
LAV-097	365,765	6,412,810	2,040	270.00	-80	291	2007-2008	Los Andes Copper
LAV-098	365,960	6,413,051	2,021	255.00	-90	0	2007-2008	Los Andes Copper
LAV-099B	365,769	6,413,119	2,014	370.00	-90	0	2007-2008	Los Andes Copper
LAV-100	365,465	6,413,438	2,140	220.00	-70	115	2007-2008	Los Andes Copper
LAV-101	366,067	6,412,581	1,956	143.70	-71	301	2007-2008	Los Andes Copper
LAV-102	366,406	6,412,466	2,047	250.00	-70	294	2007-2008	Los Andes Copper
LAV-103	366,256	6,412,405	2,012	152.75	-68	290	2007-2008	Los Andes Copper
LAV-104	365,487	6,412,798	2,156	250.00	-69	291	2007-2008	Los Andes Copper
LAV-105	365,592	6,412,761	2,092	250.00	-69	288	2007-2008	Los Andes Copper
LAV-106	365,466	6,412,598	2,105	250.00	-70	297	2007-2008	Los Andes Copper
LAV-107A	365,842	6,412,678	1,956	330.00	-70	298	2007-2008	Los Andes Copper
LAV-108	365,784	6,412,582	1,957	260.00	-69	290	2007-2008	Los Andes Copper
LAV-109	366,329	6,412,699	2,020	256.00	-74	107	2007-2008	Los Andes Copper
LAV-110	365,469	6,413,119	2,121	156.00	-70	293	2007-2008	Los Andes Copper
LAV-111	365,401	6,413,152	2,144	250.00	-69	290	2007-2008	Los Andes Copper
LAV-112	366,402	6,412,789	2,040	350.00	-65	291	2007-2008	Los Andes Copper
LAV-113	365,747	6,412,481	1,952	200.00	-69	295	2007-2008	Los Andes Copper
LAV-114	365,532	6,413,202	2,085	250.00	-70	291	2007-2008	Los Andes Copper
LAV-115	365,666	6,414,139	1,988	250.00	-70	107	2007-2008	Los Andes Copper
LAV-116	365,509	6,414,390	2,026	250.00	-60	95	2007-2008	Los Andes Copper
LAV-117	366,473	6,412,753	2,076	270.00	-75	108	2007-2008	Los Andes Copper
LAV-118	366,555	6,412,859	2,156	350.00	-71	290	2007-2008	Los Andes Copper
LAV-119	365,587	6,414,257	2,024	198.00	-69	290	2007-2008	Los Andes Copper
LAV-120	366,165	6,412,654	1,959	450.00	-67	112	2007-2008	Los Andes Copper
LAV-121A	366,294	6,413,024	2,024	250.00	-79	111	2007-2008	Los Andes Copper
LAV-122	365,683	6,412,608	2,021	250.00	-69	295	2007-2008	Los Andes Copper
LAV-123	366,228	6,412,740	1,969	270.00	-81	106	2007-2008	Los Andes Copper
LAV-124	366,055	6,412,696	1,945	717.20	-67	110	2007-2008	Los Andes Copper
LAV-125	365,869	6,412,551	1,941	300.00	-70	288	2007-2008	Los Andes Copper
LAV-126	365,874	6,412,554	1,941	258.00	-70	105	2007-2008	Los Andes Copper
LAV-127	366,555	6,412,863	2,155	350.00	-81	107	2007-2008	Los Andes Copper
LAV-128	366,278	6,412,609	2,012	280.00	-74	112	2007-2008	Los Andes Copper

Drill Hole Number	East	North	Elevation	Total Depth	Dip	Azimuth	Drilling Campaign	Company
LAV-129	365,841	6,413,081	2,033	340.00	-59	292	2007-2008	Los Andes Copper
LAV-130	365,665	6,412,837	2,081	250.00	-90	0	2007-2008	Los Andes Copper
LAV-131	366,074	6,412,474	1,973	266.00	-69	293	2007-2008	Los Andes Copper
LAV-132A	366,190	6,412,961	1,964	150.00	-82	297	2007-2008	Los Andes Copper
LAV-133	365,848	6,412,462	1,935	250.00	-64	114	2007-2008	Los Andes Copper
LAV-134A	365,623	6,412,518	2,022	250.00	-70	285	2007-2008	Los Andes Copper
LAV-135A	366,060	6,412,917	1,977	249.00	-75	292	2007-2008	Los Andes Copper
LAV-136	366,136	6,412,340	2,020	200.00	-67	293	2007-2008	Los Andes Copper
LAV-137	366,036	6,412,386	1,959	250.00	-70	295	2007-2008	Los Andes Copper
LAV-138	366,046	6,413,016	1,994	352.00	-74	292	2007-2008	Los Andes Copper
LAV-139	366,001	6,412,612	1,941	351.00	-79	279	2007-2008	Los Andes Copper
LAV-140	366,160	6,412,447	1,997	401.60	-75	276	2007-2008	Los Andes Copper
LAV-141	365,767	6,412,817	2,033	435.00	-64	284	2007-2008	Los Andes Copper
LAV-142	365,988	6,412,277	1,947	217.00	-69	286	2007-2008	Los Andes Copper
V2015-01	365,791	6,413,735	2,015	476.35	-66	110	2015-2017	Los Andes Copper
V2015-02	365,785	6,413,377	1,993	459.80	-73	290	2015-2017	Los Andes Copper
V2015-03	365,933	6,413,379	1,991	535.00	-75	298	2015-2017	Los Andes Copper
V2015-04	365,682	6,413,878	2,040	656.00	-61	120	2015-2017	Los Andes Copper
V2015-05	366,185	6,413,278	2,035	638.00	-59	294	2015-2017	Los Andes Copper
V2015-06b	366,041	6,413,855	2,104	67.00	-74	80	2015-2017	Los Andes Copper
V2015-07	366,239	6,413,137	2,022	52.00	-70	290	2015-2017	Los Andes Copper
V2015-08	366,159	6,413,542	2,154	1,001.00	-75	290	2015-2017	Los Andes Copper
V2017-01A	365,784	6,413,536	2,003	851.25	-59	105	2015-2017	Los Andes Copper
V2017-02	366,285	6,413,246	2,076	1,030.60	-65	290	2015-2017	Los Andes Copper
V2017-03	365,936	6,413,856	2,049	0.00	-75	290	2015-2017	Los Andes Copper
V2017-04	366,199	6,413,048	1,974	653.00	-68	110	2015-2017	Los Andes Copper
V2017-05	365,986	6,413,877	2,077	931.90	-80	270	2015-2017	Los Andes Copper
V2017-06	366,037	6,413,483	2,067	857.00	-70	95	2015-2017	Los Andes Copper
V2017-07	366,094	6,413,346	2,048	721.10	-80	110	2015-2017	Los Andes Copper
V2017-08	365,991	6,413,871	2,077	400.25	-69	15	2015-2017	Los Andes Copper
V2017-09B	365,784	6,413,380	1,994	804.20	-69	120	2015-2017	Los Andes Copper
V2017-10	365,679	6,413,882	2,040	1,001.00	-73	65	2015-2017	Los Andes Copper
V2017-11	365,741	6,413,739	2,029	736.00	-70	85	2015-2017	Los Andes Copper
Notes								
Drill hole V2015-08 was drilled to a depth of 725.5 m in 2015. In 2017 the drill hole was extended to a depth of 1001.0 m								
Drill hole V2017-03 was abandoned at a depth of 72 m in overburden. It has been assigned a depth of 0 m as there is no useful information from this drill hole.								

31. APPENDIX III - LAND TENURE LETTER

OSSA | ALESSANDRI
ABOGADOS

Santiago, May 10, 2019

Mr.
Antony Amberg
Los Andes Copper Ltd.
Augusto Leguía Norte 100, Of. 812
Las Condes,
Santiago-Chile.

Ref.: Information regarding Compañía Minera Vizcachitas Holding and Sociedad Legal Minera San José Mining Property.

Dear Antony:

We hereby inform you that currently the mining property of Compañía Minera Vizcachitas Holding ("CMVH") is composed of 124 mining concessions; 49 of which are exploitation concessions and 75 exploration concessions. In addition, CMVH owns 2 exploitation concessions and 33 exploration concessions that are in the process of being granted.

Sociedad Legal Minera San José, a wholly owned subsidiary of Los Andes Copper Ltd., is also the owner of one exploitation concession.

Except for the aforementioned concessions that are still to be granted, all of the foregoing concessions were legally constituted as they were granted by the court of Putaendo and duly registered in the Mining Registry of such city. Additionally, as of this date these concessions do not have any outstanding taxes or permits that need to be paid for.

The concessions that are in the process of being constituted will be legally granted by the same court and all of them have preferential rights over other possible claims in the area. Once granted, they will be registered in the Mining Registry of Putaendo.

Roger de Flor 2736 Piso 5, Las Condes, Santiago
Teléfono (56-2) 333 3292 Fax (56-2) 333 9457
www.o-a.cl

OSSA | ALESSANDRI
ABOGADOS

Please find attached to this letter a list of all the concessions that form part of the mining property of CMVH and Sociedad Legal Minera San José.

If you have any queries in relation to this letter please feel free to contact us.

Regards,

Francisco Ossa S.C.

Roger de Flor 2736 Piso 5, Las Condes, Santiago
Teléfono (56-2) 333 3292 Fax (56-2) 333 9457
www.o-a.cl

Exploitation Claims as of May 10, 2019

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 1AL 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 60	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	In Process
52	ROJO 9 1 AL 30	CIA. MRA VIZCAHITAS HOLDING	056040717-3	300	In Process

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Exploration Claims as of May 10, 2019

N°	Exploration Claim Name	Owner	ROL NACIONAL	Hectares	Validity
1	PAOLA 1	CIA. MRA VIZCACHITAS HOLDING	056041943-0	200	2020/01/17
2	PAOLA 2	CIA. MRA VIZCACHITAS HOLDING	056041942-2	200	2020/01/17
3	CHAL 1	CIA. MRA VIZCACHITAS HOLDING	056041964-3	300	2020/08/17
4	CHAL 2	CIA. MRA VIZCACHITAS HOLDING	056041965-1	300	2020/08/17
5	CHAL 3	CIA. MRA VIZCACHITAS HOLDING	056041966-K	300	2020/08/17
6	CHAL 4	CIA. MRA VIZCACHITAS HOLDING	056041967-8	300	2020/08/17
7	CHAL 5	CIA. MRA VIZCACHITAS HOLDING	056041968-6	300	2020/08/17
8	CHAL 6	CIA. MRA VIZCACHITAS HOLDING	056041969-4	300	2020/08/17
9	CHAL 7	CIA. MRA VIZCACHITAS HOLDING	056041970-8	300	2020/08/17
10	CHAL 8	CIA. MRA VIZCACHITAS HOLDING	056041971-6	300	2020/08/17
11	CHAL 9	CIA. MRA VIZCACHITAS HOLDING	056041972-4	300	2020/08/17
12	CHAL 10	CIA. MRA VIZCACHITAS HOLDING	056041973-2	300	2020/08/17
13	CHAL 11	CIA. MRA VIZCACHITAS HOLDING	056041974-0	300	2020/08/17
14	CHAL 12	CIA. MRA VIZCACHITAS HOLDING	056041975-9	300	2020/08/17
15	CHAL 13	CIA. MRA VIZCACHITAS HOLDING	056041976-7	300	2020/08/17
16	CHAL 14	CIA. MRA VIZCACHITAS HOLDING	056041977-5	300	2020/08/17
17	CHAL 15	CIA. MRA VIZCACHITAS HOLDING	056041978-3	300	2020/08/17
18	CHAL 16	CIA. MRA VIZCACHITAS HOLDING	056041979-1	300	2020/08/17
19	CHAL 17	CIA. MRA VIZCACHITAS HOLDING	056041980-5	300	2020/08/17
20	CHAL 18	CIA. MRA VIZCACHITAS HOLDING	056041981-3	300	2020/08/17
21	TOTORA 1	CIA. MRA VIZCACHITAS HOLDING	056042008-0	300	2020/11/28
22	TOTORA 3	CIA. MRA VIZCACHITAS HOLDING	056042007-2	300	2020/11/28
23	TOTORA 4	CIA. MRA VIZCACHITAS HOLDING	056042006-4	300	2020/11/28
24	TOTORA 5	CIA. MRA VIZCACHITAS HOLDING	056042005-6	300	2020/11/28
25	TOTORA 6	CIA. MRA VIZCACHITAS HOLDING	056042004-8	300	2020/11/28
26	MAITEN 1	CIA. MRA VIZCACHITAS HOLDING	056042003-K	300	2020/11/28
27	MAITEN 2	CIA. MRA VIZCACHITAS HOLDING	056042002-1	300	2020/11/28
28	MAITEN 3	CIA. MRA VIZCACHITAS HOLDING	056042001-3	300	2020/11/28
29	MAITEN 4	CIA. MRA VIZCACHITAS HOLDING	056042000-5	300	2020/11/28
30	MAITEN 5	CIA. MRA VIZCACHITAS HOLDING	056041999-6	300	2020/11/28
31	MAITEN 6	CIA. MRA VIZCACHITAS HOLDING	056041998-8	300	2020/11/28
32	MAITEN 7	CIA. MRA VIZCACHITAS HOLDING	056041997-K	300	2020/11/28
33	MAITEN 8	CIA. MRA VIZCACHITAS HOLDING	056041996-1	300	2020/11/28
34	MAITEN 9	CIA. MRA VIZCACHITAS HOLDING	056041995-3	300	2020/11/28
35	MAITEN 10	CIA. MRA VIZCACHITAS HOLDING	056041994-5	300	2020/11/28
36	MAITEN 11	CIA. MRA VIZCACHITAS HOLDING	056041993-7	300	2020/11/28
37	MAITEN 12	CIA. MRA VIZCACHITAS HOLDING	056041992-9	300	2020/11/28
38	MAITEN 13	CIA. MRA VIZCACHITAS HOLDING	056041991-0	300	2020/11/28
39	MAITEN 14	CIA. MRA VIZCACHITAS HOLDING	056041990-2	300	2020/11/28
40	MAITEN 15	CIA. MRA VIZCACHITAS HOLDING	056041989-9	200	2020/11/28
41	MAITEN 16	CIA. MRA VIZCACHITAS HOLDING	056041988-0	200	2020/11/28
42	TOTORA 2	CIA. MRA VIZCACHITAS HOLDING	056041982-1	300	2020/12/20
43	ESPINO 1	CIA. MRA VIZCACHITAS HOLDING	056042009-9	200	2020/12/20
44	ESPINO 2	CIA. MRA VIZCACHITAS HOLDING	056042010-2	200	2020/12/20
45	MAITEN 17	CIA. MRA VIZCACHITAS HOLDING	056042011-0	300	2020/12/20
46	MAITEN 18	CIA. MRA VIZCACHITAS HOLDING	056042012-9	300	2020/12/20
47	VERDE 1	CIA. MRA VIZCACHITAS HOLDING	056042013-7	300	2020/12/20
48	VERDE 2	CIA. MRA VIZCACHITAS HOLDING	056042014-5	300	2020/12/20
49	VERDE 3	CIA. MRA VIZCACHITAS HOLDING	056042015-3	300	2020/12/20
50	VERDE 4	CIA. MRA VIZCACHITAS HOLDING	056042016-1	200	2020/12/20
51	VERDE 5	CIA. MRA VIZCACHITAS HOLDING	056042017-K	200	2020/12/20
52	VERDE 6	CIA. MRA VIZCACHITAS HOLDING	056042018-8	300	2020/12/20
53	VERDE 7	CIA. MRA VIZCACHITAS HOLDING	056042027-7	300	2020/12/20
54	VERDE 8	CIA. MRA VIZCACHITAS HOLDING	056042019-6	300	2020/12/20
55	VERDE 9	CIA. MRA VIZCACHITAS HOLDING	056042020-K	300	2020/12/20
56	VERDE 10	CIA. MRA VIZCACHITAS HOLDING	056042021-8	300	2020/12/20

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Exploration Claims as of May 10, 2019

N°	Exploration Claim Name	Owner	ROL NACIONAL	Hectares	Validity
57	VERDE 11	CIA. MRA VIZCACHITAS HOLDING	056042022-6	300	2020/12/20
58	VERDE 12	CIA. MRA VIZCACHITAS HOLDING	056042024-2	300	2020/12/20
59	VERDE 13	CIA. MRA VIZCACHITAS HOLDING	056042025-0	300	2020/12/20
60	VERDE 14	CIA. MRA VIZCACHITAS HOLDING	056042023-4	300	2020/12/20
61	VERDE 15	CIA. MRA VIZCACHITAS HOLDING	056042026-9	200	2020/12/20
62	PEUMO 1	CIA. MRA VIZCACHITAS HOLDING	056042086-2	200	2021/03/25
63	PEUMO 2	CIA. MRA VIZCACHITAS HOLDING	056042085-4	200	2021/03/25
64	PEUMO 3	CIA. MRA VIZCACHITAS HOLDING	056042084-6	300	2021/03/25
65	PEUMO 4	CIA. MRA VIZCACHITAS HOLDING	056042083-8	300	2021/03/25
66	PEUMO 5	CIA. MRA VIZCACHITAS HOLDING	056042082-K	300	2021/03/25
67	PEUMO 6	CIA. MRA VIZCACHITAS HOLDING	056042081-1	300	2021/03/25
68	PEUMO 7	CIA. MRA VIZCACHITAS HOLDING	056042080-3	300	2021/03/25
69	PEUMO 8	CIA. MRA VIZCACHITAS HOLDING	056042079-K	300	2021/03/25
70	PEUMO 9	CIA. MRA VIZCACHITAS HOLDING	056042078-1	300	2021/03/25
71	PEUMO 10	CIA. MRA VIZCACHITAS HOLDING	056042077-3	300	2021/03/25
72	PEUMO 11	CIA. MRA VIZCACHITAS HOLDING	056042076-5	300	2021/03/25
73	PEUMO 12	CIA. MRA VIZCACHITAS HOLDING	056042075-7	300	2021/03/25
74	PEUMO 13	CIA. MRA VIZCACHITAS HOLDING	056042074-9	300	2021/03/25
75	PALMA 2	CIA. MRA VIZCACHITAS HOLDING	056042087-0	300	2021/03/25
76	LOICA 1	CIA. MRA VIZCACHITAS HOLDING	056042110-9	300	In Process
77	LOICA 2	CIA. MRA VIZCACHITAS HOLDING	056042112-5	300	In Process
78	LOICA 3	CIA. MRA VIZCACHITAS HOLDING	056042113-3	300	In Process
79	LOICA 4	CIA. MRA VIZCACHITAS HOLDING	056042114-1	300	In Process
80	LOICA 5	CIA. MRA VIZCACHITAS HOLDING	056042115-K	300	In Process
81	LOICA 6	CIA. MRA VIZCACHITAS HOLDING	056042116-8	300	In Process
82	LOICA 7	CIA. MRA VIZCACHITAS HOLDING	056042117-6	300	In Process
83	LOICA 8	CIA. MRA VIZCACHITAS HOLDING	056042118-4	300	In Process
84	PAICO 1	CIA. MRA VIZCACHITAS HOLDING	S/R	300	In Process
85	PAICO 2	CIA. MRA VIZCACHITAS HOLDING	S/R	300	In Process
86	PAICO 3	CIA. MRA VIZCACHITAS HOLDING	S/R	300	In Process
87	PAICO 4	CIA. MRA VIZCACHITAS HOLDING	S/R	300	In Process
88	PAICO 5	CIA. MRA VIZCACHITAS HOLDING	S/R	300	In Process
89	PAICO 6	CIA. MRA VIZCACHITAS HOLDING	S/R	300	In Process
90	PAICO 7	CIA. MRA VIZCACHITAS HOLDING	S/R	300	In Process
91	PAICO 8	CIA. MRA VIZCACHITAS HOLDING	S/R	300	In Process
92	PAICO 9	CIA. MRA VIZCACHITAS HOLDING	S/R	300	In Process
93	PAICO 10	CIA. MRA VIZCACHITAS HOLDING	S/R	300	In Process
94	PAICO 11	CIA. MRA VIZCACHITAS HOLDING	S/R	300	In Process
95	PAICO 12	CIA. MRA VIZCACHITAS HOLDING	S/R	100	In Process
96	PAICO 13	CIA. MRA VIZCACHITAS HOLDING	S/R	300	In Process
97	PAICO 14	CIA. MRA VIZCACHITAS HOLDING	S/R	200	In Process
98	PAICO 15	CIA. MRA VIZCACHITAS HOLDING	S/R	300	In Process
99	PAICO 16	CIA. MRA VIZCACHITAS HOLDING	S/R	300	In Process
100	PAICO 17	CIA. MRA VIZCACHITAS HOLDING	S/R	300	In Process
101	PAICO 18	CIA. MRA VIZCACHITAS HOLDING	S/R	300	In Process
102	PAICO 19	CIA. MRA VIZCACHITAS HOLDING	S/R	300	In Process
103	PAICO 20	CIA. MRA VIZCACHITAS HOLDING	S/R	300	In Process
104	PAICO 21	CIA. MRA VIZCACHITAS HOLDING	S/R	300	In Process
105	PAICO 22	CIA. MRA VIZCACHITAS HOLDING	S/R	200	In Process
106	PAICO 23	CIA. MRA VIZCACHITAS HOLDING	S/R	200	In Process
107	PAICO 24	CIA. MRA VIZCACHITAS HOLDING	S/R	300	In Process
108	PAICO 25	CIA. MRA VIZCACHITAS HOLDING	S/R	300	In Process

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